# EXPLORING HIGH PRESSURE SLURRY ABLATION AS A MINERAL PROCESSING TECHNOLOGY

by

David Paul Antoniak

B.Sc., The University of Calgary, 2018

## A THESIS SUBMITTED IN PARTIAL FULFILLMENT OF THE REQUIREMENTS FOR THE DEGREE OF

MASTER OF APPLIED SCIENCE

in

The Faculty of Graduate and Postdoctoral Studies

(Mining Engineering)

THE UNIVERSITY OF BRITISH COLUMBIA

(Vancouver)

August 2020

© David Paul Antoniak, 2020

The following individuals certify that they have read, and recommend to the Faculty of Graduate and Postdoctoral Studies for acceptance, a thesis entitled:

Exploring high	pressure s	lurry abl	lation as a	mineral	processing	technology
Exploring mgn	pressure s	nully uo	union us c	( IIIIII) ai	processing	teennology

submitted by	David Paul Antoniak	in partial fulfillment of the requirements for			
the degree of	Master of Applied Science				
in	Mining Engineering				
Examining Committee:					
Dr. Sanja Mis	skovic, Mining Engineering				
Supervisor					
Dr. Bern Klein, Mining Engineering					
Supervisory Committee Member					
Dr. Marek Pawlik, Mining Engineering					
Supervisory Committee Member					
Dr. Scott Dunbar, Mining Engineering					
Additional Ex	Additional Examiner				

### Abstract

In order to achieve economic operation and enable efficient separation of valuable components, most mineral processing facilities require comminution technologies. These machines liberate valuable minerals from the associated gangue by reducing the size of individual particles, or, in some operations, by removing loosely adhered particles from the hard mineral matrix. Due to the stochastic nature of particle-particle and particle-wall collisions, as well as the intrinsic inefficiencies associated with the low fraction of energy spent on particle breakage, these technologies require enormous amounts of energy to function and, therefore, research into the development and implementation of more efficient technologies is continuously ongoing.

This study assesses the performance of high pressure slurry ablation, a new size reduction, particle liberation, and slurry conditioning technology, against current industry standards. Various materials were tested with the equipment, and product samples were collected and analyzed for size distribution and size-based assaying. Where applicable, separation technologies such as gravity separation and froth flotation were used to provide further insights into the amenability of HPSA products to beneficiation. The results were then compared to the expected outcomes of typical grinding and scrubbing-attrition technologies.

The results concluded that the technology can achieve size reduction for all considered materials, can selectively liberate soft and loosely attached materials into a fraction of the original mass, and can provided elevated gravity separation recoveries in precious metals applications. However, due to the mechanical components of the pilot-scale unit used in this study limiting the percentage of solids that the unit could handle, the observed unit efficiencies are found to be lower than those of conventional size reduction units. By upgrading the pumps associated with the testing rig these limitations could be eliminated, thus, identifying areas for future work.

## Lay Summary

Within most mining operations, size reduction technologies are typically needed to liberate valuable minerals from the unwanted waste rock. This is often achieved using crushing and grinding equipment where particles are reduced to a size at which the targeted material can be efficiently recovered. Alternatively, in some applications liberation is achieved using attrition-scrubbing equipment to remove loosely adhered materials from the valuable component.

While there are many technologies available to perform these operations, all options consume a large amount of energy and, as a result, carry large operational costs. This thesis aims to investigate the performance of a new technology that has the potential to use less energy than the standard equipment. The research indicates that the new method can effectively clean unwanted material from the surface of valuable minerals in high-clay ores and potash deposits and can provide higher gold recoveries via gravity separation than standard size reduction methods.

## Preface

The work presented here within was conducted at the Coal and Mineral Processing facility with the support of Disa, LLC<sup>®</sup> and Nutrien. Under the supervision of Dr. Sanja Miskovic I was tasked with creating a testing plan, operating the HPSA technology, conducting separation work and interpreting the results.

Mewael Islet and Dustin Weaver assisted with testing in the HPSA unit and sample collection. Santiago Seiler also provided guidance during analysis of the Ecuador gold material.

# Table of Contents

Abstractiii
Lay Summaryiv
Prefacev
Table of Contents
List of Tablesix
List of Figures xi
Nomenclaturexv
Acknowledgementsxviii
1 Introduction 1
1.1 Background
1.2 Thesis Objectives
1.3 Thesis Outline
2 Literature Review
2.1 Mineral Processing Basics
2.2 Particle Liberation Technologies6
2.2.1 Overview
2.2.2 Grinding
2.2.3 Washing/Scrubbing11
2.2.4 Mechanisms of Breakage

	2.2.	5	Grinding Energy Laws	17
	2.3	Hig	h Pressure Slurry Ablation	19
	2.3.	1	History and Applications	19
	2.3.	2	Breakage Mechanisms and Theory	21
	2.4	Sum	nmary of Review	23
3	Des	ign o	of Experiment	24
	3.1	Met	hodology	24
	3.2	Exp	perimental Equipment	25
	3.3	Sele	ection of Variables	27
	3.4	Sam	ple Preparation	29
	3.5	Sch	edule of Experiments	30
	3.5.	1	Campaign 1 – Black Diamond Holdings (BDH) Tailings	30
	3.5.	2	Campaign 2 – Nutrien Potash	31
	3.5.	3	Campaign 3 – Ecuador Gold	33
	3.6	Data	a Analysis	36
4	Exp	erim	ental Results	37
	4.1	Feed	d Characterization	37
	4.1.	1	BDH Tailings	37
	4.1.	2	Nutrien Potash	39
	4.1.	3	Ecuador Gold	42
	4.2	Sele	ective Concentration	47

4.3	Size	Reduction	53
4.3	3.1	Particle Size Distributions	53
4.3	3.2	Size-Energy Relationships	60
4.4	Gold	d Recovery	64
4.4	4.1	Imaging Analysis	64
4.4	4.2	Metallurgical Balance	66
4.4	4.2	Gold Recoveries	68
5 Co	onclus	ions and Recommendations	71
5.1	Con	clusions	71
5.2	Futu	ıre Work	73
Referen	nces		74
Append	dix A	– Raw Experimental Data	80
Append	dix B -	– Particle Size Distribution Tables and Trends	87

# List of Tables

Table 1: PSD model results for BDH feed material	. 39
Table 2: Specific gravities of potash materials. Component values from Fuzesy [49]	. 40
Table 3: PSD model results for potash feed material	. 42
Table 4: PSD model results for BDH feed material	. 46
Table 5: Insol assay results for potash feed and 2-minute product classes	. 52
Table 6: Characteristic passing sizes for BDH feed and products	. 55
Table 7: Characteristic passing sizes for potash feed and 2-minute product	. 57
Table 8: Characteristic passing sizes for Ecuador gold feed and products	. 59
Table 9: Resulting stream grades of all experimental samples	. 67
Table A-1: Brine specific gravity data	. 80
Table A-2: Potash specific gravity data	. 80
Table A-3: Brine solubility data	. 80
Table A-4: Weight data for potash feed insolubles	. 81
Table A-5: Weight data for potash 2-minute product insolubles	. 81
Table A-6: Potash HPSA test energy consumption	. 82
Table A-7: Ecuador gold HPSA test energy consumption	. 82
Table A-8: Starting sample masses for each testing campaign	. 83
Table A-9: Bond work index values for BDH material	. 83
Table A-10: Bond work index values for potash material	. 83

Table A-11: Bond work index values for Ecuador gold material	83
Table A-12: Ecuador gold assay lab sample weights	84
Table A-13: Ecuador gold mass and metallurgical balance data	84
Table A-14: Internal results for several Ecuador gold lab assay samples	85
Table B-1: Size distribution data for BDH feed sample	87
Table B-2: Size distribution data for BDH 2-minute product	88
Table B-3: Size distribution data for BDH 4-minute product	89
Table B-4: Size distribution data for BDH 16-minute product	90
Table B-5: Size distribution data for potash feed sample	91
Table B-6: Size distribution data for potash 2-minute product	92
Table B-7: Size distribution data for Ecuador gold feed sample	93
Table B-8: Size distribution data for Ecuador gold HPSA 8-minute product	94
Table B-9: Size distribution data for Ecuador gold rod mill 1-minute product	95

# List of Figures

Figure 1: Liberation limited grade-recovery curves before and after comminution7
Figure 2: Cost analysis of mineral processing
Figure 3: Closed circuit configurations of a ball mill circuit: a) FCC, and b) RCC
Figure 4: Various motions of a tumbling mill: a) cascading, b) cataracting, c) centrifuging 10
Figure 5: Lab scale Denver flotation cell with attrition scrubber attachment
Figure 6: Methods of collision and resulting breakage: a) impact or compression, b) chipping or
attrition, c) abrasion. Adapted from Wills & Finch [12]
Figure 7: Resulting particle breakage due to impact/sudden compression collisions: a) falling
motion, b) impact and compressive force, c) resulting tensile force and crack propagation 14
Figure 8: Resulting particle breakage due to abrasive or attrition collisions: a) tangential motion,
b) surface forces and breakage
Figure 9: Comparison of the three comminution laws of Rittinger, Bond, and Kick. Adapted
from Mayer-Laigle et al. [39] 19
Figure 10: Simulation showing how the jet to jet configuration creates stream-based collisions
[47]
Figure 11: High pressure slurry ablation lab scale pilot unit
Figure 12: Various lab equipment: a) 9" screens, b) ROTAP, c) filter press, d) Knelson
Concentrator,
e) Denver flotation cell 26
Figure 13: Flowsheet of BDH testing campaign

Figure 14: Flowsheet of potash testing campaign	. 33
Figure 15: Flowsheet of HPSA material is Ecuador gold testing campaign	. 35
Figure 16: a) 10 x image of BDH feed 355 $\mu m$ fraction, b) 10 x image of BDH feed 125 $\mu m$	
fraction	. 37
Figure 17: a) 20 x image of BDH feed fines fraction, b) scale up of the red zone in Figure 16a,	,
with the trapped fines circled	. 38
Figure 18: a) Image of the potash feed "coarse" fraction, b) Image of the potash feed	
"intermediate" fraction	. 41
Figure 19: Microscope image of the potash feed "fine" fraction	. 41
Figure 20: SEM view of Ecuador gold material. Courtesy of Disa, LLC <sup>®</sup> [54]	. 43
Figure 21: SEM view of Ecuador gold material highlighting antimony minerals. Courtesy of	
Disa, LLC <sup>®</sup> [54]	. 44
Figure 22: a) 20x image of the Ecuador gold feed 75 µm fraction, b) 20x image of the Ecuador	r
gold feed 45 μm fraction	. 45
Figure 23: 20x image of the Ecuador gold feed fines fraction	. 45
Figure 24: BDH feed and product size fractions after classification and drying	. 47
Figure 25: Fines and 150 µm size classes of the BDH 4-minute product	. 48
Figure 26: a) 10x image of the BDH 4-minute product 355 µm fraction, b) 10x image of the	
BDH 4-minute product 125 µm fraction	. 49
Figure 27: 20x image of the BDH 4-minute product fines fraction	. 49
Figure 28: Potash 2-minute product size fractions: a) coarse, b) intermediate, c) fine	. 50

Figure 29: a) Image of the potash 2-minute product "coarse" fraction, b) Image of the potash 2-
minute product "coarse" fraction
Figure 30: Image of the potash 2-minute product "coarse" fraction
Figure 31: Particle size distributions for BDH products
Figure 32: Percentage of material retained on each screen for BDH products
Figure 33: Particle size distributions for potash 2-minute product
Figure 34: Percentage of material retained on each screen for potash 2-minute product
Figure 35: P <sub>80</sub> versus duration data for rod mill calibration
Figure 36: Particle size distribution for Ecuador gold feed and products
Figure 37: Percentage of material retained on each screen for Ecuador gold feed and products. 59
Figure 38: Size-Energy Relationship for the BDH material
Figure 39: Size-Energy Relationship for the potash material
Figure 40: Size-Energy Relationship for the Ecuador gold material
Figure 41: a) 20x image of the Ecuador gold HPSA 8-minute product 75 $\mu$ m fraction, b) 20x
image of the Ecuador gold HPSA 8-minute product 45 $\mu$ m fraction
Figure 42: 20x image of the Ecuador gold HPSA 8-minute product fines fraction
Figure 43: a) 20x image of the Ecuador gold rod mill 1-minute product 75 $\mu$ m fraction, b) 20x
image of the Ecuador gold rod mill 1-minute product 45 µm fraction
Figure 44: 20x image of the Ecuador gold rod mill 1-minute product fines fraction
Figure 45: Size-assay analysis of gold bearing particles in Ecuador gold feed and products 68

Figure 46: Knelson concentrator recovery for all experimental samples	59
Figure 47: Flotation mass yield and recovery for all experimental samples	59
Figure A-1: Ecuador gold lab assay results	35
Figure A-2: Ecuador gold overweight test lab assay results	36
Figure B-1: PSD plots and trendline fitting for BDH feed sample	37
Figure B-2: PSD plots and trendline fitting for BDH 2-minute product	38
Figure B-3: PSD plots and trendline fitting for BDH 4-minute product	39
Figure B-4: PSD plots and trendline fitting for BDH 16-minute product	<del>)</del> 0
Figure B-5: PSD plots and trendline fitting for potash feed sample9	<del>9</del> 1
Figure B-6: PSD plots and trendline fitting for potash 2-minute product9	<del>9</del> 2
Figure B-7: PSD plots and trendline fitting for Ecuador gold feed sample9	<del>)</del> 3
Figure B-8: PSD plots and trendline fitting for Ecuador gold HPSA 8-minute product	94
Figure B-9: PSD plots and trendline fitting for Ecuador gold rod mill 1-minute product	95

## Nomenclature

- AC Alternating current
- AG Autogenous
- **BDH** Black Diamond Holdings
- CMP Coal and Mineral Processing Lab
- $d_{80}-Particle\ size\ where\ 80\%\ of\ mass\ passes$
- $F_{80}$  Particle size where 80% of feed mass passes
- FCC Forward closed circuit
- g Gram
- $GS-Gaudin\ Schuhmann$
- g/t grams per tonne
- HPSA High pressure slurry ablation
- Hz-Hertz
- I-Current
- kg Kilogram
- kW Kilowatte
- L Litre
- L/min Litres per minute

MIBC – Methyl isobutyl carbinol

mL - millilitre

mm - millimetre

m/s – metres per second

 $P_{80}$  – Particle size where 80% of product mass passes

PAX - Potassium amyl xanthate

PF - Power factor

PSD – Particle size distribution

psig – Pounds per square inch gauge

RCC – Reverse closed circuit

ROM - Run of mine

RPM - revolutions per minute

RR – Rosin Rammler

SAG - Semi-autogenous

SEM – Scanning electron microscope

UBC - The University of British Columbia

 $\mu m - micrometre$ 

V – Voltage

VFD – Variable frequency drive

 $W_i - Bond \ work \ index$ 

## Acknowledgements

I would like to thank my supervisor, Dr. Sanja Miskovic without whom this research would not have been possible. I would also like to express my gratitude to the other members of my supervisory committee, Dr. Bern Klein, Dr. Marek Pawlik, and Dr. Scott Dunbar for their continuous support throughout my thesis development. Their encouragement, feedback, and adaptability were critical to the success of this work amidst the coronavirus pandemic.

I would also like to recognize the contributions of John Lee and the rest of the Disa, LLC<sup>®</sup> team for providing the equipment used in this study and results from their earlier experiments. Additionally, I would like to thank Nutrien for providing the potash samples from their operations.

I would like to acknowledge several other members of the mining department for their help throughout my degree: Santiago Seiler and Ayse Tugbe Cebeci for their guidance in the operation of the equipment in CMP, Libin Tong and Aaron Hope for providing ongoing equipment and facility support, and Dustin Weaver, Mewael Islet and Danny George for their emotional and technical support.

Last but not least I would like to express my sincerest appreciation to my family whose encouragement, love and unyielding support carried me through the inevitable struggles of a graduate degree.

# To Claire,

For picking me up every time this degree knocked me down.

You are, without a doubt, the best.

## 1 Introduction

### 1.1 Background

Within mineral processing circuits, comminution technologies are required to facilitate liberation of the valuable components from waste rock and enable their efficient concentration in the separation circuit. The liberation is often accomplished through the process of size reduction where run of mine (ROM) ore is broken down to a particle size that is several orders of magnitude smaller than the original size. In certain applications, selective comminution may be possible allowing preconcentration of certain components of the ore within a narrow size class, allowing for partial or complete separation of one phase of ore by screening prior to the downstream separation stages. The type of equipment used is based on feed composition and mineralogy, and different circuit configurations are often deployed at different operating sites.

In order to process the large daily tonnages commonly found in the industry, and due to the intrinsically low operation efficiencies, these size reduction or mineral liberation technologies carry extremely high operating costs. It has been found that the mining industry is responsible for approximately 3 - 7% of global electrical energy consumption; approximately 30 - 50% of this demand is used in size reduction, with 85 - 90% of this fraction being attributable to the grinding equipment [1]–[4]. Due to the equipment size and high rates of wear caused by the intensive particle-particle and particle-wall contacts, these units require ongoing maintenance followed by additional labour and material expenses. Combined with the growing concern regarding water consumption, these methods create sizable economic and environmental footprints.

Where pre-concentration is the primary operating target, the goal is often to liberate loosely bound soft minerals from the surface of hard mineral grains, where either the soft component (such as uranium oxides) or the hard component (such as potash) is valuable. This is generally accomplished in highly agitated attrition-scrubbing units that provide a high level of shear in the unit, causing abrasive interactions and generating liberation. However, for the same reasons stated previously this process also requires large energy inputs and continuous maintenance, which ultimately leads to the same economic and environmental challenges.

As ore bodies become more complex and grades decrease in quality, operations will have to further process the material to achieve the same degree of liberation. To make matters more challenging, the efficiencies in mining comminution circuits are notoriously poor, falling as low as 1 - 2% [4]–[6]. However, this affords opportunities for innovation as a small increase in efficiency can provide a large increase in profitability. One such innovation is the focus of this study: High Pressure Slurry Ablation (HPSA).

HPSA is a technology that employs a set of opposingly oriented nozzles to direct streams of high pressure high velocity slurry into an impinging jet region where particles are colliding causing breakage. The slurry streams collide either with each other (jet to jet) or a surface (jet to plate) based on the desired setup of the unit. The latter configuration results in an increased likelihood of particles colliding, which in theory increases the probability of particles being selected for breakage. The technology is novel and little literature exists on its application in conventional mining apart from uranium mining.

### 1.2 Thesis Objectives

This study aims to assess the potential gains to be made by using HPSA to cause mineral liberation and size reduction, and to understand the merits of the technology in its jet to plate configuration. In order to do this, several objectives were created:

- Carry out multiple campaigns in a pilot scale HPSA unit using three unique materials.
- Complete size analyses on product and feed samples to determine size reduction potential.
- Review product samples microscopically to provide insight on breakage mechanisms.
- > Determine if selective concentration exists through composition analysis where applicable.
- > Analyze how HPSA can enhance downstream separation recovery of precious metals.

### 1.3 Thesis Outline

This thesis includes a total of 5 chapters.

Chapter 1 introduces the goals of the project and delivers the necessary background information to understand these goals.

Chapter 2 reviews the current literature on relevant subjects. A basic understanding of mineral processing is presented. Best practices in industrial liberation are discussed, including energy-size reduction relationships. Finally, the HPSA technology and its current uses are explained.

Chapter 3 discusses the proposed methodology for the research. A description of the experimental equipment is given, followed by sample preparation procedures. Descriptions of the testing campaigns are provided, as well as how design parameters were selected. The chapter is concluded by the explanation of data analysis techniques that were used.

Chapter 4 performs the review of the experimental data collected. A characterization of the feed materials is presented, including size distribution and mineralogy. Selective concentration and size reduction results are discussed, supported by visual evidence and subsequent analyses. Size reduction-energy relationships and separation recoveries are compared to industry standards and the outcomes are discussed.

Chapter 5 combines the results of the previous chapter to draw conclusions and answer the goals of this study. Due to the youth of this technology, multiple opportunities exist to further this research in a variety of directions: several of them will be discussed and recommended.

## 2 Literature Review

#### 2.1 Mineral Processing Basics

The mineral processing sequence to transform ROM ore into a marketable concentrate has remained the same for centuries and commonly includes size reduction and control, enrichment, and upgrading. These stages allow operators to liberate valuable minerals through particle breakage, improve concentrate grades through gangue removal, and finally create both valuable products and safely disposable tailings [7]. This thesis will focus on the size reduction, or *comminution* step as its goals align with those of the proposed technology.

During comminution, ROM ore that can be multiple meters in diameter is broken down to particles as small as a few microns. The optimal size will be unique to each operation as it is highly dependent on physical and geomineralogical properties of the ore, as well as the limitations and requirements of the downstream units in the separation circuit. In most cases, comminution begins with a crushing step and is followed by a more targeted liberation method such as grinding or scrubbing. Crushing is used to reduce the particles in the feed to a size that can be safely and effectively processed by the following operation [8].

Various crushing technologies are available and are often run together in multiple stages. The selection of which are used often depends both on the feed top size and the required magnitude of size reduction. This requirement is often illustrated using a ratio of the feed and product stream  $d_{80}$  values. The  $d_{80}$  is a characteristic measurement that represents the particle size at which 80% of a the given stream mass is smaller. When represented as  $F_{80}$  and  $P_{80}$  for the feed and product respectively, this is commonly known as a reduction ratio and is denoted as  $F_{80}/P_{80}$ . [8].

### 2.2 Particle Liberation Technologies

#### 2.2.1 Overview

While the primary goal of crushing is to make the feed material more manageable, downstream comminution technologies operate with the target of optimizing the balance between liberation and cost. These concepts can be better represented by the known relationships between grade and recovery, and between processing operation costs and average particle size.

In an ideal world, a mining operation would be able to recover 100% of the target mineral from the deposit over the lifetime of the project. However, the only realistic way to achieve this is to eliminate the separation step from the operation; as some mineral will always be associated with gangue in the middling particles, it is impossible to achieve full recovery when separation is included. On the other hand, products with a higher grade are more valuable than those with larger gangue compositions. Yet this goal has the opposite problem of recovery: the only way to have a perfect grade is to only collect fully liberated or free particles. While a perfect grade is achievable, it often leads to poor recoveries that are unlikely to be feasible [9].

The balance of these concepts is known as the liberation-limited grade/recovery relationship. As recovery improves more gangue is naturally collected and the grade decreases. The opposite is also true: to increase product grade, middlings must be removed which subsequently reduces recovery. It has been shown that the only way to improve this relationship it through additional particle liberation, which is often achieved through comminution [10], [11]. By breaking down the ore, the amount of gangue associated with the middling particles is reduced and simultaneously new free particles are generated. This allows for an improved operating point with higher grades and recoveries, ultimately improving project outcome. This shift in the relationship is illustrated below in Figure 1.



Grade-Recovery Relationship

Figure 1: Liberation limited grade-recovery curves before and after comminution

From these curves alone one can conclude that operations should reduce their material as much as possible as this would lead to the best liberation-limited grade/recovery curve possible. However, size reduction becomes exponentially more difficult and more expensive as the average particle size shrinks. Simultaneously, the generation of ultra fine content can create additional complications in conventional mineral processing operations. At some point, the additional costs incurred from this comminution exceed the benefits to be gained. Therefore, most mills aspire to operate at the point where the overall costs are minimized. This optimal condition will be unique to each ore body and processing circuit, but Figure 2 below illustrates the general concept.



Figure 2: Cost analysis of mineral processing

From feasibility through to operation, the target of comminution circuits is to continuously optimize the balance between these two relationships: maximize liberation while minimizing costs. In most applications, size reduction is the desired goal of the process and so grinding is used to achieve this balance. In more specialized situations, where surface liberation is more relevant, scrubbing is often used instead. Both methods and common technologies will be discussed further.

### 2.2.2 Grinding

Grinding technologies aim to liberate valuable minerals by reducing their average particle size from up to 250 mm down to as small as under 15  $\mu$ m. However, the operation must avoid overgrinding the material; while the additional liberation will improve the grade/recovery relationship, their presence can have negative effects during separation, resulting in a net-negative effect on recovery [12]. Furthermore, these fines can take longer to settle out in solution, which can cause issues during dewatering and tailings management. These effects will all result in increased operational expenses, reducing project value and, thus, need to be properly managed. To achieve this goal, mills are often run in a *closed-circuit* operation, meaning a grinding mill is run in concert with a classification unit such as a hydrocyclone. In these operations, feed material is either sent directly to the mill (forward closed circuit, or FCC) or to the classifier (reverse closed circuit or RCC). The classifier removes material that has reached the desired size while sending the coarser oversize stream back for regrinding [12]. Examples of these configurations are shown below in Figure 3.



Figure 3: Closed circuit configurations of a ball mill circuit: a) FCC, and b) RCC

In many commercial mineral processing facilities, a grind size in the micron scale is often required to achieve profitable operation. This is particularly the case for ore bodies that have extremely low grades, such as in precious metals applications where gold composition is commonly well below 1%. To achieve this final grind size, multistage circuits are typically used with different grinding technologies at each stage in order to overcome the natural limitations on reduction ratio each technology exhibits. Many operations employ tumbling mills in at least one stage of these circuits, such as in the SABC (semi-autogenous (SAG) mill, ball mill, pebble crusher) circuit used commonly in industry today [13].

Tumbling mills operate by rotating a horizontal drum containing the mill charge to generate a centrifugal force. The mill is lined with steel lifters that help the force carry the material up the side of the mill wall. Once the force of gravity exceeds the centrifugal force the lifted mass falls from the shoulder of the mill and collides with the rest of the charge, resulting in breakage [14].

Several motion regimes relevant to grinding are possible in a tumbling mill, such as those illustrated in Figure 4.



Figure 4: Various motions of a tumbling mill: a) cascading, b) cataracting, c) centrifuging

Cascading and cataracting are the desired regimes for grinding and are often targeted by controlling the rotation speed of the mill; slower speeds facilitate cascading motions while higher speeds encourage cataracting. However, these regimes can only happen below the critical speed of the mill. As described by Gupta et al. the tumbling motion and therefore grinding is enabled so long as the centrifugal force is surpassed at some point by the gravitational force. Above the critical speed this fails to occur, leading to the centrifuging where all grinding action stops. To ensure this speed is not breached, most mills will measure mill speed as a percentage of critical speed, and operators will rarely surpass 90% of this value regardless of targeted operation [12], [15].

In certain applications it is necessary to produce an even finer grind than what is economically available through tumbling mills; creating product sizes of  $30 \ \mu m$  or smaller with tumbling mills while remaining profitable is virtually impossible. In these applications secondary grinding is used

to produce fine and ultrafine products, whose size classes can be nominally considered  $10 - 30 \,\mu m$ and  $1 - 10 \,\mu m$  respectively for this review. To generate these streams stirred mills are often used due to the significant energy savings that are seen when compared to tumbling mills [16].

Stirred mills have some similarities in operation to tumbling mills: both use a grinding media and a slurry as feed to liberate a valuable mineral. However, in this method a stirring mechanism is used to mix the charge within a static drum rather than rotating the mill itself. As the ore particles move within the mill they collide with each other, the grinding media, and the walls of the mill and ultimately break. The major differences between stirred mills are in stirrer geometry and orientation, however all designs fundamentally follow the above principle [17], [18].

While the above technologies are the current standard, they are all victim to the same inefficiencies. As there is no way to control exactly how particles will move within a mill, the nature of collisions contains a degree of randomness. Liberated particles may collide which offers no further improvements, and unliberated particles may collide but have insufficient energy to break [12]. These interactions result in wasted energy and decrease overall efficiency. As the abrasive collisions will generate heat via friction, losses also occur as this energy is consumed by the slurry [19]. While the exact efficiency of various mills to translate mechanical energy into liberating breakage has long been a source of debate, most researchers agree due to the above losses it is often 1% or lower [4]–[6].

#### 2.2.3 Washing/Scrubbing

While liberation is always a target in mineral processing, size reduction in certain situations is not necessary. For example, this is true in systems where loosely bound clays are connected at the surface to a coarser mineral grain. Two such applications where this is relevant are phosphate and potash deposits, where coarser grain sizes are desired for use in agricultural fertilizers. [20], [21].

11

Both phosphate and potash processing operations necessitate clay removal prior to downstream flotation to ensure successful separation. The lower settling velocities and higher surface areas of clay particles tend to result in the collecting agents undesirably adsorbing onto their surfaces, resulting in the generation of slimes [22]. These slimes then attach to the adsorption sites on the valuable mineral, rendering their surfaces hydrophilic and significantly reducing interaction with collector agents. As a result, this effective deactivation of mineral surfaces can lead to reduced recoveries, clay entrainment, and increased reagent costs and therefore must be avoided [23]–[25].

In most facilities, clay liberation is achieved using attrition scrubbers. These units operate autogenously at high mass fractions to encourage particle interactions. Slurry is continuously fed to the unit where it is agitated by impellers to promote a high frequency of vigorous particle collisions. These collisions generate the necessary forces to scrub the clay particles from grain surfaces, liberating them into the solution and allowing for desliming to occur [26]. These vessels usually contain multiple cells like the lab scale version shown in Figure 5, each of which can have a volume of up to  $10 \text{ m}^3$ .



Figure 5: Lab scale Denver flotation cell with attrition scrubber attachment.

Despite being the most popular technology for scrubbing, attrition scrubbers contain the same inefficiencies as grinding mills; there is no known mechanism to control the conditions of each particle collision and therefore the same degree of randomness is present. Additionally, as the scrubbing motion generates friction, increased amounts of heat are lost to the slurry. While these units can offset some of the losses by running at higher solids contents, these scrubbers join grinding mills in requiring high energy inputs to operate successfully.

#### 2.2.4 Mechanisms of Breakage

Comminution equipment creates liberation by placing particles under stress, typically through one of five forces: tensile, compressive, impact, shearing, or attrition. In most comminution equipment, all but tensile forces can be applied through particle collisions. These collisions occur in one of the three ways shown in Figure 6, each generating its own form of breakage. Different liberating technologies will apply some or all these mechanisms, and in nearly all cases one form will dominate. As the success of these mechanisms varies depending on particle size, how dominant each method is plays a significant role in determining the functional range of the equipment [12].



Figure 6: Methods of collision and resulting breakage: a) impact or compression, b) chipping or attrition, c) abrasion. Adapted from Wills & Finch [12].

The compressive/impact mechanism shown in Figure 6a is the most successful in coarser applications in the particle size order of 1 - 50 mm. When particles collide in this way, a

compressive impact force occurs at the location of collision acting parallel to the motion of the particle. This load results in a tensile force that acts perpendicular to the compressive force, ultimately pulling the particle apart [27]. Cracks propagate throughout the volume of the particle, manifesting in ways that split the original particle into several relatively equal sized progeny. Due to a high energy density at the site of impact, further size reduction occurs in the local area compared to the reduction exhibited throughout the rest of the mass. These interactions are illustrated below in Figure 7.



*Figure 7: Resulting particle breakage due to impact/sudden compression collisions: a) falling motion, b) impact and compressive force, c) resulting tensile force and crack propagation.* 

This mechanism is the most successful in coarse applications for two reasons. Firstly, due to their increased mass, coarse particles supply a higher collision force either through gravity or momentum, leading to a higher probability that the stress will surpass the tensile strength of the material. Secondly, the relatively large diameter of the particles results in a significant reduction of surface area. As abrasive and attrition forces are surface acting, this means they can only affect a fraction of the total particle mass, making the compressive mechanism more effective [12], [28]. Since the 1960s autogenous (AG) and SAG mills have been the equipment of choice in coarse grinding, beginning with ferrous bodies and expanding into base and precious metal deposits.

These mills are characterized by their large aspect ratio (diameter to length) and a low or nonexistent use of grinding media [29]. The large diameter of the mill allows for higher throughputs, leading to reduced costs on a per ton basis. Additionally, as particles have a larger distance to fall, collisions with higher energy will occur. Finally, as the mills can handle material similar in size to secondary and tertiary crushers, AG and SAG mills often eliminate the need for these stages [12].

While compressive breakage is effective in coarse grinding, it does have limitations. As primary mills are designed to break ore with a specific mineralogy and size distribution, their efficiency becomes vulnerable to variations in these characteristics. Additionally, these circuits generally use an additional pebble crusher on the screen oversize. This leads to better control of the product size but requires additional energy input. Finally, these impacts lead to significant wear in the mill liner, requiring maintenance and downtime [30].

Attrition and abrasion collisions, like those shown in Figure 6b and 6c, become increasingly significant as particle sizes decrease as a result of their surface acting nature. Smaller particles exhibit higher surface area to volume ratios; these forces can therefore be applied to larger portions of the ore as the pieces become smaller. This translates to applications in fine and ultrafine grinding and, when combined with impact collisions, in finer tumbling mill applications [28], [31].

These collisions generate breakage through shearing forces. The velocity vectors in these occurrences are not perfectly aligned, causing particles to collide tangentially. As this happens, small pieces are chipped or sheared away from the surface, leaving the remaining volume relatively unchanged. This leads to progeny where one large product is similar in size to the original particle and multiple products are at least a scale of magnitude smaller [32]. These interactions are shown below in Figure 8.



Figure 8: Resulting particle breakage due to abrasive or attrition collisions: a) tangential motion, b) surface forces and breakage

In tumbling mills, these forces are best seen in ball mills as they are often operated with speeds and geometries that create both cataracting and cascading motions, leading to impact and surface mechanisms. This combination allows ball mills to frequently operate as the final stage of grinding before the product is sent for separation. If secondary grinding is needed, stirred mills exclusively produce breakage using this mechanism to capitalize on the increased surface area at this size [12]. Surface mechanisms become exceedingly more important in stirred mill applications where impact breakage is non-existent. In these operations, the stirring motion suspends the charge in the mill, which allows it to rub with the mill liner or other particles in the slurry. This generates the friction that leads to abrasive breakage, a process well described by Gupta et al.: "It is generally accepted that the mechanics of grinding in vertical and horizontal stirred mills is primarily due to abrasive action and not by crushing as in ball mills. The grinding action is brought about by the tangential velocity produced by a rotating spiral, pins or perforated discs" [18].

Like impact breakage, the attrition and abrasion mechanisms have limitations and must be selectively employed to ensure successful operation. As these forces are surface acting, they fail to interact with large portions of coarse particles, thus, these mechanisms have poor performance in coarser applications. Additionally, due to the friction generated in these collisions, thermal energy is produced which is immediately lost to the slurry, resulting in reduced efficiencies. [12].

#### 2.2.5 Grinding Energy Laws

Over the past two centuries various formulas have been proposed to relate particle sizes and comminution energy requirements in comminution circuits. The earliest efforts were conducted by Peter von Rittinger in 1867 and Friedrich Kick in 1885. They postulated that the energy required to reduce the size of a particle is a function of both the initial and final sizes [33], [34]. Walker et al. (1927) recognized that these formulas share similarities and found that both can be derived to a single differential equation [35]. The equation is as follows:

$$dE = -C\frac{dX}{x^n}$$
 Equation 2.1

where dE represents the small amount of energy required to make an incremental change in diameter dX occur. C and n are constants that are dependent on the system [36]. When n = 1 and 2, integration yields Kick's and Rittinger's laws respectively:

$$E = Cln(\frac{X_1}{X_2})$$
 Equation 2.2

$$E = C\left(\frac{1}{X_2} - \frac{1}{X_1}\right)$$
 Equation 2.3

These equations show the major difference in the two earliest laws. In Kick's law, the equation proposes that energy requirements are related to the reduction ratio. This implies that energy requirements are the same whether the particles are millimetres or micrometres in size so long as the ratio is maintained. In Rittinger's law, it is proposed that energy requirements are inversely proportional to the diameter. "Since the surface of unit mass of material is proportional to (1/X),
the interpretation of this law is that the energy required for size reduction is directly proportional to the increase in surface" [37].

While both laws hold true in certain applications, neither is successful in most energy approximations. Rittinger's law works in finer applications where high surface areas are present, while Kick's law is effective in coarse applications where fracturing occurs through surpassing the tensile strength [37]. In 1952 Fred Bond proposed a formula that lies within the range of the two laws by using n = 1.5 prior to integration, resulting in the equation below [38]:

$$E = C(\frac{1}{\sqrt{X_2}} - \frac{1}{\sqrt{X_1}})$$
 Equation 2.4

Bond discussed in his work that *C* is unique to each mineral and can be determined. He considered that the amount of energy required to reduce a particle of infinite feed size down to 100  $\mu$ m is called the work index, denoted *W<sub>i</sub>*. When a characteristic passing size is used in place of *X*, the formula may be rearranged and solved for W<sub>i</sub> and C like so:

$$W_i = C\left(\frac{1}{\sqrt{100}} - \frac{1}{\sqrt{\infty}}\right) = \frac{C}{10} \rightarrow C = 10W_i$$
 Equation 2.5

$$E = 10W_i(\frac{1}{\sqrt{P_{80}}} - \frac{1}{\sqrt{F_{80}}})$$
 Equation 2.6

Values of W<sub>i</sub> for various minerals are supplied by Bond in his original work [38]. Equation 2.6 is widely accepted as the most effective approximation of grinding energy requirements in today's industry due to the balance it achieves of the proposals from Rittinger and Kick. However, due to the stringent requirements of Bond's work index test, several empirical correction factors are generally applied to improve accuracy. A graphical representation of all three laws can be seen below in Figure 9.



Figure 9: Comparison of the three comminution laws of Rittinger, Bond, and Kick. Adapted from Mayer-Laigle et al. [39]

# 2.3 High Pressure Slurry Ablation

#### 2.3.1 History and Applications

High Pressure Slurry Ablation is a new technology, only having been patented by Coates et al. in September 2012 [40]. However, the concepts used in the success of the technology date back to a 1932 patent by Andrews et al. [41] who attempted to use jet on jet contact to pulverize various minerals. In their patent, Andrews et al. discuss how the "head-on impact" of two jets can result in the size reduction of any particles within the streams. While their technology led to integrity issues (caused by unaligned jets colliding with chamber walls) they did show that significant collision energy can be achieved through pressurized slurry streams [41].

Another patent that led to the invention of HPSA is that of Dunn et al. in 1951, who attempted to increase vanadium and uranium recoveries using attrition-based scrubbing action. In their

technology, the inventors found significant levels of the valuable minerals existed in the softer mineral phase that coated the outside of harder quartz grains. Dunn et al. proposed that by submitting these grains to high attrition forces, they could separate the softer materials from the larger grains without breaking the quartz, allowing the two phases to be separated on the basis of size class [42].

Several other patents have been proposed in the time since Dunn et al. that employ one or both of the above concepts; yet Coates et al. are the first to invent a slurry-based jet to jet ablation technology. Within their patent, Coates et al. list that this technology has seen success with several materials such as silicate-gold systems, oil contaminated sands, and uranium processing [40]. However, most of the work has been done in uranium-silica deposits. Black Range Minerals Colorado, LLC discuss in their white paper how HPSA (in their paper termed "Ablation Mining Technology", or "AMT") can disassociate the uranium bearing mineral coating from the coarser silica-based sand grain. In doing so, the sand grains remain at much larger size classes than the uranium, thus allowing for a finer concentrate to be separated from a coarser gangue. In this service, it has been shown that approximately 95% of the original uranium can be concentrated in the sub 400 mesh size class, which accounts for approximately 10% of the total mass [43].

By combining the high levels of collision energy given by pressurized jet to jet collisions with the success seen in abrasive separation of a softer material, Coates et al. were able to create a technology that provides valuable concentration of the finer size class. Within their patent, they also include configurations that allow for a jet to collide with a solid surface rather than an opposing jet [40]. While it is proposed by the researchers of this study that the technology has applications in size reduction, there is little literature in this mode of operation, thus, the technology has to date been primarily considered as a scrubbing device.

## 2.3.2 Breakage Mechanisms and Theory

HPSA employs the same traditional breakage mechanisms as all comminution technologies that are outlined in Figure 6, yet where HPSA uniquely excels is the significantly reduced degree of randomness in particle collisions. Due to the narrow exit diameter of the nozzles, the size of the impinging region is sizably reduced, resulting in the probability of particles colliding being much higher. When combined with the larger collision forces that result from high particle speeds, this can increase the probability of a particle being selected for breakage, potentially leading to improved energy efficiencies in the breakage process.

As a pre-concentration technology, HPSA would be comparable in application to the attrition scrubbers discussed in Section 2.2.3. In both technologies, high shear environments are created through large particle velocities which lead to the kinetic energies required for breakage to occur. This mechanism is well described by Nienow and Conti [44] in their paper on stirred vessels, and is discussed by Bemrose and Bridgewater [45] in their review paper on attrition. As previously discussed, these shearing forces are surface acting, explaining their success in separating loosely adhered surface minerals from a coarser mineral matrix.

When compared to an attrition scrubber the HPSA has some operational benefits. Equipment wear is reduced due to the lack of stirring impellers and the decreased number of particle-wall collisions, and as the collision zone is not within a vessel there are no issues regarding flow patterns or dead zones. However, as the HPSA requires pumps to generate pressure there are additional operational considerations. Wear is reintroduced in the pump housing, and there will likely be additional mechanical loses between the pump and motor. Additionally, slurry rheology changes from Newtonian to non-Newtonian at high solids concentrations, which may impact power draw and pump wear [46]. As the literature for HPSA has currently tested the technology up to 20% solids

21

by mass the technologies may be compared on a conceptual basis, however further research must be conducted using HPSA at higher solids contents before it can be considered as a commercial alternative.

A screenshot from a simulation of the HPSA collision zone is shown below. The image depicts how the jet to jet configuration can generate high energy particle-particle collisions in the impinging region through the employment of attrition and abrasion. The technology also has the option of a jet to plate configuration, achieved by inserting a steel plate between the nozzles. This would provide a collision zone dominated by impact rather than attrition and abrasion. Therefore, it is considered that this operation would be better suited to targeting size reduction of coarser ore particles rather than pre-concentration through scrubbing and aggregate dissociation. However, no studies have yet been published with HPSA targeting size reduction, providing the opportunity that inspired this study.



Figure 10: Simulation showing how the jet to jet configuration creates stream-based collisions [47].

# 2.4 Summary of Review

Grinding and liberation technologies have become valuable to an increasing number of industries since their initial consideration over a century ago. Grinding mills are commonly used in systems where size reduction is required, and attrition scrubbers are popular in applications where maintaining feed size is desirable. To optimize process design, several authors in the literature outline empirical methods to estimate the energy required to reduce a mass from one size to another. However, due to the random nature of collisions and heat generation exhibited by these machines, breakage efficiencies remain extremely low.

High Pressure Slurry Ablation is an emerging technology that uses high velocity slurry streams to create abrasive particle collisions. This equipment has shown success in the separation of uranium mineral coatings from the underlying silica grains, achieving over 90% recovery while processing only 10% of the mass. It is proposed that the geometries of the technology can reduce the randomness of the process, therefore, offering improved efficiencies compared to common commercial equipment. This study will review the merits of this proposal through campaigns of various materials and subsequent analyses.

# 3 Design of Experiment

In order to achieve the primary goals of the project, testing was performed on three unique materials: a silica-based tailing from the Black Diamond Holding (BDH) group, a potash ore provided by Nutrien from Saskatchewan, and an arsenopyrite-based gold ore from Ecuador. Each material was tested in the pilot scale unit and products underwent size analysis for particle size distribution (PSD) generation. Further analyses such as gravity concentration and flotation were also performed on the gold material to better understand the effects of HPSA on the recoveries this technology can achieve. The results from all three experimental campaigns were then collectively reviewed.

# 3.1 Methodology

Sample materials were provided by Disa, LLC<sup>®</sup> and delivered to the UBC CMP facility for our use. As the jet to jet configuration has shown success in the literature, it was of interest to understand how impact breakage would affect the outcomes of the process. Therefore, all materials underwent testing in the HPSA unit with the jet to plate configuration. Size analysis in all tests were completed using screens that followed a  $\sqrt{2}$  sequence in accordance with industry standards. Due to the corrosive nature of potash when in contact with steel and concrete, additional measures needed to be taken to ensure all other lab assets remained undamaged during this testing scheme. For dry operations this was achieved by engaging the dust collection system and ensuring adequate ventilation in the lab size. Wet operations were confined to the pilot unit and the wet grinding room as much as possible as the concrete in this room already has a protective coating. Excess rinsing cycles and stainless-steel equipment were also used to further mitigate asset integrity risks.

# 3.2 Experimental Equipment

The critical components of the HPSA unit consist of a central collection tank, two centrifugal slurry pumps and two opposing jet nozzles. A picture of the lab scale pilot rig is provided below.



Figure 11: High pressure slurry ablation lab scale pilot unit

The two pumps used were equivalent to the Warman AH 1.5/1 centrifugal slurry pump, each powered by a Siemens 15 kW three phase induction motor. The discharge of each pump flowed through a two inch rubber hose to the nozzle inlet where an analog pressure gauge was placed. This pressure was used to confirm each pump provided equal flow. The north pump (left in photo) was also fitted with a sampling line to allow for product collection during testing.

The nozzles used in the experimental campaigns have nominal diameters of 50.80 mm (2") at the inlet and 19.05 mm (3/4") at the outlet. This allowed the pumps to create collision velocities of up to 20 m/s. Due to the modular nature of the lab equipment, these nozzles could be replaced with varying geometries if lower or higher velocities were desired. The collection tank used in the pilot

scale was a 1,000L cylindrical vessel with a conical bottom. Each test used less than 10% of this volume, therefore no overflow concerns were present. Net positive suction head of the pumps were also not relevant in this study; safety tests using water were performed with test volumes as low as 10 gallons and no cavitation was present.

Various lab equipment was employed in the analysis of samples during the various testing campaigns of this study. Nine inch screens were used with a ROTAP in all wet screening processes, as well as a pressurized filter press for dewatering. Where separation was necessary, a Knelson concentrator was used for gravity separation, and a benchtop Denver cell was used for flotation. Examples of units used in this study are provided below in Figure 12.



Figure 12: Various lab equipment: a) 9" screens, b) ROTAP, c) filter press, d) Knelson Concentrator, e) Denver flotation cell

## 3.3 Selection of Variables

To adequately represent the system three variables were chosen: the impact velocity of the particles, the average number of cycles through the unit each particle observes, and the mass fraction of the slurry. These variables can successfully encompass the factors affecting the outcome of the process and each will be reviewed in detail.

It is well known that the magnitude of a collision force is proportional to the change in velocity. Therefore, it is intuitive that an increase in collision velocity results in an increased collision force. This results in the amount of energy transferred to the particles increasing, theoretically improving liberation mechanisms. However, collision velocity is nearly impossible to directly measure in this application. To control and capture this variable, the flow rate from the pump will be used. As the pressures at the inlet and outlet of the nozzle are known, Bernoulli's equation can be applied. Due to the law of conservation of mass and incompressibility of the fluids, the flow rate will remain constant, and the resulting inlet and outlet velocities can subsequently be calculated. This flow rate can be easily controlled through the variable frequency drive (VFD) of the pump, providing the first operating variable.

The mass fraction of the slurry plays a significant role in the breakage of individual particles. By increasing the number of particles in the system, it can be expected that the number of particles colliding with the plate per unit of time is also increasing. At the same time, the number of particle to particle and particle to wall interactions in the hoses will also increase. Finally, it can also be considered that an increase in the percentage of solids in the slurry will lead to a decrease in specific energy costs as it is a measurement of energy consumption per unit mass, which is a desirable outcome of this study. As mass fraction is determined through sample preparation, this easily encompasses these effects and provides the second variable for the study.

It is intuitive that the degree of liberation can be related to the number of times each particle passes through the unit; by increasing the quantity of collisions each particle faces, you also increase the probability each particle will be selected for breakage. The measurement of this variable must be calculated through a combination of flow rate, run time, and sample volume. By using a known volume and calculated flowrate (determined by the desired velocity) the time needed for a single pass can be determined. By controlling the duration of a test, the desired number of passes can be achieved, and so run time provides the third operating variable.

The available operating range for particle speed and mass fraction are up to 20 m/s and 25% respectively. Both variables are limited by the capabilities of the pumps in use, and therefore could be exceeded in future work through equipment upgrades. The number of particle collisions has no such conditions as test duration can be increased to any desired level. Each campaign used the maximum 60 Hz frequency to obtain the 20 m/s velocity, which translates to a nozzle inlet pressure of approximately 35 psig. However, as each campaign contained different analyses and conditions for the other variables they will be discussed separately in the following sections.

As the goal of this study was to assess the merits of the technology and potentially provide proof of concept, scalability and sizing approach were not considered within the scope of work. However, it can be suggested that when increasing the size and throughput of the technology these operating variables must be kept constant; altering the conditions of particle interactions will likely alter their effectiveness. While the desired variable levels may be different for each operating target (ie: size reduction vs selective concentration), the levels that are proven to be effective for that operation must be the same for the upsized unit as they are during pilot-scale work.

# 3.4 Sample Preparation

Due to the physical geometry of the nozzles it was impossible for particles greater than the outlet size to be processed in the HPSA unit; if such a particle was present in the system asset integrity issues such as over pressurized hoses or deadheading pumps could have occurred due to a blocked flow. To eliminate this possibility all testing materials were screened at 6.73 mm before processing to ensure all particles would safely pass through the equipment. This size was selected due to a safety factor of 2.5 - 3.0 being desired and it aligned well with readily available screen sizes. Once the materials were screened they were blended to ensure material homogeneity.

To control the slurry solids percentage in each test, densities for each material were required. For the gold material this value was provided by Disa, and a value for BDH was approximated based on composition. Potash density was calculated using the displacement principle in a graduated cylinder, and tests were conducted in triplicate and the average of the results was used.

Due to its high solubility in water, all potash tests required saturated brine to be used as the carrier fluid. To generate the brine two 110 L plastic drums were used. Hot water was added to the drum, followed by raw ore while simultaneously being agitated using a stainless-steel mixer. The mixture was left to stir for an hour and then checked: if the ore had fully dissolved more ore was added, and the process was repeated. Once an hour had passed with some ore remaining undissolved, the drums were covered and allowed to settle for 48 to 72 hours. This allowed the water to cool to room temperature and the insoluble materials to segregate into a bottom layer. After this settling period the solution was considered saturated if salt crystals had formed within the drum. The brine density and solubility were then measured using displacement and evaporation tests respectively.

Using the determined densities and available material, samples were generated for experimental tests. For the BDH and gold materials a riffle splitter was used to create these portions while ensuring each sample remained representative. For the potash this was not possible due to corrosion issues. Alternatively, a cone-and-quarter method was applied while maintaining airflow and dust collection. For these tests brine was transferred from the generation drums to sample buckets via siphon. Once correctly weighed for a given test the carrier fluid was introduced to the unit and the pumps were brought up to speed. Materials were then added to the unit over a timeframe of 60 seconds to avoid shocking the system.

# 3.5 Schedule of Experiments

## 3.5.1 Campaign 1 – Black Diamond Holdings (BDH) Tailings

The test of the BDH material was run at a sample volume of approximately 30 gallons with a solids fraction of 10% by mass or 4% by volume. This was to ensure there was enough raw material should additional testing be required. Samples were collected immediately after the fresh material was added and then at 2, 4, and 16 minutes after this point. These times resulted in an average of 5, 11, and 42 passes per particle respectively. The initial sample was considered the feed for the campaign to ensure any minor grinding that occurred during ore addition was accounted for. These samples were collected in 2-gallon pails, then settled overnight prior to size analysis to facilitate segregation and decantation of any free liquid.

Through analysis of the feed a top size screen of 600  $\mu$ m (30 mesh) was determined to be adequate. Using a deck of ten screens the product was split into size classes down to 45  $\mu$ m (325 mesh). The remaining fines from this analysis were dewatered in a pressurized filter press, then all size classes were dried overnight and subsequently weighed. These weights were used to generate a resulting PSD for the product, and the dried material was then qualitatively analyzed for clay content. This was accomplished by comparing the texture of various samples as well as by collecting microscopic images of both feed and products. This process is illustrated below in Figure 13, with the blue boxes indicating testing steps and the gold boxes representing outputs.



Figure 13: Flowsheet of BDH testing campaign

#### 3.5.2 Campaign 2 – Nutrien Potash

For the testing of the potash a sample of approximately 25-gallons was used with a solids fraction of 20% by mass or 13% by volume. Products were collected at 2, 5, and 8 minutes along with the feed sample, resulting in 6, 16, and 25 passes per particle respectively. During testing the amperage of the operational pump was measured at 30 second intervals using a remote access terminal controlled through a smartphone. Using the input voltage to the motor and an average value for power factor, these amperages could be converted into power draw using the three-phase alternating current (AC) power equation. As was the case with the BDH tailings the products were collected in 2-gallon pails and allowed to settle over night to segregate free liquid.

It is important to note for the size analysis of potash that extra salt weight from the brine must be considered. As the water from the retained brine in the samples evaporates, all salt present in the brine will be deposited. Due to the higher surface area of the finer classes more brine will remain in the wet ore, resulting in larger salt deposits and greater potential for experimental error. To avoid this issue the deposited salt must be calculated and removed from the size class. A separate size analysis test was conducted to measure the amount of water removed from each size class by weighing the sample before and after drying. Using the calculated density of potash and the nominal particle size for each size class, the mass of each particle can be related to its surface area. As the amount of brine retained is proportional to surface area this allows for the determination of water retained per unit mass in each size class, a value which can be used for any potash test. The deposit weight can be found by converting this water mass to salt mass using the solubility, which can then be subtracted from the dry mass to reveal the sample weight.

As the size distribution of the ore is considered proprietary by Nutrien, all size classes will be normalized to maintain confidentiality. The screen deck used in this test consisted of thirteen screens. Fines were dewatered in a filter press and then all size classes were weighed wet and allowed to dry over night before being reweighed. However, as potash is corrosive to steel all equipment used was made of stainless steel. The filter press also contained a rubber lining to further reduce corrosion issues from leakage.

Clay content could again be viewed qualitatively through visual assessment, however in this case a quantitative review can be achieved through dissolution. Following collection of weights for the PSDs, the samples could be dissolved in an abundance of water. This would leave behind only the insols, which could be dewatered, dried, and reweighed. Comparing these weights to the original size class sample would afford the opportunity to understand the location of the fines before and after HPSA, and to review if the process provides additional separation compared to wet screening. The overall testing plan for the potash material is shown in Figure 14.



Figure 14: Flowsheet of potash testing campaign

#### 3.5.3 Campaign 3 – Ecuador Gold

Due to the availability of additional analysis equipment as well as the interest of the stakeholders, the Ecuador gold material underwent a more extensive campaign than the other materials. The HPSA test was performed using a sample of approximately 23-gallons with a solids fraction of 20% by mass or 7% by volume in order to again conserve material in the event future testing was required. In this case the feed sample was collected along with a single product at 8 minutes, or an average of 25 passes per particle. This product was split into several portions to allow multiple analyses to be completed. Unit amperage draw was again measured in 30 second intervals to generate an average power draw for the test. One product portion was used for a size analysis with a screen deck of eight screens containing a top size of 600  $\mu$ m (30 mesh) and a minimum size of 37  $\mu$ m (400 mesh). The size classes were again dewatered, dried overnight, and weighed to produce a PSD.

As the performance of HPSA in this application was to be compared to that of tumbling mills, a product set was also desired from the lab scale rod mills. However, to effectively compare the gold recovery of the two processes, the  $P_{80}$  of the two products needed to be comparable. To determine the required milling duration in the rod mill to achieve this size, two 1-kg subsamples were run in a rod mill for 5 and 15 minutes respectively at a mass fraction of 60% with 30 rods in the mill. These products were then analyzed with the same screen deck as the HPSA and the  $P_{80}$  of each product was determined. Using these values along with the passing sizes of the feed and HPSA product, a desired run time was determined. Fresh material was then ground in the rod mills, separated into several portions like the HPSA, and one portion underwent the same size analysis. The resulting product size was then compared to the HPSA product to ensure similarity.

Following the collection of the products, a portion of the rod mill and HPSA product underwent identical separation testing. A portion of the feed material also underwent this testing plan in order to see the incremental change in recovery that resulted from the size reduction processes. Sample was first fed to a Knelson concentrator for separation of any free gold. The Knelson was operated at a water flowrate of 3 L/min and a centrifugal force of 60 G's. The tails and concentrate streams were collected and the tails were temporarily set aside.

During HPSA and rod mill product testing the concentrate stream was split over a 400-mesh screen to determine if the gold preferentially concentrated in the fines. The two products were then panned down to a mass of approximately 10 - 20 g, dried, and sent to the lab for fire assay. During the raw material testing a similar approach was conducted, however only the free gold mass was desired. As the size class distinction was not of interest the screening was not conducted prior to panning. In all three iterations of the procedure the tails from the gold panning processes were then recombined with the Knelson tails and served as the feed for a flotation test.

The flotation tests were conducted using MIBC as a frother and PAX as a collector due to the ore having a high sulphide content. Both reagents were dosed at 50 g/t. Slurry pH was maintained above 9.0 to ensure effective interaction between the reagents and particle surfaces. Air was bubbled into the cell at 5 L/min, and the agitator was set to approximately 1300 RPM. Froth was collected for 2 minutes, at which point it was visually concluded that all material had been floated.

Like the Knelson products it was relevant to know if the gold preferentially concentrated into the finer portion. To determine this the concentrate and tailings products were also separated over a 400 mesh screen before being dewatered and dried. Once dry the weights were collected to determine flotation mass yield, then a 30 g portion was split from each product to be sent for fire assay. To illustrate this testing scheme and the resulting products, the flowsheet for the HPSA products is provided below as Figure 15.



Figure 15: Flowsheet of HPSA material is Ecuador gold testing campaign

# 3.6 Data Analysis

 $P_{80}$  values were calculated for each data set for comparative review. To ensure the best fit line was used for all size analyses, both Gaudin-Schuhmann (GS) and Rosin-Rammler (RR) methods were applied in this study. Both trendlines were fit to the respective data and the better fit was used for calculation of characteristic passing size. This was determined by the trendline with the highest  $R^2$ value. If neither method provided an effective passing size estimation, a third-degree polynomial function was fit to the curve up to the GS theoretical top size for better representation. All trendlines and data sets are provided in the appendix.

To review the specific energy requirements of the HPSA technology, a size-energy relationship for each material was determined using collected amperage readings and compared to Bond's equation as shown in Chapter 2.

During the Ecuador gold testing scheme, assay data for each product was provided by ALS. Using this data and the collected weights a full metallurgical balance was generated. Head grade of the material was determined as the average of the testing campaigns, and this average was later used to calculate process recoveries. As these assays provided data at  $\pm 400$  mesh, they were also used to determine if the grinding operations concentrated the material into one size class.

# 4 Experimental Results

# 4.1 Feed Characterization

# 4.1.1 BDH Tailings

For the purposes of calculating mass fraction, the specific gravity of this material was considered to be 2.65. This value was chosen as it is expected that the material is made up nearly entirely by quartz and secondarily by clays, all of which have specific gravities similar to that of silica. Using a microscope, photos were collected of the feed material at a coarse, intermediate, and fine size. These images with their respective size classes and magnifications are provided below in Figure 16 and 17, as well as a scaled up portion of the coarse fraction image.



Figure 16: a) 10 x image of BDH feed 355 µm fraction, b) 10 x image of BDH feed 125 µm fraction



*Figure 17: a) 20 x image of BDH feed fines fraction, b) scale up of the red zone in Figure 16a, with the trapped fines circled.* 

These images show that even with wet screening the clays begin to collect in the fines. As this is a common outcome for the desliming of surface bound clays, it supports the expectation that the sample displays the loosely adhered mineralogy which was a key to success in the uranium-oxide project, providing evidence that the tailings are a good candidate for the HPSA technology. However, there are some residual clays in the non-fine portions. These unscreened fines are visible as the darker flecks on the surface of the coarser particles, such as the cluster magnified in Figure 17b. This inefficiency indicates that concentrations beyond that of desliming are possible, further supporting the candidacy of the material for testing. No cleavage is present in the images of this mineral, rather the particles exhibit a conchoidal fracture pattern. This is a common characteristic for quartz particles and supports our assumption of a quartz grain [48].

The results of the size analysis on the BDH feed material are presented below in Table 1. The full data set as well as the trendlines are available in the appendix.

Model	P <sub>80</sub> (µm)	$\mathbb{R}^2$
Gaudin-Schuhmann	359	98.24
Rosin-Rammler	294	95.12
Polynomial Fit	327	99.66

Table 1: PSD model results for BDH feed material

This material has an evenly spread size distribution with a relatively small top size. As nearly all material passes through a screen less than 1 mm in size, this indicates that little to no prescreening would be necessary prior to feeding this sample to the HPSA. The GS trendline is the better fit in this application with an  $R^2$  value of 98.24, however it overestimates the  $F_{80}$  value. Therefore, the third order polynomial was more effective at determining this value for discussion.

It is worth noting that only 10% of the total sample mass is originally present in the  $-45 \,\mu\text{m}$  fraction. As wet screening was used to accomplish this task, the assumption can be made that a large portion of this can be contributed to the clays being partially washed from the grains. Even if the fraction were to double in size due to the HPSA scrubbing, the large majority of the quartz can be reclaimed if the clays are effectively concentrated into this fraction. This further supports our initial analysis that the material is a good candidate for HPSA testing.

#### 4.1.2 Nutrien Potash

The average measured specific gravities for the saturated brine and potash materials are displayed below in Table 2. For context, values for the relevant components of the ore are also provided.

Material	Specific Gravity
Saturated Brine	1.24
Potash Sample	2.03
Sylvite	1.98
Halite	2.16
Insoluble Clays	2.60

Table 2: Specific gravities of potash materials. Component values from Fuzesy [49]

The calculated potash value aligns well with expectations, being slightly higher than that of sylvite. The material is predominantly mined from the Esterhazy member, a portion of the potash deposit found in the southeastern corner of Saskatchewan. The potash from this area is known to be predominantly made of halite (NaCl) and sylvite (KCl) with minor fractions of carnallite (KMgCl<sub>3</sub>·6H<sub>2</sub>O) and water insoluble clays [20], [49]. As the halite and insols both have higher gravities than the sylvite, the presence of these minerals likely resulted in the higher value calculated in the lab.

Using the evaporation test an average salt solubility was determined to be approximately 455 g/L of water. This value translates to a water mass fraction of approximately 69% which matches well with literature values for Saskatchewan deposits [20], [50]. As the brine was collected from buckets showing crystallization products, it was deemed saturated and ready for use.

Pictures were again collected from three size classes and viewed under a microscope to review the surface properties of each particle. However, in this case the scale of the microscope and the particle size classes have been removed as Nutrien considers the size distribution of the ore proprietary. Therefore, they have not been included to maintain confidentiality. These images are provided below as Figures 18 and 19.



Figure 18: a) Image of the potash feed "coarse" fraction, b) Image of the potash feed "intermediate" fraction



Figure 19: Microscope image of the potash feed "fine" fraction

These images support the findings within literature that the clays are present on the surface of the salt crystals and filling in the spaces of the matrix. The darker shade of the fines is a result of clay elevated clay content, indicating once again that initial screening is sufficient to partially deslime the material. Some cubic cleavage can be observed in the salt crystals, which is a characteristic

crystal structure for sylvite and halite particles. The orange-red hue seen in many of the particles can be attributed to discouloring from ferrous oxides [49].

The size distribution trends for the potash are below in Table 3. To maintain the previously mentioned confidentiality, all sizes have been normalized to the best fit characteristic passing size, which belongs to the polynomial fit.

Model	Normalized P <sub>80</sub> (-)	$\mathbb{R}^2$
Gaudin-Schuhmann	1.09	98.59
Rosin-Rammler	1.03	97.94
Polynomial Fit	1.00	99.43

Table 3: PSD model results for potash feed material

Like the BDH sample this material exhibits a flatter distribution that is spread across multiple size classes. The GS trendline is again the better fit in this application with an  $R^2$  value of 98.59, while a third order polynomial would be more effective for determining  $F_{80}$ . However, this distribution shows a much lower percentage in the finest fraction at less than 3%. This could provide competitive clay concentrations to that of the attrition scrubber should the clays be effectively confined to this size class.

#### 4.1.3 Ecuador Gold

The Ecuador gold material was unique in this study because assay data were available from Disa. Several lab analyses were conducted, and one such report is provided in the appendix. These reports all indicate that the material has relatively high contents of arsenic (As), antimony (Sb), iron (Fe), and sulphur (S), as well as the expected gold (Au). Refractory and invisible gold has long been known to coexist in association with As and Sb sulphides such as arsenopyrite and stibnite, and is well documented in the literature [51]–[53]. Therefore, it was concluded these are likely the dominant minerals in the sample. Disa was also able to provide a bulk specific gravity of the material at 3.20, which was used for calculating testing volumes and mass fractions.

While the ore consistently shows gold presence across the various lab reports, the exact amount varies between analyses. Therefore, while it can be concluded that the ore is gold-bearing at a relevant grade, the true head grade must be determined through a metallurgical balance if it is to be used in subsequent analyses. Disa was also able to provide scanning electron microscope (SEM) results for the material, shown below as Figures 20 and 21. Both images represent the same 2 mm scope of view with precious metals in the lighter grains, with the latter picture also highlighting antimony bearing grains.



Figure 20: SEM view of Ecuador gold material. Courtesy of Disa, LLC<sup>®</sup> [54]



Figure 21: SEM view of Ecuador gold material highlighting antimony minerals. Courtesy of Disa, LLC<sup>®</sup> [54]

These images show that while there is a significant amount of gold bearing particles, they are often poorly liberated from the gangue minerals. As the minerals in this material are disseminated within grains rather than cemented to the surfaces, this likely means the process will be more successful through size reduction and downstream separation rather than selective concentration. The overlap of highlighted Sb minerals in Figure 20 with the gold bearing particles in Figure 21 further supports that the gold is associated with the Sb-bearing grains. This understanding supports our decision to use a xanthate collector during flotation.

While selective concentration of the gold sample was not a target, understanding the breakage mechanisms remains valuable in helping understand the results of the recovery processes. As a result, images were again collected of a coarse, intermediate, and fine size for the feed and are illustrated below with their respective sizes in Figures 22 and 23.



Figure 22: a) 20x image of the Ecuador gold feed 75  $\mu$ m fraction, b) 20x image of the Ecuador gold feed 45  $\mu$ m fraction



Figure 23: 20x image of the Ecuador gold feed fines fraction

These images support the SEM analysis where the gold bearing As-Sb particles are well mixed within the ore; the reflective grey minerals are visible in each size class spread between the cloudy white pieces of quartz, which makes up the majority of the ore. These images alone

provide little insight but will clarify breakage mechanisms when compared to products from the same size classes.

The size distribution for the gold material is provided below as Table 4. GS and RR approaches were again used in being consistent with the previous campaigns, and datasets are provided in the appendix. While the  $R^2$  of the GS curve is below 99.00, it provides an effective approximate for the  $F_{80}$  value, and therefore a polynomial function was not necessary.

 Model
 P<sub>80</sub> (μm)
 R<sup>2</sup>

 Gaudin-Schuhmann
 87.24
 98.37

 Rosin-Rammler
 76.26
 97.56

Table 4: PSD model results for BDH feed material

Similar to the BDH tailings this sample has nearly all of its mass passing at less than 1 mm, meaning little to no pre-screening would be necessary prior to HPSA operation. However, in this case a significant fraction of the material is passing the finest screen at over 50%. This supports our initial observation that selective concentration may not be as effective in this application as substantial losses may be a result.

# 4.2 Selective Concentration

Following size classification, all fractions of the BDH products at and below 212  $\mu$ m were qualitatively reviewed visually and physically inspected for clay content. Figure 24 contains a visual representation of these fractions, with a close-up of the fines and 150  $\mu$ m portion shown in Figure 25.



Figure 24: BDH feed and product size fractions after classification and drying



Figure 25: Fines and 150 µm size classes of the BDH 4-minute product

It can clearly be seen that higher clay concentrations are present in the fines of each product stream regardless of processing time, providing visual evidence of selective concentration. On a textural basis each non-fine fraction presents liberated particles that move freely when touched. Little to no clay dust remains on the object used to move the samples, indicating the clays have been removed. Meanwhile, the fines contain large amounts of agglomerated flakes after the dewatering process. This is due to clay particles acting as a binder, creating a disc within the filter press and cracking into flakes upon drying.

To better understand the outcomes identified by Figures 24 and 25, portions of the 4-minute product were collected from the same size classes as those in Section 4.1 and viewed under a similar magnification. Photographs of these portions under the microscope are provided in Figures 26 and 27.



Figure 26: a) 10x image of the BDH 4-minute product 355 µm fraction, b) 10x image of the BDH 4minute product 125 µm fraction



Figure 27: 20x image of the BDH 4-minute product fines fraction

The microscopic images provide further evidence that most of the clay material has concentrated into the -45  $\mu$ m size class. Little to no clays are present on the surfaces of the coarse and intermediate grains, allowing for a view of the cleaned and rounded surfaces of the quartz grains. Both the iron-bearing (red) and native (white) quartz particles resemble their similar sized feed

images from Section 4.1, yet in this case further clay removal is present. This indicates that while wet screening is effective for desliming, processing in the HPSA unit can provide further improvements. While the various quartz grains can still be seen in the fines, the deeper colour is attributable to the darker insoluble minerals, further indicating selective concentration is present in this sample.

In similar fashion to the BDH products, the potash 2-minute sample was visually reviewed following size classification. Several fractions of the product are provided below in Figure 28, with microscopic magnification of these classes shown in Figures 29 and 30. As was the case for the feed, size specifics have been withheld to maintain confidentiality.



Figure 28: Potash 2-minute product size fractions: a) coarse, b) intermediate, c) fine



Figure 29: a) Image of the potash 2-minute product "coarse" fraction, b) Image of the potash 2-minute product "coarse" fraction



Figure 30: Image of the potash 2-minute product "coarse" fraction

These images mirror the outcomes of the previous campaign, with the darker coloured clays concentrating into the finest size fraction. Cleavage is again prominent in the non-fine fractions while it is obscured in the fines by particle size of the insolubles present. This adds more weight to the initial conclusions that HPSA can successfully generate selective concentration in surface-

bound systems and provides improved results over wet screening. However, in this case a quantitative content analysis via dissolution was possible. Insols composition was measured in each size class for the feed and 2-minute product and the results are provided below in Table 5.

Normalized Size Class (-)	Feed Content (%)	2-Minute Content (%)
2.21	3.43	0.00
1.56	2.29	0.00
1.10	8.57	0.00
0.79	9.71	5.56
0.55	6.86	7.41
0.40	5.71	3.70
0.28	6.29	8.33
0.20	6.29	12.04
0.14	2.86	2.78
0.10	0.57	0.00
0.07	0.57	0.00
0.05	0.00	0.00
0.03	0.00	0.00
Fines	46.86	60.19
Total Insols (% of sample weight)	1.29	1.40

Table 5: Insol assay results for potash feed and 2-minute product classes

This data indicates that both wet screening only and HPSA followed by screening are effective at concentrating the clays, however it supports that the ablation technology can enhance concentration, achieving an additional 13% of clay removal. The results also support the findings in literature that the Esterhazy has low insols content, with both samples showing around 1%. It should be noted that the 0.20 portion of both samples have slightly elevated percentages due to

minor contamination, however the removal of these contaminants would only improve results and therefore was considered usable. Even though the jet to plate configuration generates impact-based breakage, it is evident that it is successful in achieving its purpose. As the literature has already concluded that the jet to jet is effective in this application from the uranium results, it can be concluded that the HPSA is effective in removal of surface gangue particles regardless of configuration.

# 4.3 Size Reduction

#### 4.3.1 Particle Size Distributions

The PSDs for each of the BDH products are collectively plotted below in Figure 31 with the feed distribution also shown for reference. Figures including the GS and RR trends for each product like those in Section 4.1 are provided in the appendix. Initial screening of the 2-minute product was only completed down to  $125 \,\mu$ m, and later attempts to rescreen these fines into subfractions proved ineffective. Therefore, this portion of the trend has been removed from the figures below, however the raw data are available for view in the appendix. To better illustrate the distribution of mass throughout the size classes, trends of percentage retained versus particle size were also created and are provided below as Figure 32. Finally, the P<sub>80</sub> values for each product are given in Table 6. As both trend methods often overestimated the characteristic size, a best fit polynomial was used to improve the estimate.


Figure 31: Particle size distributions for BDH products



Figure 32: Percentage of material retained on each screen for BDH products

Sample	P <sub>80</sub> (µm)	$\mathbb{R}^2$
Feed	327	99.66
2-minute	308	99.91
4-minute	297	99.55
16-minute	281	100.00

Table 6: Characteristic passing sizes for BDH feed and products

These datasets show that the HPSA unit can generate size reduction in this application, however the achieved comminution is relatively small. The images in Section 4.2 indicate that the clays have been collected into the fines, but little additional surface area is generated on the quartz grains through comminution. This means that the energies provided by the HPSA during the test was not enough to break the harder silicates. However, while the level of breakage is minor in this application this is desirable for the BDH tailings as the primary target was to achieve selective concentration.

The size distribution of the potash 2-minute product is shown below. During the analysis of the 5 and 8-minute samples an equipment malfunction occurred, rendering the data unusable. The remaining values were normalized by dividing all sizes by the  $F_{80}$  of the sample. Like the BDH material a polynomial fit was used to accurately calculate the  $P_{80}$  of the product. Feed trends are again provided for reference, and figures showing the GS and RR fits are provided in the appendix.



Figure 33: Particle size distributions for potash 2-minute product



Figure 34: Percentage of material retained on each screen for potash 2-minute product

Sample	Percent passing at F <sub>80</sub> (%)	$\mathbb{R}^2$
Feed	80	99.43
2-minute	97	99.57

Table 7: Characteristic passing sizes for potash feed and 2-minute product

While selective concentration was still the primary target, size reduction in the potash was much more prominent than the silica tails, with two minutes being enough to bring nearly all the material to passing the  $F_{80}$ . This is likely a result of potash particles being coarser than the BDH quartz, allowing the impact mechanisms of the jet to plate to work more effectively. Additionally, the images of the potash clearly show the standard cubic cleavage of halite and sylvite grains, which further facilitates breakage.

Using the  $P_{80}$  of the Ecuador gold ablation product in conjunction with the rod mill calibration, it was found that a run time of approximately 1-minute would provide a grind product of similar size. Polynomials were again used for better fit of the  $P_{80}$  values, and the GS and RR trendlines are again available for both products in the appendix. The results of the calibration curve as well as the HPSA 8-minute and rod mill products are shown below.



Figure 35: P<sub>80</sub> versus duration data for rod mill calibration



Ecuador Gold Particle Size Analysis - PSD

Figure 36: Particle size distribution for Ecuador gold feed and products



Figure 37: Percentage of material retained on each screen for Ecuador gold feed and products

Sample	P <sub>80</sub> (µm)	$\mathbb{R}^2$
Feed	87	98.37
HPSA 8-minute	66	99.17
Rod Mill 1-minute	62	99.88

Table 8: Characteristic passing sizes for Ecuador gold feed and products

When reviewing the characteristic sizes, it is seen that little grinding takes place. This could again be a result of the impact mechanisms employed in the jet to plate configuration being less effective in finer applications. For the rod mill it is likely due to the shorter run time rather than breakage mechanisms; from the calibration tests much finer products are achieved by increasing the run time due employment the surface-based breakage mechanisms, which are effective for finer particles.

The major conclusion from analyzing these products comes from comparing the quantity of fines. It is apparent from the curves that the HPSA can generate a similar characteristic size and trend shape to that of the rod mill. However, the HPSA limits the generation of additional fines smaller than the 53  $\mu$ m class. This can be valuable in certain applications where coarser particles are desired such as gravity concentration or flash flotation, and therefore is worth noting.

#### 4.3.2 Size-Energy Relationships

Using the collected amperage readings from the potash and Ecuador gold testing, a relationship between size reduction and energy consumption can be generated for the HPSA tests. As the pump was connected to a three-phase, alternating-current motor these amperages could be directly converted to power using the three-phase power equation, shown below as Equation 4.1.

$$P = \sqrt{3} * V * I * PF$$
 Equation 4.1

Where V, I and PF stand for voltage in volts, current in amps, and the unitless power factor. The pumps being used were connected to a source of 480 volts, but no power factor was provided on the pump nameplate. A value of 0.86 was therefore assumed based on typical values for similar sized motors [55]. Once the power was calculated it could be multiplied by the quotient of run time over sample mass to provide a specific energy value in kWh/t. For the BDH test where sample mass changed as products were collected, each interval between sample draws was calculated individually and summed before multiplication. The calculation for all three campaigns can both be represented by Equation 4.2.

$$E = P * \sum_{i=1}^{n} \left[ \frac{(t_i - t_{i-1})}{m_i} \right]$$
 Equation 4.2

Where E is the specific energy in kWh/t, P is the calculated power draw in kW, i is the sample to be collected, t is the time of sample collection in hours, m is the mass in the HPSA unit at the time of sample i in kg, and n is the total number of collected samples. Each mass can be calculated by subtracting all previously removed sample masses from the original mass at the beginning of operation.

The energy draws were both constant and similar for each campaign, therefore, the average amperage was used to calculate a relationship for the BDH campaign as this was done at a comparable solids fraction. For discussion, theoretical values for each material at solids fractions seen in commercial mills are also shown. Finally, results for Bond's equation were created for comparison using work indices from various sources [56], [57]. The range of potential specific energy values are shown below with the size-energy relationships.



**BDH Size-Energy Results** 

Bond Range HPSA Results (10%) Theoretical HPSA (50%)

Figure 38: Size-Energy Relationship for the BDH material



Figure 39: Size-Energy Relationship for the potash material



Ecuador Gold Size-Energy Results

Figure 40: Size-Energy Relationship for the Ecuador gold material

The BDH trend indicates that like other liberation technologies the HPSA energy requirements increase exponentially as product size is decreased. However, at the tested levels all trends show that the HPSA requires a higher energy input than standard tumbling mills to produce size reduction. This especially becomes the case following 3 to 4 minutes of grinding time as the variation between the theoretical results provided by Bond's equation and the test results becomes substantial. Combining these trends with the size analysis and concentration results leads to the conclusion that the HPSA technology in its current batch configuration is most effective when run with short durations.

As the campaigns were run at relatively low mass fractions (and a long run time in the case of the Ecuador gold) there is significant room for improved energy outcomes. By maintaining a similar sample volume but increasing the solids contents to percentages commonly used by conventional technologies, the differential between Bond's prediction and the experimental HPSA results can be significantly reduced. Simultaneously, the increased number of particles in the system would lead to a higher number of particle collisions, likely providing additional size reduction or selective concentration. However, the increased mass fraction would have the negative rheological impact of an increased viscosity due to the non-Newtonian nature of high mass fraction slurries. As a result, further testing is required to determine the optimum mass fraction which balances these effects and minimizes energy requirements.

While these lower values still fail to be the more efficient option, they provide incentive to pursue additional testing with the technology. However, it is important to note that due to the batch nature of the pilot scale HPSA, the actual and theoretical energy requirements would be comparable to an open-circuit process. To be truly comparable to a typical operation the HPSA would need to be connected to some form of clarifier, allowing for closed-circuit operation. Therefore, while it is

anticipated that future work will result in improved energy efficiencies and liberation results, it is recommended that the pilot scale unit be altered to allow for higher mass fractions and to provide the choice of open or closed-circuit operation before further testing is conducted.

### 4.4 Gold Recovery

#### 4.4.1 Imaging Analysis

To provide background to our recovery conclusions, portions of the HPSA and rod mill products were collected following size analysis and reviewed under a microscope. These images are provided below with their size classes and magnification as Figures 41 and 42 and Figures 43 and 44 for the HPSA and rod mill products respectively.



Figure 41: a) 20x image of the Ecuador gold HPSA 8-minute product 75 µm fraction, b) 20x image of the Ecuador gold HPSA 8-minute product 45 µm fraction



Figure 42: 20x image of the Ecuador gold HPSA 8-minute product fines fraction



Figure 43: a) 20x image of the Ecuador gold rod mill 1-minute product 75 µm fraction, b) 20x image of the Ecuador gold rod mill 1-minute product 45 µm fraction



Figure 44: 20x image of the Ecuador gold rod mill 1-minute product fines fraction

While it is expected that the rod mill particles would exhibit smoother edges than those from the HPSA due to their respective breakage mechanisms, it is interesting to note that both products display similar surface characteristics. This may be indicative of some surface acting mechanisms in the jet to plate configuration, such as in the pump chamber or through the pressurized tubing. What is clear is that these images show similar colouring and As-Sb content to those in Section 4.1, which supports the notion that selective concentration is not recommended in this application.

#### 4.4.2 Metallurgical Balance

To determine the original location of the gold particles and what incremental gains in recovery can be made, a metallurgical balance was conducted for each of the testing samples and the feed. Using samples from the  $\pm 400$  mesh fractions of each separation product, these balances afforded the opportunity to calculate the head grade of the raw material and provided insight into how fine the majority of the gold bearing particles were. The resulting grades and recovered free gold are provided below in Table 9, and all assay and mass results are available in the appendix.

	Gold Panning	Float Conc.	Float Tails	Head
Test	Recovery (%)	Grade (g/t)	Grade (g/t)	Grade (g/t)
Fresh	2.48	75.28	53.66	55.04
HPSA 8-minute	7.25	53.67	54.10	55.41
Rod mill 1-minute	4.56	62.76	50.38	55.78

Table 9: Resulting stream grades of all experimental samples

It should be noted that complications arose at the lab during three assays due to overrange grades, and in one case due to insufficient sample mass. Internal data was provided by the lab where available, but caution was recommended by ALS as it was expected that the tolerance ranges for these values could be at or above 20%. However, upon further review the lab was able to conduct the applicable overrange test for one of the samples and provided a confirmed grade. The internal value showed an error of less than 2% when compared to the confirmed result, and therefore the internal values of the other two samples were deemed effective for our analysis. These samples resulted in completed balances for feed and rod mill product.

Using the two completed balances an average head grade was calculated, which was then used to calculate the remaining unknown product grade from the HPSA flowsheet. As the resulting value was comparable to the lab results for the products of the rod mill and feed, it was considered successful. These balances resulted in a head grade of 55.41 g/t with a tolerance range of less than 1%. This is a relatively high grade for ROM gold ores, which leads to the conclusion this material may be a side product or tailing stream from an existing operation. The low variance in calculated head grades also suggests that our assumption of the internal values being usable was justified.

From the assays it can be shown which size class the gold predominantly is present in for both the feed and the HPSA and rod mill products. The comparison of these analyses is provided below as Figure 45.



Figure 45: Size-assay analysis of gold bearing particles in Ecuador gold feed and products

From the illustration it is evident that the gold is more abundant in the finer class, however this effect is only slightly more prominent in the HPSA product. Therefore, it is not considered a significant result in this study, confirming our initial observation that selective concentration may not be effective in this application.

#### 4.4.2 Gold Recoveries

The free gold recoveries of each unit are presented below in Figure 46. The mass yield and gold recovery of the flotation processes are also given below in Figure 47. The recoveries in both figures are calculated with respect to feed to the individual separation unit, not the raw material.



Figure 46: Knelson concentrator recovery for all experimental samples



Ecuador Gold - Flotation Outcomes

#### Figure 47: Flotation mass yield and recovery for all experimental samples

Figure 46 illustrates that regardless of liberation method, recovery via gravity concentration remains low. This is indicative of minor free gold content, which was already anticipated based on composition and metallurgical balancing; arsenopyrite-based gold ores are known to largely

contain refractory gold that is not amenable to gravity concentration, and any upstream processing likely already removed the free gold available in this materials original deposit. This proves that gravity concentration is not recommended for this material in its current form.

Despite the low recoveries the testing did show that the HPSA unit provided increased recoveries when compared to the rod mill product. The impact breakage of the jet to plate HPSA allowed larger pieces of gangue to be removed from individual middling particles compared to the surface acting removal through the rod mill, resulting in a relatively larger increase in particle bulk density. As the fundamental principles of gravity concentration rely on density differentials, this allowed more gold-bearing particles to be captured in the concentrate rather than being lost to tails.

Figure 47 shows that comminution improved the flotation results of both materials, as is generally expected following comminution. However, like gravity separation the flotation recoveries remain lower than expected. This can likely be attributed to the chemistry not being optimized, and results could be improved through an increased dosage of PAX and MIBC, or perhaps through a different collecting agent. Increasing the run time could also have a positive effect on both mass yield and recovery, potentially allowing more concentrate to be collected.

In this situation the rod mill sample performed best, showing gold recoveries that were nearly 10% higher than the HPSA product. It has previously been shown that the rod mill generates additional -325 and -400 mesh particles, and due to the higher surface areas available these particles are more amenable to adsorption with flotation collectors. When combined with the size-assay results of Figure 45 this would explain the higher gold recoveries in the rod mill product. It is expected that for such a conclusion the differential in mass yields would be higher, yet due to the evidence of other sources this can likely be attributed to measurement and experimental error.

70

## 5 Conclusions and Recommendations

#### 5.1 Conclusions

This study endeavoured to assess the potential success of high pressure slurry ablation as a liberation technology in various applications. Three materials of interest were tested in the lab scale unit at UBC and subsequently underwent size and content analyses to accomplish this goal. The results of these testing campaigns, as well as our recommendations, will be provided to Disa, LLC<sup>®</sup> as the owner of the technology.

Through the review of the appropriate literature, as well as the experimental results from the testing campaigns, the following conclusions regarding the technology can be drawn:

- The results of the size-energy analyses indicate that there is little benefit to be gained by processing in the HPSA unit after a certain run time. A significant portion of the breakage occurs during the initial cycles of the unit, and further operation results in excessive energy consumption for minor additional size reduction.
- Due to the relatively high gold head grade and low available recoveries from gravity concentration, the Ecuador gold material presented to Disa is likely a by-product or a high-grade tailings stream from an existing operation. The flowsheet used in this operation likely contains gravity concentration but fails to provide high overall recoveries, explaining why the group has reached out to Disa to improve their processing outcomes.

- The jet to plate configuration of the HPSA offers limited generation of additional fines, with products often showing fine fractions of similar size to the feed. This can be of value in applications where coarse particles are more valuable such as the potash industry, or in facilities where coarse materials are amenable to downstream separation processes such as flash flotation.
- Selective concentration is achievable using HPSA in the jet to plate configuration in ore bodies where the gangue is loosely connected to the surface of the valuable mineral. The results of the BDH and potash campaigns indicate that the technology is effective at providing clay removal at as low as 5 cycles per particle in the unit.
- The HPSA equipment can generate size reduction in all tested materials, but currently fails to provide a more efficient alternative to common liberation technologies. This is a result of the open-circuit concept provided by a batch operation, as well as the limited available throughputs. Therefore, new equipment that allows for classification and higher solids handling must be considered before the ablation unit can become competitive.
- The jet to plate configuration of the HPSA unit has the potential to provide higher gravity concentration recoveries in precious metal applications. Due to the reduced generation of fines and impact-based breakage mechanisms, the ablation technology has been shown to provide elevated gravity recoveries when compared to tumbling mills. This would be directly beneficial in circuits that traditionally employ gravity concentrators, such as deposits with coarser middlings or higher free gold content.

#### 5.2 Future Work

Several suggestions for future work presented themselves during this study:

- In all tested applications the current HPSA pilot unit was unable to meet similar mass fractions to the current industrial technologies. This will result in significant scale up issues as it cannot reach the desired throughput rates for economic operation. To rectify this, upgraded pumps should be installed that can handle higher solids fractions. However, these mass fractions will have implications on the results of this study and the rheology of the fluids, and thus similar tests should be conducted with this new equipment.
- While the qualitative analysis of clay content was effective in presenting the success of HPSA in the BDH and potash applications, a quantitative analysis would provide further detail. As such, these samples should undergo compositional analyses such as SEM to determine the exact quantity of clays in each size class and verify the above conclusions.
- While it has been shown in the literature that the jet to jet configuration is successful for surface bound mineralogy, little research has been done in ores such as the gold material in this study. As such, it is recommended that a similar campaign to this one be conducted on the Ecuador gold with the jet to jet configuration to see if attrition-based breakage provides similar outcomes. It is also recommended that this future study collect a sample at a shorter run time to promote improved specific energy consumption within the unit.
- Due to the availability of a VFD, the pilot unit can vary the flow rate of the test sample. This in turn impacts the collision velocity and number of cycles each particle experiences. For materials that show positive outcomes from campaigns such as the ones in this study, it is recommended that a more detailed design of experiment be conducted to review the impact of individual variables and their second order interactions on the relevant targets.

## References

- J. Jeswiet and A. Szekeres, "Energy Consumption in Mining Comminution," *Procedia CIRP*, vol. 48, pp. 140–145, 2016.
- [2] K. R. Rábago, A. B. Lovins, and T. E. Feiler, "Energy and Sustainable Development in the Mining and Minerals Industries," *Mining, Miner. Sustain. Dev.*, no. 41, pp. 1–7, 2002.
- [3] K. Holmberg, P. Kivikytö-Reponen, P. Härkisaari, K. Valtonen, and A. Erdemir, "Global energy consumption due to friction and wear in the mining industry," *Tribol. Int.*, vol. 115, pp. 116–139, 2017.
- [4] S. Alvarado, J. Algüerno, H. Auracher, and A. Casali, "Energy-Exergy Optimization of Comminution," *Energy*, vol. 23, no. 2, pp. 153–158, 1998.
- [5] D. W. Fuerstenau and A. Z. Abouzeid, "The energy efficiency of ball milling in comminution," *Int. J. Miner. Process.*, vol. 67, no. 1–4, pp. 161–185, 2002.
- [6] D. Tromans and J. A. Meech, "Fracture toughness and surface energies of minerals: theoretical estimates for oxides, sulphides, silicates and halides," *Miner. Eng.*, vol. 15, no. 12, pp. 1027–1041, 2002.
- [7] Metso, "Basics in Minerals Processing." 2018.
- [8] B. A. Wills and J. A. Finch, "Crushers," in *Wills' Mineral Processing Technology*, Elsevier, 2016, pp. 123–146.
- [9] S. J. Neethling and J. J. Cilliers, "Grade-recovery curves: A new approach for analysis of and predicting from plant data," *Miner. Eng.*, vol. 36–38, pp. 105–110, 2012.
- [10] J. D. Miller, C.-L. Lin, L. Hupka, and M. I. Al-Wakeel, "Liberation-limited

grade/recovery curves from X-ray micro CT analysis of feed material for the evaluation of separation efficiency," *Int. J. Miner. Process.*, vol. 93, no. 1, pp. 48–53, 2009.

- [11] J. F. Medina, "Liberation-limited grade/recovery curves for auriferous pyrite ores as determined by high resolution X-ray," The University of Utah, 2012.
- B. A. Wills and J. A. Finch, "Grinding Mills," in *Wills' Mineral Processing Technology*, Oxford, UK: Elsevier, 2016, pp. 147–179.
- [13] S. Palaniandy, M. S. Powell, J. Seppelt, P. Griffin, A. Moore, and K. Kermanshahi,
  "Utilising a Tertiary Stirred Mill To Recover Grind Size After Expansion of an SABC Circuit," in SAG Conference, 2015, pp. 1–17.
- [14] R. K. Rajamani, B. K. Mishra, R. Venugopal, and A. Datta, "Discrete element analysis of tumbling mills," *Powder Technol.*, vol. 109, no. 1–3, pp. 105–112, 2000.
- [15] A. Gupta and D. S. Yan, "Tubular Ball Mills," in *Mineral Processing Design and Operations*, Elsevier, 2016, pp. 189–240.
- [16] A. Jankovic, "Variables affecting the fine grinding of minerals using stirred mills," *Miner. Eng.*, vol. 16, no. 4, pp. 337–345, 2003.
- [17] O. Celep, N. Aslan, I. Alp, and G. Taşdemir, "Optimization of some parameters of stirred mill for ultra-fine grinding of refractory Au/Ag ores," *Powder Technol.*, vol. 208, no. 1, pp. 121–127, Mar. 2011.
- [18] A. Gupta and D. S. Yan, "Stirred Mills Ultrafine Grinding," *Miner. Process. Des. Oper.*, pp. 287–316, 2016.
- [19] J. Bouchard, G. LeBlanc, M. Levesque, P. Radziszewski, and D. Georges-Filteau,

"Breaking down energy consumption in industrial grinding mills," in *Canadian Mineral Processors Conference*, 2017.

- [20] C. F. Perucca, "Potash processing in Saskatchewan A review of process technologies," in *CIM Bulletin*, 2003, vol. 96, no. 1070, pp. 61–65.
- [21] A. M. Abouzeid, "Physical and thermal treatment of phosphate ores An overview," Int.
  J. Miner. Process., vol. 85, pp. 59–84, 2008.
- [22] X. Ma and M. Pawlik, "Adsorption of Guar Gum on Potash Slimes," *Can. Metall. Q.*, vol. 46, no. 3, pp. 321–328, 2007.
- [23] E. Forbes, K. J. Davey, and L. Smith, "Decoupling rehology and slime coatings effect on the natural flotability of chalcopyrite in a clay-rich flotation pulp," *Miner. Eng.*, vol. 56, pp. 136–144, 2014.
- [24] B. J. Arnold and F. F. Aplan, "The Effect of Clay Slimes on Coal Flotation, Part I: The Nature of the Clay," *Int. J. Miner. Process.*, vol. 17, pp. 225–242, 1986.
- [25] Y. Yu, L. Ma, M. Cao, and Q. Liu, "Slime coatings in froth flotation: A review," *Minerals Engineering*. 2017.
- [26] A. Sablok, "Physiochemical Characterization of a Carbon Containing Phosphate Ore," The University of British Columbia, 2019.
- [27] B. A. Wills and J. A. Finch, "Comminution," in *Wills' Mineral Processing Technology*, 2016, pp. 109–122.
- [28] X. Chen, Y. Peng, and D. Bradshaw, "The effect of particle breakage mechanisms during regrinding on the subsequent cleaner flotation," *Miner. Eng.*, vol. 66, pp. 157–164, 2014.

- [29] A. J. Lynch and C. A. Rowland, "Tumbling Mills," in *The History of Grinding*, 2005, pp. 95–131.
- [30] B. K. Loveday and H. Dong, "Optimisation of Autogeneous Grinding," *Miner. Eng.*, vol. 13, no. 13, pp. 1341–1348, 2000.
- [31] M. Gao and E. Forssberg, "Prediction of product size distributions for a stirred ball mill," *Powder Technol.*, vol. 84, no. 2, pp. 101–106, 995.
- [32] J. M. Menacho, "Some Solutions for the Kinetics of Combined Fracture and Abrasion Breakage," *Powder Technol.*, vol. 49, no. 1, pp. 87–96, 1986.
- [33] P. von Rittinger, *Lehrbuch der Aufbereitungskunde in ihrer neuesten Entwicklung und Ausbindung systematisch dargestellt*. Berlin, Germany: Ernst & Kern, 1867.
- [34] F. Kick, Das Gesetz der proportionalen Widerstande und seine Anwendungen. Leipzig, Germany, 1885.
- [35] W. H. Walker, W. K. Lewis, and W. H. McAdams, *Principles of Chemical Engineering*. New York, US: McGraw-Hill, 1927.
- [36] P. C. Kapur and D. W. Fuerstenau, "Energy-Size Reduction 'Laws' Revisited," Int. J. Miner. Process., vol. 20, pp. 45–57, 1987.
- [37] J. F. Richardson, J. H. Harker, and J. R. Backhurst, "Particle Size Reduction and Enlargement," in *Chemical Engineering*, 2003, pp. 95–145.
- [38] F. C. Bond, "The third theory of comminution," Min. Eng., vol. 4, p. 484, 1952.
- [39] C. Mayer-Laigle, R. K. Rajaonarivony, N. Blanc, and X. Rouau, "Comminution of Dry Lignocellulosic Biomass: Part II. Technologies, Improvement of Milling Performances,

and Security Issues," Bioengineering, vol. 5, no. 2, 2018.

- [40] J. A. Coates, D. H. Scriven, C. Coates, and E. Coates, "Devices, Systems, and Methods for Processing Heterogeneous Materials," US 8646705 B2, 2014.
- [41] N. H. Andrews and W. J. Willoughby, "Method of Pulverizing Minerals and Similar Materials," US 1948609, 1932.
- [42] H. E. Dunn and W. T. Bradley, "Process for Concentrating Carnotite Ores," US 2562024, 1951.
- [43] Black Range Minerals Colorado and Thompson & Pugsley, "Description of Ablation Mining Technology Applied to Uranium Deposits," 2015.
- [44] A. W. Nienow and R. Conti, "Particle abrasion at high solids concentration in stirred vessels," *Chem. Eng. Sci.*, vol. 33, no. 8, pp. 1077–1086, 1978.
- [45] C. R. Bemrose and J. Bridgwater, "A review of attrition and attrition test methods," *Powder Technol.*, vol. 49, no. 2, pp. 97–126, 1987.
- [46] J. J. Stickel and R. L. Powell, "Fluid mechanics and rheology of dense suspensions," *Annu. Rev. Fluid Mech.*, vol. 37, pp. 129–149, 2005.
- [47] D. S. Weaver, "Numerical Study of Dense Turbulent Jet Flow using CFD-DEM."
- [48] Department of Geology at the University of Minnesota, "Quartz," 2020. [Online].Available: https://www.esci.umn.edu/courses/1001/minerals/quartz.shtml.
- [49] A. Fuzesy, "Potash in Saskatchewan," 1982.
- [50] S. Titkov, "Flotation of water-soluble mineral resources," Int. J. Miner. Process., vol. 74,

no. 1-4, pp. 107-113, 2004.

- [51] Y. Zhu, F. An, and J. Tan, "Geochemistry of hydrothermal gold deposits: A review," *Geosci. Front.*, vol. 2, no. 3, pp. 367–374, 2011.
- [52] L. J. Cabri, S. L. Chryssoulis, J. De Villiers, J. H. G. Laflamme, and P. R. Buseck, "The Nature of 'Invisible' Gold in Arsenopyrite," *Can. Mineral.*, vol. 27, no. 3, 1989.
- [53] F. An and Y. Zhu, "Native antimony in the Baogutu gold deposit (west Junggar, NW China): Its occurrence and origin," *Ore Geol. Rev.*, vol. 37, no. 3–4, pp. 214–223, 2010.
- [54] D. Bagdonas, C. Nye, F. McLaughlin, E. Phillips, and S. Quillinan, "Preliminary Report for Disa, LLC."
- [55] Engineering Toolbox, "Power Factor Inductive Load," 2004. [Online]. Available: https://www.engineeringtoolbox.com/power-factor-electrical-motor-d\_654.html. [Accessed: 04-Apr-2020].
- [56] Doering International GmbH, "Grinding Media size formula." [Online]. Available: https://www.doeringsinn.de/fileadmin/masterTemplate/Resources/Public/JavaScript/rechner/Mahlkoerperform el\_nach\_Bond\_en.pdf.
- [57] F. C. Bond, "Crushing and Grinding Calculations Part II," *Br. Chem. Eng.*, vol. 6, no. 8, pp. 543–548, 1961.

# Appendix A – Raw Experimental Data

Run	Volume (mL)	Mass (g)	Specific Gravity (-)
1	128	157.8	1.233
2	200	246.2	1.231
3	111	136.7	1.232
4	149	184.1	1.236
5	187	231.4	1.237
6	115	143	1.243

Table A-1: Brine specific gravity data

Table A-2: Potash specific gravity data

Initial Volume (mL)	Initial Weight (g)	Final Volume (mL)	Final Weight (g)	Specific Gravity (-)
150	185.1	177	240.3	2.044
150	185.3	215	317.3	2.031
200	246.4	256	359.6	2.021

Table A-3: Brine solubility data

Run	Pan (g)	Full (g)	Empty (g)	Water (mL)	Salt (g)	Solubility (g/mL)
1	80.2	328.1	157.7	170.4	77.5	0.4548
2	85.3	252.2	137.5	114.7	52.2	0.4551
3	88.4	224.9	131.1	93.8	42.7	0.4552

Normalized Particle Size (-)	Filter (g)	Total (g)	Insols $(g)^*$	Insols (%)
2.21	7.9	8.5	0.6	3.43
1.56	7.9	8.3	0.4	2.29
1.10	7.8	9.3	1.5	8.57
0.79	7.9	9.6	1.7	9.71
0.55	8.1	9.3	1.2	6.86
0.40	8	9	1	5.71
0.28	7.8	8.9	1.1	6.29
0.20	7.9	9	1.1	6.29
0.14	7.9	8.4	0.5	2.86
0.10	7.9	8	0.1	0.57
0.07	7.8	7.9	0.1	0.57
0.05	7.9	7.7	0	0.00
0.03	8	8	0	0.00
Fines	8	16.2	8.2	46.86

Table A-4: Weight data for potash feed insolubles

\*Negative insols calculations were deemed errors from moisture originally in the filter and therefore were set to zero.

Normalized Particle Size (-)	Filter (g)	Total (g)	Insols $(g)^*$	Insols (%)
2.21	7.9	7.7	0	0.00
1.56	7.9	7.8	0	0.00
1.10	7.9	7.9	0	0.00
0.79	7.9	8.5	0.6	5.56
0.55	7.9	8.7	0.8	7.41
0.40	7.9	8.3	0.4	3.70
0.28	7.9	8.8	0.9	8.33
0.20	7.9	9.2	1.3	12.04
0.14	8	8.3	0.3	2.78
0.10	7.9	7.9	0	0.00
0.07	7.8	7.7	0	0.00
0.05	7.9	7.7	0	0.00
0.03	8	7.7	0	0.00
Fines	7.8	14.3	6.5	60.19

Table A-5: Weight data for potash 2-minute product insolubles

\*Negative insols calculations were deemed errors from moisture originally in the filter and therefore were set to zero.

Time (s)	Power Draw (A)
30	11.75
60	12.30
90	12.18
120	12.10
150	12.24
180	12.16
210	12.12
240	12.23
270	12.17
300	12.17
330	12.10
360	12.21
390	12.05
420	12.09
450	12.26
480	12.17

Table A-6: Potash HPSA test energy consumption

Table A-7: Ecuador gold HPSA test energy consumption

Time (s)	Power Draw (A)
30	11.81
60	11.66
90	11.50
120	11.12
150	11.50
180	11.62
210	11.64
240	11.51
270	10.93
300	10.71
330	11.20
360	10.66
390	10.70
420	10.94
450	10.63
480	10.29

Material	Starting Mass (kg)
BDH Tailings	12.11
Potash	25.52
Ecuador Gold	20.47

Table A-8: Starting sample masses for each testing campaign

Description	Value (kWh/t)	Reference Source	
"Quartz"	12.77	Bond [57]	
"Silica"	13.53	Bond [57]	
"Silica Sand"	16.46	Bond [57]	
"Quartz"	15	Doering [56]	
"Silica Sand"	16	Doering [56]	

Table A-9: Bond work index values for BDH material

Table A-10: Bond work index values for potash material

Description	Value (kWh/t)	Reference Source	
"Potash ore"	8.88	Bond [57]	
"Potash salt"	8.23	Bond [57]	
"Potash"	9	Doering [56]	
"Potash ore"	7 - 9	Perucca [20]	

Table A-11: Bond work index values for Ecuador gold material

Description	Value (kWh/t)	Reference Source
"Gold ore"	14.83	Bond [57]
"Pyrite ore"	8.9	Bond [57]
"Pyrites"	10	Doering [56]

				Final	Sample
Material	Source	Size Class	Bag Weight (g)	Weight (g)	Weight (g)
Fresh	Pan Concentrate	+400	2.95	14.32	11.37
Rod Mill	Pan Concentrate	+400	2.87	21.49	18.62
Rod Mill	Pan Concentrate	- 400	2.91	13.14	10.23
HPSA	Pan Concentrate	+400	2.88	26.85	23.97
HPSA	Pan Concentrate	- 400	2.84	11.05	8.21
Fresh	Flotation Tails	- 400	3.00	94.32	91.32
HPSA	Flotation Tails	- 400	2.99	67.73	64.74
Rod Mill	Flotation Tails	- 400	2.91	94.39	91.48
	Flotation				
Fresh	Concentrate	- 400	2.97	57.63	54.66
	Flotation				
HPSA	Concentrate	- 400	2.90	80.91	78.01
	Flotation				
Rod Mill	Concentrate	- 400	2.98	63.80	60.82
Fresh	Flotation Tails	+ 400	2.92	69.73	66.81
HPSA	Flotation Tails	+ 400	2.98	71.83	68.85
Rod Mill	Flotation Tails	+400	2.96	53.71	50.75
	Flotation				
Fresh	Concentrate	+ 400	3.01	68.86	65.85
	Flotation				
HPSA	Concentrate	+ 400	2.93	73.52	70.59
	Flotation				
Rod Mill	Concentrate	+400	2.90	87.07	84.17

Table A-12: Ecuador gold assay lab sample weights

Table A-13: Ecuador gold mass and metallurgical balance data

	Fresh Feed	HPSA 8-minute	Rod Mill 1-minute
Pan Concentrate (g)	11.37	32.18	28.85
Flotation Concentrate (g)	385.26	533.45	838.24
Flotation Tails (g)	1283.35	833.55	1164.46
Flotation Mass Yield (%)	23.09	39.02	41.86
Pan Concentrate Grade (g/t)	203.00	174.66	177.89
Flotation Concentrate Grade (g/t)	75.28	53.67	62.76
Flotation Tails Grade (g/t)	53.66	54.10	50.38
Feed Grade	55.04	55.41	55.78
Pan Recovery	2.48	7.25	4.56
Flotation Recovery	31.95	39.81	48.97
Total Recovery	34.43	47.06	53.53



ALS Canada Ltd. 2103 Dollarton Hwy North Vancouver 8C V7H 0A7 Phone: +1 (604) 984 0221 Fax: +1 (604) 984 0218 www.alsglobal.com/geochemistry To: UBC NBK MINING INT. 517-6350 STORES ROAD VANCOUVER BC V6T 1Z4 Page: 2 - A Total # Pages: 2 (A) Plus Appendix Pages Finalized Date: 1-APR-2020 Account: UNBMIN

Project: Gold by Fire Assay

							KTIFICA	NALIS	15	VA20057887	
Sample Description	Method Analyte Units LOD	WEI-21 Recvd Wt. kg 0.02	Au-AA25 Au ppm 0.01	Au-GRA21 Au ppm 0.05	Au-CON01 Au ppm 0.07						
DA1 DA2		0.02 0.02	>100 >100	NSS 157.5							
DA3 DA4 DA5 DA6 DA7		0.02 0.02 0.02 0.10 0.06	>100 >100 NSS 64.7 61.4	NSS NSS							
DA8 DA9 DA10 DA11 DA12		0.10 0.06 0.08 0.06 0.06	59.1 84.1 67.7 71.8 39.5								
DA13 DA14 DA15 DA16 DA17		0.08 0.06 0.06 0.06 0.08	42.2 36.1 59.9 33.4 51.0								

Figure A-1: Ecuador gold lab assay results

Table A-14: Internal results for several Ecuador gold lab assay samples

Method	Au-AA25				
Analyte	Internal Au result (ppm)				
DA1	203				
DA3	215				
DA4	133				



ALS Canada Ltd. 2103 Dollarton Hwy North Vancouver BC V7H 0A7 Phone: +1 (604) 984 0221 Fax: +1 (604) 984 0218 www.alsglobal.com/geochemistry To: UBC NBK MINING INT. 517-6350 STORES ROAD VANCOUVER BC V6T 1Z4

Page: 2 - A Total # Pages: 2 (A) Plus Appendix Pages Finalized Date: 10-APR-2020 Account: UNBMIN

	)		Project: UNBMIN_VA20057887				
			 CERTIFICATE OF ANALYSIS	VA20080290			
Sample Description	Method Analyte Units LOD	Au-GRA21 Au ppm 0.05					
DA4		130.5					

Comments: \*\*Originally from workorder VA20057887

Figure A-2: Ecuador gold overweight test lab assay results

## Appendix B – Particle Size Distribution Tables and Trends

Sieve Size	Weight (g)	Weight (%)	Nominal Aperture Size (µm)	Undersize (%)	Oversize (%)	RR Factor
30 mesh	4.6	0.49%	600	99.51%	0.49%	5.31
40 mesh	32.1	3.44%	425	96.07%	3.93%	3.24
45 mesh	75.8	8.12%	355	87.94%	12.06%	2.12
50 mesh	131.7	14.11%	300	73.83%	26.17%	1.34
70 mesh	301	32.25%	212	41.58%	58.42%	0.54
100 mesh	117.7	12.61%	150	28.96%	71.04%	0.34
120 mesh	44.9	4.81%	125	24.15%	75.85%	0.28
200 mesh	84.3	9.03%	75	15.12%	84.88%	0.16
325 mesh	40	4.29%	45	10.83%	89.17%	0.11
Fines	101.1	10.83%	0	0.00%	100.00%	0.00

Table B-1: Size distribution data for BDH feed sample

#### **BDH Feed**



Figure B-1: PSD plots and trendline fitting for BDH feed sample

Sieve Size	Weight (g)	Weight (%)	Nominal Aperture	Undersize (%)	Oversize (%)	RR Factor
30 mesh	1.5	0.25%	600	99.75%	0.25%	5.98
40 mesh	8.8	1.49%	425	98.26%	1.74%	4.05
45 mesh	27.1	4.58%	355	93.68%	6.32%	2.76
50 mesh	97.4	16.47%	300	77.21%	22.79%	1.48
70 mesh	178.1	30.11%	212	47.10%	52.90%	0.64
100 mesh	104.7	17.70%	150	29.40%	70.60%	0.35
120 mesh	58	9.72%	125	20.27%	79.73%	0.23
200 mesh	42.8	7.18%	75	13.09%	86.91%	0.14
325 mesh	23.3	3.91%	45	9.19%	90.81%	0.10
Fines	54.8	9.19%	0	0.00%	100.00%	0.00

Table B-2: Size distribution data for BDH 2-minute product



**BDH 2-Minute** 

Figure B-2: PSD plots and trendline fitting for BDH 2-minute product

Sieve Size	Weight (g)	Weight (%)	Nominal Aperture Size (µm)	Undersize (%)	Oversize (%)	RR Factor
30 mesh	0.6	0.13%	600	99.87%	0.13%	6.68
40 mesh	5.7	1.19%	425	98.68%	1.32%	4.33
45 mesh	12	2.51%	355	96.17%	3.83%	3.26
50 mesh	75.1	15.71%	300	80.46%	19.54%	1.63
70 mesh	142.5	29.81%	212	50.66%	49.34%	0.71
100 mesh	61.1	12.78%	150	37.88%	62.12%	0.48
120 mesh	25.8	5.40%	125	32.48%	67.52%	0.39
200 mesh	52.1	10.90%	75	21.59%	78.41%	0.24
325 mesh	22.1	4.62%	45	16.96%	83.04%	0.19
Fines	81.1	16.96%	0	0.00%	100.00%	0.00

Table B-3: Size distribution data for BDH 4-minute product



**BDH 4-Minute** 

Figure B-3: PSD plots and trendline fitting for BDH 4-minute product
Siovo Sizo	Waight (g)	$W_{aight}(0/)$	Nominal Aperture	Undersize (9()	$O_{\text{Moreizo}}(0/)$	DD Easter
Sleve Size	weight (g)	weight (%)	Size (µm)	Undersize (%)	Oversize (%)	KK Factor
30 mesh	0.1	0.03%	600	99.97%	0.03%	8.19
40 mesh	1.9	0.53%	425	99.44%	0.56%	5.19
45 mesh	5.1	1.42%	355	98.03%	1.97%	3.93
50 mesh	47.3	13.13%	300	84.90%	15.10%	1.89
70 mesh	77.5	21.51%	212	63.39%	36.61%	1.00
100 mesh	52.6	14.60%	150	48.79%	51.21%	0.67
120 mesh	19.8	5.50%	125	43.30%	56.70%	0.57
200 mesh	41.7	11.57%	75	31.72%	68.28%	0.38
325 mesh	23	6.38%	45	25.34%	74.66%	0.29
Fines	91.3	25.34%	0	0.00%	100.00%	0.00

Table B-4: Size distribution data for BDH 16-minute product



Figure B-4: PSD plots and trendline fitting for BDH 16-minute product

		Normalized			
Weight (g)	Weight (%) Particle Size		Undersize (%)	Oversize (%)	RR Factor
28.9	2.13%	2.21	97.87%	2.13%	3.85
38.3	2.81%	1.56	95.06%	4.94%	3.01
162.8	11.97%	1.10	83.09%	16.91%	1.78
195.7	14.38%	0.79	68.72%	31.28%	1.16
396.4	29.13%	0.55	39.58%	60.42%	0.50
180.5	13.26%	0.40	26.32%	73.68%	0.31
82.6	6.07%	0.28	20.25%	79.75%	0.23
87.3	6.41%	0.20	13.84%	86.16%	0.15
40.0	2.94%	0.14	10.90%	89.10%	0.12
27.0	1.98%	0.10	8.92%	91.08%	0.09
31.4	2.31%	0.07	6.61%	93.39%	0.07
21.9	1.61%	0.05	5.00%	95.00%	0.05
34.8	2.56%	0.03	2.44%	97.56%	0.02
33.3	2.44%	0.00	0.00%	100.00%	0.00

Table B-5: Size distribution data for potash feed sample

## Potash Feed



Figure B-5: PSD plots and trendline fitting for potash feed sample

		Normalized			
Weight (g)	Weight (%)	Particle Size (-)	Undersize (%)	Oversize (%)	RR Factor
0.2	0.02%	2.21	99.98%	0.02%	8.35
0.3	0.04%	1.56	99.94%	0.06%	7.44
7.2	0.94%	1.10	99.00%	1.00%	4.61
47.8	6.22%	0.79	92.78%	7.22%	2.63
196.9	25.59%	0.55	67.19%	32.81%	1.11
134.6	17.50%	0.40	49.69%	50.31%	0.69
99.5	12.93%	0.28	36.76%	63.24%	0.46
83.6	10.87%	0.20	25.89%	74.11%	0.30
67.5	8.78%	0.14	17.11%	82.89%	0.19
28.1	3.65%	0.10	13.46%	86.54%	0.14
28.8	3.75%	0.07	9.71%	90.29%	0.10
18.8	2.44%	0.05	7.27%	92.73%	0.08
24.3	3.16%	0.03	4.11%	95.89%	0.04
31.6	4.11%	0.00	0.00%	100.00%	0.00

Table B-6: Size distribution data for potash 2-minute product

Potash 2-Minute



Figure B-6: PSD plots and trendline fitting for potash 2-minute product

Sieve Size	Weight (g)	Weight (%)	Nominal Aperture Size (microns)	Undersize (%)	Oversize (%)	RR Factor
30 mesh	3.01	0.60%	600	99.40%	0.60%	5.12
50 mesh	4.92	0.98%	300	98.43%	1.57%	4.15
100 mesh	32.66	6.48%	150	91.95%	8.05%	2.52
140 mesh	32.57	6.46%	106	85.49%	14.51%	1.93
200 mesh	45.46	9.02%	75	76.47%	23.53%	1.45
270 mesh	57.69	11.44%	53	65.03%	34.97%	1.05
325 mesh	31.77	6.30%	45	58.73%	41.27%	0.88
400 mesh	28.79	5.71%	38	53.02%	46.98%	0.76
Fines	267.28	53.02%	0	0.00%	100.00%	0.00

Table B-7: Size distribution data for Ecuador gold feed sample



Figure B-7: PSD plots and trendline fitting for Ecuador gold feed sample

Sieve Size	Weight (g)	Weight (%)	Nominal Aperture Size (microns)	Undersize (%)	Oversize (%)	RR Factor
30 mesh	1.89	0.13%	600	99.87%	0.13%	6.62
50 mesh	2.97	0.21%	300	99.66%	0.34%	5.67
100 mesh	34.74	2.46%	150	97.20%	2.80%	3.57
140 mesh	60.95	4.31%	106	92.88%	7.12%	2.64
200 mesh	116.39	8.24%	75	84.65%	15.35%	1.87
270 mesh	183.27	12.97%	53	71.67%	28.33%	1.26
325 mesh	166.32	11.77%	45	59.90%	40.10%	0.91
400 mesh	74.08	5.24%	38	54.66%	45.34%	0.79
Fines	772.24	54.66%	0	0.00%	100.00%	0.00

Table B-8: Size distribution data for Ecuador gold HPSA 8-minute product



HPSA 8-minute

Figure B-8: PSD plots and trendline fitting for Ecuador gold HPSA 8-minute product

Sieve Size	Weight (g)	Weight (%)	Nominal Aperture Size (microns)	Undersize (%)	Oversize (%)	RR Factor
30 mesh	0.06	0.01%	600	99.99%	0.01%	9.12
50 mesh	0	0.00%	300	99.99%	0.01%	9.12
100 mesh	4.12	0.75%	150	99.24%	0.76%	4.88
140 mesh	20.85	3.80%	106	95.44%	4.56%	3.09
200 mesh	46.12	8.40%	75	87.04%	12.96%	2.04
270 mesh	68.26	12.44%	53	74.60%	25.40%	1.37
325 mesh	40.18	7.32%	45	67.28%	32.72%	1.12
400 mesh	26.88	4.90%	38	62.38%	37.62%	0.98
Fines	342.37	62.38%	0	0.00%	100.00%	0.00

Table B-9: Size distribution data for Ecuador gold rod mill 1-minute product





Figure B-9: PSD plots and trendline fitting for Ecuador gold rod mill 1-minute product