THE INFLUENCE OF BLASTING ON KEMESS HYPOGENE ORE MILLING

by

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Abstract

Traditionally, optimization of drill and blast practices have taken place with sole respect to minimizing mine unit cost (as opposed to the operation as a whole), whilst maintaining fragmentation at a level that is considered “acceptable” to mine operations. Judgments as to “acceptable” are largely based on qualitative measures and the consequences of varying blast practices on subsequent downstream processes are secondary. The objective of this work was to evaluate both the technical and economic consequences of blast practices on comminution processes at the Kemess South Project. Computer modeling indicates that without changes to crusher operating parameters, there does not appear to be much room for improving the performance of the comminution circuit. This indicates that current blasting at the mine is at or near optimal for the mill’s current configuration.

Experiments to attempt to detect the effect of blasting on the crushability and grindability of rock were carried out on several different samples, including a copper-gold porphyry ore from Kemess Mine, taconite from Minnesota, granodiorite from the Kingston area, and limestone from the Kingston area. The main focus of this study was the Kemess ore. Examining rock samples from blasted and un-blasted ore, the samples that were blasted showed a significant increase in the density of very small cracks within individual fragments, which would indicate that they should break more easily during crushing and grinding. Drop-weight impact tests investigating the grindability of these samples suggested that there was improved performance with blasting effort, although the difference between some samples was not statistically significant. Preliminary testing of the taconite and the limestone indicated that crushability and grindability may be improved by blasting.
Using mineral processing simulation software, the second part of the study investigated the effects of changing the ore size distribution that is fed to the Kemess Primary crusher. It was concluded that the Kemess Semi-Autogenous Grinding (SAG) mill throughput could be increased by increasing fragmentation and making adjustments to the primary crusher closed side set to take advantage of the increased fragmentation. These changes were calculated to be economically advantageous.
Acknowledgements

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# Table of Contents

**ABSTRACT** ........................................................................................................................................ II

**ACKNOWLEDGEMENTS** ................................................................................................................ IV

## CHAPTER 1: GOALS AND LITERATURE REVIEW ........................................................................... 1

1.1 Thesis Overview and Goals ............................................................................................................. 1
1.2 Introduction to Literature Review .................................................................................................. 1
1.3 Blasting and Fragmentation Effects on Mining and Comminution Operations ..................... 7
1.4 Micro-Fractures and Work Index .................................................................................................. 14
1.5 Economic Rationale and Modelling .............................................................................................. 17
1.6 Kemess Mine .................................................................................................................................. 23
1.6.1 Geologic and Setting .................................................................................................................. 23
1.6.2 Mining Method and Equipment ................................................................................................. 25
1.6.3 Milling Circuit ............................................................................................................................ 25

## CHAPTER 2: SMALL-SCALE BLASTING EXPERIMENTS .......................................................... 27

2.1 Introduction ..................................................................................................................................... 27
2.2 Experimental Set-up ....................................................................................................................... 27
2.3 2-Dimensional Blast Numerical Modelling ............................................................................... 30
2.4 Micro-fractures Created by Blasting ............................................................................................. 32
2.4.1 Micro-fracture Detection and Measurement ........................................................................... 33
2.4.2 Results and Discussion .............................................................................................................. 36

## CHAPTER 3: MILLING PROPERTY CHANGES INDUCED BY BLASTING ........................ 38

3.1 Introduction ..................................................................................................................................... 38
3.2 MinnovEX Test Procedures .......................................................................................................... 39
  3.2.1 MinnovEX Feed Crushing Parameter ..................................................................................... 40
  3.2.2 MinnovEX Semi-Autogenous Grinding Power Index (SPI) .................................................... 43
3.3 Drop Weight Testing ....................................................................................................................... 45
  3.3.1 Principles and Calibration of the Queen’s Drop-Weight Tester ............................................. 45
  3.3.2 Calculations for Drop-Weight Sample Testing ...................................................................... 49
  3.3.3 Drop-weight Testing Methods ................................................................................................. 50
  3.3.4 Analysis of Drop-weight Fragments ...................................................................................... 53
3.4 Ball Mill Testing ............................................................................................................................. 60
  3.4.1 Locked Cycle Testing .............................................................................................................. 60
CHAPTER 4: KEMESS BLASTING AND COMMINUTION COMPUTER MODELLING ................................................. 69
  4.1 Introduction .......................................................................................................................................... 69
  4.2 Base Case Fragmentation Study ........................................................................................................ 73
    4.2.1 Blasting Model ............................................................................................................................ 73
    4.2.2 Crusher Model Fitting ................................................................................................................ 78
    4.2.3 SAG Circuit Model Fitting .......................................................................................................... 81
    4.2.4 Ball Mill Circuit Model Fitting .................................................................................................... 85
  4.3 Base Case Combined Circuit Simulation Fit ...................................................................................... 88

CHAPTER 5: EFFECTS OF CHANGING BLASTING PARAMETERS ON CIRCUIT OPERATIONS .......... 91
  5.1 Introduction .......................................................................................................................................... 91
  5.2 Constraints for Modelling .................................................................................................................. 91
  5.3 Factorial experiments: Blasting Fragmentation and Crusher Closed Side Set Changes .................. 92
  5.4 Economic Analysis ............................................................................................................................. 107

CHAPTER 6: PRELIMINARY INVESTIGATION INTO THE EFFECT OF BLASTING ON THE
  COMMINUTION PROPERTIES OF OTHER MATERIALS ........................................................................ 112
  6.1 Taconite Drop Weight Tests ............................................................................................................ 113
  6.2 Granodiorite Drop-Weight Tests .................................................................................................... 116
  6.3 Limestone Drop-Weight tests .......................................................................................................... 119

CHAPTER 7: SUMMARY OF RESULTS, DISCUSSION AND CONCLUSION .................................. 122
  7.1 Experimental Results and Conclusions .......................................................................................... 122
  7.2 Discussion .......................................................................................................................................... 123
  7.3 Future Work at Kemess ................................................................................................................... 125

REFERENCES .............................................................................................................................................. 126

APPENDIX A: BOND WORK INDEX PROCEDURE ............................................................................. 132
APPENDIX B: KEMESS ORE DROP-WEIGHT RESULTS ..................................................................... 134
APPENDIX C: BALL MILL LOCKED CYCLE TESTING ........................................................................ 144
APPENDIX D: BALL MILL BATCH CYCLE TESTING .......................................................................... 152
APPENDIX E: TAConITE DROP WEIGHT TEST RESULTS ................................................................. 156
APPENDIX F: GRANODIORITE DROP-WEIGHT TEST DATA ................................................................. 161
APPENDIX G: LIMESTONE DROP-WEIGHT TEST DATA ................................................................. 167
APPENDIX H: ACRONYMS .................................................................................................................... 179
List of Tables

TABLE 1-1: MINING AND CRUSHING COSTS (KANCHIBOTLA, 2000) .................................................. 11
TABLE 1-2: CRACK TYPE AND FREQUENCY IN BLASTED AND UNBLASTED ROCK SAMPLES (NIELSEN, 1999) ........................................................................................................ 16
TABLE 1-3: WORK INDICES (W_i) IN KWH/T FOR VARIOUS ROCKS AT DIFFERENT BLAST ENERGIES (NIELSEN AND KRISTIANSEN, 1996). ................................................................. 16
TABLE 1-4: MINE MATERIAL TYPES .................................................................................................... 24
TABLE 1-5: KEMESS HYPOGENE ORE ALTERATION DOMAINS LISTED IN DESCENDING ORDER OF HARDNESS (KEMESS, 2003) .......................................................... 24
TABLE 2-1: PRE-BLAST SAMPLE INFORMATION .................................................................................. 29
TABLE 2-2: BASIC PARAMETERS FOR BLASTING FAILURE MODEL ................................................................ 32
TABLE 2-3: MICROFRACKURE DENSITY INDUCED IN KEMESS D1 HYPOGENE ORE .................. 36
TABLE 3-1: SUMMARY OF MILLING PROPERTY TESTS ....................................................................... 38
TABLE 3-2: FEED SIZE CLASSES FOR t_{10} TESTING (NAPIER-MUNN ET AL., 1996) ............... 51
TABLE 3-3: PLANNED IMPACT ENERGIES FOR EACH SIZE-ENERGY COMBINATION (KWH/T) .................................................................................................................................................... 51
TABLE 3-4: KEMESS HO BLASTED SPECIFIC IMPACT ENERGIES (KWH/T) FOR D1, D2 AND D5 ........................................................................................................................................... 52
TABLE 3-5: KEMESS HO UNBLASTED SPECIFIC IMPACT ENERGIES (KWH/T) FOR D1, D2, AND D5 ........................................................................................................................................... 52
TABLE 3-6: HO D1 BLASTED AND UNBLASTED PARAMETER FITTING RESULTS ......................... 58
TABLE 3-7: HO D2 BLASTED AND UNBLASTED PARAMETER FITTING RESULTS ......................... 58
TABLE 3-8: HO D5 BLASTED AND UNBLASTED PARAMETER FITTING RESULTS ......................... 58
TABLE 3-9: BALL MILL WORK INDEX FEED AND PRODUCT SIZES .................................................. 61
TABLE 3-10: REGRESSION RESULTS FOR FIRST ORDER BALL MILL BATCH TESTING ......... 68
TABLE 3-11: GRINDING RATE CONSTANTS FOR -2.36 +1.7 MM FEED (MINUTE^{-1}) ............... 68
TABLE 4-1: BLASTING MODEL BASE CASE PARAMETERS FOR CRUSHER FEED PREDICTION ............................................................................................................................................... 77
TABLE 4-2: CRUSHER MODEL FITTED AND ACTUAL OPERATING DATA. FITTED DATA IS DENOTED BY THE COLUMN HEADING FIT AND ACTUAL DATA BY EXP. TPH REFERS TO TONNES PER HOUR. .................................................................................................................. 80
TABLE 4-3: B LINE SAG MILL DIMENSIONS AND OPERATING DATA ............................................. 81
TABLE 4-4: SAG CIRCUIT MODEL FITTING DATA ................................................................................ 82
TABLE 4-5: SAG CIRCUIT FITTED MODEL PARAMETERS .................................................................. 84
TABLE 4-6: B LINE BALL MILL DIMENSIONS AND OPERATING DATA ........................................... 85
TABLE 4-7: CYCLONE DIMENSIONS AND OPERATING DATA ......................................................... 86
TABLE 4-8: BALL MILL CIRCUIT MODEL FITTING RESULTS .............................................................. 86
List of Figures and Illustrations

FIGURE 1-1: TRADITIONAL VIEW OF PARAMETERS CONSIDERED FOR BLAST OPTIMIZATION (KANCHIBOTLA, 2003).............................................................. 3
FIGURE 1-2: RANGE OF PARTICLE SIZE OPERATION FOR VARIOUS COMMINUTION PROCESSES AND EQUIPMENT (ICE, 1974)................................................................. 6
FIGURE 1-3: THE EFFECT OF FRAGMENTATION ON UNIT COST PER METRIC TONNE (MICHAUD AND BLANCHET, 1996).................................................................................. 8
FIGURE 1-4: ENVIRONMENTAL CONSIDERATIONS FOR INCREASED BLASTING EFFORT (DA GAMA AND JIMENO, 1993).................................................................................. 10
FIGURE 1-5: CUMULATIVE COSTS IN MINING AND PROCESSING RELATED TO EXPLOSIVES POWDER FACTOR (PF) (KANCHIBOTLA, 2000).......................... 12
FIGURE 1-6: RELATIONSHIP BETWEEN SEMI-AUTOGENOUS GRINDING (SAG) MILL FEED 20% PASSING SIZE (F20) AND SAG MILL THROUGHPUT/FEED, IN TONNES PER HOUR (TPH) (KANCHIBOTLA, 2000)............. 13
FIGURE 1-7: MICRO-FRACTURES OF PRE AND POST BLAST ROCK CORES (OLSON ET AL., 1973)................................................................................................. 15
FIGURE 1-8: OVERALL COST RELATIONSHIP FOR TACONITE MINING (ELORANTA, 1999) 17
FIGURE 1-9: ENVISIONED COST RELATIONSHIPS AS A FUNCTION OF BLASTING EFFORT (NIELSEN AND KRISTIANSEN, 1996)........................................................................... 18
FIGURE 1-10: RELATIVE MINING AND COMMINUTION COSTS FOR TACONITE MINING (PASTIKA ET AL., 1995)................................................................................................. 19
FIGURE 1-11: NET PROFIT INCREASE REALIZED BY INCREASED THROUGHPUT VS. POWDER FACTOR, EXPRESSED IN KG/T OF 70/30 HEAVY ANFO (PALEY AND KOJOVIC, 2001) ............................................................... 20
FIGURE 1-12: RELATIONSHIP BETWEEN THROUGHPUT TONNES PER HOUR (TPH) AND FLOTATION RECOVERY (NAPIER-MUNN ET AL., 1996).............................. 22
FIGURE 1-13: KEMESS MILLING CIRCUIT (GRAY ET AL., 2003)......................................................... 26
FIGURE 2-1: SCALE BLASTING CONFIGURATION FOR IMPACTING ROCK SAMPLES WITH STRESS WAVE TO SIMULATE BLASTING ON ROCK SAMPLE (NOT TO SCALE)........................................................................ 28
FIGURE 2-2: BLAST FRAGMENT SCREENING RESULTS................................................................. 29
FIGURE 2-3: MODELLING RESULTS SHOWING STRESS WAVE PASSING FROM WATER INTO THE ROCK ......................................................................................... 31
FIGURE 2-4: LOADING FAILURE FROM SMALL SCALE BLASTING............................................. 32
FIGURE 2-5: BLASTED THIN-SECTION COMPOSITE PICTURE ...................................................... 34
FIGURE 2-6: UNBLASTED KEMESS D1 COMPOSITE IMAGE......................................................... 35
FIGURE 3-1: RELATIONSHIP BETWEEN MINNOVEX $C_F$ PARAMETER AND CRUSHER F80 PRODUCT (DOBBY ET AL., 2001)................................................................. 41
FIGURE 3-2: MINNOVEX FEED CRUSHING PARAMETER RESULTS ($C_F$) FOR 6 KEMESS HO ORE SAMPLES FROM ALTERATION DOMAINS D1, D2 AND D5. ............. 42
FIGURE 3-3: COMPARISON OF SPI VALUES FOR KEMESS HO DOMAINS D1, D2, AND D5
FIGURE 3-4: QUEEN'S DROP WEIGHT TESTER (QDWT) SAMPLE STAGE
FIGURE 3-5: CONFIGURATION OF THE QUEEN'S DROP-WEIGHT TESTER
FIGURE 3-6: PREDICTED GATE TIME USING MODIFIED G CONSTANT FOR QDWT AND ACTUAL GATE TIME
FIGURE 3-7: CORRELATION BETWEEN BOND WORK INDEX AND ABRASION PARAMETER $T_a$ WITH THE VALUE OF $A^*B$ (NAPIER-MUNN ET AL., 1996)
FIGURE 3-8: $t_{10}$ PLOT FOR BLASTED AND UNBLASTED KEMESS HYPOGENE D1 ORE
FIGURE 3-9: $t_{10}$ PLOT FOR BLASTED AND UNBLASTED KEMESS D2 HYPOGENE ORE
FIGURE 3-10: $t_{10}$ PLOT FOR BLASTED AND UNBLASTED KEMESS D5 HYPOGENE ORE
FIGURE 3-11: PLOT SCATTER FROM $t_{10}$ TESTING (GRUNDSROM ET AL., 2001)
FIGURE 3-12: LOCKED CYCLE BOND WORK INDEX RESULTS, IN KWH PER TONNE
FIGURE 3-13: D1 BALL MILL PRODUCTS AFTER 30, 60 AND 90 REVOLUTIONS (U=UNBLASTED, B=BLASTED)
FIGURE 3-14: D2 BALL MILL PRODUCTS AFTER 30, 60, AND 90 REVOLUTIONS (U=UNBLASTED, B=BLASTED)
FIGURE 3-15: D5 BALL MILL PRODUCTS AFTER 30, 60, 90 REVOLUTIONS (U=UNBLASTED, B=BLASTED)
FIGURE 3-16: RATE PLOT FOR D1 BLASTED AND UNBLASTED FEED SAMPLES
FIGURE 3-17: RATE PLOT FOR D2 BLASTED AND UNBLASTED FEED SAMPLES
FIGURE 3-18: RATE PLOT FOR D5 BLASTED AND UNBLASTED FEED SAMPLES
FIGURE 4-1: JKSIMMET SIMULATOR CIRCUIT CONFIGURATION OVERVIEW
FIGURE 4-2: THROUGHPUT-RECOVERY RELATIONSHIP FOR HO ORE (AXES INTENTIONALLY LEFT BLANK)
FIGURE 4-3: BASE CASE FRAGMENTATION AT PF=0.584 KG/M$^3$
FIGURE 4-4: CRUSHER MODEL FIT TO EXPERIMENTAL DATA
FIGURE 4-5: SAG CIRCUIT MODEL FIT TO MASS-BALANCED DATA
FIGURE 4-6: SAG MILL BREAKAGE RATE CURVE
FIGURE 4-7: BALL CIRCUIT MODEL FIT TO MASS-BALANCED DATA
FIGURE 4-8: BALL MILL BREAKAGE R/D* CURVE
FIGURE 4-9: UNIFIED B LINE SIMULATION PREDICTIONS FOR THE BASE CASE
FIGURE 5-1: DIFFERENT FEED SIZE DISTRIBUTIONS FOR POWDER FACTORS RANGING FROM 0.4-1.0 KG/M$^3$
FIGURE 5-2: COMBINED CIRCUIT SIMULATION FOR 770 TPH @ PF=0.75 KG/M$^3$, CSS=110 MM
FIGURE 5-3: COMBINED CIRCUIT SIMULATION FOR 790 TPH @ PF=1.0 KG/M$^3$, CSS=110 MM

x
Chapter 1: Goals and Literature Review

1.1 Thesis Overview and Goals

The fragmentation of rock, from in-situ blasting to the final product that is passed on to separation circuits is a complex process involving many parties and different technologies. The interaction between these technologies and parties gives rise to a set of scientific relationships. The goal of this thesis is to try and understand what these relationships are for a particular operation, the Kemess Mine. Specifically:

- Does blasting create microfractures in hypogene ore? Laboratory testing using typical samples of hypogene ore will be conducted.
- Understand the relationship between blasting and how it affects the grinding and crushing properties of the ore, using a variety of laboratory tests.
- Create a computer model that includes blasting and comminution operations and try to quantify how blast fragmentation affects downstream comminution operations. Theoretically, it should be possible to increase grinding circuit throughput by making changes to blasting and comminution equipment operational parameters.
- Conduct a preliminary investigation into the effects of blasting on other rock types to see if their crushing and grinding properties can be modified by blasting.

1.2 Introduction to Literature Review

It is envisioned that that the whole process of mining and milling can be optimized as a single system. Typically, the mine and the mill for any given site are operated as separate economic entities (i.e., separate budgets), each focusing on minimizing their individual costs under the management assumption that this will minimize total cost and
maximize profitability. Certain levels of technical cooperation between the units are required for the management of the operation as a whole and are usually limited to grade, tonnages and metallurgical requirements for ore being mined and processed. The required level of fragmentation can be thought of as the coarsest acceptable fragmentation to the mine and mill. The mine’s chief concern with fragmentation is the production of large fragments, which can lead to operational problems, such as difficult digging and loading. From the mill’s standpoint, the required minimum fragmentation is which the crusher throughput can reliably keep the mill supplied with ore. If the mine and mill are operationally satisfied with the level of fragmentation produced by blasting, and there is no apparent opportunity to further reduce blasting costs, fragmentation is regarded as optimum (Kanchibotla, 2003). Figure 1-1 graphically illustrates what the parameters of blast optimization are traditionally thought to be, with a specific focus on mining and no consideration of milling processes. The factors considered in Figure 1-1 that contribute to the unit cost of mining ($/t) refer only to costs that are paid for by the mine. The total mining cost is given as the sum of the unit blasting, digging and hauling cost. Optimum blasting occurs when mining costs have been minimized, but this is not necessarily valid since it assumes that blasting only impacts mining costs.
There are reasons for the reduced scope (i.e., only in the mine) of blasting evaluation. Mining is highly technical business, and operationally there are significant divisions within areas of technical expertise. The mining industry at large has not implemented significant efforts for the quality control of raw materials (e.g. ore feeds) beyond minimum satisfactory levels, whereas many other industries have highly developed programs for the procurement and quality control of raw materials (Pease et al., 1998), e.g. Run-of-Mine (ROM) ore.

Research suggests the definition of optimum blasting effort, shown in Figure 1-1 may not be an optimum at all, when viewed from the perspective of the whole operation.

Figure 1-1: Traditional view of parameters considered for blast optimization (Kanchibotla, 2003)
(Eloranta, 1999). Blasting directly affects the size distribution of the ore that is sent to the mill, and also may affect other parameters such as the apparent ‘hardness’ of the ore undergoing comminution. Small fractures in ore particles that are being fed to the mill are hypothesized to be the cause of this apparent reduction in the ‘hardness’ of ore undergoing comminution. Therefore, it seems reasonable that milling operations and not just mining operations need to be considered when evaluating how ‘optimized’ blasting is.

For a typical comminution circuit, quality and quantity of the (primary) crusher product affect the operation and products of the rod mill, and so on. Therefore, it would seem reasonable that the fragmentation by blasting would affect the crusher product, and thus all processes downstream. Simkus and Dance (1998) observed that the Semi-Autogenous Grinding (SAG) mill throughput was almost entirely dependant on the feed size distribution. Also, blasting may have a more fundamental role to play in downstream mechanical comminution from the crusher to the final regrind. While fragmentation produced by blasting is readily observed, there are effects on the rock being blasted other than fragmentation. Blasting also affects the mechanical properties of the rock fragments themselves, causing them to become weaker than in-situ. Logically, this would affect downstream crushing and grinding activities. Current comminution technology breaks rocks by the application of mechanical energy. This applied mechanical energy takes advantage of existing flaws in the material, extending existing cracks and creating new ones in areas of weakness until the material fails and the particle breaks. Therefore, if blasting creates cracks in ore particles, then blasted particles should fail more readily in response to applied mechanical force than equivalent unblasted particles. Extending this idea, the prevalence of cracks should increase with increasing levels of applied blasting energy. In practical terms, the effect of blasting rock
should show as a reduction in the required specific energy for particle size reduction over a fixed particle size range for a mechanical crushing/grinding device. Increasing applied blasting energy should show a proportional reduction in the required specific energy for particle size reduction over a fixed particle size range for a mechanical crushing/grinding device.

The types of flaws and fractures created by blasting may also favour mineral liberation and consequently affect separation processes such as flotation. If blasting preferentially creates fractures that are inter-granular, it is reasonable to think that this would favour liberation; conversely, it could also hamper mineral liberation if intra-granular fractures are produced. Ultimately, the nature of the flaws could significantly affect the desirability of fragmenting and fracturing of ore using blasting. This topic, however, is beyond the scope of this study.

Figure 1-2 shows the particle size ranges that different processes of the comminution chain operate over, with some size overlaps occurring at every stage of size reduction. Virtually all studies into the effects of blasting focus on the ‘coarse’ size range in Figure 1-2. Crushers, Autogenous Grinding (AG) mills, Semi-Autogenous Grinding (SAG) mills and various tumbling mills are also amongst the most widely employed types of comminution equipment, and function mainly in the coarse range. Thus, most operations would be most interested in how blasting would affect the operation of these pieces of equipment.
Studies summarized in Eloranta (1999) indicate that changes in blasting practice can impact downstream comminution processes through two mechanisms: (1) rock fragmentation and (2) changes to comminution properties due to micro-fractures induced in the rock caused by blasting (Kemeny et al., 2003). There are also mining operational benefits to be recognized as well from improved blasting fragmentation and throw control for mining operations such as higher truck fill factors and improved loader/excavator productivities (Michaud and Blanchet, 1996). Summarized by Grundstrom et al. (2001), the areas of operations that can benefit by adjusting drill and blast practices are as follows:

1. Improvements in loader/excavator productivity through muck pile diggability and bucket fill factors.
2. Increase in crusher throughput (tonnes processed per hour) due to changes in ROM material size distributions.

3. Reduction in energy consumption for downstream processes including crushing and grinding.

4. Improvements in mill throughput and the reduction in unit process energy consumption.

5. Reduction in blast induced damage and ore dilution resulting in increased final product.

6. Potential for increased liberation of valuables leading to enhanced mill recoveries.

While all these topics are very relevant to optimizing the mine-to-mill interface, items 2, 3, and 4 are the focus of this study.

1.3 **Blasting and Fragmentation Effects on Mining and Comminution Operations**

Operationally, there are advantages to increased fragmentation in the pit resulting in increased efficiencies. The study performed by Mackenzie (1966) indicated there was a correlation between shovel productivity and blasting fragmentation. While this relationship exists intuitively, the study looked at it from scientific viewpoint and appears to be the first study of its kind. It spawned much of the interest in the current research field generally referred to as mine-to-mill, drill-to-mill or variations thereof that focus on relating drill and blast activities to other mining processes. The results of a study by Michaud and Blanchet (1996) are summarized in Figure 1-3, correlating decreasing unit costs (stated in $/tonne) with increasing fragmentation. The Fragmentation Index in
Figure 1-3 refers to the ICI Fragmentation Index, where a higher index number refers to a decreasing degree of fragmentation. Total cost reductions were realized in the form of increased fill factors for both trucks and loaders with increased fragmentation alone. Grinding operations do not need to be considered to show that increasing blasting fragmentation can lower the overall unit cost. The conclusion is similar to Eloranta (1999), whereby mining unit cost savings can be realized with increasing fragmentation without accounting for any cost savings in grinding.

Figure 1-3: The effect of fragmentation on unit cost per metric tonne (Michaud and Blanchet, 1996)
There seems to be little debate about the validity of these claims, but others have not drawn the same results. Possibly this is due to differences between mining operations. The cost conclusions drawn by da Gama and Jimeno (1993) suggest there is no benefit to increasing fragmentation beyond what is necessary. The study by da Gama and Jimeno (1993) takes a different approach to optimization by trying to calculate what is the minimum possible economic powder factor. They also acknowledge that there are other factors that need to be considered when addressing the issue of blasting outside of technical and economic parameters. Environmental and safety considerations must be made when considering blast designs and practice, and conceptually these ideas are summarized in Figure 1-4 and they may be of more consequence than purely economic factors. Figure 1-4 indicates what the relationship between the degree of fragmentation (increasing to the right) and the environmental impacts of primary and secondary blasting. At a certain level of fragmentation, the combined environmental effects of primary and secondary blasting are at a minimum. The degree of blasting fragmentation is governed not by what is economically most desirable, but which offers the lowest overall environmental impact.
Environmental considerations are important for operations near residential areas, but little consideration is given to them in all (other) studies reviewed considering increasing blasting effort. For remote mines, complaints from the neighbours may not be a concern, but these mines have equipment and infrastructure is sensitive to blasting effects such as fly-rock. Thus, the environmental effects of blasting may be a factor that they must consider. Ultimately, for a given operation, environmental considerations may ultimately override economic optimization.

Kanchibotla (2000) suggests that mining and crushing costs increase with increasing powder factor (the amount of explosives used per unit of rock blasted, abbreviated PF) and there are no unit cost savings found when examining costs from drill and blast to the crusher. Three different hypothetical blasting configurations were examined by
Kanchibotla (2000), using 3 different powder factors and thus 3 different levels of fragmentation. The predicted unit operation costs under each scenario are summarized in Table 1-1. The higher costs incurred for drilling and blasting at a higher powder factor than scenario (1) (nominal) are not offset by cost reductions in other parts operations, up to and including the crusher. Kanchibotla (2000) did not realize any unit cost savings until grinding and separation process costs are accounted for; these costs are summarized in Figure 1-5. This is in disagreement with Michaud and Blanchet (1996), Eloranta (1999) and others, who found that optimizing drill and blast could produce an overall lower unit mining cost. How changes in blasting affect the unit costs of different mining/milling processes likely varies significantly between mine sites and ore types, and is difficult to make generalizations about.

**Table 1-1: Mining and crushing costs (Kanchibotla, 2000)**

<table>
<thead>
<tr>
<th>Scenario</th>
<th>1</th>
<th>2</th>
<th>3</th>
</tr>
</thead>
<tbody>
<tr>
<td>Powder Factor (kg/m³)</td>
<td>0.58</td>
<td>0.66</td>
<td>0.96</td>
</tr>
<tr>
<td>Drill and blast ($/t)</td>
<td>$0.28</td>
<td>$0.32</td>
<td>$0.46</td>
</tr>
<tr>
<td>Excavate and haul ($/t)</td>
<td>$1.00</td>
<td>$1.00</td>
<td>$0.95</td>
</tr>
<tr>
<td>Grade control ($/t)</td>
<td>$0.20</td>
<td>$0.20</td>
<td>$0.40</td>
</tr>
<tr>
<td>Crushing ($/t)</td>
<td>$1.75</td>
<td>$1.75</td>
<td>$1.66</td>
</tr>
<tr>
<td>Total Cost:</td>
<td>$3.23</td>
<td>$3.27</td>
<td>$3.47</td>
</tr>
</tbody>
</table>

They may both be correct however; if the blasting is sufficiently suboptimal at the studied operation, it may be possible to show unit cost savings for the mining and crushing operations alone by adjusting blasting. If blasting is already at (or close to) an optimum with respect to mining and crushing unit costs, then changes may only increase costs. The ratio of fixed costs (e.g. overheads, capital) to variable costs (e.g. power, maintenance) also may play a role. If fixed costs are relatively high (to variable costs), then increasing production would reduce the unit cost to a greater degree than if variable costs were relatively high.
Figure 1-5: Cumulative costs in mining and processing related to explosives Powder Factor (PF) (Kanchibotla, 2000)

The majority of the benefit from increasing blasting effort found by Kanchibotla (2000) is not in unit cost savings, but come in the form of increased mill throughput, which increases the revenue and gross profit. Overall unit costs do not change appreciably. In the blasting scenarios summarized in Table 1-1, mill throughput increases 14% from cases 1 to 2, and 18% from scenarios 1 to 3, as the bottle neck in the mill, the SAG mills, are able to process more ore due to the finer feed size being provided to them from the drill-blast-crush steps of size reduction. No consideration is made for the ‘softening’ of the ore due to increased fracture density being introduced by blasting energy, i.e. the Bond Work Index is assumed to remain constant. The relationship
between the ability of the SAG mill to process material and feed size to the SAG mill is shown Figure 1-6, where an increase in the feed size causes mill throughput to drop.

![Figure 1-6: Relationship between Semi-Autogenous Grinding (SAG) mill feed 20% passing size (F20) and SAG mill throughput/feed, in Tonnes per Hour (TPH) (Kanchibotla, 2000)](image)

Paley and Kojovic (2001) found similar results to those of Kanchibotla (2000), in that on a unit basis, the actual cost does not change significantly by increasing drill and blast fragmentation. Benefits are largely the consequence of increasing mill throughput. The cost of adding additional capacity needs to be considered before any conclusions can be drawn about the benefits of such large increases in throughput. Both assume that increased ore production from the mine is possible at a fixed unit cost. In the long term this is true, but in the near term this would necessitate the purchase of more mining equipment to meet the increased ore demand. It seems unlikely that an operation would have 14-18% slack in ore production capacity. If capital expenditure was made so that higher production was possible, logically unit costs should decrease due to scales of economy, if all other factors are ignored. However, they remain the same in scenarios 1 and 2 in Table 1-1. This can be regarded as a conservative assumption.

It is relatively easy to visualize how rock fragmentation caused by blasting can reduce energy requirements in grinding. Blasting itself can be thought of as the first stage of
comminution. What also may be occurring during blasting is that the material is being ‘damaged’. Micro-fractures are being created in the rock by the high stress experienced during blasting, which later translate into reduced energy requirements for crushing and grinding. The degree to which this occurs is consequence to this study, and is discussed in the following section.

1.4 Micro-Fractures and Work Index

Blasting creates small fractures in rock, beyond fragmentation. An investigation by Olson et al. (1973) revealed that fractures (0.1-1.0 mm in length) could be created in rock by blasting and the fracture density, that is the ratio between the total length of micro-fractures per unit of thin section area was related to the intensity of the blast that the rock was subjected to. In Olson’s et al. (1973) investigation, blast holes were drilled and then shot, and thin sections were made from pre-blast and post blast core samples taken from in and around the borehole. The thin-sections were then examined for fracture density. Figure 1-7 depicts the relationship between the density of micro-fractures and the size of the explosive charge used, clearly indicating that the density of micro-fractures increased non-linearly with increasing charge size. Thus, it should follow that rock fragments created by blasting should be affected similarly.
Figure 1-7: Micro-fractures of pre and post blast rock cores (Olson et al., 1973)

Experiments by Nielsen (1999) indicate that blasting can induce micro-fracturing as shown in Table 1-2, and the level of micro-fracturing is dependant on the amount of explosive energy applied. Table 1-2 lists the number of cracks that were counted by analyzing thin-sections of blasted rock. In Table 1-2 “Small Cracks” are those that only occur on the boundary between two mineral grains, or those that appear in a single grain without bisecting it. “Boundary Cracks” are those that appear along the boundary between several grains, and “Grain Cracks” cut entirely through one or more grains (Nielsen, 1999). The number following the sample name in Table 1-2 indicated the number of detonating cords used to fragment the rock, i.e. the amount of blasting energy.
the rock was subjected to. The suffix “Standard” indicates that the sample was an unblasted control sample. All of the samples tested by Nielsen (1999) show a strong correlation between blasting intensity and crack frequency.

Table 1-2: Crack type and frequency in blasted and unblasted rock samples (Nielsen, 1999)

<table>
<thead>
<tr>
<th>Sample</th>
<th>Small Cracks</th>
<th>Boundary Cracks</th>
<th>Grain Cracks</th>
</tr>
</thead>
<tbody>
<tr>
<td>Taconite N 1</td>
<td>97</td>
<td>97</td>
<td>45</td>
</tr>
<tr>
<td>Taconite N 2</td>
<td>146</td>
<td>130</td>
<td>72</td>
</tr>
<tr>
<td>Taconite U Standard</td>
<td>55</td>
<td>48</td>
<td>15</td>
</tr>
<tr>
<td>Taconite U 1</td>
<td>73</td>
<td>116</td>
<td>23</td>
</tr>
<tr>
<td>Taconite U 2</td>
<td>84</td>
<td>103</td>
<td>35</td>
</tr>
<tr>
<td>Ilmenite Standard</td>
<td>40</td>
<td>9</td>
<td>31</td>
</tr>
<tr>
<td>Ilmenite 1</td>
<td>74</td>
<td>15</td>
<td>48</td>
</tr>
<tr>
<td>Ilmenite 2</td>
<td>97</td>
<td>18</td>
<td>80</td>
</tr>
</tbody>
</table>

Nielsen and Kristiansen (1996) studied changes to the work index for various rocks due to micro-cracks produced by varying powder factors, and concluded that significant changes in the work index are possible, and that increasing powder factor resulted in the reduction of the work index for taconite. It should be noted that they did not use the standard Bond (1961) test. Their results are summarized in Table 1-3.

Table 1-3: Work Indices ($W_i$) in kWh/t for various rocks at different blast energies (Nielsen and Kristiansen, 1996)

<table>
<thead>
<tr>
<th>Ore Type and Explosive</th>
<th>$P_{80}$</th>
<th>$W_i$</th>
<th>Percent -0.104 mm</th>
<th>$W_{bls}/W_{ref}$</th>
<th>Increase -0.104 mm (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Nepheline Reference</td>
<td>1090</td>
<td>4.4</td>
<td>22.8</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Nepheline 1 cord</td>
<td>900</td>
<td>4.6</td>
<td>23.3</td>
<td>0.85</td>
<td>2.2</td>
</tr>
<tr>
<td>Nepheline 2 cords</td>
<td>730</td>
<td>3.9</td>
<td>25.5</td>
<td>0.72</td>
<td>11.8</td>
</tr>
<tr>
<td>Taconite Reference</td>
<td>2910</td>
<td>14.4</td>
<td>31.4</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Taconite 1 cord</td>
<td>1420</td>
<td>6.7</td>
<td>34.0</td>
<td>0.47</td>
<td>8.3</td>
</tr>
<tr>
<td>Taconite 2 cords</td>
<td>730</td>
<td>3.9</td>
<td>37.2</td>
<td>0.27</td>
<td>18.5</td>
</tr>
<tr>
<td>Ilmenite Reference</td>
<td>5030</td>
<td>40.0</td>
<td>15.2</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Ilmenite 1 cord</td>
<td>4630</td>
<td>32.2</td>
<td>15.5</td>
<td>0.81</td>
<td>2</td>
</tr>
<tr>
<td>Ilmenite 2 cords</td>
<td>3500</td>
<td>18.8</td>
<td>17.1</td>
<td>0.47</td>
<td>12.5</td>
</tr>
</tbody>
</table>
The changes observed by Nielsen and Kristiansen (1996) were substantial, with work index reductions of over 50% compared to the reference (control) sample for the taconite and illmenite samples listed in Table 1-3. These were significantly higher than those obtained by Victorov (1983), who concluded that throughput increases of 6-7% are possible in grinding circuits due to increasing powder factor; however both studies predict a possible throughput increase.

1.5 Economic Rationale and Modelling

For economic modelling, there are several distinct, if not independent, approaches that can be taken for mine to mill optimization, and definition of them will help to focus the scientific work. Ultimately, the possibility for cost savings and/or profit increase is the impetus for research in this area.

![Powder Factor vs Total Cost](image)

Figure 1-8: Overall cost relationship for taconite mining (Eloranta, 1999)

One of the primary approaches used to determine where the optimum balance between blasting, crushing, and grinding lies in the analysis of unit cost, as seen in Figure 1-8 and Figure 1-9, where savings in downstream functions outweigh the increased cost of
higher fragmentation. This is possibly due to the fact that blasting costs are relatively small compared to the overall costs for mining and processing ore, as in Figure 1-9. An increase in drill and blast costs can be offset by a relatively small decrease in crushing and grinding costs.

![Diagram of cost relationships](image)

**Figure 1-9: Envisioned cost relationships as a function of blasting effort (Nielsen and Kristiansen, 1996)**

Assuming there is little to no effect on downstream processes beyond comminution improvements, this scope of analysis is valid. Research by Nielsen and Kristiansen (1996), Paley and Kojovic (2001), and Grundstrom et al. (2001) suggest that there are significant positive effects as a result of optimized run-of-mine fragmentation on downstream processes, in the form of increased throughput and increased liberation.
Graphically, the left-hand four curves in Figure 1-9 can be related to the individual unit costs in Figure 1-10, which summarizes the relative magnitude between increased blasting effort and the unit costs downstream operations for taconite mining.

![Graphical Representation of Taconite Mining Costs]

**Figure 1-10: Relative mining and comminution costs for taconite mining (Pastika et al., 1995)**

Nielsen and Kristiansen (1996), Paley and Kojovic (2001), and Grundstrom et al. (2001) did not consider, in any significant technical detail, the effects of increasing throughputs on post-comminution processing such as flotation and thickening, and this may affect their assessment of the benefits of increasing blasting effort. Grundstrom et al. (2001) conducted work similar to that of Paley and Kojovic (2001), with the goal of increasing SAG mill throughput (the bottleneck) at the Porgera Gold Mine by increasing fragmentation and altering crusher settings. Computer modelling of the mill concluded that a mill throughput increase of 25% is possible, and initial validation trials suggested that this was possible. The results of the analysis by Paley and Kojovic (2001) are summarized in Figure 1-11. It was found that the annual profit generated by the mine could be increased substantially by increasing the throughput in the mill via changes to
drill and blast practice. By making the feed to the crusher finer, the throughput of the mill bottleneck, the SAG mills, could be increased. Little long-term economic analysis was performed, but again, the effect of reduced recovery as a result of increased throughput needs to be quantified before any conclusions can be drawn regarding economic benefits.

Figure 1-11: Net profit increase realized by increased throughput vs. powder factor, expressed in kg/t of 70/30 heavy ANFO (Paley and Kojovic, 2001)

The reduced recoveries expected at higher throughput rates were included in their concentrate production (i.e., ore requirements increase at a rate faster than the increasing concentrate value), but the overall effect of reduced recoveries on the project as a whole were overlooked. If recovery losses are minor, then this may not be a large concern, since losses in the value of the in-situ mineralization would be minor, but this loss is not quantified by Paley and Kojovic (2001). Any rigorous economic analysis
using revenues from increased concentrate production needs to examine the effects of increasing throughput on recovery and concentrate grade (or quality, such as water content), as this could potentially negate any benefits from higher throughput.

Nielsen (1999) discusses the idea of two distinct approaches for mine to mill optimization: cost minimization or profit maximization. Both are equally valid, but represent two distinct philosophical approaches. In terms of total dollar gain, the savings provided by the work of Eloranta (1999) and Nielsen (1999) are relatively modest when compared to the profit increases due to throughput increases projected by the work of Paley and Kojovic (2001). The increased cash flow from process savings of \( \sim \$0.25/t \) is significantly less, even for a large mine, than the revenue increases projected by increased throughput. Neither work can be described as comprehensive, but the indication is that the bulk of the benefit of improving the mine-to-mill interface lies in increased revenue, not reduced cost. One area that is not quantified by either party is the possibility of capital savings that would be possible with higher throughputs.

Recovery issues are addressed to an extent by Kanchibotla (2000). Kanchibotla (2000) predicts that as mill throughput increases, recoveries either remain constant or increase, citing increased liberation due to increased blasting effort as the cause. This in itself is plausible but conventional wisdom would tend to disagree. In the case where the mill throughput is below expectation and there is sufficient capacity within separation circuits to handle higher throughputs (Simkus and Dance, 1998) this may be possible. In cases where throughput gains are above the nominal capacity of separation circuits, however, it is questionable whether recoveries would remain constant or increase, as the available residence time for ore in the flotation circuits would decrease. Figure 1-12 summarizes this concept.
Figure 1-12: Relationship between throughput tonnes per hour (tph) and flotation recovery (Napier-Munn et al., 1996)

Fine particles are another problematic issue. Kanchibotla (2000) views the generation of fines in the pit as a positive thing, stating that “…fines are essentially “free” throughput for the mill”. Fundamentally this is true, since no further comminution is required for downstream processes, but fines present operational problems. If the mill is designed for fines, i.e. there is some kind of pre-grinding classification so that the fines generation can be taken advantage of, this may be a safe assumption, but often this is not the case. Also, for many particular types of mining operations such as aggregate production, operations producing lump ores (e.g. some iron ore operations), and coal operations, increased fines production is detrimental to the operation and not economically desirable. The increased fines generation in the pit is also cited as one of the reasons that higher mill throughputs are possible. It has been the author’s experience that fines can detrimentally affect pit operations, especially in wet conditions, as they tend to adhere to and clog almost every type of mobile equipment, and cause traction/mobility problems. Fines also oxidize rapidly, often resulting in flotation problems.
1.6 Kemess Mine

1.6.1 Geologic and Setting

Rock samples and operational data that were used in Chapters 2-6 of this study were provided by Kemess Mine. The Kemess Mine is a gold/copper porphyry deposit owned and operated by Northgate Minerals (NGX-T), located in North-Central British Columbia in the Omenieca Mountain Range. The Kemess South deposit, which is the deposit being mined, is characterized as a Lower Jurassic calc-alkaline porphyry system consisting of a relatively flat-lying, homogenous quartz monzodiorite sill. The deposit is underlain by Stuhini Group mafic volcanic rocks, and is unconformable overlain by Sustut Group sedimentary rocks (Diakow and Metcalf, 1997 and Werniuk, 2003). Reserves for Kemess South as of Dec. 31, 2002 were 109 mt grading 0.712 g/t gold and 0.234% copper (Northgate Annual Report, 2002).

Kemess South has been affected by two phases of alteration. The first (hypogene) alteration introduced potassium and sericite with pyrite (1-5% of the rock volume), chalcopyrite, magnetite-hematite, bornite and molybdenite, with lesser pyrrhotite, tetrahedrite and native gold, plus intense stockwork veining. The second (supergene) alteration was a prolonged period of acid weathering, and lead to retrograde alteration to chlorite and pyrophyllite, with mineralized zones containing native copper, chalcocite, bornite, chalcopyrite, hematite, native gold, and rare silver (Werniuk, 2003).

For mining purposes, there are fourteen different material types identified by the mine and are listed in Table 1-4, and will be referred to by their acronym for brevity. The four ore types are named according to their alteration, as described above. Hypogene Ore (HO) is a result of the first alteration; Leach Cap Ore (LO) is from the top of the leach zone and has been stripped of sulphide minerals and essentially ‘caps’ the ore. Copper
leached from the LO zone has been transported downward, enriching the Supergene Ore (SO), and Transition Ore (TO), exists in the boundary between the SO and the HO zones.

<table>
<thead>
<tr>
<th>Material Code</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>AG</td>
<td>Acid Generating Waste</td>
</tr>
<tr>
<td>GAG</td>
<td>Graphitic AG material</td>
</tr>
<tr>
<td>HO</td>
<td>Hypogene Ore</td>
</tr>
<tr>
<td>LCW</td>
<td>Leach Waste</td>
</tr>
<tr>
<td>LO</td>
<td>Leach Cap Ore</td>
</tr>
<tr>
<td>NAG</td>
<td>Non Acid Generating Waste</td>
</tr>
<tr>
<td>OB</td>
<td>Pit Overburden</td>
</tr>
<tr>
<td>ORG</td>
<td>Pit Organic Overburden</td>
</tr>
<tr>
<td>PAG</td>
<td>Potential Acid Generating Waste</td>
</tr>
<tr>
<td>SG</td>
<td>Low Grade Supergene</td>
</tr>
<tr>
<td>SO</td>
<td>Supergene Ore</td>
</tr>
<tr>
<td>TKA</td>
<td>Takla Sediments AG from Pit</td>
</tr>
<tr>
<td>TKN</td>
<td>Takla Sediments NAG from Pit</td>
</tr>
<tr>
<td>TO</td>
<td>Transition Ore</td>
</tr>
</tbody>
</table>

Table 1-4: Mine Material Types

The main focus of this thesis will be blast optimization of the HO zone, as this ore type comprises the majority of the deposit and is also the hardest rock, with the lowest mill throughput rates of all ore types. The HO ore has been itself classified into five different types (and domains) based on alteration and ranked according to hardness and associated mill throughput and are summarized in Table 1-5.

Table 1-5: Kemess hypogene ore alteration domains listed in descending order of hardness (Kemess, 2003)
Rock samples from Domains 1, 2, and 5 were examined for changes in grindability due to blasting.

1.6.2 Mining Method and Equipment

The Kemess South deposit is being mined as an overlapping double-pit with the two pit frustums aligning approximately east-west. Bench height is 15 m with the exception of pioneering work, where bench height varies. Two 311 mm rotary drills are responsible for production drilling, a diesel powered Ingersoll Rand 351 Pit Viper and an electrically powered P&H 100XP. Explosives and accessories are provided and delivered by BXL Bulk Explosives operating under contract, with blasts being loaded, stemmed, and initiated by mine employees, and blast patterns are designed by the Kemess Mines Technical Services Department. The two main production shovels are a P&H 2800XPB (32 m$^3$ bucket), which is mainly responsible for waste removal and P&H 2300 (25 m$^3$ bucket) for ore. Secondary production equipment consists of a LeTourneau L1400 front-end loader and Hitachi EX3500 which is scheduled to be retired in 2003 upon completion of overburden stripping. The vast majority of material, both waste and ore is transported by the mine's fleet of 13 Euclid R260 haul trucks with a rated capacity of 238 metric tonnes, with occasionally small amounts being moved by smaller equipment.

1.6.3 Milling Circuit

The general flow sheet for mineral processing at Kemess is summarized in Figure 1-13. ROM ore is dumped directly into the crusher, which then feeds into a 72 hr capacity stockpile. The two lines shown in Figure 1-13 are fed from a common stockpile, which stores approximately 72 hours of mill feed. The ore is then passed by two reclaim feeders into 2 identical parallel grinding lines (referred to as the A and B Lines), each of which consist of a SAG mill operating in closed circuit with a screen, followed by a ball
mill operating in closed circuit with cyclones. For further detail regarding the
comminution circuit equipment please see Chapter 4:

![Diagram of mill process](image)

**Figure 1-13: Kemess milling circuit** (Gray et al., 2003)

The final product, the cyclone overflow is then passed to the flotation circuit, where it
passes first through rougher flotation and then to the first cleaner and cleaner scavenger
cells, with coarse material being passed to the regrind circuit. Rejected material from
the rougher float is sent to tails. After being floated by the first cleaners, selected
material is then sent to the cleaner columns, where floated material is sent to the
thickener and the clarifier to be processed into final concentrate. Tails from the cleaner
columns is then sent to the column scavengers, from which floated material reports to
the thickener/clarifier and tails report back to the regrind circuit.

26
Chapter 2: Small-Scale Blasting Experiments

2.1 Introduction

It has been shown that the work index of a material can be affected by blasting (Nielsen and Kristiansen, 1996) and that blasting reduces the energy required to grind ore to the size required in crushing and grinding processes. To test whether blasting would effect any changes in work index blasting Kemess Hypogene ore, coarse grab-samples of Kemess Hypogene ore were collected from the open-pit by the Mine Geologist that appeared to be representative of alteration domains D1, D2 and D5. The samples themselves were coarse irregular rock fragments at least 20 cm in size. D3 and D4 domain samples were not available at the time the samples were taken in May 2003. All samples taken were blast fragments and had been subjected to blasting. The samples were then split into 2 groups, one which was to serve as the control which henceforth is referred to as ‘Unblasted’ and the other group to be subjected to a blast induced stress wave, which will be referred to as ‘blasted’. Both groups were then subjected to laboratory testing to ascertain whether the secondary blasting had affected physical grinding properties.

2.2 Experimental Set-up

To assess the impacts of blasting on irregularly shaped rock samples in a controllable fashion, a method of blasting the rock using a 150 g Pentex primer as a donor coupled to the rock with water was employed (Orica, 2004). The primer charge was submerged 34 mm longitudinally in the water to couple the charge to the water for the transmission of the stress wave from the blast. This configuration was preferred since the pressures in the water were known from other experimental studies. The nearest part of the specimen to be blasted was located 76 mm from the bottom of the submerged primer, as
shown in Figure 2-1. The objective of the blasting experiments was to subject the specimens to a dynamic compression stress wave similar to what would be experienced by a significant amount of material around a blast hole. The dynamic compression would exceed the strength of the rock and cause it to fragment as in a 'real' blast.

![Figure 2-1: Scale Blasting configuration for impacting rock samples with stress wave to simulate blasting on rock sample (not to scale)](image)

The entire apparatus was placed in a closed blasting chamber at the Queen’s University Blasting Test Site and the charge was detonated. The resulting fragments were then collected for analysis in the mineral processing laboratory.

Two sets of samples were blasted, to provide materials for different types of comminution testing. Approximate physical dimensions of the samples are listed in Table 2-1.
Table 2-1: Pre-blast sample information

<table>
<thead>
<tr>
<th>Block Information</th>
<th>X</th>
<th>Y</th>
<th>Z</th>
<th>Mass (kg)</th>
</tr>
</thead>
<tbody>
<tr>
<td>D1</td>
<td>36</td>
<td>25</td>
<td>23</td>
<td>28.92</td>
</tr>
<tr>
<td>D2</td>
<td>36</td>
<td>25</td>
<td>20</td>
<td>18.87</td>
</tr>
<tr>
<td>D5</td>
<td>33</td>
<td>19</td>
<td>17</td>
<td>20.86</td>
</tr>
</tbody>
</table>

After blasting the fragments were collected and screened using the Queen’s Civil Engineering’s Gilson Screening machine, as it is the only machine at the university able to screen such large coarse samples. The results are shown in Figure 2-2.

Figure 2-2: Blast fragment screening results

A large proportion of the material did not pass through the 50 mm screen, which was the largest aperture available, and thus there is no resolution of the size distributions for all samples at coarser sizes. The D2 and D5 have 50% passing size of 38.7 mm and 49.0 mm, respectively, and the trend suggests that the D1 sample would have a 50% passing
size slightly over 50 mm. Due to the nature of the experiment, it was not possible to relate fragmentation to normal blasting parameters, such as powder factor, but numerical modelling was used to understand the nature of the damage imparted to the rock. This is discussed in the next section.

### 2.3 2-Dimensional Blast Numerical Modelling

To assess the level of damage imparted on the block and obtain an understanding of the level of stress that the specimen was subjected to, the experiment was modelled in the two dimensional numerical modelling software package AUTODYN-2D, developed by Century Dynamics (Century Dynamics, 1995). AUTODYN-2D contains fully integrated engineering analysis codes designed for non-linear dynamic problems, such as explosion, blast and impact events (Fairlie, 1998). The constitutive model used to simulate the Kemess HO ore was a high strength concrete model which has approximately similar physical parameters. The donor charge (the primer) and the water are modelled using an Eulerian grid, which is well suited for modelling materials that are expected to undergo large displacements. A Lagrangian grid was used to model the rock, as it was not expected to undergo much displacement within the relevant modelling time (Fairlie, 1998).
Figure 2-3: Modelling results showing stress wave passing from water into the rock

Figure 2-3 shows numerical modelling results after 53 µs. The model is set up so that the axis of symmetry is vertical through Figure 2-1, and is set on its side in Figure 2-3, with the shape of the specimen approximated by a rectangle. The primer has completely detonated and the shockwave has transferred from the water into the rock, imparting pressures well above the failure threshold. Ten ‘targets’, which can be thought of as being virtual instruments, were placed in the model to record pressures in the specimen along the axis of symmetry. The indicated that failure threshold was surpassed, and inspection of the samples after blasting confirmed this.

Figure 2-4 shows the failure map of material using a Mohr-Coulomb failure criterion after 125 µs, with the parameters for the material tested listed in Table 2-2. The material has failed. The blast wave impacted the specimen and cracking has taken place throughout the sample, fragmenting it.
Figure 2-4: Loading failure from small scale blasting

Table 2-2: Basic parameters for blasting failure model

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Density (g/cc)</td>
<td>2.85</td>
</tr>
<tr>
<td>Bulk Modulus (GPa)</td>
<td>43.52</td>
</tr>
<tr>
<td>Shear Modulus (GPa)</td>
<td>22.59</td>
</tr>
</tbody>
</table>

The modelling indicated that the pressure experienced by the rock is not unlike what would be experienced by material undergoing regular blasting at a mine, and that the damage imparted to the rock would be similar.

### 2.4 Micro-fractures Created by Blasting

To assess whether the blasting created any micro-fractures in the rock, samples of blasted and unblasted D1 material was examined for micro fracture density. Only one type of rock was selected for this analysis because this type of analysis is done externally to the Queen’s Mining Department. The Department lacks the necessary equipment and expertise to conduct this type of work, which can be costly and time-consuming.
intensive. The micro-fracture analysis was performed by Dr. B Mohanty and Dr. M.H.B. Nasseri at the University of Toronto. The goal of the analysis was to see if blasting had increased the micro-fracture density of the rock, presumably making it softer and more amenable to mechanical size reduction.

2.4.1 Micro-fracture Detection and Measurement

The first step in quantifying micro-fractions is to create thin-sections of the rock for photographing on a standard petrographic microscope. The samples of the blasted and unblasted D1 material consisted of one rock each approximately the size of a fist. For each sample, two petrographic thin-sections (3.0 x 2.0 cm in size) were cut from relatively orthogonal (but arbitrarily oriented) planes for a total of 4 thin-sections. Each thin-section was then digitally photographed in over 30 separate pictures and then combined to create a total image of the thin-section as shown in Figure 2-5 and Figure 2-6. The 'squares' in Figure 2-5 and Figure 2-6 are a result of the total image of the thin section being put together to create a composite image from the many smaller photos that were taken. The dark lines indicate where the fractures in each thin section were located. Fracture densities were obtained using CorelDraw and Intercept, two computer packages (Laneau and Robin, 1996). The task of creating the mineral grain and fracture mosaic image from the original thin section is extremely demanding (of human time) since the grains and fractures must be traced manually. Human discretion is also required in deciding what is a true fracture in the thin-section, since some discontinuities appear to be cracks although they may only be interfaces between two different mineral types in the rock. To do this effectively requires an experienced eye (Nasseri, 2004).
Figure 2-5: Blasted thin-section composite picture
Figure 2-6: Unblasted Kemess D1 composite image
The Intercept software uses the ‘Intercept’ method to detect (Launeau and Robin, 1996) and quantify flaws in the material and generate the corresponding micro-fracture density. It can compute other micro-fracture metrics such as length and orientation, but for the purposes of this study, density (cm/cm$^2$) is the most relevant. The intercept method scans the image along sets of parallel lines oriented over a 180° span, counting the number of discontinuities encountered by each line. The fracture density for the rock sample can then be calculated by averaging the value of the micro-fracture density from the two thin-sectioned planes (Nasseri et al., 2005). A complete description of the method used for deriving the micro-fracture densities is presented by Nasseri et al. (2005).

2.4.2 Results and Discussion

The results of the analysis are summarized in Table 2-3. The difference found in the micro-fracture density between the two samples is large. The blasted sample had a micro-fracture density of more that 3 times the unblasted control sample, which would indicate that blasting caused a substantial increase in the micro-fracture density in the rock.

Table 2-3: Microfracture density induced in Kemess D1 Hypogene ore

<table>
<thead>
<tr>
<th>Plane</th>
<th>Unblasted D1 Sample Fracture Density (cm/cm$^2$)</th>
<th>Blasted D1 Sample Fracture Density (cm/cm$^2$)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Side</td>
<td>1.9</td>
<td>5.0</td>
</tr>
<tr>
<td>Front</td>
<td>1.7</td>
<td>7.9</td>
</tr>
<tr>
<td>Average</td>
<td>1.8</td>
<td>6.5</td>
</tr>
</tbody>
</table>

The sample for each group was a single fragment of rock, from which two thin-sections in orthogonal planes were created and analyzed. The two fragments chosen for analysis appeared to be representative of the sample. A more thorough approach would be to
perform micro-fracture density analysis on a number of sampled fragments from each blast to increase statistical confidence of the analysis. This however, is not practical since it is so time-consuming (Nasseri, 2004).
Chapter 3: Milling Property Changes Induced by Blasting

3.1 Introduction

To assess if there were changes in the milling properties of the samples that were subjected to the primer blast wave from the primer (“blasted”) against the control samples (“unblasted”), the samples were subjected to a variety of laboratory tests. The tests used are indicative of the behaviour of the material undergoing crushing, Semi-Autogenous Grinding (SAG), and ball milling. Typically, these are considered to be the rate limiting processes for many mills (Kanchibotla, 2001) and hence are of the most interest. Nielsen and Kristiansen (1996) also suggest that the largest source of grinding performance improvements come from the reduction of the work index due to blasting as opposed to strictly optimizing blast fragmentation. The blasted and unblasted samples were subjected to the MinnovEX Crusher Factor and SAG Power Index (SPI) Test (Kosick et al., 2001), $t_{10}$ drop weight tests (Napier-Munn et al., 1996), and the Bond Work Index Test (Bond, 1961). The tests, representative of all stages of mechanical comminution were analyzed to observe if feed and product size played any potential role in the comparison of the blasted and unblasted samples.

Table 3-1: Summary of milling property tests

<table>
<thead>
<tr>
<th>Test</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>MinnovEX Feed Crushing</td>
<td>A proprietary test that is used to characterize the behaviour of material undergoing crushing, using laboratory-scale crushers. It is used in conjunction with the MinnovEX SPI test.</td>
</tr>
<tr>
<td>MinnovEX SAG Power Index</td>
<td>A proprietary test that used to describe the specific power consumption requirements of a material undergoing size reduction in a SAG mill. It employs a laboratory-scale SAG mill for testing, and only requires a small amount (&lt;5 kg) of sample material.</td>
</tr>
</tbody>
</table>
This test is described by Napier-Munn et al. (1996), and was devised at the Julius Kruttschnitt Mineral Research Centre (JKMRC). It is used to determine the energy requirements for rock breaking in crushing and SAG/AG mill size fractions. It involves a drop-weight apparatus that crushes a single particle under controlled conditions. A complete test (to characterize a material) requires compiling data from numerous single particle test results. It can test larger particles e.g. >50 mm in size, which form a large fraction of blasted material.

As described by Bond (1961), this is a commonly used test to predict the energy requirements of material that is undergoing comminution in a ball mill in a closed circuit. It is very commonly used in industry, and is widely accepted. It is performed on a dry sample however, and not a slurry as it would be in an actual mill.

A test that investigates how fast a monosize feed grinds in a laboratory ball mill (the same ball mill as used in the locked cycle testing), as described by Austin and Weller, (1982). It provides an indication of how fast a certain size fraction is grinding under controlled conditions.

### 3.2 MinnovEX Test Procedures

The MinnovEX Sag Power Index (SPI) test (Starkey, 1997) was used to determine if there were changes in the comminution properties if the material was to be processed in a SAG mill. The samples were also subjected to the MinnovEX feed crusher parameter 

(C_f) (Dobby et al., 1999), providing an indication of any changes to the material properties of the ore in the size fractions fed to and produced by a crusher. Both tests are optimized and designed for small samples sizes (<5 kg of feed material), as their intended use is for assessing the milling properties of ore based on exploration drill core samples. Hence, comprehensive estimations of the physical milling properties of a deposit can be made before the production phase. These tests are ideal diagnostic test given the limited amount of material produced by the small-scale blasting experiments.
3.2.1 MinnovEX Feed Crushing Parameter

The MinnovEX Feed Crushing Parameter is used to predict the cumulative 80% and 50% passing sizes of the crusher product. It is a proprietary test that measures the percent reduction of ¾" (18 mm) feed material by laboratory crushers in several stages of reduction (Kosick et al., 1999). It is referred to as the “feed” crushing parameter since it is intended to assist in the prediction of the crusher product that is to be fed to the downstream AG/SAG mill(s). Three blasted Kemess HO ore samples from alteration domains D1, D2 and D5 (the different ore alteration zones discussed in Section 1.6.1) were tested and compared to 3 control samples from the same domain. The objective was to determine whether blasting had any influence on the $C_F$ values of the Kemess ore samples.

The $C_F$ value is related to the crusher 50% passing size product (F50) and the 80% passing size products (F80) through Expressions (1) and (2). The relationship for the F80 product and Expression (1) is graphically expressed in Figure 3-1. Expressions (1) and (2) also show that the crusher performance is related to the SAG Power Index value, so truly the F50 and the F80 crusher product sizes are related to the first two terms in expressions (1) and (2) (independent of machine settings). It should be noted that neither of these equations takes into account the feed size of the material to the crusher, i.e. these expressions cannot predict the changes to the crusher product due to changes in the crusher feed size distribution (Dobby et al., 1999).

\[
F80 \propto C_F^{n_1} SPI^{n_2} CSS^{n_3} \quad (1)
\]

\[
F50 \propto C_F^{n_1} SPI^{n_2} CSS^{n_3} \quad (2)
\]

Where:

$C_F$ = MinnovEX Feed Crushing Parameter
SPI = MinnovEX SAG Power Index (discussed in Section 3.2.2)

CSS = Crusher Closed side set

n1, n2, n3 = Constants

F50, F80 = Crusher product 50% and 80% passing

![Graph showing relationship between MinnovEX F parameter and crusher F80 product](image)

**Figure 3-1: Relationship between MinnovEX $C_F$ parameter and crusher F80 product**

(Dobby et al., 2001)

It has been found by MinnovEX that the $C_F$ parameter varies inversely with ease of comminution, i.e. a higher $C_F$ value indicates that the ore is softer and produces a finer product. The results summarized in Figure 3-2 show that the differences in the $C_F$ values for the samples subjected to the primer pulse compared the control samples.
Figure 3-2: MinnovEX Feed Crushing Parameter results ($C_F$) for 6 Kemess HO ore samples from alteration domains D1, D2 and D5.

The Y-axis in Figure 3-2 is intentionally left blank as the precise data is proprietary. The intention is to show the relative difference. All of the blasted samples have a higher $C_F$ value than their control sample, indicating that the blasted samples will produce a finer crusher product given identical crusher feed size distributions and operating conditions. The largest difference in crushing behaviour is observed in the D1 material, and is quite large compared to the differences between the D2 and the D5 domain samples. This would suggest that the crushability of D1 material would be significantly enhanced (i.e. softer) when undergoing size reduction in the primary crusher. This would agree with the observations in Section 2.4. which indicated a higher degree of microfracturing in the blasted D1 sample relative to the unblasted sample.
3.2.2 MinnovEX Semi-Autogenous Grinding Power Index (SPI)

The MinnovEX SPI test is a proprietary laboratory-scale test designed for prediction of the grinding behaviour and power requirements of rocks undergoing SAG milling (Starkey, 1997). The test apparatus is a 30 cm diameter mill with a 3:1 aspect ratio charged with 15% steel (Baeza and Villanueva, 2004). The feed is 80% passing 13.2 mm, and the SPI is derived from the time required to grind the initial feed to 80% passing 1.7 mm (Kosick and Bennet, 1999). The time required to reduce the feed charge to the required 80% passing product size is related to the specific grinding energy (i.e. the SPI) as shown in Equation (3) (Amelunxen et al., 2001).

\[
kWh / t_{SAG/AG} = k \left( \frac{S}{\sqrt{T_{80}}} \right)^N * f_{SAG}
\]

(3)

Where:

\(k, N\) = Proprietary Constants

\(T_{80}\) = Mill product 80% passing size

\(S\) = Time required to grind to 80% passing 1.7 mm

\(f_{SAG}\) = A circuit specific function to account for differences in operating circuit configurations

\(kWh / t_{SAG/AG}\) = The SPI, required specific grinding energy

\(t / h_{SAG/AG} = \frac{kW_{SAG/AG}}{kWh / t_{SAG/AG}}\)

(4)

Where:

\(t / h_{SAG/AG}\) = Mill throughput in tonnes/hr

\(kW_{SAG/AG}\) = Mill available power

\(kWh / t_{SAG/AG}\) = SPI value
Figure 3-3 shows the results for the SPI tests converted into kWh/t using constant values for Equation (3) which can be related to the throughput capacity of the mill by Equation (4) (Amelunxen et al., 2001). Typical values for feed and $T_{80}$ (product) sizes were used in the calculation. The D1 blasted material and D5 blasted material have an SPI value that is lower than that of their unblasted counterparts. This indicates that the D1 and D5 blasted material would have a lower grinding energy requirement than the D1 and D5 unblasted material. Initial proprietary crushing tests indicated that the D1 blasted material should be softer, with the D2 and D5 material only slightly changed.

![SPI Comparison For Kemess HO](image)

**Figure 3-3: Comparison of SPI Values for Kemess HO Domains D1, D2, and D5**

The difference in the D2 samples is small, both in absolute and relative terms. The absolute differences in the D1 and the D5 samples is small, at slightly over 1 kWh/t, but the relative difference in the D1 sample is large at approximately 19% relative to the
original sample. In processing terms, this would amount to a 19% reduction in required power cost to grind the same amount of ore to the same product size, or conversely, increase the throughput without changing the SAG mill product size. A reduction in the SPI would be expected, given the increase in the microfracture density observed in the D1 samples in Section 2.4.2.

3.3 Drop Weight Testing

The goal of drop-weight testing was to determine if there had been any apparent change in the properties of the rock in classes that would be representative of crushing and SAG milling process. Drop weight testing allows for the measuring of material comminution properties at particles sizes larger than found in other laboratory bench tests, such as rod and ball mill work index testing (Bond, 1961). An in-house drop-weight machine was developed for the drop-weight testing, which is discussed in the following section.

3.3.1 Principles and Calibration of the Queen's Drop-Weight Tester

The operating principle of the Queen’s Drop-Weight Tester (QDWT) is simple. It consists of dropping a known mass on a rock fragment of known mass from a predetermined height. Thus, the specific energy imparted on the particle can be calculated. The fragments from the smashed particle are collected and sieved, permitting analysis of the relationship between specific impact energy and particle size (Napier-Munn et al., 1996).

The QDWT was designed and constructed by Chadwick Engineering of Kingston Ontario to the specifications of the Queen’s Mining Department. The apparatus consists of a long vertical guide tube in which a cylindrical 20 kg steel weight of a slightly smaller
diameter moves. The weight is raised by a pulley, and released by an electromagnet allowing it to fall down the pipe and impact the sample on the sample stage as shown in Figure 3-4. The sample stage is enclosed by a removable steel shroud that prevents rock fragments from being ejected so that they can be collected and analyzed. Installed along the guide tube are two infra-red timing gates that allow the velocity of the drop-weight to be measured. These were used in calibrating the ‘effective’ gravitational constant of the machine, since it was anticipated that there would be some drag on the drop-weight from frictional resistance between the drop-weight and walls of the guide tube.

![Figure 3-4: Queen's Drop Weight Tester (QDWT) sample stage](image)

The two timing-gate switches are optical sensors, and they are connected to an oscilloscope to observe the timing of the gates closing. Figure 3-5 shows the layout of
the apparatus and the location of the timing gates. At constant acceleration, the difference in velocity at two different points can be calculated by using Equation (5).

\[ \Delta x = \frac{v^2 - v_o^2}{2a} \]  

(5)

Where:

\( \Delta x \) = displacement of the drop-weight (downwards direction taken as positive)

\( v \) = Final velocity

\( v_o \) = Initial velocity

\( a \) = calculated acceleration due to gravity

Figure 3-5: Configuration of the Queen's Drop-weight Tester

Re-arranging Equation (5) for \( v \), given an initial velocity of zero yields Equation (6) and Equation (7) that can be used to calculate the velocity of the drop-weight at the upper and lower timing gates.
\[ v_u = \sqrt{2a\Delta x_1} \]  
\[ v_f^2 = v_u^2 + 2a\Delta x_2 \]  

The average velocity between two points at constant acceleration is given by Equation (8). Substituting Equations (6) and (7) into Equation (8) yields Equation (9).

\[
\frac{-v}{2} = \frac{v_u + v_f}{2} = \frac{\Delta x_2}{\Delta t}
\]  

\[
\frac{\Delta x_2}{\Delta t} = \frac{\sqrt{2a\Delta x_1} + \sqrt{2a\Delta x_1 + 2a\Delta x_2}}{2}
\]  

Re-arranging for \(\Delta t\) yields equation (10):  

\[
\Delta t = \frac{2\Delta x_2}{\sqrt{2a\Delta x_2} + \sqrt{2a(\Delta x_1 + \Delta x_2)}}
\]  

Using Equation (10) a correction for the gravitational change due to friction can be made for the QDWT. Using a least squares fitting procedure, the calculated acceleration (as in Equation (10)) constant that provides the best prediction for the observed gate times is 9.36 m/s\(^2\) over the range of heights tested. Figure 3-6 shows a comparison between the actual gate times and the predicted gate times using the fitted gravitational constant of 9.36 m/s\(^2\).
Figure 3-6: Predicted gate time using modified g constant for QDWT and actual gate time

The one limitation of the timing gate is that its position does not allow for gate times from a drop height less than 335 mm above the sample stage. Thus, data below that drop point could not be collected and included in the calibration.

3.3.2 Calculations for Drop-Weight Sample Testing

The relationship between drop height and specific energy is:

\[ Ep = mg\Delta h \]  \hspace{1cm} (11)

Dividing the potential energy of the mean mass of the sample particle batch yields the specific comminution energy imparted on the batch.

\[ Eis = \frac{Ep}{m} \] \hspace{1cm} (12)

Where:

\[ Ep = \text{potential energy (j)} \]

\[ m = \text{drop-weight mass (kg)} \]
\( g \) = acceleration due to gravity

\( \Delta h \) = height (m)

\( Eis \) = specific energy (j/kg)

\( \bar{m} \) = mean particle mass (kg)

Substituting Equation (11) into (12) and converting energy units to kWh/t yields equation (13):

\[
Eis = \frac{0.0026m\Delta h}{\bar{m}}
\]  

(13)

Where:

\( m \) = Drop-weight mass (kg)

\( \Delta h \) = Drop height (m)

\( \bar{m} \) = Sample particle mass (g)

It should be noted that the drop height, \( \Delta h \), is taken as the initial height of the drop-weight minus the final height of the drop weight, since the drop-weight will come to a rest on top of the sample at some small elevation above the sample stage base. This situation generally results in a specific impact energy slightly less than predicted. Equation (13) is then re-arranged to Equation (14) to calculate the required drop height given a desired \( Eis \) and mean particle mass.

\[
\Delta h = \frac{Eis\bar{m}}{0.0026m}
\]  

(14)

Using Equations (13) and (14) samples can be tested at known specific impact energies.

### 3.3.3 Drop-weight Testing Methods

For drop-weight testing, a procedure similar to that proposed by Napier-Munn et al. (1996) was followed. First, rock fragments from the blasted and the unblasted samples were screened to the following size fractions listed in Table 3-2, which were suggested
by Napier-Munn et al. (1996). The reason for using the narrow $\sqrt{2}$ size intervals is to try and approximate a single size range, i.e. so that replicate testing of a single size particle can be achieved.

**Table 3-2: Feed size classes for $t_{10}$ testing (Napier-Munn et al., 1996)**

<table>
<thead>
<tr>
<th>Batch</th>
<th>-63+53mm</th>
<th>-45+37.5mm</th>
<th>-31.5+26.5mm</th>
<th>-22.4+19mm</th>
<th>-16+13.2mm</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>0.1</td>
<td>0.1</td>
<td>0.25</td>
<td>0.25</td>
<td>0.25</td>
</tr>
<tr>
<td>2</td>
<td>0.25</td>
<td>0.25</td>
<td>1</td>
<td>1</td>
<td>1</td>
</tr>
<tr>
<td>3</td>
<td>0.5</td>
<td>1</td>
<td>2.5</td>
<td>2.5</td>
<td>2.5</td>
</tr>
</tbody>
</table>

Napier-Munn et al. (1996) suggest a size interval of –63+53 mm of feed particles as indicated in Table 3-2, but due to the lack of particles available in the size range it was not included. Also, sample size restrictions allowed for only 10 particles of each energy-size combination to be tested, whereas Napier-Munn et al. (1996) indicate that 20-30 particles are normally used by the JKMRC in their testing procedure. The specific impact energies for each energy-size combination that samples were planned to be tested at are listed in Table 3-3. Each set of 10 particles was weighed, and the average particle mass for that particular energy-size combination was used to determine the drop height required to impart the required specific impact energy according to Equations (13) and (14) in Section 3.3.3. Each particle is broken separately at the pre-determined height for each particular energy-size combination, and after all the particles from a specific energy-size combination have been broken, the resulting fragments are combined and then screened.

**Table 3-3: Planned impact energies for each size-energy combination (kWh/t)**

<table>
<thead>
<tr>
<th>Batch</th>
<th>-45 +37.5 mm</th>
<th>-31.5 +26.5 mm</th>
<th>-22.4 +19 mm</th>
<th>-16 +13.2 mm</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>0.1</td>
<td>0.25</td>
<td>0.25</td>
<td>0.25</td>
</tr>
<tr>
<td>2</td>
<td>0.25</td>
<td>1</td>
<td>1</td>
<td>1</td>
</tr>
<tr>
<td>3</td>
<td>0.5</td>
<td>2</td>
<td>2.5</td>
<td>2.5</td>
</tr>
</tbody>
</table>
The difference between the planned drop energy in Table 3-2 and Table 3-3 for Batch #3 -31.5+26.5 mm material is because the QDWT is not capable of generating a specific impact energy of 2.5 kWh/t for that particle size class. Table 3-4 and Table 3-5 summarize the actual drop-weight impact energies for each size-energy combination of Kemess ore tested.

Table 3-4: Kemess HO Blasted specific impact energies (kWh/t) for D1, D2 and D5

<table>
<thead>
<tr>
<th>Batch</th>
<th>-45 +37.5 mm</th>
<th>-31.5 +26.5 mm</th>
<th>-22.4 +19 mm</th>
<th>-16 +13.2 mm</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>0.08</td>
<td>0.23</td>
<td>0.22</td>
<td>0.18</td>
</tr>
<tr>
<td>2</td>
<td>0.24</td>
<td>0.99</td>
<td>0.99</td>
<td>0.98</td>
</tr>
<tr>
<td>3</td>
<td>0.72</td>
<td>2.01</td>
<td>2.52</td>
<td>2.50</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Batch</th>
<th>-45 +37.5 mm</th>
<th>-31.5 +26.5 mm</th>
<th>-22.4 +19 mm</th>
<th>-16 +13.2 mm</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>0.09</td>
<td>0.23</td>
<td>0.20</td>
<td>0.16</td>
</tr>
<tr>
<td>2</td>
<td>0.24</td>
<td>1.00</td>
<td>0.99</td>
<td>0.98</td>
</tr>
<tr>
<td>3</td>
<td>0.55</td>
<td>2.01</td>
<td>2.52</td>
<td>2.51</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Batch</th>
<th>-45 +37.5 mm</th>
<th>-31.5 +26.5 mm</th>
<th>-22.4 +19 mm</th>
<th>-16 +13.2 mm</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>0.09</td>
<td>0.23</td>
<td>0.21</td>
<td>0.17</td>
</tr>
<tr>
<td>2</td>
<td>0.25</td>
<td>0.96</td>
<td>1.01</td>
<td>1.01</td>
</tr>
<tr>
<td>3</td>
<td>0.49</td>
<td>2.01</td>
<td>2.51</td>
<td>2.51</td>
</tr>
</tbody>
</table>

Table 3-5: Kemess HO Unblasted specific impact energies (kWh/t) for D1, D2, and D5

<table>
<thead>
<tr>
<th>Batch</th>
<th>-45 +37.5 mm</th>
<th>-31.5 +26.5 mm</th>
<th>-22.4 +19 mm</th>
<th>-16 +13.2 mm</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>0.09</td>
<td>0.25</td>
<td>0.25</td>
<td>0.33</td>
</tr>
<tr>
<td>2</td>
<td>0.25</td>
<td>1.01</td>
<td>1.02</td>
<td>1.09</td>
</tr>
<tr>
<td>3</td>
<td>0.73</td>
<td>2.38</td>
<td>2.56</td>
<td>2.60</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Batch</th>
<th>-45 +37.5 mm</th>
<th>-31.5 +26.5 mm</th>
<th>-22.4 +19 mm</th>
<th>-16 +13.2 mm</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>0.09</td>
<td>0.23</td>
<td>0.21</td>
<td>0.15</td>
</tr>
<tr>
<td>2</td>
<td>0.24</td>
<td>0.95</td>
<td>0.99</td>
<td>0.96</td>
</tr>
<tr>
<td>3</td>
<td>0.50</td>
<td>2.01</td>
<td>2.49</td>
<td>2.52</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Batch</th>
<th>-45 +37.5 mm</th>
<th>-31.5 +26.5 mm</th>
<th>-22.4 +19 mm</th>
<th>-16 +13.2 mm</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>0.08</td>
<td>0.22</td>
<td>0.20</td>
<td>0.13</td>
</tr>
<tr>
<td>2</td>
<td>0.24</td>
<td>0.94</td>
<td>0.98</td>
<td>0.95</td>
</tr>
<tr>
<td>3</td>
<td>0.49</td>
<td>1.80</td>
<td>2.52</td>
<td>2.53</td>
</tr>
</tbody>
</table>
The reason for the variance in specific impact energies between the planned specific impact energies in Table 3-3 and the actual specific energies in Table 3-4 and Table 3-5 is that the drop-weight comes to a rest at some unknown and unpredictable height above the reference point, as it is resting on the fragmented sample, as discussed in Section 3.3.2. Overall all, the variances are minor, with no systematic differences and a maximum difference in specific drop-weight energies of approximately 10%.

3.3.4 Analysis of Drop-weight Fragments

From work at the Julius Krushnitt Mineral Research Centre (JKMRC) it has been shown that for most ores, under single particle breakage tests, the daughter (i.e. resultant) products form geometrically similar distributions relative to the original size particle. The daughter products from a particle fragmented by a drop-weight impact event follow a nomenclature system where $t_n$ refers to the amount of material passing $1/n$th the size of the original particle. For instance, the amount of material passing $1/25$th of the original particle size would be referred to as the $t_{25}$ value. The original particle size was taken to be the mean of the apertures of the two sieves that defined the particle size range. A set of $t_n$ curves can be combined together to form a 'breakage map', i.e. the size distribution of the daughter products as a function of the degree of breakage. The breakage map uses the amount of $t_{10}$ material as an index to the degree of breakage. The relationship between the specific impact energy that a sample is subjected to and the percent passing $1/10$th of the original particle size has been described by Equation (15) for most ores (Napier-Munn et al., 1996).

$$t_{10} = A(1 - e^{-E/E_{in}b})$$

(15)

Where:
\[ t_{10} = \text{Percent passing 1/10th the original particle size} \]

\[ A, b = \text{Fitted constants} \]

\[ Eis = \text{Specific impact energy (kWh/t)} \]

The value for \( A \) is usually around 0.5 for brittle ores, but can vary. Generally, the values of \( A \) and \( b \) are hard to characterize, i.e. it is difficult to assign ‘typical’ values for a certain rock type. This is more easily done with more established tests such as the Bond Ball Mill Work Index (BWI) (Napier-Munn et al., 1996). The comminution properties of each particular ore tend to be represented by a unique combination of \( A \) and \( b \) values.

The \( t_{10} \) value and its relationship to specific impact energy in Equation (1) is of special interest because it is used for the prediction and modelling of SAG mills, and the value of \( A * b \) shows some correlation with the Bond Work Index and the abrasiveness index \( t_a \), which is used to characterize an ore’s susceptibility to comminution by abrasion as shown in Figure 3-7. The \( A * b \) value corresponds to the zero-energy slope of the \( t_{10} \) curve, which can be thought of as the rate at which the material is breaking at low specific drop-weight energies (Napier-Munn et al., 1996). If the material has been softened by blasting, it should show a slightly higher \( A * b \) value than the unblasted sample, suggesting that it has a lower specific grinding energy value. Thus, if there is a change in the material properties induced by blasting there should be a significant difference between the blasted and the unblasted samples' \( A \) and \( b \) values.
Figure 3-7: Correlation between Bond Work Index and abrasion parameter $t_a$ with the value of $A*b$ (Napier-Munn et al., 1996)

Plotting $t_{10}$ values for the products of the impact tests vs. their specific impact energies listed in Table 3-4 and Table 3-5 yields Figure 3-8, Figure 3-9, and Figure 3-10, to which Equation (15) was fitted using a non-linear least-squares routine. The fit indicates that the blasted samples produced higher $t_{10}$ values (i.e. finer products) than the unblasted samples at lower specific impact energies, which was the expectation. To ascertain if the differences between the blasted and the unblasted samples were statistically significant, the upper and lower 95% confidence values for $A*b$ were computed and are listed in Table 3-6, Table 3-7, and Table 3-8. Reference data for the drop weight testing can be found in Appendix B.
Figure 3-8: $t_{10}$ plot for blasted and unblasted Kemess Hypogene D1 ore

The results of the drop-weight testing is plotted in Figure 3-8, Figure 3-9, and Figure 3-10. While all 3 samples showed that the blasted material was softer, the difference was only statistically significant for the D1 sample at the 95% confidence limit, which would be consistent with the SPI testing. In the SPI testing, the D1 material showed the largest difference between the blasted and unblasted samples. Visual inspection of the D2 and the D5 samples suggest that there is a difference, but it is not significant at the 95% confidence limit. This indicates that the D1 material, in practice, could be influenced by blasting and that its behaviour under SAG milling would be affected.
Figure 3-9: $t_{10}$ plot for blasted and unblasted Kemess D2 Hypogene ore

Figure 3-10: $t_{10}$ plot for blasted and unblasted Kemess D5 Hypogene ore
Table 3-6: HO D1 blasted and unblasted parameter fitting results

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Unblasted</th>
<th>Blasted</th>
</tr>
</thead>
<tbody>
<tr>
<td>$A$</td>
<td>0.490</td>
<td>0.503</td>
</tr>
<tr>
<td>$b$</td>
<td>1.423</td>
<td>2.475</td>
</tr>
<tr>
<td>$r^2$</td>
<td>0.94</td>
<td>0.91</td>
</tr>
<tr>
<td>Initial slope (A*b)</td>
<td></td>
<td></td>
</tr>
<tr>
<td>95% lower confidence</td>
<td>0.504</td>
<td>0.809</td>
</tr>
<tr>
<td>95% upper confidence</td>
<td>0.795</td>
<td>1.396</td>
</tr>
</tbody>
</table>

Table 3-7: HO D2 blasted and unblasted parameter fitting results

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Unblasted</th>
<th>Blasted</th>
</tr>
</thead>
<tbody>
<tr>
<td>$A$</td>
<td>0.540</td>
<td>0.495</td>
</tr>
<tr>
<td>$b$</td>
<td>0.971</td>
<td>0.989</td>
</tr>
<tr>
<td>$r^2$</td>
<td>0.98</td>
<td>0.96</td>
</tr>
<tr>
<td>Initial slope (A*b)</td>
<td></td>
<td></td>
</tr>
<tr>
<td>95% lower confidence</td>
<td>0.425</td>
<td>0.367</td>
</tr>
<tr>
<td>95% upper confidence</td>
<td>0.574</td>
<td>0.565</td>
</tr>
</tbody>
</table>

Table 3-8: HO D5 blasted and unblasted parameter fitting results

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Unblasted</th>
<th>Blasted</th>
</tr>
</thead>
<tbody>
<tr>
<td>$A$</td>
<td>0.543</td>
<td>0.507</td>
</tr>
<tr>
<td>$b$</td>
<td>0.544</td>
<td>0.698</td>
</tr>
<tr>
<td>$r^2$</td>
<td>0.98</td>
<td>0.98</td>
</tr>
<tr>
<td>Initial slope (A*b)</td>
<td></td>
<td></td>
</tr>
<tr>
<td>95% lower confidence</td>
<td>0.236</td>
<td>0.280</td>
</tr>
<tr>
<td>95% upper confidence</td>
<td>0.339</td>
<td>0.403</td>
</tr>
</tbody>
</table>

There is a statistical difference between the D1 unblasted and blasted samples, but not in the D2 and the D5 samples. There may be several causes for the lack of difference between the other two samples, relating to both the nature of the test itself and the samples used:

- The most pertinent explanation is that the blast experiments did not effect any significant change (as to be observable) in the samples.

- The test itself does not appear precise enough to detect small to moderate changes in the resistance to comminution (i.e. $A$ and $b$ values). Examining Figure 3-11 provides some reinforcement for the idea that the drop weight test
may not be ideal for detecting small changes in resistance to comminution. Figure 3-11 is taken from a study published by JKMRC Fellows (Grundstrom et al., 2001), and shows some data points from a $t_{10}$ test performed for the study, and the amount of scatter indicates that the derived values for $A$ and $b$ are variable at a high level of confidence, similar to the Kemess Hypogene $t_{10}$ test results; this would suggest that the variance is not unique to the testing procedure and equipment employed at Queen’s University.

![Plot scatter from $t_{10}$ testing (Grundstrom et al., 2001)](image)

**Figure 3-11: Plot scatter from $t_{10}$ testing (Grundstrom et al., 2001)**

- The $t_{10}$ test incorporates some sampling bias if one is examining blasted or mechanically fragmented material. The test requires similar quantities at identical sizes for each test sample. Intuition suggests that for a given particle, its size would be an indicator of the amount of energy imparted in that particle. If this was the case, it would be expected that if one were to compare two samples of the same size and of the same material they would have very similar
comminution properties. Granted, the level of imparted energy and its effect may be of different quality if imparted dynamically as in blasting or more statically as in crushing, but this is beyond the scope of this study.

What can be concluded from \( t_{10} \) testing is that the actual comminution qualities for the D1 coarse SAG feed was influenced by the lab-scale blasting experiments, although not the D2 and D5 materials.

### 3.4 Ball Mill Testing

#### 3.4.1 Locked Cycle Testing

To investigate if there was any change in the specific comminution energy of the material at smaller feed sizes than tested by the MinnovEX SPI test and drop-weight testing, samples of the blasted and unblasted materials were subjected to locked cycle ball mill Bond Work Index (BWI) testing (Bond, 1961 and Napier-Munn et al., 1996). The standard feed size for the ball mill BWI test is material that has been crushed to \(-2.36\) mm, which makes it similar in size to the products of the aforementioned tests, and the feed and product 80% passing sizes for each individual test are listed in Table 3-9. The ball mill BWI is a widely used test in industry, as it has been found to agree reasonably well with observations in ‘real world’ operational mills, in the sense that it indicative of relative behaviour. It does not provide an absolute indication of the energy required to grind material. For the Bond testing, the laboratory procedure implemented was from the MINE 331 Undergraduate laboratory manual and can be found in Appendix A.
Table 3-9: Ball mill work index feed and product sizes

<table>
<thead>
<tr>
<th>Sample</th>
<th>F80 (um)</th>
<th>P80 (um)</th>
</tr>
</thead>
<tbody>
<tr>
<td>D1 Unblasted</td>
<td>1710</td>
<td>164</td>
</tr>
<tr>
<td>D1 Blasted</td>
<td>2005</td>
<td>168</td>
</tr>
<tr>
<td>D2 Unblasted</td>
<td>2090</td>
<td>168</td>
</tr>
<tr>
<td>D2 Blasted</td>
<td>1935</td>
<td>170</td>
</tr>
<tr>
<td>D5 Unblasted</td>
<td>1980</td>
<td>168</td>
</tr>
<tr>
<td>D5 Blasted</td>
<td>1850</td>
<td>162</td>
</tr>
</tbody>
</table>

The unadjusted (for efficiency and size factors) ball mill BWI test results are summarized in Figure 3-12. The blasted samples all have lower work indices than the unblasted samples, with the D1 having the largest absolute and relative difference between the two samples. There is virtually no difference between the D5 samples, and the D2 blasted material is slightly softer than the unblasted material. Mosher and Tague (2001) suggest that for a given laboratory, the precision of the BWI is between 4-13%. Conservatively assuming that the repeatability for the tests conducted is within 13%, then there is a small but significant difference between the D1 blasted and unblasted samples, whereas there is no difference between the D2 and D5 blasted samples. At the other extreme, assuming a repeatability of 4%, there is a significant difference between the D1 samples, a small difference between the D2 samples and again, no difference between the D5 samples. Possible explanations for the differences in the factors will be discussed in Chapter 7: Data for the locked cycle testing can be found in Appendix C.
3.4.2 Batch Testing

Batch ball mill testing was carried on a –2.36 +1.7 mm monosize feed fraction as described by Austin and Weller, (1982) to see if there was any change in the grinding rate of this size fraction. 700 ml of feed material was prepared for each sample of blasted and unblasted material, and then was ground in the ball mill (the same mill used for the locked cycle BWI test) for 30, 60, and 90 revolutions. At each stage, the mill was emptied, the contents screened and then returned to the mill for further grinding. The screening results at the 30, 60, and 90 intervals for blasted and unblasted D1, D2, and D5 samples are given in Figure 3-13, Figure 3-14 and Figure 3-15. The D1 and D2 unblasted samples both produced finer products than the blasted samples, which was unexpected and counter-intuitive, as the locked cycle tests indicated that the blasted
material had a lower BWI. The D5 blasted material was finer after 90 revolutions than the unblasted material, but the difference is small and probably not significant, similar to the locked cycle test results. Data for the results of the batch testing can be found in Appendix D.

![D1 Blasted Vs. Unblasted](image)

**Figure 3-13:** D1 ball mill products after 30, 60 and 90 revolutions (U=unblasted, B=blasted)
Figure 3-14: D2 ball mill products after 30, 60, and 90 revolutions (U=unblasted, B=blasted)

Figure 3-15: D5 ball mill products after 30, 60, 90 revolutions (U=unblasted, B=blasted)
Another calculation is to examine the rate of destruction of the feed size class \(-2.36 + 1.7\) mm material and compare the relative rates of grinding. Assuming that the rate of breakage is 1\(^{st}\) order (i.e. it remains constant) Equation (16) is valid (Austin and Weller 1982).

\[
\frac{dm_i(t)}{dt} = -s_i m_i(t) \tag{16}
\]

Where:

\[m_i(t) = \text{The percentage of original feed after time } t\]

\[s_i = \text{Rate of breakage}\]

For the analysis here, time was to be taken as number of revolutions turned by the mill, thus the units for \(s_i\) are minutes\(^{-1}\). The integrated solution for Equation (16) is Equation (17) (Napier-Munn et al., 1996):

\[
\log(m_i(t)) = \log(m_i(0)) - \frac{s_i t}{\ln(10)} \tag{17}
\]

Thus by plotting the log of the percent feed remaining vs. the grinding time, (in this case taken minutes) the slope of the line can be calculated and hence the grinding rate for the feed size fraction inside the ball mill can be calculated. Rate plots for the blasted and unblasted D1, D2 and D5 material samples are shown in Figure 3-16, Figure 3-17, and Figure 3-18, including the start point of the test at time zero with 100\% of the material in the original size fraction. The horizontal axis has been converted to minutes using the measured mill speed of 76 rpm, and the vertical axis the log of the percent remaining of the original amount of feed material after time \(t\). Therefore the slope of the line should be the term \(s_i/\ln(10)\) in Equation (17).
Figure 3-16: Rate plot for D1 blasted and unblasted feed samples

Figure 3-17: Rate plot for D2 blasted and unblasted feed samples
After inspecting the plots, it was decided that the data would only include the measurements taken after the test had started, since over those 3 intervals the results seem to be linear, which is consistent with the first-order grinding rate assumption. If the test start point is included (at time zero before any grinding has occurred), the 1st order rate assumption does not appear to be correct, so only the post-start data points were used for the fitting of the constant terms in Equation (17). The fitted value for the constants in Equation (17), the intercept (b) and the slope (m) are listed in Table 3-10, and the calculated rate constants \( k \) calculated by removing the constant \( 1/\ln(10) \) are listed in Table 3-11.

**Figure 3-18: Rate plot for D5 blasted and unblasted feed samples**

![Graph showing rate plot for D5 blasted and unblasted feed samples.](image)
Table 3-10: Regression results for first order ball mill batch testing

<table>
<thead>
<tr>
<th></th>
<th>Unblasted</th>
<th>Blasted</th>
</tr>
</thead>
<tbody>
<tr>
<td>D1</td>
<td></td>
<td></td>
</tr>
<tr>
<td>b</td>
<td>-0.12</td>
<td>-0.13</td>
</tr>
<tr>
<td>m</td>
<td>-0.27</td>
<td>-0.21</td>
</tr>
<tr>
<td>D2</td>
<td></td>
<td></td>
</tr>
<tr>
<td>b</td>
<td>-0.11</td>
<td>-0.08</td>
</tr>
<tr>
<td>m</td>
<td>-0.32</td>
<td>-0.34</td>
</tr>
<tr>
<td>D5</td>
<td></td>
<td></td>
</tr>
<tr>
<td>b</td>
<td>-0.12</td>
<td>-0.11</td>
</tr>
<tr>
<td>m</td>
<td>-0.30</td>
<td>-0.32</td>
</tr>
</tbody>
</table>

Table 3-11: Grinding rate constants for -2.36 +1.7 mm feed (minute⁻¹)

<table>
<thead>
<tr>
<th>Sample</th>
<th>Unblasted</th>
<th>Blasted</th>
</tr>
</thead>
<tbody>
<tr>
<td>D1</td>
<td>-0.63</td>
<td>-0.49</td>
</tr>
<tr>
<td>D2</td>
<td>-0.74</td>
<td>-0.78</td>
</tr>
<tr>
<td>D5</td>
<td>-0.70</td>
<td>-0.73</td>
</tr>
</tbody>
</table>

By inspection, The D2 and D5 blasted samples were grinding at a slightly higher rate than their unblasted counterparts, but the difference is very small. Unexpectedly, the D1 unblasted material appeared to grinding faster than the blasted material. It is difficult to determine whether the result is statistically significant, since only one round of testing was performed and the differences in the grinding rates are relatively small. It would be conservative to presume that there are no differences between the blasted and the unblasted samples. The batch ball mill test is not directly comparable to the locked cycle BWI test, since it only uses a monosize (~2.36 +1.7 mm in this case) feed material. It should however, characterize the grinding behaviour of the larger particles in the feed for the conventional Bond Ball Mill Work Index test. The D2 and D5 were similar to Bond test results, but the D1 was not. It showed that the blasted material was grinding slower than the unblasted material. This would suggest that this particular size of material was not affected by blasting. Another explanation may be that the sampling was biased.
Chapter 4: Kemess Blasting and Comminution Computer Modelling

4.1 Introduction

Fragmentation from blasting has been shown to affect the performance of downstream crushing and SAG/AG milling processes (McKee et al., 1995) and this topic is investigated in this section. The objective was to construct a realistic “Fragmentation System” as described by Hustrulid (1999) that extends from blasting to classification of the final product for the flotation circuits and investigate the impacts of changing blasting and crushing parameters, to see if any economic benefits could be achieved. The system was constructed in 3 separate parts:

1. Blasting and Crushing
2. The SAG Circuit (including the SAG mill, SAG screen and water feeder)
3. The ball mill circuit (Including ball mill, cyclones, and water feeder)

Computer modelling was done with the software package JKSimMet (JKTech, 2000), which is a steady-state comminution simulator. The computer model was based on operational data collected during a circuit survey by Tinney et al. (2003). Ultimately, they were linked in the following section allowing the sensitivities of downstream operations to upstream operations to be modelled. The reason for the construction of the model (i.e. model fitting) in 3 separate parts is because of a software limitation that only allows for the fitting of 10 model parameters in a trial, and also does not include a blasting fragmentation modelling routine. Figure 4-1 shows an overview of the circuit configuration in JKSimMet, with all the process units that were simulated and how they are connected. The blasting model, as mentioned previously, was considered separately.
Figure 4-1: JKSimMet Simulator circuit configuration overview

The model was based on operational data collected by Tinney et al. (2003), which included a circuit survey of all relevant process streams and equipment parameters. Flotation and recovery analysis, as well as re-grind material are not considered in this section as it is beyond the scope of this document. Recovery performance seems to be insensitive to throughput, as shown in Figure 4-2, so provided that material being passed to flotation remains at the nominal size, this should be a safe assumption. Figure 4-2 shows daily operational mill recoveries at various throughputs, processing Hypogene ore from domains D1-D5. The axes are intentionally left blank, as the data is proprietary.
Figure 4-2: Throughput-recovery relationship for HO ore (axes intentionally left blank)

While it is likely that changes to the comminution properties of rock as a result of blasting correlate to some degree with blast fragmentation, the relationship between blasting and work index modifications is not well understood. The relationship between blasting and grindability in Chapter 3: produced ambiguous results, thus they will not be considered and are assumed to remain constant. This may not be universally true, as some testing on controlled materials has suggested that this in fact does occur (Katsabanis et al., 2003). The goal of this section and the following is to understand how blasting fragmentation affects crushing and grinding independently of changes to the work index of the material being processed. Also, the work index of the D5 material, which is used for modelling blasting and milling did not show a significant change in response to blasting, thus the assumption that the work index should remain constant should be valid. Operationally, detecting small changes in the comminution properties of rock is
difficult, and a large degree of the success achieved by the implementation of blast optimization studies at operating mines has come from correlating blast fragmentation with downstream comminution performance. A large proportion of the literature is devoted to examining the impacts of fragmentation on throughput, as it is easier to monitor in real time as part of the process. The prediction of fragmentation based on blast design is well established and methods for predicting blasting fragmentation based on the work of Kuznetsov (1973) Cunningham (1987), and Djordjevic (1999) are widely used and accepted in industry.

For modelling of mill processes, the software package JKSimMet v5.1 (JKTech, 2000) was used. Brief descriptions of the blasting and crusher models used are provided since the variance analysis relied on making changes to parameters of those two particular models, and for all other downstream models a more thorough background can be found in Napier-Munn (1996) and JKTech (2000). Circuit and model data was taken from a grinding survey conducted by a consultant in 2003 (Tinney et al., 2003). Two circuit surveys and studies on the Kemess B Line comminution circuit (one of the two parallel circuits) were conducted in early 2003 to investigate possible circuit improvements. One study considered the mill processing relatively softer ore, a blend of D1 and D2 material, whereas the other study considered the mill processing harder D5 type ore. It was concluded in the soft ore survey that the ball mill limited mill throughput, and while processing the harder ore the rate limiting process was the SAG mill. It is logical that changes to the size distribution to ROM ore from blasting will have a larger effect on circuit performance with the harder D5 ore since the rate limiting step is farther upstream, thus this study will focus on modelling circuit performance changes due to feed fragmentation using equipment operating parameters determined in the hard ore (D5 material) study. Unless the changes to the Bond Ball Mill Work Index can be
reduced by blasting (which produced inconsistent results) it seems unlikely that throughput can be improved when processing the harder ore.

To evaluate changes of blasting and crushing parameters, a comminution ‘base case’ was constructed, and is outlined in Section 4.2. It is based on the circuit survey conducted by the mill on April 2, 2003, and represents standard blasting and comminution operations, with the nominal tonnage for the base case being 738 tonnes per hour. Changes in blasting parameters and crusher parameter changes are investigated in Chapter 5. The economic analysis of the proposed changes can be found in Section 5.4.

4.2 **Base Case Fragmentation Study**

4.2.1 **Blasting Model**

The model used for predicting rock fragmentation from blasting is the Djordjevic (1999) model, which is based on the work by Cunningham (1987) and Kuznetsov (1973). The major difference between Djordjevic’s (1999) method and previous methods of predicted fragmentation is that it is a Two Component Model (TCM). It predicts the fines component of the blast due to compressive failure around the borehole while the coarse fraction outside the immediate area of the borehole is predicted using Kuznetsov (1973) and Cunningham’s (1987) work. The prediction of fines generated by the compressive crushing action around the borehole is one of the main shortcomings of the Kuznetsov-Rammler model, which consistently under-predicts the amount of fines generated by blasting. The generation of fines from blasting plays a critical role in blasting optimization, since the fines generated can greatly help increase SAG throughput, and the two component model was selected since it does this well (Grundstrom et al., 2001). The TCM predicts the fragment sizes produced by blasting by using a weighted
combination of two Rosin-Rammler distributions as expressed in Equation (18) (Djordjevic, 1999). The 1\textsuperscript{st} term describes the coarse part of the distribution and the second the fine component of the distribution. The relative contribution to the distribution made by each of the two components is split by estimating the volume around the borehole that fails in shear compression, which is the zone primarily responsible for the contribution of fines in the muckpile. The balance of the material is assumed to undergo tensile failure, and contributes to the coarse component of the distribution.

\[
p(x) = 100 \left[ 1 - \left( 1 - F_c \right) e^{\left(-0.693 \left( \frac{x}{a} \right)^d \right)} - F_c e^{\left(-0.693 \left( \frac{x}{c} \right)^d \right)} \right]
\]  

(18)

Where:

\begin{align*}
  p(x) & = \text{Percent passing size } x \\
  F_c & = \text{Percentage of material fragmented by crushing around the borehole} \\
  a, c & = \text{Mean fragment size, 50\% passing size for the coarse fraction and the fine fraction} \\
  b, d & = \text{Uniformity exponent for the coarse and fine fraction.}
\end{align*}

By using a thick-walled cylinder failure criterion, it can be shown that for a circular opening under loading the radius of the zone undergoing shear failure is described by Equation (19) (Djordjevic, 1999).

\[
x = \frac{r}{\sqrt{\frac{24T_o}{P_b}}} 
\]  

(19)

Where:

\begin{align*}
  r & = \text{Blasthole radius}
\end{align*}
\[ T_o = \text{Tensile strength of the rock} \]
\[ P_b = \text{Borehole pressure (applied load)} \]
\[ x = \text{Ultimate radius of shear failure} \]

The applied load, in this case the pressure exerted on the borehole walls by the explosive can be estimated using Equation (20) (Persson et al., 1994).

\[ P_b = \frac{\rho VOD^2}{8} \]  
\[ (20) \]

Where:
\[ P_b = \text{Borehole pressure} \]
\[ VOD = \text{Explosive Velocity Of Detonation} \]
\[ \rho = \text{Density of the explosive} \]

Equations (21) and (22) (Cunningham, 1987) were used for modelling the coarse end of the distribution, while the coefficients \( c \) and \( d \) for a copper mine were used from Djordjevic (1999) for the prediction of the fines component of the distribution. Ideally, these coefficients would be measured in a laboratory using small-scale blasts, but due to limited sample size and practical difficulties they were not. The coefficients \( c \) and \( d \) in Equation (19) were used for fine fragmentation should provide a realistic estimate, however, and be adequate for modelling purposes.

\[ x_{50} = A \left( \frac{V}{Q} \right)^{0.8} Q^{0.167} \left( \frac{115}{E} \right)^{0.633} \]  
\[ (21) \]

Where:
\[ x_{50} = 50\% \text{ passing size (cm)} \]
\[ A = \text{Rock factor based on rock mass properties} \]
\[ V = \text{Volume of rock to be blasted (m}^3) \]
Relative weight strength (ANFO = 100)

\[ E = \left(2.2 - \frac{14B}{D}\right) \sqrt{\frac{R+1}{2} \left(1 - \frac{W}{B}\right) \frac{L}{H}} \]  

(22)

Where:

- **B** = Burden (m)
- **D** = Charge diameter (mm)
- **W** = Standard deviation of drilling accuracy
- **R** = Spacing-to-burden ration
- **L** = Charge length (m)
- **H** = Height of bench (m)

Ideally, the blasting model is calibrated to actual observed fragmentation, but due to lack of data, this was not possible. The author took some photographs of mine fragmentation, but without the benefit actually being able to screen and calibrate the data, it is difficult to infer the actual size distribution of ROM ore since the camera cannot accurately resolve material smaller than approximately 5-10 cm. There are ‘fines correction’ factors that can be applied, but without actual calibration to actual sieve data (which in itself is a very time and cost intensive task, and for this particular operation not feasible due to the remoteness and scale of the operation) they are arbitrary and thus the estimating the amount of fines generated based on the tensile strength of the material should be more accurate. A more complete discussion about the problem of fines resolution with image-analysis fragmentation software is presented by Latham et al. (2003).

The properties of the base case explosive used by the mine and relevant blasting parameters are listed in Table 4-1, which are representative of the blasting parameters used for the material that was being fed to the mill during the circuit survey.
Table 4-1: Blasting model base case parameters for crusher feed prediction

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Powder Factor (kg/m³)</td>
<td>0.584</td>
</tr>
<tr>
<td>Pattern Type</td>
<td>Equilateral</td>
</tr>
<tr>
<td>Spacing (m)</td>
<td>9.75</td>
</tr>
<tr>
<td>Burden (m)</td>
<td>8.45</td>
</tr>
<tr>
<td>Subgrade (m)</td>
<td>2</td>
</tr>
<tr>
<td>Collar (m)</td>
<td>6.5</td>
</tr>
<tr>
<td>Hole diameter (mm)</td>
<td>270</td>
</tr>
<tr>
<td>Drilling Std. Dev. (m)</td>
<td>0.6</td>
</tr>
<tr>
<td>VOD (m/s)</td>
<td>5500</td>
</tr>
<tr>
<td>Exp. Density (kg/m³)</td>
<td>1200</td>
</tr>
<tr>
<td>A (rock factor)</td>
<td>8</td>
</tr>
<tr>
<td>RWS (to ANFO)</td>
<td>87</td>
</tr>
<tr>
<td>Rock UTS (MPa)</td>
<td>5.25</td>
</tr>
</tbody>
</table>

The VOD of the explosive was measured to be 5500 m/s in a 270 mm borehole, with density and the Relative weight strength (RWS) data for the explosive was provided by the manufacturer (BXL, 2004). The official VOD of the explosive provided by the manufacturer is 5000 m/s at 200 mm, but it is common for manufacturers to be very conservative when estimating field VOD’s, which would explain the discrepancy between the two. The Kuznetsov rock factor was estimated to be 8, and the Uniaxial Tensile Strength (UTS) was measured using a point-load tester (ISRM, 1985). All other information was provided by Kemess Mines Ltd.
Using the blasting model, the base case fragmentation for ROM ore was calculated. The resulting crusher feed size distribution is shown in Figure 4-3. The 50% and 80% passing sizes are 40.5 cm and 81.3 cm, respectively, and a topsize cut of 2.0 m was assumed for the crusher feed. This data was then applied to fitting the crusher model, which is outlined in the next section.

**4.2.2 Crusher Model Fitting**

Run-of-mine ore is dumped directly from the trucks into a 1.5 x 2.3 m Metso primary crusher reducing it to -15 cm, whereupon it is fed via conveyor into a 48 hr ore stockpile (Werniuk, 2003). The crusher model was fit to the ROM fragmentation predicted by the blasting model and to mass balanced, sampled SAG feed material from the Kemess B Line SAG mill when it was determined that the line was operating at a steady state.
While there is only one crusher feeding both the Kemess A and B SAG lines, only the B line is modelled, i.e. the crusher is running at half capacity. This is valid since the crusher model breakage is not dependant on feed rate as modelled here using a single instance of operation. Ideally, with multiple detailed surveys crusher breakage sensitivities to feed rate can be established, but this is beyond the scope of this study. The crusher pendulum power will be used to set the base case for model power prediction, that is, the amount of power used is completely dedicated to crushing rock, as it were crushed in a drop-weight testing, and not accounting for efficiency factors and the no-load power consumption levels associated with the crusher. While this will not provide a ‘realistic’ power draw for the crusher, it is adequate for setting the level of power draw for the base case simulation, to which subsequent simulations can be compared and constrained by. The base case pendulum power was fitted and found to be 184.7 kW, based on the guidelines developed by Napier-Munn et al. (1996).

The crusher model is described in detail in JKTech (2000). The model was fitted as outlined by JKTech (2000) for a single set of data, using a crusher Closed Side Set (CSS) of 119 mm and an eccentric throw of 46 mm, to the blast fragmentation and the SAG feed particle size distributions. The mathematical underpinnings of model fitting, are described by Napier-Munn (1996). Whiten standard deviations were used for estimating the error on all particles size distributions and all distributions, to provide realistic estimates of the circuit survey variances. Ideally multiple samples that are taken at different times and sieved separately are used for circuit particle size distribution estimates since they allow for the true precision and quality of the data (as well as confirming the steady-state assumption) to be quantified. It has, however, been found from experience that the Whiten error estimates provide a good approximation in lieu (Napier-Munn et al., 1996).
The resulting model fit is shown in Figure 4-4, which depicts the actual feed and product size distributions with the ones that are predicted by the model after fitting. The model fit is good, with a sum of error standard deviations at 1.19, which should tend to be 1 if the data and the error estimates are in agreement and the model is appropriate (JKTech, 2000). Figure 4-4 shows that the model predictions are close to experimental data and there is no systematic problem with the fit.

![Crusher Model Fit](image)

**Figure 4-4: Crusher model fit to experimental data**

**Table 4-2: Crusher model fitted and actual operating data.** Fitted data is denoted by the column heading Fit and actual data by Exp. TPH refers to Tonnes Per Hour.

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Port</th>
<th>TPH Solids</th>
<th>TPH Solids</th>
<th>% Solids</th>
<th>% Solids</th>
<th>P80 (mm)</th>
<th>P80 (mm)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Exp</td>
<td>Fit</td>
<td>Exp</td>
<td>Fit</td>
<td>Exp</td>
<td>Sim</td>
</tr>
<tr>
<td>ROM Feed Combiner</td>
<td></td>
<td>738</td>
<td>738</td>
<td>98.11</td>
<td>98.11</td>
<td>812.9</td>
<td>812.9</td>
</tr>
<tr>
<td>Kemess Crusher</td>
<td>Product</td>
<td>738</td>
<td>738</td>
<td>98.11</td>
<td>98.11</td>
<td>104.8</td>
<td>105</td>
</tr>
</tbody>
</table>

It should be noted that the feed size data is treated as being absolute, which is why the feed size data perfectly matches the predicted feed size data for simulation. The
crusher model appears to be satisfactory, predicting the product size distribution well. For the base case, the pendulum crushing power, that is, the power calculated used strictly for crushing rocks is taken as being the base load.

4.2.3 SAG Circuit Model Fitting

The dimensions and operating data for the B line SAG mill and circuit are listed in Table 4-3. The screen data was modelled as being a separation efficiency curve, so no physical data was required since the model can be fitted. Details regarding the models used for the SAG mill and the screen are presented by Napier-Munn (1996). The $A$ and $b$ values for SAG comminution properties and mass-balanced particle distribution data samples used for fitting the SAG circuit were provided by Tinney et al. (2003).

Table 4-3: B Line SAG mill dimensions and operating data

<table>
<thead>
<tr>
<th>B SAG Mill</th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Internal Diameter</td>
<td>9.70 m</td>
</tr>
<tr>
<td>Belly Length</td>
<td>4.65 m</td>
</tr>
<tr>
<td>Centre Line Length</td>
<td>5.71 m</td>
</tr>
<tr>
<td>Feed Trunnion Diameter</td>
<td>2.15 m</td>
</tr>
<tr>
<td>Cone Angle</td>
<td>18.3 deg.</td>
</tr>
<tr>
<td>Ball Load</td>
<td>11.5 vol.%</td>
</tr>
<tr>
<td>Total Filling</td>
<td>17.5 vol.%</td>
</tr>
<tr>
<td>Speed</td>
<td>71.5 %Cs</td>
</tr>
<tr>
<td>Installed Power</td>
<td>8948 kW</td>
</tr>
<tr>
<td>Power Draw* (Balls Only)</td>
<td>5624 kW</td>
</tr>
<tr>
<td>Power Draw* (Filled)</td>
<td>6652 kW</td>
</tr>
<tr>
<td>Make-up Ball Size</td>
<td>127 mm</td>
</tr>
<tr>
<td>Grate Size</td>
<td>51 mm</td>
</tr>
<tr>
<td>Grate Open Area</td>
<td>6.90%</td>
</tr>
<tr>
<td>Pebble Port Size</td>
<td>None</td>
</tr>
<tr>
<td>Pebble Port Open Area</td>
<td>0%</td>
</tr>
</tbody>
</table>

* corrected for DCS/Multilin offset

The $A$ and $b$ values for D5 ore measured by Tinney et al. (2003) were 0.507 and 1.02 respectively, while as measured in this study were 0.542 and 0.55 for D5 unblasted ore. While they are different, the values they measured were for the particular instance of the
circuit survey, and the mill operating parameters reflect these values, thus they will be used for modelling.

Model fitting for the SAG circuit was performed in two stages since the SAG mill and screen are operating in a closed circuit and they are heavily dependant on one another. First, models were fit individually, then together as a circuit and the two fit results were compared. The individual fits should be close to the circuit fits if the fit is good, since fitting the whole circuit at once may allow for one of the models to hide or ‘absorb’ inadequacies in another model. The models were fitted to the SAG product and screen oversize particle distributions, since the screen undersize was not sampled, and it was assumed that for the screen, 100% of the water was associated with the fines. These assumptions were necessary as the software is limited to fitting only 10 parameters at once. The individual fits were close, and the sum of error standard deviations was 1.46.

Table 4-4: SAG circuit model fitting data

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Port</th>
<th>TPH Solids</th>
<th>TPH Solids</th>
<th>% Solids</th>
<th>% Solids</th>
<th>P80 (mm)</th>
<th>P80 (mm)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Exp</td>
<td>Fit</td>
<td>Exp</td>
<td>Fit</td>
<td>Exp</td>
<td>Fit</td>
</tr>
<tr>
<td>Ore Reclalm</td>
<td>Combiner</td>
<td>738</td>
<td>738</td>
<td>98.11</td>
<td>98.11</td>
<td>104.8</td>
<td>104.5</td>
</tr>
<tr>
<td>SAG B Screen</td>
<td>Oversize</td>
<td>145.4</td>
<td>146.1</td>
<td>98.99</td>
<td>100</td>
<td>39.98</td>
<td>38.47</td>
</tr>
<tr>
<td>SAG B Screen</td>
<td>Undersize</td>
<td>738</td>
<td>738</td>
<td>59.06</td>
<td>59.08</td>
<td>3.75</td>
<td>3.709</td>
</tr>
<tr>
<td>BSide Sag</td>
<td>Product</td>
<td>883.4</td>
<td>884.1</td>
<td>63.2</td>
<td>63.37</td>
<td>9.153</td>
<td>10.27</td>
</tr>
</tbody>
</table>

Inspection of Table 4-4 and Figure 4-5 show that the model fits appear to be in good agreement with the experimental data. The experimental data for the SAG screen undersize shown in Figure 4-5 are for the mass balanced data, which was not used in fitting, but provide a reference for what the screen undersize should be. Solids and water flow stream rates are in good agreement with mass-balanced values and all model values appear realistic, with the fitted circulating load being in good agreement with the observed load.
Figure 4-5: SAG circuit model fit to mass-balanced data

Figure 4-6 shows the SAG breakage rate as a function of size with a dip for material approximately 25-50 mm in size, which is typical of a breakage rate curve (Napier-Munn et al., 1996). The extremely high breakage rate for large material in Figure 4-6 is normal for a mill with a high steel charge 11.5%, as in this case. The dip in the breakage rates observed at the 25-50 mm size is typical of SAG mills. Particles in this size are too large to be broken easily by the grinding media and other particles in the mill and are small enough so that they do not have enough energy to break themselves easily. Calculated mill power draw is 6479 kW versus 6652 kW and calculated filling is 17.86% vs. 17.5% as measured.
Figure 4-6: SAG mill breakage rate curve

Table 4-5: SAG circuit fitted model parameters

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Parameter</th>
<th>Fitted Value</th>
<th>Fit SD</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sag Mill Water Feeder</td>
<td>Exp New W Addin</td>
<td>496.9</td>
<td>34.39</td>
</tr>
<tr>
<td>SAG B Screen</td>
<td>Alpha</td>
<td>12.14</td>
<td>1.006</td>
</tr>
<tr>
<td>SAG B Screen</td>
<td>D50c</td>
<td>14.37</td>
<td>0.29</td>
</tr>
<tr>
<td>BSide Sag</td>
<td>xg</td>
<td>63.45</td>
<td>1.269</td>
</tr>
<tr>
<td>BSide Sag</td>
<td>xm</td>
<td>0.49</td>
<td>0.704</td>
</tr>
<tr>
<td>BSide Sag</td>
<td>BrConst1</td>
<td>2.039</td>
<td>0.177</td>
</tr>
<tr>
<td>BSide Sag</td>
<td>BrConst2</td>
<td>-0.854</td>
<td>0.247</td>
</tr>
<tr>
<td>BSide Sag</td>
<td>BrConst3</td>
<td>-0.0121</td>
<td>0.187</td>
</tr>
<tr>
<td>BSide Sag</td>
<td>BrConst4</td>
<td>1.099</td>
<td>0.429</td>
</tr>
<tr>
<td>BSide Sag</td>
<td>BrConst5</td>
<td>0.236</td>
<td>0.817</td>
</tr>
</tbody>
</table>

A summary of the model fit parameters is listed in Table 4-5. The SAG Screen Alpha refers to a parameter that describes the separation efficiency curve. Over all, the model fit provides a realistic and plausible model of the B line SAG circuit.
4.2.4 Ball Mill Circuit Model Fitting

The procedure for fitting the equipment operating in the B Line ball mill circuit was similar to that for the SAG Circuit, with the ball mill and the cyclones being fitted individually, and then in closed circuit to ensure that one model wasn’t compensating for problems with the other. The model used for the ball mill was the standard perfect mixing model, and for the cyclones the Nageswararao model, both of which are described in Napier-Munn et al. (1996). The Bond ball mill work index value used was taken from Tinney et al. (2003), and cyclone and ball mill dimensions and operating data is listed in Table 4-6 and Table 4-7. The ball mill model does not predict the power draw of the mill, but given that the measured power of the mill is substantially less than available maximum power (> 1 MW difference), it is unlikely that the power consumption of the mill will vary drastically unless there are large changes to its operating parameters.

Table 4-6: B line ball mill dimensions and operating data

<table>
<thead>
<tr>
<th>B Ball Mill</th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Internal Diameter</td>
<td>6.26 m</td>
</tr>
<tr>
<td>Belly Length</td>
<td>10.87 m</td>
</tr>
<tr>
<td>Centre Line Length</td>
<td>12.42 m</td>
</tr>
<tr>
<td>Discharge Trunnion Diameter</td>
<td>2.09 m</td>
</tr>
<tr>
<td>Cone Angle</td>
<td>20.4 deg.</td>
</tr>
<tr>
<td>Ball Load</td>
<td>25.0 vol.%</td>
</tr>
<tr>
<td>Total Filling</td>
<td>28.7 vol.%</td>
</tr>
<tr>
<td>Speed</td>
<td>73.1 %Cs</td>
</tr>
<tr>
<td>Installed Power</td>
<td>8948 kW</td>
</tr>
<tr>
<td>Power Draw* (Balls Only)</td>
<td>n/a</td>
</tr>
<tr>
<td>Power Draw* (Filled)</td>
<td>7785 kW</td>
</tr>
<tr>
<td>Make-up Ball Sizes</td>
<td>50 wt.% 51 mm</td>
</tr>
<tr>
<td></td>
<td>50 wt.% 64 mm</td>
</tr>
</tbody>
</table>
Table 4-7: Cyclone dimensions and operating data

<table>
<thead>
<tr>
<th>B Cyclone Data</th>
</tr>
</thead>
<tbody>
<tr>
<td>Inlet Diameter</td>
</tr>
<tr>
<td>Cylinder Diameter</td>
</tr>
<tr>
<td>Cylinder Length</td>
</tr>
<tr>
<td>Vortex Diameter</td>
</tr>
<tr>
<td>Apex Diameter</td>
</tr>
<tr>
<td>Cone Angle</td>
</tr>
<tr>
<td>Number Op.</td>
</tr>
</tbody>
</table>

The fit is good, with a sum of standard deviations of 1.06 for the model parameter fit, and the model fits agree well with the experimental data as shown in Figure 4-7 and appear to be realistic. Table 4-8 summarizes the fitting of the ball mill circuit, with process parameters being in reasonable agreement. The model slightly under predicts both the cyclone overflow and underflow 80 % passing sizes, but the difference between the mass-balanced and fitted data are both well within one sieve size interval.

Table 4-8: Ball mill circuit model fitting results

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Port</th>
<th>TPH Solids</th>
<th>TPH Solids</th>
<th>% Solids</th>
<th>% Solids</th>
<th>P80 (mm)</th>
<th>P80 (mm)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Exp</td>
<td>Fit</td>
<td>Exp</td>
<td>Fit</td>
<td>Exp</td>
<td>Fit</td>
</tr>
<tr>
<td>B Side Ball Mill</td>
<td>Product</td>
<td>2176</td>
<td>2176</td>
<td>71.72</td>
<td>71.72</td>
<td>0.515</td>
<td>0.492</td>
</tr>
<tr>
<td>Cyclone B Sump</td>
<td>Product</td>
<td>2914</td>
<td>2914</td>
<td>52.87</td>
<td>52.87</td>
<td>0.878</td>
<td>0.82</td>
</tr>
<tr>
<td>B Side Cyclone</td>
<td>Underflow</td>
<td>2176</td>
<td>2176</td>
<td>71.72</td>
<td>71.72</td>
<td>1.498</td>
<td>1.351</td>
</tr>
<tr>
<td>B Side Cyclone</td>
<td>Overflow</td>
<td>738</td>
<td>738</td>
<td>29.78</td>
<td>29.78</td>
<td>0.134</td>
<td>0.122</td>
</tr>
</tbody>
</table>
Figure 4-7: Ball circuit model fit to mass-balanced data

Figure 4-8: Ball mill breakage R/D* curve

The breakage rate curve shown in Figure 4-8 appears normal for a ball mill, and the fitted cyclone operating data is very close to the measured cyclone operating data in
Table 4-9. Table 4-10 summarizes the ball mill circuit fitted model parameters. The fit SD’s are low, indicating a good fit and that the models are appropriate.

**Table 4-9: Fitted and experimental cyclone operating data**

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Experimental</th>
<th>Calculated</th>
<th>Wtd. Error</th>
</tr>
</thead>
<tbody>
<tr>
<td>Water Split To O/F (%)</td>
<td>67</td>
<td>66.97</td>
<td>0.0102</td>
</tr>
<tr>
<td>Corrected D50, mm (Total)</td>
<td>0.156</td>
<td>0.155</td>
<td>0.0077</td>
</tr>
<tr>
<td>Operating Pressure, kPa</td>
<td>95.1</td>
<td>94.52</td>
<td>0.0585</td>
</tr>
</tbody>
</table>

**Table 4-10: Ball mill and cyclone circuit fitted data**

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Parameter</th>
<th>Fitted Value</th>
<th>Fit SD</th>
</tr>
</thead>
<tbody>
<tr>
<td>B Side Cyclone</td>
<td>KD0</td>
<td>1.55E-04</td>
<td>1.13E-06</td>
</tr>
<tr>
<td>B Side Cyclone</td>
<td>KQ0</td>
<td>678.8</td>
<td>12.19</td>
</tr>
<tr>
<td>B Side Cyclone</td>
<td>Alpha</td>
<td>3.197</td>
<td>0.18</td>
</tr>
<tr>
<td>B Side Cyclone</td>
<td>Cal WS</td>
<td>66.97</td>
<td>0.0239</td>
</tr>
<tr>
<td>B Side Ball Mill</td>
<td>In R/D1</td>
<td>1.103</td>
<td>0.0271</td>
</tr>
<tr>
<td>B Side Ball Mill</td>
<td>In R/D2</td>
<td>3.403</td>
<td>0.031</td>
</tr>
<tr>
<td>B Side Ball Mill</td>
<td>In R/D3</td>
<td>3.662</td>
<td>0.167</td>
</tr>
<tr>
<td>Balll Water Feeder</td>
<td>Exp New W Addin</td>
<td>1229</td>
<td>0.889</td>
</tr>
</tbody>
</table>

A summary of the model fit parameters is listed in Table 4-9. Solids and water flow stream rates are in good agreement with mass-balanced values and all model values appear realistic, with the fitted circulating load being in good agreement with the observed load. Overall, the ball circuit model fit provides a realistic and plausible model of the B line ball mill circuit.

**4.3 Base Case Combined Circuit Simulation Fit**

Once the models and fits from the 3 previous sections were completed, they were simulated as one unified side of the mill, and the results are summarized in Figure 4-9 and Table 4-11.
Figure 4-9: Unified B Line simulation predictions for the base case

The fits appear to be similar to the mass-balanced streams with the exception of the cyclone underflow, which as simulated is too fine at small sizes (< 200 \( \mu \) ). This is not a critical concern since for modelling purposes the crusher and SAG mills will be the two units most affected and there shouldn’t be any major impact on the ball mill circuit, and the accurate prediction of the cyclone overflow stream will be the critical product
produced by the ball circuit, which is predicted reasonably well. Also, it is expected that the ball mill circuit would show the largest aberrations from mass-balance data since it is farthest downstream from the actual source, and therefore its feed stream subjected to the largest cumulative error resulting from less-than-perfect simulation predictions from upstream operations. The predicted power for the SAG mill is 6491 kW vs. 6652 kW as measured, and filling is 17.95% vs. 17.5% as measured, similar to the fit result. The model appears to be robust, with fitted parameters being realistic and simulated values close to their measured values. Most importantly, it should respond to changes in the feed size in a realistic manner. The following chapters investigate changes to blasting and crushing parameters using the model described here.
Chapter 5: Effects of Changing Blasting Parameters on Circuit Operations

5.1 Introduction

The objective of this section was to investigate the effect of changing the Powder Factor (PF) and crusher Closed Side Set (CSS) settings on circuit performance. The goal of the adjustments was to increase circuit throughput without violating any processing constraints. A two factorial process was used for determining the different throughput capacities of the circuit operating under different feed size distributions (changed by altering the PF) and crusher CSS settings.

5.2 Constraints for Modelling

In order to assess the possibility of throughput increases within the SAG and ball mill circuits, as a result of different feed sizes, constraints need to be set to reflect the practical upper limits for production. The software itself will accept any feed rate that is mathematically possible within a very narrow set of pre-defined constraints (such as cyclone roping). It will not automatically determine what the maximum feed rate is, given a set of constraints; this must be determined manually, through a process of trial and error, given specific equipment constraints. Generally speaking, the constraints will be such that no particular piece of process equipment will operate beyond the base case simulation values, e.g. the SAG filling cannot exceed 17.95% as in the base case simulation.

The constraints for the simulations are as follows:

- Crusher calculated pendulum power is restricted to a maximum of 184.7 kW, as in the base case simulation.
• SAG simulated power is restricted to 8948 kW (maximum installed power) and the maximum volumetric filling is restricted to 17.95%, as simulated in the base case.

• Cyclone pressure is limited to 138 kPa as indicated by Tinney et al. (2003) indicating the practical maximum for cyclone operating pressure.

• Recirculating load for the SAG circuit is to be capped at 25% of the SAG feed rate (Tinney et al., 2003)

• Cyclone underflow density is limited to 80 % solids, as this is the predicted limit (by the software) where roping will occur.

• Final product 80% passing size is limited to the base case simulation size of 0.125 mm +/- 0.009, since this is the divergence from the original value.

In accordance with these constraints, throughput increases will be examined until one of the previous conditions is violated; this will be considered the maximum possible throughput.

5.3 Factorial experiments: Blasting

Fragmentation and Crusher Closed Side Set

Changes

This section investigates the changes induced in the whole comminution chain by making changes to the blasting Powder Factor (PF) and the crusher Closed Side Set (CSS). Three different powder factors (not including the base case) were examined in the JKSimMet model of the Kemess B Line, with one being lower than the base case of 0.584 kg/m³ and the other two higher. The feed size distributions (including the base case) generated using the two-component blasting model are shown in Figure 5-1, and
summary data is listed in Table 5-1. It should be noted that the ‘n’ factor listed in Table 5-1 is for the coarse fraction of the distribution, which describes the majority of the distribution by mass; the distribution itself is a weighted sum of two Rosin-Rammler curves, and cannot be characterized by a single uniformity coefficient.

Table 5-1: Summary of feed size distributions for the crusher produced at different powder factors by the blasting model.

<table>
<thead>
<tr>
<th>PF (kg/m³)</th>
<th>P50 (mm)</th>
<th>P80 (mm)</th>
<th>n</th>
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<tr>
<td>0.4</td>
<td>56.3</td>
<td>114.3</td>
<td>1.25</td>
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<tr>
<td>0.584</td>
<td>40.5</td>
<td>81.3</td>
<td>1.30</td>
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</table>
To select different operating scenarios to see whether an increase in throughput was obtainable, all combinations of feed blasted powder factor and crusher closed side set were examined at the nominal base case throughput of 738 Tonnes Per Hour (tph). Combinations where there appeared to be potential for throughput increases would then be tested further. Table 5-3 shows the difference between the operating parameters selected to be constraints, with a negative value indicating that the computed process parameter value is lower than the base case simulated value, which are shown by the light shading, indicating that for that process parameter slack capacity exists. The base case process parameter (trial 5) is dark-shaded. The exceptions are the cyclone operating pressure, SAG power consumption and the cyclone overflow (final product size), which are stated in absolute terms. Trials selected for throughput increase are numbers 7, 8, 10 and 11 since all process parameters are below those of the base case or not appreciably different than base case, indicating that there is slack capacity in the system to handle more throughput. Four scenarios (as shown in Table 5-3) were selected for throughput increase trials.

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<tr>
<td>0.75</td>
<td>32.4</td>
<td>65.0</td>
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<td>1</td>
<td>24.7</td>
<td>50.4</td>
<td>1.35</td>
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Table 5-2: Process parameter description for throughput trials

<table>
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<th>Parameter</th>
<th>Description</th>
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<tr>
<td>PF (kg/m³)</td>
<td>Feed blasted powder factor</td>
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<tr>
<td>CSS (mm)</td>
<td>Crusher Closed Side Set</td>
</tr>
<tr>
<td>B SAG Filling (%)</td>
<td>B SAG Filling difference from base case. A positive value indicates that filling is below base case and there is extra capacity</td>
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<tr>
<td>B Crusher Power (%)</td>
<td>Crusher power consumption relative to base case. A positive value indicates that power consumption is less than the base case.</td>
</tr>
<tr>
<td>B SAG Power (kW)</td>
<td>B SAG mill power consumption</td>
</tr>
<tr>
<td>B SAG Recirc. (%)</td>
<td>B SAG mill recirculating load</td>
</tr>
<tr>
<td>B Ball Recirc. (%)</td>
<td>B Ball mill recirculating load</td>
</tr>
<tr>
<td>B Cyclone Pressure (kPa)</td>
<td>B Cyclone operating pressure</td>
</tr>
<tr>
<td>B Cyclone Overflow P80 (mm)</td>
<td>B Cyclone overflow (final product) size</td>
</tr>
</tbody>
</table>

The PF and CSS combinations that were selected for examining throughput increases were calculated using incremental steps of 10 tph increases in throughput starting at 750 tph. The throughput was then raised until one or more of the operating constraints was violated. Water addition for the SAG and ball mill water feeders was raised linearly with the throughput increases to keep slurry densities approximately the same as in the base case. The last trial where all of the constraints were respected was considered to be the maximum throughput produced with the given PF and CSS values.

Table 5-4 summarizes the results from the throughput-increase trials, with the white shaded cells indicating interim steps (trials 12, 13, and 18-21) the dark-shaded trials indicating the maximum tested throughput for a given PF/CSS combination (trials 14, 22, and 23) and the light shaded cells unfeasible trials that violated one or more of the stated process constraints (trials 15, 16, and 24). Three of the four scenarios tested for a potential throughput increases indicated the possibility for such an increase.
| Table 5.4: Summary of throughput increases trials at varying PF and CSS combinations |
|----------------------------------|-----------------|-----------------|-----------------|-----------------|-----------------|-----------------|-----------------|-----------------|-----------------|
| Trial Number                    | CSS (mm)        | Pf (kg/m)       | Tonnage per Hr  | Refill (%)      | B SAG Power (%) | B Crusher (%)   | B SAG (%)       | B Cyclone Overflow (%) | B Cyclone Pressure (kPa) |
|---------------------------------|-----------------|-----------------|-----------------|-----------------|-----------------|-----------------|-----------------|--------------------------|
| 0.125                            | 0.750           | 0.0750          | 110             | 70%            | 9.82             | 10.00           | 9.98            | 10.00                     |
| 0.750                            | 1.000           | 1.000           | 110             | 70%            | 9.82             | 10.00           | 9.98            | 10.00                     |
| 1.000                            | 1.750           | 1.750           | 110             | 70%            | 9.82             | 10.00           | 9.98            | 10.00                     |
| 1.750                            | 2.000           | 2.000           | 110             | 70%            | 9.82             | 10.00           | 9.98            | 10.00                     |
| 2.000                            | 2.500           | 2.500           | 110             | 70%            | 9.82             | 10.00           | 9.98            | 10.00                     |
| 2.500                            | 3.000           | 3.000           | 110             | 70%            | 9.82             | 10.00           | 9.98            | 10.00                     |
| 3.000                            | 3.500           | 3.500           | 110             | 70%            | 9.82             | 10.00           | 9.98            | 10.00                     |
| 3.500                            | 4.000           | 4.000           | 110             | 70%            | 9.82             | 10.00           | 9.98            | 10.00                     |
| 4.000                            | 4.500           | 4.500           | 110             | 70%            | 9.82             | 10.00           | 9.98            | 10.00                     |
| 4.500                            | 5.000           | 5.000           | 110             | 70%            | 9.82             | 10.00           | 9.98            | 10.00                     |
| 5.000                            | 5.500           | 5.500           | 110             | 70%            | 9.82             | 10.00           | 9.98            | 10.00                     |

Table 5.3: Powder factor increases and CSS settings selected for further investigation.
The two limiting factors for throughput as examined were either SAG filling or crushe
power. The maximum throughputs for trials 14, 22, and 23 that could support a
significant throughput increase above the nominal throughput of 738 tph were 770 tph at
a PF of 0.75 kg/m$^3$ and a CSS of 110 mm, 790 tph at a PF of 1.0 kg/m$^3$ at a CSS of 110
mm, and 750 tph at the same PF at the nominal CSS 119 mm. The prevailing circuit
conditions are summarized in Figure 5-2 (trial 14), Figure 5-3 (trial 22) and Figure 5-4
(trial 23), which compare the simulated particle size distributions with the real observed
mass-balanced data. The simulated particle size distributions and are not drastically
different from the observed mass-balanced data. Figure 5-2 and Figure 5-3 have a
slightly finer crusher product over the entire distribution, whereas the scenario depicted
in Figure 5-4 has a crusher product that is finer at smaller particle sizes but is relatively
unchanged at the coarser end of the distribution, which may explain the modest increase
in throughput.

Figure 5-2: Combined circuit simulation for 770 tph @ PF=0.75 kg/m$^3$, CSS=110 mm
Cumulative Passing (%)

Figure 5-3: Combined circuit simulation for 790 tph @ PF=1.0 kg/m³, CSS=110 mm

Figure 5-4: Combined circuit simulation for 750 tph @ PF=1.0 kg/m³, CSS=119 mm
Examining the crusher product size differences relative to each other depicted in Figure 5-5, it becomes apparent that the main difference between the base case trial and the trials that produced throughput increases of 770 tph and 790 tph is that they seemed to be facilitated by a finer crusher product distribution between 150 and 20 mm. This would be expected behaviour for this mill running under a high steel load (11.5%), which causes it to behave more like a ball mill and respond positively to reducing feed size coarseness. Much of the available grinding energy in the mill is being created by balls, and is less dependent on coarse feed for grinding media, as would be the case under a low steel charge or in a fully autogenous mill (Napier-Munn et al., 1996).

Comparing the base case with the 750 tph trial, the coarse fraction of the distribution is very similar to the base case, but is finer at smaller sizes (<20 mm), which would indicate that the contribution of fines generated in blasting to the overall SAG throughput is minimal, even at a PF of 1.0 kg/m$^3$.

Cyclone pressure in all trials begins to rise slightly with increasing throughputs, peaking at 106.3 kPa at 790 tph, but is well below the maximum pressure of 138 kPa. The cyclone overflow product begins to increase slightly, to a predicted maximum of 80% passing 0.127 mm at 790 tph, although the change is small (0.002 mm) and it is doubtful if it is meaningful.
Figure 5-5: Comparison of the crusher product sizes at maximum throughput scenarios
(Note: the x-axis has been plotted linearly to highlight differences in the distributions)

Examining the feed and product streams summarized in Table 5-5, there is little deviation from the base case simulation, similar to the results from the particle size distribution graphs for the circuit streams. Slurry percent solids are all very similar since the water feeders for the SAG and ball mill circuit were adjusted to reflect the increasing throughput. There is an absolute increase in the ball mill recirculating load of 152 tonnes, which is relatively small compared to the total circulating load and expressed as a percentage, a very small increase. The pulp density of the ball mill decreased slightly as well, which according to the Morrell model for tumbling mill power prediction indicates that the ball mill power draw would drop slightly (Napier-Munn, 1996), not making it a concern.
Examining the results in Table 5-4, it would appear that for the maximum throughput trial at a PF of 1.0 kg/m$^3$ and CSS of 110 mm (trial 22) there is potential for a further increase in throughput. The constraint that is violated causing the maximum to be 790 tph is the SAG filling level, which exceeds the base case simulation. There is however, a significant amount of crusher power available, which could be used to reduce the SAG feed further. Reducing the CSS to 100 mm could potentially reduce the SAG load to acceptable levels and allow more increases in throughput.

The trial for increased throughput at a PF of 1.0 kg/m$^3$ and a CSS of 100 mm was performed in the same manner as the first set, by increasing throughput in 10 tph increments until one of the operating constraints were violated. Table 5-7 shows these interim steps as being trials 1 and 2. The maximum throughput before an operating constraint was violated was 820 tph, with the constraining factor being the available crusher pendulum power.
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Table 5: Operating conditions summary for max throughput trials
Figure 5-6: Stream particle sizes at 820 tph, PF = 1.0, CSS = 100 mm

The final maximum throughput is summarized by trial 3 in Table 5-7. The size differences between this trial and the real mass-balanced data are quite significant, shown in Figure 5-6. The crusher product is finer than in all other trials, and the SAG screen undersize and SAG product are considerably finer than the base simulation case and the mass-balanced data. Operational data for this trial are summarized in Table 5-6 and Table 5-7. Again, in Table 5-6 unshaded cells indicate interim steps up to a maximum throughput, light shaded cells indicate that the trial exceeded a process parameter maximum and dark shaded cells indicate a maximum throughput trial that does not violate any process parameters. For the SAG circuit, there is a significant decrease in the SAG filling and the SAG circulating load relative to the base case, the result of the SAG product being much finer than in all other previous simulations. The
SAG screen oversize decreases very slightly, but the undersize is much finer, with the 80% passing size reduced to 2.62 mm from 3.67 mm, resulting in a finer feed for the ball mill circuit and the ball mill itself.

Examining Figure 5-6, the ball mill recirculating load decreases from the base case simulation, which suggests that the ball mill power requirements should be slightly lower than in the base case simulation and should not be a concern. The cyclone overflow size is slightly finer than the base case simulation at 0.121 mm. The cyclone pressure increases with increasing throughput, to a value of 108.2 kPa, but this is still well within the practical operating limit of 138 kPa for cyclone pressure.

Two other simulations involving changes to blasting practices are also introduced in Table 5-6 to evaluate their impact on mill throughput. The first listed is a change in the drill diameter to 311 mm (trial 5), from the base case diameter of 270 mm while keeping the PF constant at the base case value of 0.584 kg/m$^3$, and the second is shortening of the blast hole collar to 5 m from the base case value of 6.5 m (trial 6), again, keeping the PF at the base case value. The crusher feed sizes obtained by changing these parameters in the blasting model are shown in Figure 5-7. In both instances throughput and crusher CSS were left at base case levels. The goal was not to quantify how much throughput was reduced by, but to determine whether the effect of introducing such changes would be negative or not. Both changes indicated a negative impact on throughput, with both requiring increases in crusher power, and an increase in SAG filling for the 5 m collar to accommodate the base case throughput of 738 tph.
In conclusion, it is technically feasible to increase throughput as modelled. Only a very small (~12 tph) throughput increase was found by solely increasing the PF without making any adjustments to the crusher CSS. The economic implications of this increase are examined in the next section.
### Table 5-7: Operating conditions summary for max throughput trials

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<th>Operating Conditions</th>
<th>Tph</th>
<th>CSS (mm)</th>
<th>PF (%)</th>
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<tr>
<td>1</td>
<td>738</td>
<td>6 780</td>
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<tr>
<td>2</td>
<td>810</td>
<td>8 800</td>
<td>1.000</td>
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<tr>
<td>3</td>
<td>820</td>
<td>3 820</td>
<td>1.000</td>
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<tr>
<td>4</td>
<td>830</td>
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<td>5</td>
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<tr>
<td>6</td>
<td>900</td>
<td>6 900</td>
<td>1.000</td>
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### Table 5-6: Summary of throughput increases trials at PF=1.0 kg/m³ CSS = 100 mm and other blasting changes

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<th>107</th>
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### Table 5-5: Summary of throughput increases trials at PF=1.0 kg/m³ and other blasting changes

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Substantial throughput increases were possible only when a reduction in the crusher CSS was facilitated by higher levels of ROM ore fragmentation. This would agree with Adam and Sidall (1998) who found that changes to ROM ore fragmentation had a small effect on throughput unless crusher setting changes were made. They found that the crusher largely negated changes in the feed size, and that the critical item for increasing throughput was to make the SAG feed finer, which was largely controlled by the crusher settings.

5.4 Economic Analysis

For economic evaluation, two trials from the previous section will be examined, as by inspection, there are only two that are candidates for improvements. The first is 770 tph at a CSS of 110 mm and a PF of 0.75 kg/m$^3$ and the other is 820 tph at a CSS of 100 mm at a PF of 1.0 kg/m$^3$. One presents a lower blasting cost, whereas the other offers a higher rate of mill production (throughput). All financial numbers are presented relative to the base case, as actual cost and profit values are proprietary and not published. For economic analysis, two metrics are examined: cost per tonne processed and overall revenue. Table 5-8 shows the base case blasting parameters. The 2 trials examined for economic evaluation are summarized in Table 5-9, with trial #1 being the base case, and numbers 2 and 3 being the two maximum throughputs calculated.
Table 5-8: Base case blasting parameters for cost analysis

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<tr>
<td>Bench Height</td>
<td>15 m</td>
</tr>
<tr>
<td>Qe Per Hole</td>
<td>721.4 kg</td>
</tr>
<tr>
<td>Spacing</td>
<td>9.8 m</td>
</tr>
<tr>
<td>Burden</td>
<td>8.4 m</td>
</tr>
<tr>
<td>Rock Type</td>
<td>HO</td>
</tr>
<tr>
<td>Rock Density</td>
<td>2790 kg/m³</td>
</tr>
<tr>
<td>Volume Rock Broken</td>
<td>1235.3 kg/m³</td>
</tr>
<tr>
<td>Tonnes rock Broken</td>
<td>3446.5 t/hole</td>
</tr>
<tr>
<td>Drill meters</td>
<td>17 m/hole</td>
</tr>
<tr>
<td>Drill Meters per tonne</td>
<td>0.00493 m/t</td>
</tr>
</tbody>
</table>

Table 5-9: Summary of trials selected for economic analysis

<table>
<thead>
<tr>
<th>Trial</th>
<th>1</th>
<th>2</th>
<th>3</th>
</tr>
</thead>
<tbody>
<tr>
<td>Powder Factor (kg/m³)</td>
<td>0.584</td>
<td>0.750</td>
<td>1.000</td>
</tr>
<tr>
<td>Crusher Closed Side Set (mm)</td>
<td>119</td>
<td>110</td>
<td>100</td>
</tr>
<tr>
<td>Throughput (tph)</td>
<td>738</td>
<td>770</td>
<td>820</td>
</tr>
</tbody>
</table>

Figure 5-8 shows the relative unit costs for the 2 increased throughput scenarios. There is an increase in the mining unit cost, as expected, due to higher drill and blast costs, but this is more than offset by the decreased milling unit cost, resulting in a very slightly decreased overall unit cost for processing and mining. The reason for the unit milling cost declining is due to the lessening influence of fixed costs. Both of the higher throughput trials are favourable, yielding reduced unit costs. Table 5-10 summarizes the factors that were included in the economic analysis for cost. The methodology applied broke up the mining and milling costs into two types, fixed and variable.
Figure 5-8: Relative unit costs for mining and milling compared to base case costs

Table 5-10: Factors considered in economic analysis

<table>
<thead>
<tr>
<th>Business Area</th>
<th>Item Type</th>
<th>Item</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mining</td>
<td>OPEX Fixed</td>
<td>Assay Lab</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Metallurgy</td>
</tr>
<tr>
<td></td>
<td></td>
<td>G &amp; A</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Power Distribution</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Substations</td>
</tr>
<tr>
<td></td>
<td>OPEX Variable</td>
<td>Tailings</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Crusher</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Reclaim</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Flotation</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Dewatering</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Grinding</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Reclaim Water</td>
</tr>
<tr>
<td>Milling</td>
<td>OPEX Fixed</td>
<td>Payroll</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Misc.</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Maintenance</td>
</tr>
<tr>
<td></td>
<td>OPEX Variable</td>
<td>Power</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Consumables</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Steel</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Liners</td>
</tr>
</tbody>
</table>
Fixed variables are not dependant on the amount of material being mined and processed, while variable costs do and were assigned a value of dollars per ton and were scaled according to the tonnage being processed in each scenario. The two were then summed and divided by the amount of tonnes processed to yield the unit cost for each scenario. For examining the revenue effects of increasing throughput, a net smelter return was used which incorporates realistic feed grades and metal prices at the time of writing, late 2004.

Figure 5-9 shows the rise in revenue relative to the base case, assuming a fixed NSR and constant gold and copper recovery rates. The increase in revenue is substantial, with the highest case showing an increase in revenue over 10%. To provide an idea what this would mean for the company, the base case daily throughput is 35,424 tpd (738 tph * 2 lines * 24 hrs), whereas under the maximum throughput scenario of 820 tph the daily throughput is 39,360 tpd for a daily difference in throughput of 3,936 tpd. In the 2003 annual report (Northgate Exploration, 2003) it states that for each 500 tpd increase in throughput, annual cash flow increases 1.5 million US dollars.
Figure 5-9: Daily revenue relative to base case

It appears that benefits from increasing throughput by adjusting PF and CSS are economically desirable, with the two maximum throughputs of 770 tph and 820 tph both indicating positive financial benefits, with the latter having the greatest of the two trials.
Chapter 6: Preliminary Investigation into the Effect of Blasting on the Comminution Properties of Other Materials

As shown here, the Kemess Hypogene ore showed limited response in grinding properties due to blasting, but this is not necessarily true for other materials. Eloranta (1999) provides a brief summary of studies that have concluded (some of which are discussed in this study) that changes to the work index in rock is possible. Katsabanis et al. (2003) found small reductions in the work index of granodiorite and the sonic p-wave velocities of the rock were reduced by blasting in small-scale experiments. To help guide the future direction for research, taconite, granodiorite and limestone were subjected to a partial $t_{10}$ test, to see if they showed any response to blasting in SAG mill feed sized particles. These particular rock types were chosen as samples with very low variability are easy to procure, and this makes them ideal for scientific tests. The test was not a full $t_{10}$ test intended to fully characterize the comminution properties of SAG mill like the tests in Section 3.3, but rather to see if there is a difference in blasted rocks vs. unblasted rocks in the $t_{10}$ value at constant drop-weight impact energy. Only 4-5 size-energy sample combinations were used and screened (as opposed to the full 12 for a full $t_{10}$ characterization), at a variety of powder factors to see if there was an indication of some change in the material, and whether future research would be of interest. Also, it would provide an indicator as to how the $t_{10}$ value could be affected by the powder factor, i.e. the level of blast energy that the rock was subjected to. For details regarding testing, the reader is referred to Section 3.3.
6.1 Taconite Drop Weight Tests

The work of Nielsen and Kristiansen (1996) indicated that the Bond ball mill work index of taconite was influenced by blasting. Two samples of taconite from an iron ore mine in the Mesabi range in Minnesota were tested, labelled # 15 and # 16 both of which had been previously blasted. Both samples were large single-lump samples taken from ROM ore that were then cut into two for testing. One half was mechanically fragmented using laboratory crushers, to be used as the control sample and the other was then cut into a square piece approximately 15 cm x 15 cm x 15 cm and blasted using a single hole, loaded with an amount of 75 grain detonating cord that would be energetically equivalent to a powder factor of 0.5 kg/m$^3$ using regular ANFO. Holes were drilled in the center of the sample for the detonating cord, and water was used as the coupling medium. The samples were blasted in the blasting chamber at the Queen’s University Blasting Test Site, and the rock fragments created by blasting were collected for analysis. The particles from the control sample and the blasted sample were then screened into the 4 size bins, and 20 particles from each size bin were broken using the Queen’s University Drop-Weight Tester (QDWT) at the specific drop-weight impact energy listed in Table 6-1.

Table 6-1: Feed bin selected sizes for taconite $t_{10}$ testing

<table>
<thead>
<tr>
<th>Feed Bin Size</th>
<th>Specific Impact Energy (kWH/t)</th>
<th>Average Size (mm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>-45 +37.5 mm</td>
<td>0.25</td>
<td>41.25</td>
</tr>
<tr>
<td>-31.5 +26.5 mm</td>
<td>1.00</td>
<td>29.00</td>
</tr>
<tr>
<td>-22.4 +19 mm</td>
<td>1.00</td>
<td>20.70</td>
</tr>
<tr>
<td>-16 +13.2 mm</td>
<td>1.00</td>
<td>14.60</td>
</tr>
</tbody>
</table>

The average size column is the average particle size for each size bin, which is simply the mean of the upper and lower screen aperture sizes, thus the $t_{10}$ (or the 1/10th)
passing size is simply the average size divided by 10. After being broken in the QDWT the fragments from each bin were collected and screened and the amount of material passing the $t_{10}$ sizes were computed. The results are shown in Figure 6-1 and Figure 6-2.

Figure 6-1: Taconite Sample #15 blasted and unblasted $t_{10}$ comparison
Figure 6-2: Taconite Sample #16 blasted and unblasted $t_{10}$ comparison

For seven of the eight size-energy combinations tested for taconite samples # 15 and # 16, the material blasted at a powder factor of 0.5 kg/m$^3$ produced more material passing $t_{10}$ size than the unblasted material after being subjected to equal drop-weight specific energies. There is one exception for Sample # 16, where the -16+13.2 mm feed sample size where the blasted sample produced less material passing the $t_{10}$ size than the unblasted material, indicating that the blasted sample was harder than the unblasted sample. This is against the expectation, and a plausible explanation for this is sampling variance and/or random error. It is also possible that previous blasting had affected part of the ‘unblasted’ control sample as well. Table 6-2 and Table 6-3 summarize the actual drop energies that the sample particles were subjected to. All of the comparison groups are close and the difference between the blasted and unblasted specific drop-weight
energies is within 1.2%. Data for the taconite drop weight testing can be found in Appendix E.

Table 6-2: Actual drop-weight specific impact energies for taconite sample #15 (kWh/t)

<table>
<thead>
<tr>
<th>Size Range</th>
<th>Blasted</th>
<th>Unblasted</th>
</tr>
</thead>
<tbody>
<tr>
<td>45 - 37.5 mm</td>
<td>0.242</td>
<td>0.239</td>
</tr>
<tr>
<td>31.5 - 26.5 mm</td>
<td>0.989</td>
<td>0.985</td>
</tr>
<tr>
<td>22.4 - 19 mm</td>
<td>0.974</td>
<td>0.972</td>
</tr>
<tr>
<td>16 - 13.2 mm</td>
<td>0.946</td>
<td>0.944</td>
</tr>
</tbody>
</table>

Table 6-3: Actual drop-weight specific impact energies for taconite sample #16 (kWh/t)

<table>
<thead>
<tr>
<th>Size Range</th>
<th>Blasted</th>
<th>Unblasted</th>
</tr>
</thead>
<tbody>
<tr>
<td>45 - 37.5 mm</td>
<td>0.250</td>
<td>0.250</td>
</tr>
<tr>
<td>31.5 - 26.5 mm</td>
<td>0.990</td>
<td>0.988</td>
</tr>
<tr>
<td>22.4 - 19 mm</td>
<td>0.976</td>
<td>0.975</td>
</tr>
<tr>
<td>16 - 13.2 mm</td>
<td>0.948</td>
<td>0.951</td>
</tr>
</tbody>
</table>

Seven of the eight size-energy combinations showed increased $t_{10}$ values as a result of blasting. This indicates that the taconite material was affected by blasting, softening it so that it fragmented more easily than the unblasted sample. This would help substantiate Nielsen and Kristiansen’s (1996) findings that the milling properties of taconite are affected by the application of blasting energy, although the magnitude may be less. Thus, it would seem that further investigation into the effect of blasting on taconite is warranted since its milling properties seem to show a response to blasting.

### 6.2 Granodiorite Drop-Weight Tests

The reason for testing granodiorite was the same as testing the taconite; to see if its comminution properties were affected by blasting. Samples of granodiorite (that were intended for monuments) were provided by a stone dealer in Kingston for testing. The procedure was the same as for the taconite material in the previous section, although instead of having only a blasted sample and a control sample, a control sample and two blasted samples were used. The two blasted samples were shot at ANFO equivalent powder factors of 0.5 and 1.0 kg/m$^3$ at the Queen’s University Blasting Test Site. For details on the blasting procedure and the material tested, the reader is referred to
The samples were then processed the same way as the taconite particles in the previous section, however the larger sample sizes allowed the use of a 5th coarse size energy fraction. For each size-energy combinations 20 particles were used, as suggested by Napier-Munn et al. (1996). The actual drop-weight specific impact energies that each sample was subjected to are listed in Table 6-4, and the difference between any set of samples is within 1.5%. Data for the granodiorite drop weight tests can be found in Appendix F.

**Table 6-4: Actual drop-weight specific impact energies for all granodiorite samples (kWh/t)**

<table>
<thead>
<tr>
<th>Powder Factor</th>
<th>-63 +53 mm</th>
<th>-45 +37.5 mm</th>
<th>-31.5 +26.5 mm</th>
<th>-22.4 +19 mm</th>
<th>-16 +13.2 mm</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.0</td>
<td>0.143</td>
<td>0.242</td>
<td>0.989</td>
<td>0.975</td>
<td>0.945</td>
</tr>
<tr>
<td>0.5</td>
<td>0.144</td>
<td>0.242</td>
<td>0.992</td>
<td>0.976</td>
<td>0.885</td>
</tr>
<tr>
<td>1.0</td>
<td>0.099</td>
<td>0.240</td>
<td>0.992</td>
<td>0.981</td>
<td>0.955</td>
</tr>
</tbody>
</table>

Figure 6-3 summarizes the $t_{10}$ results for the granodiorite samples. There is a consistent trend for all of the sample groups, with the degree of breakage (i.e. the $t_{10}$ value) increasing as a function of Powder Factor (PF).
Figure 6-3: $t_{10}$ values for granodiorite, at constant specific drop-weight energy

Also, the coarser the particle, the larger the relative difference between the $t_{10}$ values for the three different powder factors. The smaller particles were tested at a higher specific impact energy, and the drop-weight test becomes more inefficient as the specific drop-weight energy increases, so the smaller relative difference between the powder factors of the smaller size classes may be caused by the increasing comminution inefficiency of the drop-weight, and not by anything specific to the particles. It can be concluded, however, that the blasting increases the degree of breakage the material exhibits at constant drop-weight energy. This is consistent with the results found by Katsabanis et al. (2003).
6.3 **Limestone Drop-Weight tests**

Square samples of finished limestone, measuring 20 x 20 x 30 cm were blasted and then broken using the same procedure as for granodiorite, except samples blasted at four different powder factors ranging from 0.25 kg/m$^3$ to 1.0 kg/m$^3$ ANFO equivalent using 75 and 150 grain PETN detonating cord, using water for coupling. The rock fragments created by the blast were then collected and subjected to drop-weight testing. As well, a control sample of unblasted material was mechanically fragmented and then subjected to drop-weight $t_{10}$ testing. The actual drop weight energies that the samples were subjected to are listed in Table 6-5, and the variation of the drop-weight energies between similar samples are within 3%. Data for the limestone drop weight tests can be found in Appendix G.

**Table 6-5: Actual drop-weight specific impact energies for all limestone samples (kWh/t)**

<table>
<thead>
<tr>
<th>PF = 0 kg/m³</th>
<th>-45 +37.5 mm</th>
<th>-31.5 +26.5 mm</th>
<th>-22.4 +19 mm</th>
<th>-16 +13.2 mm</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.236</td>
<td>0.988</td>
<td>0.974</td>
<td>0.945</td>
<td></td>
</tr>
<tr>
<td>PF = 0.25 kg/m³</td>
<td>0.238</td>
<td>0.988</td>
<td>0.978</td>
<td>0.949</td>
</tr>
<tr>
<td>PF = 0.5 kg/m³</td>
<td>0.238</td>
<td>0.990</td>
<td>0.978</td>
<td>0.960</td>
</tr>
<tr>
<td>PF = 0.75 kg/m³</td>
<td>0.240</td>
<td>0.989</td>
<td>0.977</td>
<td>0.963</td>
</tr>
<tr>
<td>PF = 1.0 kg/m³</td>
<td>0.240</td>
<td>0.982</td>
<td>0.980</td>
<td>0.971</td>
</tr>
</tbody>
</table>

Examining the $t_{10}$ breakage results in Figure 6-4, there is an apparent trend in the coarsest size fraction of -45 +37.5 mm particles. With increasing powder factor, the $t_{10}$ breakage increased, as observed with the taconite and the limestone. There appears to be significant scatter in the $t_{10}$ breakage among the smaller size fractions of the limestone, and that there are other factors present other than blasting that are affecting the degree of breakage of the particles. A $t_{10}$ breakage index of approximately 50% represents the upper degree of breakage that can be accomplished using single drop
drop-weight testing, as it becomes very inefficient beyond this breakage degree. It has been found that 50% represents the upper $t_{10}$ breakage limit for many rock types (Napier-Munn et al., 1996).

The Effect of Varying Powder Factors on the $t_{10}$ value of Limestone

Figure 6-4: $t_{10}$ values for Limestone at constant specific drop-weight energy

Many of the $t_{10}$ values are over 50% for the limestone, which suggests that the comminution efficiency of the drop-weight test was becoming inefficient, and the scatter in the $t_{10}$ breakage is being caused by this making the test ineffective for the three smaller size-energy combinations. The choice of a specific impact energy of 1.0 kWh/t was too high for the limestone, and masked any possible observable differences between samples blasted at different powder factor. The difference in the $t_{10}$ values of the -45 +37.5 mm rock is pronounced however, and would indicate that this rock is a
good candidate for further research relating energy used to blast rock to comminution properties of the rock.
Chapter 7: Summary of Results, Discussion and Conclusion

7.1 Experimental Results and Conclusions

The experimental results using laboratory scale testing did not reveal any continuous or apparent trend in changes to the comminution properties of the Kemess Hypogene ore alteration domain types D1, D2, and D5 as a result of blasting.

- Analysis of thin-sections of blasted and unblasted Kemess D1 ore (Section 2.4) showed that the blasted ore had a higher density of micro-fracturing than the unblasted ore.

- Drop-weight $t_{10}$ testing used for characterizing the impact breakage characteristics of rock for AG/SAG milling, described in section 3.3 indicated that there was a statistically significant difference between the D1 blasted and unblasted samples, and there was no statistically significant difference between the blasted and unblasted Kemess Hypogene D2 and D5 ore samples. This is consistent with the results of the fracture density measurements of the D1 thin-sections.

- The locked-cycle ball mill Bond Work Index (BWI) of Kemess Hypogene D1 and D2 samples was reduced by blasting, with the samples that were blasted using the water-impact test having a lower BWI than the unblasted material. Kemess Hypogene D5 material did not show any change.

- Batch ball mill testing using a monosize feed sample, described in Section 3.4.2 found that the unblasted Kemess D1 ore was actually grinding faster than the blasted ore. There was no observed change in the D2 and the D5 samples.
• Using computer-based mathematical models, different levels of blasting (i.e. changing the powder factor) were examined with different crusher Closed Side Set (CSS) settings to optimize throughput and the overall economic performance of the mine-mill system. Base case testing indicates that the blasting is currently very close or at an optimum given the current mill configuration. There does not appear to be an opportunity to increase throughput without making changes to the crusher. The models indicate that it is possible to optimize the system, and to do so the crusher CSS and level of ROM ore fragmentation need to be suitably matched.

7.2 Discussion

For the metallurgical testing of the Kemess Hypogene Ore, the expected result was that the blasted samples would have a reduced resistance to comminution, and would have lower Bond ball mill work index and higher $t_{10}$ breakage values. The reduction of the resistance to comminution by rock induced by blasting has been observed by Nielsen and Kristiansen (1996) for a variety of rock types, and it was expected that this would be true for the Kemess Hypogene ore samples. For the D1 ore, blasting did increase the density of micro-fractures within the rock fabric. This translated into a reduced crushing and grinding energy requirement for all of the tests with the exception of the monosize ball mill batch testing, which produced ambiguous results. This was not the case for the D2 and D5 rock types. A potential explanation is that the tests applied to the materials were not able to determine which size fractions were affected by blasting and the microfracturing density required to produce a significant effect. There is also the possibility that the natural variation in the hardness of the rock (even within domain types) may be greater than the change that can be induced by blasting, making a
relationship between the two difficult (if not impossible) to detect. Should this be the case, many repetitive tests would be necessary to generate a large enough set of results so that the effect of blasting could potentially be observed with statistical confidence. Also, the results as found would suggest that the inconsistent results here could be attributed to sampling error that would be largely associated with collection of rock samples. The samples used from Kemess were selected in the pit based upon location and appearance. They may have not been subjected to the same amount of initial blast energy, nor would (or could) they be identical since the testing methods used in this study are destructive. The potential reward for determining if such a relationship exists is quite large however, and would justify the expenditure on further investigation.

Based on the mineral processing simulations, it would appear that the primary factor when relating blasting practices to changes in crushing and grinding performance is fragmentation. The mineral processing simulator showed that with changes to crusher settings, increased fragmentation caused by blasting can be turned to advantage in the comminution circuit, where the economic crushing and grinding performance gains can exceed the increased cost of blasting. This is similar to the observations of Dance (2001).

The work done in Chapter 6: suggests that further investigation to the effect of blasting on taconite, granodiorite and limestone is worth investigating. All three material types indicated that some changes to the comminution properties could be effected by blasting in particle sizes encountered in crushing and SAG milling. These material types would be good candidates for further research.
7.3 Future Work at Kemess

The economic implications of increased mill throughput facilitated by increasing fragmentation in the mine are promising and warrant further investigation. The results presented here are a result of mathematical models and require real-world observations to substantiate predictions. The first step in this would be a validation study, to see if the model is robust in predicting the response of the mill under real operating conditions, which would encompass at least one test blast at a higher powder factor that when compared to base-case operations, would offer quantifiably increased fragmentation. To do so, a rigorous study of current blasting conditions and resulting fragmentation using on-line image analysis software would be needed. When this can be established, the mill should be fed the finer ore when the stockpile between the crusher and the SAG mills has been drawn down to reduce the effect of stockpile segregation on the feed, and give the truest mill response. Establishing the relationship between blasting and milling is a difficulty and time intensive proposition, but the costs are relatively small compared with the potential reward.
References


Appendix A: Bond Work Index Procedure
Project 3: Bond Work Index

Purpose

To determine the work index of an ore using the F.C. Bond method.

Materials

1. F.C. Bond work index mill with standard ball mill charge
2. Graduated cylinder
3. Analysette Sieve Shaker and screens

Procedure

1. Do a mesh screen analysis on about 250 grams of the assigned -6 mesh ore sample. Determine the feed 80% passing size \( F_{80} \) and the weight percent -65 mesh.
2. Tightly pack the assigned ore in a 1000 ml graduated cylinder to the 700 cm\(^3\) mark. Use the vibrator for this. Weigh the ore.
3. Calculate the “Ideal Potential Product”. This is the net weight of product that must be produced each cycle to allow the mill to run with a 250% recirculating load. Refer to the paper by Rene John Deister for complete calculations.
   
   \[ \text{Ideal potential product} = \frac{\text{feed weight}}{3.5} \]

4. Grind the 700 cm\(^3\) sample in the Bond ball mill for an arbitrary 100 revolutions. Remove the sample from the mill and then screen through a 65 mesh screen. Replace the removed -65 mesh material with fresh -6 mesh feed.
5. Calculate the net grams of -65 mesh product. (total -65 mesh product minus the “in-feed -65 mesh” and then the net grams per revolution.
6. The Ideal Potential Product minus the “in-feed -65 mesh” for the next cycle divided by the net grams per revolution from the previous cycle gives you the number of revolutions for the next cycle.
7. Put the replenished ore charge back into the mill and grind for the calculated number of revolutions.
8. Remove the ore charge, screen it, replace the removed – 65 mesh, calculate the revolutions for the next cycle and grind again. Repeat this procedure until three consecutive cycles have produced products of consistent weight close to that of the Ideal Potential Product.
Appendix B: Kemess Ore Drop-Weight Results
## Final Screening Results Summary For D1 Kemess HO Material Batch #1

### D2 Blasted Batch 1 Size and Screening Results

#### Screening Results

All masses in grams

<table>
<thead>
<tr>
<th>Mass Retained</th>
<th>-63 +53 mm</th>
<th>-45 +37.5 mm</th>
<th>-31.5 +26.5 mm</th>
<th>-22.4 +19 mm</th>
<th>-16 +13.2 mm</th>
</tr>
</thead>
<tbody>
<tr>
<td>53</td>
<td>37.5</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>26.5</td>
<td>390.71</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>19</td>
<td>307.35</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>13.2</td>
<td>201.17</td>
<td>96.74</td>
<td>5.54</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>9.5</td>
<td>84.04</td>
<td>78.66</td>
<td>17.73</td>
<td>2.61</td>
<td>0</td>
</tr>
<tr>
<td>6.7</td>
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### D2 Unlasted Batch 1 Size and Screening Results

#### Screening Results

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Total Mass: 3203.35 1250.09 414.88 122.56 36.29
## Final Screening Results Summary For D1 Kemess HO Material Batch #2

### D1 Blasted Batch 2 Size and Screening Results

**Screening Results**
All masses in grams

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**Total Mass:**
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### D1 Unlasted Batch 2 Size and Screening Results

**Screening Results**
All masses in grams

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**Total Mass:**
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### Final Screening Results Summary For D1 Kemess HO Material Batch #3

#### D1 Blasted Batch 3 Size and Screening Results

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**Total Mass:** 3142.62 1066.81 294.55 125.31 45.68

#### D1 Unlasted Batch 3 Size and Screening Results

**Screening Results**

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**Total Mass:** 3142.62 1066.81 294.55 125.31 45.68
## Final Screening Results Summary For D2 Kemess HO Material Batch #1

### D2 Blasted Batch 1 Size and Screening Results

**Screening Results**

All masses in grams

**Mass Retained**

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**Total Mass:** 0 1077.57 418.04 127.22 41.57

### D2 Unlasted Batch 1 Size and Screening Results

**Screening Results**

All masses in grams

**Mass Retained**

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**Total Mass:** 0 932.1 359.1 126.05 39.79
### Final Screening Results Summary For D2 Kemess HO Material Batch #2

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#### D2 Unlasted Batch 2 Size and Screening Results

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### D2 Blasted Batch 3 Size and Screening Results

**Screening Results**
All masses in grams

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**Total Mass:** 0 899.35 343.24 119.8 42.79

### D2 Unlasted Batch 3 Size and Screening Results

**Screening Results**
All masses in grams

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**Total Mass:** 0 844.88 353.79 121.36 40.58
## Final Screening Results Summary For D5 Kemess HO Material Batch #1

### D5 Blasted Batch 1 Size and Screening Results

**Screening Results**

All masses in grams

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Total Mass: 0 1051.11 390.35 165.47 45.74

### D5 Unlasted Batch 1 Size and Screening Results

**Screening Results**

All masses in grams

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### Final Screening Results Summary For D5 Kemess HO Material Batch #2

#### D5 Blasted Batch 2 Size and Screening Results

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- All masses in grams
- Mass Retained

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**Total Mass:** 0 1037.46 404.43 146.87 48.21

#### D5 Unlasted Batch 2 Size and Screening Results

**Screening Results**
- All masses in grams
- Mass Retained

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**Total Mass:** 0 828.25 384.41 114.85 31.1
# Final Screening Results Summary for D5 Kemess HO Material Batch #3

## D5 Blasted Batch 3 Size and Screening Results

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All masses in grams

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Total Mass: 0 954.21 335.41 126.07 41.69

## D5 Unlasted Batch 3 Size and Screening Results

### Screening Results

All masses in grams

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Total Mass: 0 852.53 419.09 130.48 44.11
Appendix C: Ball Mill Locked Cycle Testing
Mine: Kemess
Sample: SPI 6026 D1 Unblasted
Date of Test 6-Jan
Tested By DH
Feed Weight (g) 1128.2
Density (kg/m3) 1611.71
Feed -65 14.27% (From Lab Calculations)
Ideal Potential Product (g) 322.34

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<th>Net g / Rev.</th>
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Product

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F80 1710 um (from Graph)
P80 164 um (from Graph)
Pi 212 (-65 mesh)
Gbp 2.07

Material WI (kWh/t) 12.68
**Mine:** Kemess  
**Sample:** SPI 6027 D1 Blasted  
**Date of Test:** 7-Jan  
**Tested By:** DH

**Feed Weight (g):** 1082.31  
**Density (kg/m3):** 1546.16  
**Feed -65:** 8.86% (From Lab Calculations)  
**Ideal Potential Product (g):** 309.23

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### Feed Sample

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### Product

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| F80           | 2005 um (from Graph)  |
| P80           | 167.7 um (from Graph) |
| Pi            | 212 (-65 mesh)        |
| Gbp           | 1.53                  |

**Material WI (kWh/t):** 10.09
Mine: Kemess
Sample: 6028 D2 Unblasted
Date of Test Jan 18 2009
Tested By DH
Feed Weight (g) 1096.08
Density (kg/m3) 1565.83
Feed -65 11.79%

Ideal Potential Product (g) 313.17

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Feed Sample

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<th>% of Wt (%)</th>
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Product

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F80 2090 um (from Graph)
P80 168 um (from Graph)
Pi 212 (-65 mesh)
Gbp 2.06

Material WI (kWh/t) 12.98
**Mine:** Kemess  
**Sample:** SPI 6029 D2 Blasted  
**Date of Test:** Jan 24 2004  
**Tested By:** DH

Feed Weight (g) 1107.48  
Density (kg/m³) 1582.11  
Feed -65 11.71% (From Lab Calculations)

Ideal Potential Product (g) 316.42

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**Feed Sample**

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**Product**

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F80 1935 um (from Graph)  
P80 170 um (from Graph)  
Pi 212 (-65 mesh)  
Gbp 1.93

**Material WI (kWh/t)** 12.00
Mine: Kemess  
Sample: SPI 6030 D5 Unblasted  
Date of Test Jan 29 2004  
Tested By DH  

Feed Weight (g) 1149.02  
Density (kg/m3) 1641.46  
Feed -65 9.51% (From Lab Calculations)  

Ideal Potential Product (g) 328.29  

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Feed Sample  

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Product  

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F80 1980 um (from Graph)  
P80 168 um (from Graph)  
Pi 212 (-65 mesh)  
Gbp 1.59  

Material WI (kWh/t) 10.39
Mine: Kemess  
Sample: SPI 6031 D5 Blasted  
Date of Test: Feb 3 2004  
Tested By: DH

Feed Weight (g) 1141.84  
Density (kg/m³) 1631.20  
Feed -65 14.83% (From Lab Calculations)

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<th>In Mill Feed</th>
<th>Net Product</th>
<th>Net g / Rev.</th>
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Feed Sample

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<th>% of Wt (%)</th>
<th>Cumulative Retained (%)</th>
<th>Percent Passing (%)</th>
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Product

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<th>Weight (g)</th>
<th>% of Wt (%)</th>
<th>Cumulative Retained (%)</th>
<th>Percent Passing (%)</th>
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F80 1850 um (from Graph)  
P80 162 um (from Graph)  
Pi 212 (-65 mesh)  
Gbp 1.56

Material WI 10.33
Appendix D: Ball Mill Batch Cycle Testing
### Kemess D1 Unblasted Batch Ball Milling Results (Sample)

<table>
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<th>Size (mm)</th>
<th>Weight Retained (g)</th>
<th>Revolutions</th>
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<td>+0.3</td>
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<td>238.24</td>
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### Kemess D1 Blasted Batch Ball Milling Results (Sample)

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<thead>
<tr>
<th>Size (mm)</th>
<th>Weight Retained (g)</th>
<th>Revolutions</th>
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<tr>
<td>+2.36</td>
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### Kemess D1 Unblasted Batch Ball Milling Results

#### Percent Passing (%)

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<th>Size (mm)</th>
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<th>Revolutions</th>
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<tr>
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<td>6.0%</td>
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### Kemess D1 Blasted Batch Ball Milling Results

#### Percent Passing (%)

<table>
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<tr>
<th>Size (mm)</th>
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<th>Revolutions</th>
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### Kemess D2 Unblasted Batch Ball Milling Results (Sample)

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### Kemess D2 Blasted Batch Ball Milling Results (Sample)

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### Kemess D2 Unblasted Batch Ball Milling Results

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### Kemess D2 Blasted Batch Ball Milling Results

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### Kemess D5 Unblasted Batch Ball Milling Results (Sample)

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### Kemess D5 Blasted Batch Ball Milling Results (Sample)

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### Kemess D5 Unblasted Batch Ball Milling Results

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<tr>
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<td>24.9%</td>
</tr>
<tr>
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<td>12.7%</td>
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<tr>
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### Kemess D5 Blasted Batch Ball Milling Results

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Appendix E: Taconite Drop Weight Test Results
### HTC #15 Unblasted

#### Particle Mass (g)

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<th>-31.5 +26.5 mm</th>
<th>-22.4 +19 mm</th>
<th>-16 +13.2 mm</th>
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| Weight (g) | 3147.56   | 165.85       | 137.32         | 145.39       | 159.65       |
| Average Weight (g) | 157.379 | 48.26        | 17.294         | 6.283        |              |
| Stddev Weight (g)  | 0.00      | 47.65        | 14.44          | 5.09         | 2.13         |

| Num. Particles | 20 | 20 | 20 | 20 |

Eis Energy (kWh/t) | 0.15 | 0.29 | 1  | 1  | 1

### HTC #15 Blasted

#### Particle Mass (g)

<table>
<thead>
<tr>
<th>Size Class</th>
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<th>-22.4 +19 mm</th>
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</table>

| Weight (g) | 3360.74    | 1025.17      | 339.83         | 122.49       |              |
| Average Weight (g) | 168.037 | 51.2585      | 16.9915        | 6.1245       |              |
| Stddev (g)  | 47.02      | 20.97        | 4.24           | 2.28         |              |

| Num. Particles | 20 | 20 | 20 | 20 |

Eis Energy (kWh/t) | 0.15 | 0.29 | 1  | 1  | 1

### Drop Height (mm)

<table>
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<tr>
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<th>-63 +93 mm</th>
<th>-45 +37.5 mm</th>
<th>-31.5 +26.5 mm</th>
<th>-22.4 +19 mm</th>
<th>-16 +13.2 mm</th>
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| E Energy (kWh/t) | 0.15 | 0.29 | 1  | 1  | 1

### Drop Height (mm)

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<tr>
<th>Size Class</th>
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<th>-45 +37.5 mm</th>
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<th>-22.4 +19 mm</th>
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Hibbing Taconite Samples
Samples # 15 Unblasted and Blasted

**HTC #15 Unblasted**
Screening Results
All masses in grams
Mass Retained

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**HTC #15 Blasted**
Screening Results
All masses in grams
Mass Retained

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<th>-22.4 +19 mm</th>
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#### Drop Height (mm) and Weight (g)

- **Average Weight (g):** 199.267, 51.7285, 18.051
- **Std Dev Weight (g):** 0.00, 59.31, 16.44
- **Num. Particles:** 20, 20, 20

#### Size Class

- **-63 +53 mm:** 1, 1
- **-45 +37.5 mm:** 1, 1
- **-31.5 +26.5 mm:** 1, 1
- **-22.4 +19 mm:** 1, 1
- **-16 +13.2 mm:** 1, 1

### HTC #16 Blasted

#### Particle Mass (g)

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#### Drop Height (mm) and Weight (g)

- **Average Weight (g):** 133.3545, 52.6705, 15.402
- **Std Dev Weight (g):** 0.00, 45.26, 20.32
- **Num. Particles:** 20, 20, 20

#### Size Class

- **-63 +53 mm:** 1, 1
- **-45 +37.5 mm:** 1, 1
- **-31.5 +26.5 mm:** 1, 1
- **-22.4 +19 mm:** 1, 1
- **-16 +13.2 mm:** 1, 1

Eis Energy (kwh/t): 0.15, 0.25

#### Drop Height (mm)

- **HTC #16 Unblasted:** 955.15, 991.80, 346.10
- **HTC #16 Blasted:** 1053.41, 308.18, 131.85

#### Stdev Weight (g)

- **HTC #16 Unblasted:** 59.31, 16.44
- **HTC #16 Blasted:** 45.26, 20.32
# Hibbing Taconite Samples
## Samples # 16 Unblasted and Blasted

### HTC #16 Unblasted

**Screening Results**

All masses in grams

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### HTC #16 Blasted

**Screening Results**

All masses in grams

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Appendix F: Granodiorite Drop-Weight Test Data
## DW Test Information Sheet
### Single Batch

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| Weight (g) | 6518.3 | 3087.0 | 721.4 | 262.4 | 89.4 |
| Average Weight (g) | 325.9 | 154.3 | 58.1 | 8.3 | 4.5 |
| Stdev Weight (g) | 102.5 | 191.6 | 15.9 | 4.4 | 1.4 |
| Num. Particles | 20 | 20 | 20 | 20 | 20 |

| Eis Energy (kwh/t) | 0.15 | 0.25 | 1 | 1 | 1 |

#### Particle Mass (g)

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| Adjusted (Actual) Eis | 0.143 | 0.242 | 0.989 | 0.975 | 0.945 |
### Final Drop Heights Granodiorite Unblasted

All Heights in millimeters

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### Screening Results Granodiorite Unblasted

All masses in grams

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**Total Mass:** 6505.77 2222.98 717.91 260.14 88.77
DW Test Information Sheet
Single Batch

| Sample: | Granodiorite Blasted powder factor=0.5 k/m³ | g (const)= 9.36 |
| Date of test: | May 12 2004 | Dm (kg)= 20.06 |
| Technician: | Brian |

**Constants**

**Particle Mass (g)**

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**Size Class**

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All Heights in millimeters

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**Average:** 37 21.9 6 5.6 4.2

### Screening Results Powder Factor 0.5 kg/m³
All masses in grams

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**Total Mass:** 7597.80 2356.73 776.02 243.72 102.12
**Final Drop Heights Granodiorite Powder Factor 1.0 kg/m³**

All Heights in millimeters

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**Screening Results Granodiorite Powder Factor 1.0 kg/m³**

All masses in grams

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Appendix G: Limestone Drop-Weight Test Data
## DW Test Information Sheet

### Single Batch

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### Constants

| Sample: | g (const)= 9.36 |

### Particle Mass (g)

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| Average Weight (g) | 0.0 | 97.1 | 34.2 | 11.4 | 4.0 |
| Stdev Weight (g) | 0.0 | 28.9 | 12.7 | 3.7 | 1.5 |
| Num. Particles | 0 | 20 | 20 | 20 | 20 |

| Els Energy (kwh/t) | 0.0 | 0.25 | 1 | 1 | 1 |

### Drop Height (mm)

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All Heights in millimeters

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### Screening Results Limestone Unblasted
All masses in grams

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DW Test Information Sheet
Single Batch

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Stdev Weight (g)  
Num. Particles  
Eis Energy (kwh/t)  
Size Class -63 +53 mm -45 +37.5 mm -31.5 +26.5 mm -22.4 +19 mm -16 +13.2 mm  
Drop Height (mm)  

170
## Final Drop Heights Limestone Powder Factor=0.25
All Heights in millimeters

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## Screening Results Limestone Powder factor=0.25
All masses in grams

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## DW Test Information Sheet
### Single Batch

**Sample:** Limestone Powder Factor=0.5
**Date of test:**
**Technician:** BS

### Constants

| Sample: g (const)= | 9.36 |
| Date of test: Dm (kg)= | 20.06 |

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**Weight (g)**

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<th>295.1</th>
<th>108.1</th>
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</table>

**Average Weight (g)**

| Average Weight (g) | 0.0 | 80.7 | 39.7 | 14.8 | 5.4 |

**Stdev Weight (g)**

| Stdev Weight (g) | 0.0 | 25.1 | 15.0 | 5.1 | 2.1 |

**Num. Particles**

| Num. Particles | 0 | 20 | 20 | 20 | 20 |

**Eis Energy (kwh/t)**

| Eiss Energy (kwh/t) | 0.25 | 1 | 1 | 1 | 1 |

**Size Class**

<table>
<thead>
<tr>
<th>Size Class</th>
<th>-63 +53 mm</th>
<th>-45 +37.5 mm</th>
<th>-31.5 +26.5 mm</th>
<th>-22.4 +19 mm</th>
<th>-16 +13.2 mm</th>
</tr>
</thead>
</table>

**Drop Height (mm)**

| Drop Height (mm) | 0.0 | 386.8 | 761.4 | 282.9 | 103.6 |

Limestone Powder Factor=0.5
### Final Drop Heights Limestone Powder Factor = 0.5
All Heights in millimeters

<table>
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<tr>
<th>Size Class</th>
<th>-63 +53 mm</th>
<th>-45 +37.5 mm</th>
<th>-31.5 +26.5 mm</th>
<th>-22.4 +19 mm</th>
<th>-16 +13.2 mm</th>
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### Screening Results Limestone Powder Factor = 0.5
All masses in grams

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<th>-22.4 +19 mm</th>
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**DW Test Information Sheet**

**Single Batch**

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<td>Dm (kg)= 20.06</td>
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<td>Technician:</td>
<td>BS</td>
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## Constants

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## Particle Mass (g)

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<td>13.65</td>
<td>6.46</td>
<td></td>
</tr>
</tbody>
</table>

| Weight (g) | 0.0 | 1960.7 | 719.5 | 277.5 | 117.7 |
| Average Weight (g) | 0.0 | 98.0 | 36.0 | 13.9 | 5.9 |
| Stdev Weight (g) | 0.0 | 41.8 | 13.8 | 5.1 | 1.5 |
| Num. Particles | 0 | 20 | 20 | 20 | |

**Eis Energy (kwh/t)**

| 0 | 0.25 | 1 | 1 | 1 |

## Size Class

<table>
<thead>
<tr>
<th>Size Class</th>
<th>-63 +53 mm</th>
<th>-45 +37.5 mm</th>
<th>-31.5 +26.5 mm</th>
<th>-22.4 +19 mm</th>
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174
### Final Drop Heights Limestone Powder Factor=0.75
All Heights in millimeters

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### Screening Results Limestone Powder Factor=0.75
All masses in grams

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**DW Test Information Sheet**  
**Single Batch**

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**Particle Mass (g)**

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| EIS Energy (kwh/t) | 0 | 0.25 | 1 | 1 | 1 |

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## Final Drop Heights Limestone Power Factor=1.0

All Heights in millimeters

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Average: 0 14.3 12.35 5.35 3.85

## Screening Results Limestone Power Factor=1.0

All masses in grams

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Total Mass: 0.00 1439.25 714.86 276.79 88.05
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<th>Description</th>
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<tr>
<td>BWI</td>
<td>Bond Work Index: the specific energy required to grind a material, as measured using the Bond Ball Mill Work Index Test (Bond, 1961)</td>
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<tr>
<td>CSS</td>
<td>Closed Side Set: the closed side setting of the crusher</td>
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<tr>
<td>HO</td>
<td>Hypogene Ore: a type of ore found at the Kemess South Deposit</td>
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<td>PF</td>
<td>Powder Factor: describes the amount of explosive used per unit of rock being blasted</td>
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<td>ROM</td>
<td>Run-of-Mine: describes typical fragmented ore produced by the mine</td>
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<td>SAG</td>
<td>Semi-Autogenous Grinding: refers to a type of grinding that is partially autogenous</td>
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<td>SAG Power Index: A proprietary test developed by MinnovEX</td>
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<td>Tonnes per hour: Rate of material being processed</td>
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