INTEGRATED MINING, PRE-CONCENTRATION AND WASTE DISPOSAL SYSTEMS FOR THE INCREASED SUSTAINABILITY OF HARD ROCK METAL MINING

By

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Abstract

In the hard rock metal mining industry, both in Canada and globally, a decreasing number of economic mineral deposits are found at shallow to medium depth, and most of the deposits that remain are close to sub-economic and are required to be mined at high tonnages in order to show a return. The majority of remaining deposits are presented in challenging geological or geotechnical settings making the deposit sub-economic. The integration of ore pre-concentration and waste disposal functions into the hard rock metal mining system is proposed as a novel interpretation of Mine-Mill Integration for improving the economics and environmental impact of exploiting such deposits. The proposed approach seeks to reject waste as early as possible in the mining cycle, and safely dispose of it as backfill. This is proposed as a 'Lean' alternative to improving the economics of mining simply by increasing the throughput. 'Lean' philosophy seeks to design out overburden, smooth production, and eliminate waste from the manufacturing system. It is suggested that the proposed approach addresses all three areas, and is thus an important strategy to be considered for mining companies wishing to simultaneously improve their efficiency, economics and environmental performance, thus increasing their sustainability.

Technologies specific to the success of the approach including pre-concentration systems, composite fill systems, and continuous mechanized mining methods are discussed. The impacts and benefits of integrating these technologies are defined and quantified through research, testwork, systems design and analysis. Custom geo-metallurgical evaluation tools incorporating mineralogical, metallurgical and geotechnical methods have been developed to assess ores in terms of their potential for the adoption of the proposed approach. A computerized parametric evaluation model has been developed to quantify the potential impacts and benefits using data from these evaluations. Data from over 26 case studies combined with the literature indicates that the opportunity for ore pre-concentration appears to be a general case in hard rock ores. A wide range of impacts and benefits arising from this potential have been identified and quantified through the research, indicating positive overall outcomes for the majority of cases studied.

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(M)tpa	-	(Million) Tons Per Annum
(N)AG	-	(Non) Acid Generating
(S)AG	-	(Semi) Autogenous Grinding
AC	-	Alternating Current
ARD	-	Acid Rock Drainage
ASTM	-	American Society Of Testing And Materials
CDN$ (m)	-	Canadian Dollars (millions)
CHF	-	Cemented Hydraulic Fill
CRF	-	Cemented Rock Fill
d_{80}	-	Screen Size At Which 80% By Weight Of Sample Passes
dm^3	-	Cubic Decimetre
DMS	-	Dense Media Separation
EIT	-	Engineer In Training
f_{80}	-	d80 Of Feed Material To Process
g/t	-	Grammes Per Tonne
HM	-	Heavy Media
HPGR	-	High Pressure Grinding Roll
ICP	-	Inductively-Coupled Plasma Mass Spectrometry
kg	-	Kilogrammes
kHz	-	Kilohertz
kW	-	Kilowatts
kW/m^3	-	Kilowatts Per Cubic Metre
kWh/a	-	Kilowatt Hours per Annum
$m^2(^3)$	-	Metres Squared (Cubed)
m^3/hr	-	Metres Cubed Per Hour
MJ/m^2	-	Mega Joules Per Metre Squared
MMI	-	Mine-Mill Integration
MPa	-	Megapascals
mV	-	Millivolts
Ø	-	Diameter
p_{80}	-	d_{80} Of Product From A Process

PCI	-	Peripheral Component Interconnect PC Card
PGE / PGM	-	Platinum Group Element (Metal)
QEM/SEM	-	Quantitative Scanning Electron Microscopy
RD	-	Relative Density
RGB	-	Red-Green-Blue
ROM	-	Run-Of-Mine
RQD	-	Rock Quality Designation
SG	-	Specific Gravity
tpd	-	Tonnes Per Day
TPM	-	Total Precious Metals
UCS	-	Uniaxial Compressive Strength
US$ (m)	-	United States Dollars (millions)
VCR	-	Ventersdorp Contact Reef
VMS	-	Volcanogenic Massive Sulphide
Wt%	-	Proportion By Weight
σ	-	Stress

Glossary

Automation	-	autonomous monitoring and control of mechanical processes by means of electronics and microprocessors
Backfill	-	waste material placed in underground excavations for mechanical support
Barren	-	containing no, or low metal values
Batching	-	the process of preparing concrete mixtures in discrete batches
Binder	-	cement or other cementitious material used to bind other particles together
Blasthole	-	large bore hole drilled in rock to house explosives for breaking the rock
Breccia	-	rock composed of angular fragments in a matrix cementing material of typically different composition
Bridge	-	an electrical circuit of typically 4 resistors used to measure a voltage
Bushveld	-	igneous intrusive complex in South Africa
Closure (1)	-	the process of closing and rehabilitating a mine site
Closure (2)	-	time-dependent collapse of an excavation in hard rock
Colorimetric	-	the measurement of spectral values in reflected light
Comminution	-	the process of reducing the particle size of a mineral by crushing or grinding
Composite Fill	-	backfill created by mixing traditional rock fill with paste
Concentrate	-	a mineral product of sufficient metal grade to be considered for treatment in a furnace
Core Logging	-	visual examination of drill core to determine its mineralogical and geotechnical features
Dense Media	-	suspension of fine, dense particles in water used to raise the apparent density of the fluid
Disseminated	-	fine, discrete particles of valuable mineralization evenly distributed in the host rock
Drawpoint	-	location from which ore is extracted from a stope by LHD
Extraction	-	proportion the original mineral resource which is removed by mining
Fill Sequence	-	the order in which stopes are mined, and then again backfilled
Float	-	light, barren rock which floats in the dense media separation process
Flotation	-	separation of valuable from non-valuable minerals by means of differences in surface chemistry
Footwall	-	barren rock located below the longitudinal axis of the orezone

Fuzzy Logic	-	Matrix transformation function used to give an approximation instead of a precise result
Gabbro	-	basic, intrusive igneous rocks
Gangue	-	non-valuable components in a mineral assemblage, typically silica and metal oxides
Geo-metallurgy	-	interdisciplinary field linking the mineralogical and geo-technical properties of an ore with its behaviour during mineral beneficiation
Grade Control	-	the practice of monitoring and controlling the metal content of the ore delivered from the mining operation to the mill
Grade Distribution	-	the distribution of metal values by particle size in an ore
Greenhouse Gas	-	gases such as carbon dioxide and water vapour which are known to increase the retention of radiation within the Earth's atmosphere
Hangingwall	-	barren rock located above the longitudinal axis of the orezone
Hard Rock	-	rock that specifically requires drilling and blasting for excavation
Hydraulic Hoisting	-	the transport of mineral slurries vertically by means of pumping
Hydrodynamic	-	the measure of forces and motion in liquids
Igneous	-	rock of magmatic origin that cooled and solidified within the Earth's crust
INCO	-	International Nickel Company Of Canada
Leaching	-	dissolution of valuable minerals from the rock using acid or complexing agents such as cyanide
LHD	-	front end loader for loading, hauling and dumping broken rock underground
Mafic	-	igneous rock with relatively low silica content formed under equilibrium redox conditions
Marginal	-	mining operations with low profit margins likely to lose money under adverse economic conditions
Massive	-	valuable mineralization occurring as large, homogenous inclusions in the host rock
Mesotexture	-	the supra-microscopic disposition of mineralization in rock
Meta-sedimentary	-	metamorphosed sedimentary rocks
Microtexture	-	microscopic disposition of mineralization in rock

Mineralogical	-	qualitative and quantitative evaluation of minerals in rock
Mining Cycle	-	the sequence of activities in the mining of rock: drilling, blasting, mucking, ground control, haulage, hoisting
Ore / Orebody	-	collective of mineralized rock within a mineral deposit which is economically valuable
Ore Zone	-	well bounded localized area of valuable mineralization (reef or seam)
Overburden	-	typically fractured, weathered waste material overlying a near-surface mineral deposit
Oxide	-	valuable metals compounded with oxygen in rock e.g. Cr_2O_3
Paleo-Placer	-	fossilized alluvial mineral deposit
Parametric	-	the use of arbitrary values as independent variables in a model or equation in order to determine the value of the dependent variables
Particle Size Distribution	-	the proportion by weight of particles in a sample falling within various size classes
Paste	-	a concentrated suspension of fine tailings, mixed with cement and used as backfill in mining
Photometric	-	measurement of the properties of transmitted or reflected light
Pillar	-	vertical columns of ore left behind in the orebody in order to provide local or regional mechanical stability
Porphyry	-	a fine-grained igneous host rock containing conspicuous coarse phenocrysts of alkali felspar or other distinctive minerals
Positive Displacement	-	method of pumping fluids by mechanically by means of an expanding and contracting chamber
Productivity	-	the output of an industrial process measured per unit input
Radiometric	-	measurement of the level and nature of radioactivity in materials
Recovery	-	percentage by mass of the valuable mineral recovered to the concentrate in a metallurgical process
Selectivity	-	measure of the ability of a mining method to extract valuable mineralization only
Siliceous	-	containing a high proportion of silica or alumino-silicates
Sill Pillar	-	extensive horizontal pillar left in massive orebodies for regional support
Sink	-	dense, valuable rock which sinks in the dense media separation process
Size-Assay	-	process to determine the distribution of metal values by particle size

Sloughing	-	inadvertent collapse of typically hangingwall rock into ore after blasting
Slump	-	measure of the consistency of typically a concrete mix
Slurry	-	a heterogenous suspension of solids in water
Stripping Ratio	-	the ratio by mass of non-valuable to valuable material removed from an open pit mine
Sulphide	-	valuable metals compounded with sulphur such as Cu_2S and NiFeS
Surface Footprint	-	total surface area covered by the structures and excavations related to the mine
Sustainability	-	combined economic, environmental and social accounting as applied to the mining industry
Tele-operation	-	operation of multiple units of mobile mechanical equipment by one operator using telemetry
Tertiary Regrind	-	third stage grinding of an ore in order to further liberate the valuable mineral
Throughput	-	the mass flowrate of dry ore passing through a process
Ultramafic	-	igneous rock with low silica content formed under slightly oxidising conditions
UG2	-	chromite seam of the Upper Group of the Bushveld Igneous Complex
Value Chain	-	the definition and ordering of sequential value-adding steps in an industrial process
Volcanogenic	-	mineralization of extrusive volcanic origin
Waste	-	collective definition of all gangue material generated during the mining and processing of an ore into a concentrate

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1. Introduction and Thesis Outline

1.1 Introduction

In the Hard Rock Mining Industry, both in Canada and globally, a decreasing number of economic mineral deposits are found at shallow to medium depth, and most of those shallow deposits that remain are close to sub-economic grade and are required to be mined at high tonnages in order to show a return (Scoble, 1994). A large proportion of the remaining deposits are higher grade deposits which are presented in situations of either extreme depth, complex structure, poor ground conditions, or remote geography, one or a combination of which factors results in extreme pressure on costs, making the deposit sub-economic. Variations in the quantity and quality of ore delivered also negatively affects the performance of the overall mining system as the process control system takes time to adjust to these variances. Grade control and the maximizing of the grade of ore delivered to surface is thus of paramount importance in these situations. The commodity market which the industry supplies continues to be cyclical (Baxter & Parks, 1957) and economic pressures are further exacerbated in periods of declining or low commodity prices.

A common solution to improving the economics of mining low grade orebodies is to increase the mining rate, with consequent decreases in unit costs and an increase in capital costs. However, increasing the mining rate reduces the mine life and significantly increases the physical and environmental footprint of mines, with negative impacts on public perception and closure requirements. Increasing the mining rate can also exacerbate ground control problems, with unforeseen negative impacts on costs. There is significant public pressure on the industry to reduce surface footprint, and it has been suggested that the productivity, economics as well as the environmental acceptability of mining marginal orebodies would be greatly improved by introducing innovative processes for the reduction of waste into the mining activity (Parsons & Hume, 1997; Warhurst & Bridge, 1996; Feasby & Tremblay, 1995). An effective approach for deep or otherwise marginal deposits would thus be to increase the mining rate, while simultaneously minimizing dilution, ground control problems and reducing the surface environmental footprint of the mine. This concept has found wide support from industry proponents including Rio Tinto, BHP and Placer Dome (Batterham, 2003, 2004; Hames, 2005; Cross, 2006), from whom the ultimate interpretation of the approach is presented in terms of positing the ‘invisible mine’ where the maximum of the mining and processing activity is undertaken below surface. However, little work has been done in terms of a practical realization of this vision. A groundbreaking approach is required to achieve this and thus improve the feasibility of exploiting such deposits. The focus of this thesis is the specification, design and evaluation of an integrated set of technologies, the acceptance of which significant steps towards the realization of this concept can be made. Substantial efforts in the area of concept dissemination, technology transfer, and skills development together with appropriate change management are now required to begin implementation.

The mining industry is innovative, and the potential for successful transfer of this technological approach into the industry is considered high. Substantial innovation has recently been seen in other fields such as process and environmental technologies, innovations which include Outokumpu and INCO’s adoption of the flash smelting process, developments in pre-concentration and waste disposal at Mount Isa Mines in Australia, the ISA Mill, Jameson Cell and the ISASmelt process also at Mt Isa, and ongoing developments in tailings and ARD management in Canada and worldwide (Bamber & Scoble, 2007). Recent developments in hydrometallurgy have enabled the lower cost processing of lower-grade copper and gold ores. Novel technologies such as Pressure Acid Leach and heap leaching have increased the range of ores that can be processed economically. Low-pressure, low-temperature metal leaching technology is enabling the exploitation of lateritic type base metal ores as well as methods such as in-situ leaching. Notable improvements have been made in the area of sulphur- and particulate emission levels at base metal smelters such as Copper Cliff, Falconbridge and Kidd Creek, as well as at Anglo’s Waterval smelter in Rustenburg, South Africa. There have also been ongoing developments globally in area of process control and operational data management, led by companies such as Western Mining, INCO and Outokumpu. However, underground hard rock mining faces particular challenges in the area of innovation, and it is recognized that substantial care in technology transfer and change management is required for success.

There are a number of competing strategies whose effectiveness must be compared to the proposed approach in order for the benefits to be fully realized. Highly selective or resue mining methods can increase the viability of mining marginal deposits, particularly in narrow-vein situations. Resue mining is a two-phase mining process for narrow vein ores whereby waste in the mining face is drilled, blasted, and stowed separately underground first, followed by drilling, blasting and removal of the mineralized zone. Selective blast mining methods (SBM) have also been suggested where the narrow vein ore and waste are blasted simultaneously, but are segregated due to a blast design which ‘throws’ the ore and waste into different areas of the stope (Bock, 1998). However, adopting the abovementioned methods impact negatively on the productivity of the method when compared to bulk mechanized mining at conventional widths, and other solutions must be found. Alternatives to resue mining in narrow-vein situations have thus been developed. Randfontein Estates in South Africa experimented with thick-seam mechanized gold mining (‘TM3’) in the 1980’s. However the resultant decrease in head grade was considered economically unacceptable. A mechanized alternative to the resue mining of steeply-dipping narrow vein gold deposits of the Witwatersrand was proposed by Pickering et al (1999). A similar mining method has since been adopted at Placer Dome’s South Deep Mine in some sections: it was found that the cost of the bulk method was 30% lower than the conventional method, however the planned grade of ore delivered from the thick seam mechanized sections was some 30% lower than from the conventional longwall mining sections, which can be attributed solely to additional dilution arising from the mining method (James & Rafffield, 1996). The development of low-profile diesel LHD’s has now enabled higher-productivity methods to be employed while maintaining grade in mining heights as low as 1.5m at dips of up to 12º, and many manufacturers now offer reliable, high productivity mining suites for mechanized mining at these mining widths. More recent efforts to increase the efficiency and effectiveness of underground hard rock mining include tele-operation of remote LHDs, continuous face advance techniques and Vertical Crater Retreat (VCR) Mining have been made, with mixed results. In the tele-operation of mining fleets in order to improve productivity underground there has been a significant effort, particularly by INCO locally in Canada. The Mine Automation Project (MAP) was a CDN$20 million, 5 year long research program involving collaboration between INCO, Tamrock, Dyno Nobel and CANMET, for the purposes of fully automating the mucking process (Graham & Morrison, 2003). The project was taken to the stage of an operating underground stope but further development of the concept was abandoned in 2002. Continuous mining systems are also attractive as a substantial component of the potential underground mining productivity is lost between conventional drill, blast and muck cycles. Recent research into these systems is aimed at overcoming this problem. In 1995, INCO and Noranda entered into a 10 year, $35m partnership as HDRK Ltd with Tamrock and Wirth GmbH of Germany in order to develop a continuous hard rock development machine (Dessuerault et al, 2002; Graham 2001). A modified Eimco T60 roadheader capable of mining medium hard rock at acceptable rates was installed in a Manitoba mine, but further installations have not been forthcoming. A second effort comprising a C1000 Oscilloader in series with a mobile horizontal crusher and feeder unit, feeding an extendable belt which would follow the mining face by extending the head end of the belt was taken to the prototype stage, but the project was not pursued further.

More effective efforts have been made at adapting mining methods for continuous operation and greater productivity. The Vertical Crater Retreat (VCR) method was developed in the 1970s as a means of increasing stope sizes and smoothing the production of ore from these stopes and facilitating the automation of mucking and hauling operations (Gertsh & Bullock, 1998). This increase in stope sizes was facilitated through the development of large bore underground drills and spherical charging of the holes, thus eliminating raise boring and slot cutting from the development cycle. VCR was adopted widely by INCO in the 1980's as a replacement for massive cut and fill stopes in at their Ontario and Manitoba operations. However, VCR, as with TM^3 and block caving, has not achieved the level of continuity and productivity originally envisioned, and furthermore results in a massive decrease in the grade of ore delivered from the stopes when compared to cut and fill methods, and thus results of the implementation are widely considered to have been negative and several mining sections at INCO have since reverted to cut and fill techniques (Cochrane, 2006).

Several other innovative techniques for the enhanced mining of base metal hard rock ores have previously been researched at the US Bureau of Mines. These include in-situ leaching of copper and gold, as well as the leaching of previously unresponsive base metal ores by means of novel lixiviants, either in-situ or by heap leaching on surface (Davin, 1979; Rogich, 1982). However, the technology for leaching of unconventional base metal ores is presently inefficient and delivers unacceptable economics. Other solutions for the enhanced extraction of base metal ores must be found. The pre-concentration of Run–of- Mine ore (ROM) and subsequent disposal of the waste rejects underground, prior to transport and conventional treatment of the ore by fine particle processes such as flotation or leaching, is proposed as an extension of the concept of

Mine Mill integration philosophy as a more general approach to improving the efficiency, economics and environmental performance of hard rock metal mining. It has been previously stated that “pre-concentration is the rule rather than the exception” (Salter & Wyatt, 1991) yet the concept has not found wide application in the industry. Opposition to the concept in industry ranges from basic perceptions of unacceptable metal losses, high cost and low capacity of the technologies to a lack of understanding of the systemic impact of applying the concept. Further challenges to the application of pre-concentration, particularly when integrated into the underground environment are cultural: mining engineers do not understand mineral processing sufficiently well, and metallurgists are unwilling to extend their activities into the underground environment. Since 2003, strategic research at UBC has been directed primarily at developing applications for the technology at both surface and underground operations for several mining companies both in Canada as well as globally. Through the research, the benefits of ore pre-concentration, particularly when it is allied to technologies for the simultaneous preparation and disposal of the waste rejects and applied underground, have been shown to be sufficient to merit an objective consideration of this approach. The concept of ore pre-concentration itself is not new, and has been practiced on surface through modern mineral processing technologies such as Dense Media Separation (DMS) and sorting since the 1930’s (Munro et al, 1982). The concept of underground pre-concentration is also not new (Agricola, 1556; Lloyd, 1979); however, there are currently no examples of underground ore pre-concentration plants in operation. Several examples of sorting plants which operate underground are cited; however, sorting in this application is typically used for classification and diversion of the whole ore stream, and thus cannot be strictly considered pre-concentration (Kruuka & Briocher, 2002).

The integration of pre-concentration and waste disposal activities into the mining process is considered a specific interpretation of the philosophy of Mine-Mill or Mine-to-Mill Integration (MMI). The Mine-Mill Integration philosophy arises from the desire to integrate the downstream processes such as ore handling and mineral processing more closely with the mining activity in order to improve efficiencies in the whole operation (Stephenson et al, 1971). Several exponents of this philosophy can be found in the literature, and typical aspects include blast fragmentation analysis, blast fragmentation – comminution interactions, and the application of down-the-hole electromagnetic or optical sensing for improved geo-metallurgical characterization of the in-situ ore in preparation for extraction and processing. The integration of automated ore pre-concentration and waste disposal systems into the mining environment, whether on surface or underground, is proposed as an interpretation of this philosophy as a means of reducing the selectivity required of the mining method, increasing productivity and lowering costs while simultaneously reducing the quantity and increasing the grade of the ore delivered to downstream processes (Bamber et al, 2005). Due to the prospective rejection and disposal of a significant quantity of waste from the ROM ore stream, the capacity and thus size of surface facilities such as milling, tailings disposal and associated surface infrastructure would also be significantly reduced compared to the bulk mining scenario. The concept has been widely supported in principle by various industry and academic proponents (Hames, 2005; Batterham, 2003; Cross, 2006; Scoble et al, 2006; Martens, 2007) yet has presently not found application in the industry.

It is suggested that present resistance to the application of the concept on technical or economic grounds is unfounded, and the thesis seeks to define appropriate parameters for the performance of the technologies on typical hard rock ores as well as identify and quantify the benefits for a range of operational situations. Pre-concentration technologies seek to identify and reject barren siliceous waste from the Run-of-Mine ore stream by means of several available coarse-particle separation processes such as optical sorting, conductivity sorting, or dense media separation (DMS) and reject it. The process results in the production of a large quantity of coarse siliceous waste and thus consideration of appropriate solid waste disposal strategies is also required for a complete evaluation. It is suggested through the research that the opportunity for the pre-concentration of ore is a general case in hard rock metal mines, and that there is a specific window of operational features such as ore mineralogy, operating cost structure, level of capitalization and organizational topography within which the technical, economic and environmental impacts and benefits of pre-concentration are positive. Where applicable, the adoption of the integrated mining, pre-concentration and waste disposal approach can reduce the quantity of material transported, processed and disposed of on surface thus significantly reducing the surface footprint of mines. Ore reserves are increased through the reduction in costs and it is thus believed that through these impacts, the sustainability of hard rock mining is significantly increased.

A wide range of ores have been tested, and results from the research and testwork have been augmented by additional supporting data from the literature. The research was conducted using a team approach; central to this thesis is the design of the research programme, research planning and supervision, target generation and the conduction of specific site investigations on the research targets, sampling and testwork, analysis and evaluation of results. Additional supporting testwork was provided by two Masters students working in the Mine-Mill Integration Group: Mark Stephenson on the testing of conductivity sorting on the Pipe ores, and Trent Weatherwax on the metallurgical and geotechnical testing of the Xstrata ores. It is thus suggested from the results that there is a more universal case for ore pre-concentration, either on surface and underground, the adoption of which is considered an appropriate application of Lean Manufacturing in Hard Rock Mining, a concept which has had broad success in the manufacturing sector in improving production efficiency. An appropriate envelope of overall cost to ore value ratios has been defined for mining operations within which a significant degree of waste rejection at acceptable metal recoveries can deliver substantial technical, economic and environmental benefits for the enhanced exploitation of these deposits.

1.2 Significance of the Research

The anticipated impact of the proposed approach on hard rock mining in terms of delivering these technical advantages, economic benefits and improvements in environmental performance and public perception is considerable, and there are several mineral districts in particular which are expected to benefit from the application of the research outcomes. These include:

- low grade ultramafic Ni ores of the Thompson Nickel Belt type
- deep massive sulphide ores of the Sudbury Igneous Complex for example Onaping Depth and Nickel Rim
- narrow vein UG2- type ores of the Bushveld Igneous Complex situated below 1500m which are presently uneconomical to exploit through conventional methods
- Elsberg and VCR-type ores of the Witwatersrand basin situated below 3000m which are presently uneconomical to exploit
- Cordilleran-type copper porphyry ores at block cave operations such as the Candelaria and North Parkes mine

Based on the anticipated impact of the research, two commercial spin-offs have been developed in the course of the research. One is focused on the evaluation of opportunities and design of the technologies described for mining companies both in Canada as well as globally. A second company is being been formed to focus on commercializing the sensing technologies that have been developed in the course of the research.

1.3 Research Contribution

Significant challenges to the continued sustainability of hard rock mining are extant. A novel integrated mining, processing and waste disposal approach is proposed, where this potential can be identified, for improving the sustainability of extracting a wide class of mineral deposits compared to the conventional approach. A general case for the pre-concentration of ore prior to conventional treatment by grinding and flotation has been identified in the course of the research which is considered a significant contribution. The research tools that have been developed through the course of the work are also a significant contribution in terms of enabling the rapid and cost-effective assessment of a particular opportunity for pre-concentration, together with a rapid quantification of the expected benefits. Innovative improvements in laboratory techniques such as dense media testing, and the characterization of the optical and conductivity properties of ore with reference to sorting have also been made. In particular, the induction-balance sensor that has been designed and tested is a particularly unique contribution, enabling the characterization and grade evaluation of a class of low grade ores previously unresponsive to conventional conductivity methods. The future potential of this sensing technology in terms of exploration, down-the-hole sensing and ore sorting has yet to be explored.

1.4 Thesis Outline

Ore pre-concentration, prior to conventional fine-particle processing such as leaching or flotation, and the subsequent disposal of the solid waste in the mining void, is proposed as a specific interpretation of mine-mill integration in order to improve efficiency, economics, and environmental impact for the increased sustainability of hard rock metal mining. The thesis is a comprehensive treatment of the design, application and evaluation of the ore characteristics as well as the mining technologies applicable to the realization of this concept.

In Chapter 2 the philosophy of mine-mill integration, and in particular the integration of ore pre-concentration and waste disposal technologies into the hard rock mining system, is presented as an application of Lean Manufacturing for the increase in efficiency of the. Chapter 3 presents a review of the enabling technologies such as sorting and dense media separation, as well as related waste disposal technologies such as paste fill, cemented rock fills, and composite ‘rocky’ paste fills. A discussion of the integration of these technologies into the hard rock mining environment is also presented. In Chapter 4 the geo-metallurgical experimental methodology for assessing the potential for ore pre-concentration of a candidate ore, characterizing the products of pre-concentration, characterizing the waste rejects for disposal, as well as designing the appropriate pre-concentration and fill disposal systems is presented. Several examples from the testwork are cited to illustrate the results of a typical investigation.

A computerized parametric methodology for the valuation and evaluation the concept versus the conventional case has been developed over the course of the research. The model uses the results of the geo-metallurgical evaluation as well as additional field data on the deposit for the assessment of the impacts and benefits of implementing the integrated mining, processing and waste disposal approach on a typical case study. Chapter 5 presents the features of this model which has been used to evaluate the potential of the approach on a number of deposits.

Case studies conducted in the course of the research have typically comprised literature review, site investigation, sampling, testwork and evaluation using the methods described in Chapters 4 and 5. Ores from over 26 case study deposits have been sampled and tested, and complemented by additional data for ores from the literature. Case study results from the research to date are summarized in Chapter 6. Additional details of the case studies are presented in the Appendices.

Chapter 7 presents a discussion of the results obtained to date, data supporting the concept of a general case, conclusions as to the significance of the research as well as recommendations for future development of the concept.

2 On the Application of Integrated Mining, Processing and Waste Disposal Systems in the Hard Rock Mining Industry

2.1 Introduction

The pre-concentration of Run–of- Mine ore (ROM) and subsequent disposal of the waste rejects underground, prior to transport and conventional treatment of the ore by fine particle processes such as flotation or leaching, is proposed as an extension of the concept of Mine Mill Integration philosophy, as an improved approach over conventional techniques such as more selective mining, resue techniques or simply increased mining rate, for improving the efficiency, economics and environmental performance of hard rock metal mining. The approach is proposed as an effective application of the Lean Manufacturing philosophy (Shingo, 1992) to Hard Rock Metal Mining. Success in ore pre-concentration involves the identification and rejection of a significant quantity of barren, siliceous gangue material from the Run-of-Mine (ROM) ore prior to conventional fine-particle processing. It is important to properly classify what is meant by ROM ore in this context. ROM ore typically includes both valuable and non-valuable components in the form of dilution. Dilution of the ROM ore is usually defined as follows:

$$\partial = \left[\frac{twaste}{twaste + tore} \right] \times 100\% \quad \text{- (1)}$$

Where t_{waste} is the quantity of non-valuable rock component

Levels of planned dilution can vary widely depending on the mining method (Table 2.1).

Table 2.1- Planned Dilution by Mining Method (Canadian Mining Journal, 2007)

Method	**Ave (%)**	**Min (%)**	**Max (%)**
Shrinkage stoping	13.3	5.0	20.0
Open Stoping	14.0	5.0	32.0
Blasthole	15.6	6.0	25.0
Cut & fill	15.7	5.0	30.0
Longhole	23.3	15.0	30.0
VRM	27.5	18.0	37.0
Caving	52.0	52.0	52.0

A second observation to make in this context is that the mineralization in most hard rock ores, including ores such as porphyry ores conventionally considered low grade and homogeneous, are in fact highly heterogeneous in terms of the macro-mineralogy of the orezone. Other factors impact on the relative value of ore delivered to surface when compared to the value of the ore resource. In the process of creating the ore reserve by delineating the resource into ore blocks using the Smallest Mining Unit method (SMU), material which is not mineralized is included in the definition of ore, thus reducing the grade of the reserve; furthermore material which is mineralized is inadvertently excluded from the reserve through simple lack of resolution in the method, thus reducing overall extraction of the resource. ROM ore, in addition to planned dilution, thus typically includes additional barren material which may be rejected by the pre-concentration process, including unplanned dilution, barren gangue minerals included in the definition of 'ore', and gangue minerals (such as metal oxides and sulphides) directly associated with the valuable metal component (Figure 2.1).

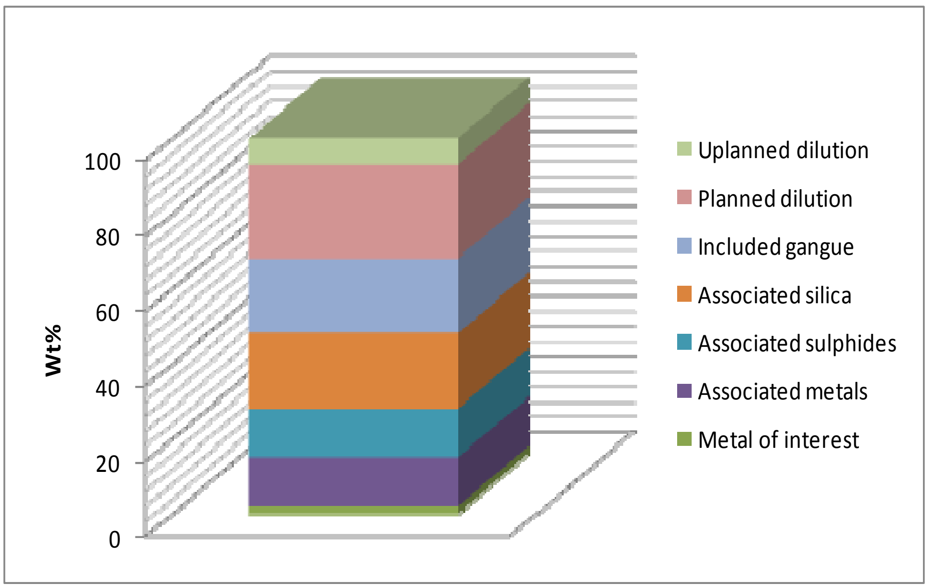

Figure 2.1 Typical Composition of ROM Ore

A large proportion of the ROM ore thus comprises both planned, unplanned, and internal dilution by gangue materials, as well as valuable minerals from material defined as ore as well as valuable minerals from material defined as 'waste'. Unplanned dilution also varies widely by mining method, in similar proportions. Loss of valuable fines has also been consistently noted to result in negative deviations to the head grade of ROM ore, particularly in gold, as well as high-grade base metal operations. The grade of ROM ore can thus vary significantly from plan (Howard et al, 2005), and this presents huge challenges in the efficient operation and control of the production system. Both grade and tonnage variations can be principally attributed to variations in the level of ore dilution as defined above, as well as the loss of valuable fines. The aim of an appropriate process strategy would be to maximize the liberation of these gangue components at a coarse particle size, identify and reject them from the ROM ore as early as possible in the mining cycle, while retaining valuable fines components, thus maximizing the retention of the valuable components. Ore quantity is minimized while maximizing quality. The capacity required of all systems downstream of this process is significantly reduced and is thus considered to be a Lean approach to hard rock mining.

Lean Manufacturing represents a philosophy for the structuring, operation, control and management of production systems for the elimination of waste, unnecessary work and hence an increase in efficiency of the industrial system, the application of which has resulted in enormous improvements in productivity and efficiency at many corporations. It is a concept that developed in the context of the motor manufacturing industry in the 1950's, and has found its most significant application at Toyota Motor Corporation in the form of the Toyota Production System. Toyota is considered the most globally successful motor manufacturer, and the Lean Manufacturing philosophy is widely credited as the single most important contributing factor to this success. More recently, Lean Manufacturing as it is applied at Toyota has been studied and put into practice at companies such as GM, Motorola and Boeing Corp (Yingling et al, 2000;). The fundamental concepts of Lean Manufacturing as originally applied at Toyota can be broken down into three principal areas (Shingo, 1992):

- design out overburden (*muri*)
- smooth out production (*mura*)
- and eliminate waste (*muda)* from the manufacturing system

Overburden can comprise unnecessary manufacturing capacity, systems or people. The smoothing out of production entails efforts to remove variance from the quantity as well as quality of the product in question. The elimination of waste embodies the physical removal of unnecessary product (as in machining), the removal of waste products (whether solid, liquid, or gaseous), as well as the elimination of re-work due to poor product quality. Seven types of waste are typically identified in this context (Ohno, 1988): wasteful processes; overproduction; excess inventory; rejects/rework; transport; waiting time; and unnecessary motion or activities.

Lean Manufacturing principles are typically applied to the behaviour and actions of workers and supervisors, which encourage the identification and removal of examples of such waste from the manufacturing system through the application of '*Kaizen*' (Shingo, 1992). Only in rare cases does the application of the Lean Manufacturing philosophy lead to the re-structuring of a business or process, although this can result in substantial benefits in terms of the objective of Lean Manufacturing. The application of ore pre-concentration and subsequent waste disposal technologies to the hard rock mining system is expected to result in a substantial reduction in system overburden, and a significant smoothing of production through the rejection of significant amounts of waste as early as possible in the mining cycle. It is this application of pre-concentration and waste disposal to the basic design and organization of the Hard Rock Mining production system which is the focus of this thesis. In order to evaluate these impacts to the capacity and performance of hard rock mining system, the system must first be adapted and configured appropriately for analysis in terms of the Lean Manufacturing concept.

2.2 The Application of Lean Manufacturing Concepts to the Hard Rock Metal Mining System

Lean manufacturing philosophy is fundamentally and originally a concept for addressing these issues of waste and inefficiency in discrete production systems such as manufacturing. However, with adaptation, the concepts of Lean Manufacturing can be applied to continuous systems such as the mining system (Yingling et al, 2000; Vagenas et al, 1995). Companies such as Rio Tinto and Barrick Gold Corp have applied some concepts of Lean Thinking to their operations (Krawchyk & Ghuldu, 2006; Dunstan et al, 2006), particularly at milling and smelting operations, with some success. However, these previous applications of Lean Thinking have focused mostly on organizational systems and personnel behaviour in the mining environment. Six Sigma, for example, is a system for the adoption of waste removing behaviours which has found wide application in the industry. However, Six Sigma itself is fundamentally a measure of quality alone, and does not embody the full breadth and depth of thinking implied in Lean Manufacturing as it does not address potential changes to the fundamental characteristics of the production system, and the results of applying Six Sigma in order to improve the quality of the core product (ore) in the mining industry have been at best marginal.

Vagenas, Scoble and Baiden (1995) explore a more structural interpretation of Lean mining through the application of automation and tele-operation to reduce the total complement of mining equipment required for a given production rate. However, even this interpretation of Lean mining fails to address the significant problem of varying quantity and quality of the product of the mining system. Further application of the concept is required to address this. Product quality variations in mining can be significant: the mass flowrate, particle size and chemical composition of ROM ore delivered from the mining operation to the metallurgical operation is known to vary widely on a daily basis (Hinde et al 1986; Lloyd & van der Walt, 1986; Tamlyn, 1994; Willis, 2000; Howard et al, 2005). Reducing these diurnal fluctuations is a significant challenge, and achieving significant and permanent gains in ore quality requires more than just changes to the attitude, behaviour and performance of the personnel operating production system. Structural changes are required in the way that ore is delineated, mined and processed in order to achieve the next step forward in the quality and consistency of production from mines. Lean Manufacturing has been previously examined in this particular context (Steelman et al, 2007), where the potential applications of Lean Thinking to core activities within the mining system (such as mine planning, material handling and mineral processing) as well as on common services and support functions to the core process (such as procurement) were highlighted. Key questions asked in the Lean review process were:

1. What are the system bottlenecks?
2. What is the efficiency, reliability, utilization and productivity of the system?
3. How effectively are common services and support functions delivered?
4. How well does the management system support the existing production system?
5. How well does the operation of the system match the planning of the operations?
6. How is ore dilution, grade and ore chemistry managed?
7. What is the variability of the feed- and product quality in the process?

The evaluation and recommendations of the study focused mostly on issues 1 – 5. Addressing issues 5 and 6 was seen as a significant challenge, as the present hard rock mining paradigm is primarily focused on economies of scale where quality of the product i.e. the ore is seen as secondary to achieving these economies of scale. Modifications of the process management system, or adjustments to the behaviours and attitude of operating personnel only are considered ineffective in addressing this issue. It is posited that the achievement of meaningful improvements in total product quality requires a re-thinking of mining production philosophy and a restructuring of the production system itself.

2.3 Ore Pre-concentration and Waste Disposal as an Example of the Application of Lean Manufacturing in Mining

In terms of addressing this key issue of product deviation in terms of quantity and quality, it must firstly be noted that ore, and more importantly the metallic component of the ore, is the primary product in all steps of the mining process, and variance in the quality of the product, occurs on a continuous basis in a typical operation in 4 primary ways:

- excess of ore (overproduction)
- paucity of ore (underproduction)
- under quality ore (below grade)
- over-quality ore (above grade)

The advent of such situations cause unplanned deviations within the production system, and can cause substantial losses due to loss of product during the lead time in which the system adjusts to accept the new product quantity or quality. Losses can occur in delivering low grade ore as the increased content of gangue material and deleterious elements can negatively affect recovery to concentrate in the mill. Losses also occur in the delivery of ore of higher grade than planned due to slow responses in the plant control system in adjusting process parameters such as sump densities or reagent dosing appropriately. Such deviations can arise due to a total system failure, as in a hoist or shaft failure, or simply the arrival of a batch of above-grade ore in the flotation circuit, where sulphides may be inadvertently lost to tailings due to the overloading of the froth. Any of these situations may occur individually or simultaneously (e.g. overproduction of low grade ore), further exacerbating the projected loss. Data from field operations confirms this observation. Previous studies of ore quality variability at BHP Pilbara indicate that variations of up to 2% in head grade and variations of between 0 and 200% of the planned tonnage to the mill led to a long term decrease of up to 1% in recovery of metal to final product (Kamperman et al, 2001).

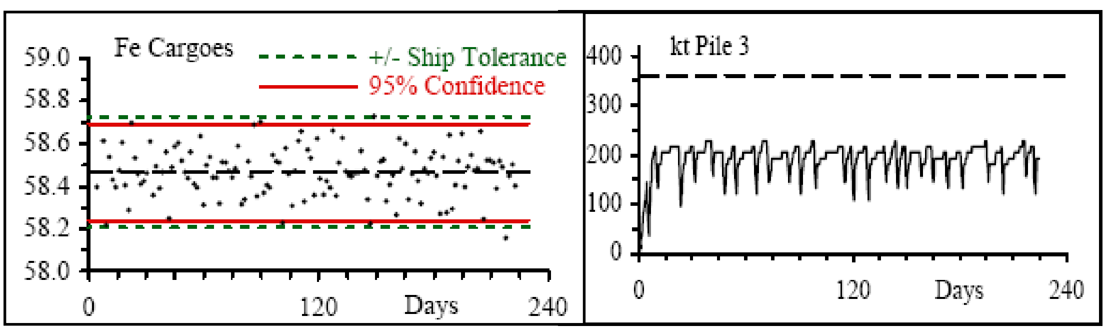

Figure 2.2 - Variance in iron grade and tonnage shipped ex Pilbara, BHP Billiton, WA (after Kamperman et al 2001)

The results of studies on the impact of variable of ore quality and quantity on operational performance at the Ulanskii and Krivbas mines in Russia made by the Leningrad Mechanobr Institute through sampling, modeling and simulation support this data (Kazanskii, 1973; Fugzan et al, 1971). An increase of 18% in copper grades, coupled with a reduction in grade variability of 35% was shown to improve overall metal recovery at Ulanskii by 0.84%. Several methods for stabilizing ore quality at the mine, including increasing the selectivity of mining, and removal of ore dilution were recommended. In the study by Epelman and Filiminov (1972) at Krivbas iron ore mine, it was shown that ore grade and tonnage varied significantly against plan and that a reduction in variation in the Fe content of the ore of between 2-4%, coupled with a reduction of 10% in the deviation of tonnage from plan at the mine resulted an increase of 21500 tonnes (5%) in metal recovered to concentrate. It was identified during the study that quality deviations from stope to stope were unavoidable; however ore quality stabilization by either more selective mining, increased grade control, tactical blending of ore streams underground or in fact separation of the individual ore streams of various quality were suggested as an economic alternative to the base case. Such deviations lead to unnecessary capacity in transport and treatment processes and furthermore that are problematic to the efficient operation of the mill. Several remedies to this deviation were recommended: increasing mining selectivity; improved grade control in the stopes; re-planning of mining sections by ore type; stockpiling and blending of the ore underground; transport of all distinct ore types in discrete streams; surface stockpiling and blending through re-handling (Kazanskii, 1973; Fugzan et al, 1971; Epelman & Filiminov, 1972). All of the above remedies are, however, considered costly in return for comparatively marginal benefits and thus require careful consideration at the planning stage. The strategic application of ore pre-concentration, either on surface ore underground, and the subsequent disposal of the solid waste prior to delivery of the ore to the mill is suggested as a more efficient and cost effective means of addressing these unplanned deviations in ore quality and quantity that have been identified as deleterious to the mining and processing operation.

The integration of the pre-concentration step into the surface or underground mining system implies structural changes to the basic mining system. Several aspects of the mining system thus must be considered in preparation for the evaluation of the integration of these processes as a Lean Manufacturing approach. Fundamental differences between the mining industry as a continuous production system, and a typical discrete manufacturing entity exist (Table 2.2), and thus a reduction of the typical hard rock mining system into discrete processes must be made.

Table 2.2 – Functional comparison of manufacturing and mining systems

Mining	**Manufacturing**
Process is continuous and cannot be arbitrarily stopped	Systems and products are discrete and production is discontinuous
Geologically and geographically constrained	Manufacturing sites can be located where market and infrastructure allow
Inherently variable and challenging environment	Production occurs within a stable and regulated environment
Operations are typically in rural or remote environments	Located near large centres transportation infrastructure
Quality of the raw material itself is highly variable	Raw materials and supplies are procured to specified standard and quality
Products are shipped as raw materials to the next process	Products are typically final products which go to market

For this purpose, the concept of the Value Chain, as originally posited by Porter (1980) is applied (Figure 2.3).

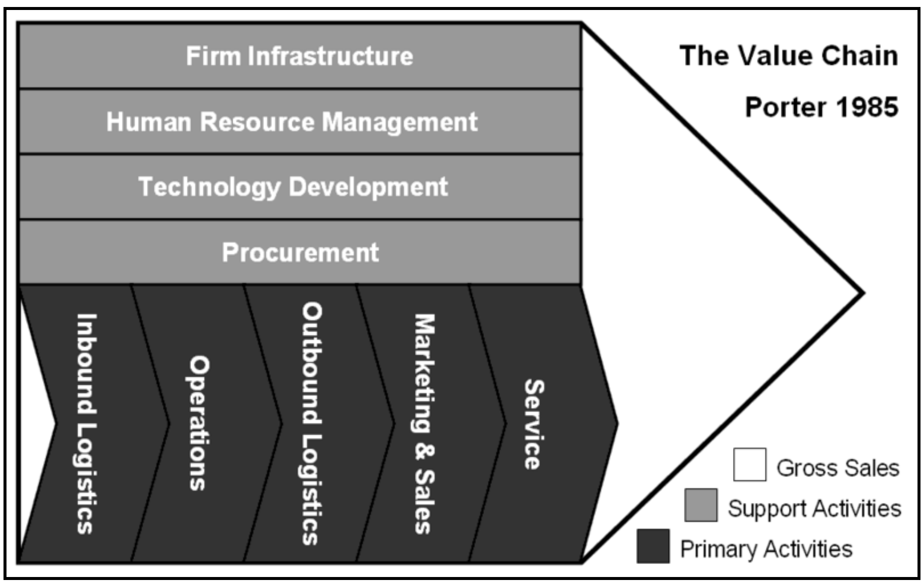

Figure 2.3 - Generic Manufacturing Value Chain (after Porter, 1980)

Like Lean Manufacturing, the Value Chain concept was originally posited for discrete systems such as supply-chain or manufacturing enterprises, however the concept has found wide application in the mining industry, as it provides a convenient way to describe operations to investors as the physical boundaries between primary activities such as mining, hoisting, surface transport and beneficiation provide convenient junctures at which to break this otherwise continuous process chain into discrete steps. For practical purposes in this thesis, a logical boundary for consideration is set at the smelter, at which point the mineral concentrate becomes an impure metal. The lower battery limit of the system is considered to be the mineral resource itself. An important interaction that has been identified in the analysis is the interaction between the mineral resource and the minable reserve, thus implying a significant impact on the sustainability of mining operations. Secondly, we introduce conventional waste disposal activities in the form of tailings disposal and the handling of development waste as an additional step in the mining value chain. Finally, the integration of coarse particle separation and solid waste disposal technologies into the mining process introduces three areas of recycle to the traditional system. Firstly, barren development waste or overburden that immediately arises during the mining sequence may be returned to the void. The balance of unreturned development waste or overburden is traditionally taken to surface and stored in dumps. The second entails the processing of this waste as fill for the underground stope. In the surface context there may be several different classes of solid waste, for example reactive- and non-reactive wastes, which must also be processed for appropriate disposal. A third area of recycle is introduced to in those operations utilizing hydraulic- or paste fills, where fine tailings traditionally disposed of on surface are prepared for placement as either cemented or uncemented fill material in the underground stopes. These steps must be incorporated into the value chain in order to represent the integrated mining, processing and waste disposal system in discrete steps, with recycle streams as appropriate (Figure 2.4).

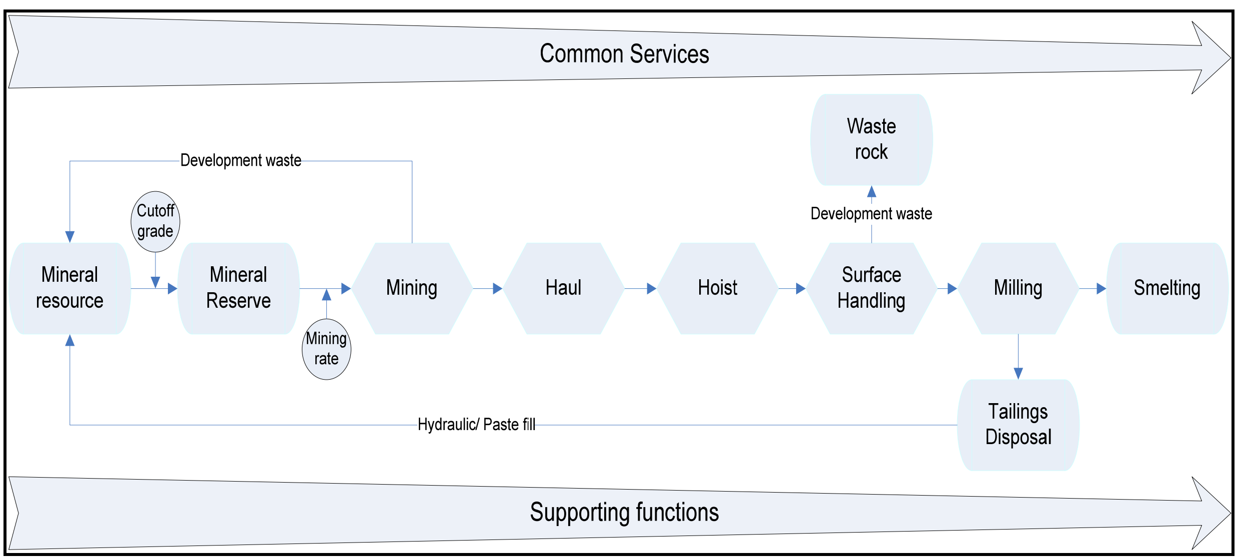

Figure 2.4 –Mining Value Chain with Recycle

Each step in the value chain has certain properties which must be constantly evaluated in the terms of the ‘Leanness’ of the mining system. Four key properties for consideration are the capacity and cost of each step in the chain, the complement of personnel required to operate and manage each step, and ultimately a measure of the quality of the product delivered by the process to the next step in the chain. Secondary properties may include measures of productivity, reliability, and utilization for each step of the process. In this thesis the focus is on primary properties, however some consideration of these secondary properties is made in specific cases. The focus in this dissertation is on primary impacts to the capacity, capital cost, operating cost, operational efficiency (in terms of metal recovery) and waste disposal requirements of the hard rock mining system. Capacity requirements in mining are typically fixed between the mining and the milling rate. The capacity / processing rate of the mining step in the value chain is typically determined as a function of the size and grade of the ore reserve, for example through the use of Taylor's formula for underground mines or by Lerchs-Grossman type optimisation in the case of open pits (Gentry & O'Neil, 1984). Until a concentration step occurs, the capacity, and thus the cost and personnel requirements for each subsequent step is determined (and constrained) by the output of the previous step in the chain. This is represented in the flowchart (Figure 2.4) by the size of the block. By removing dilution, as well as interstitial gangue, pre-concentration reduces the quantity, improves the quality, and reduces the variance in the quality of the ore arriving at the following step in the value chain, thus freeing the downstream processing from this typical constraint. We can thus evaluate the impact of introducing the pre-concentration step on surface prior to conventional fine-particle processing on the traditional mining value chain, for example as at Mount Isa Mines in Australia (Figure 2.5):

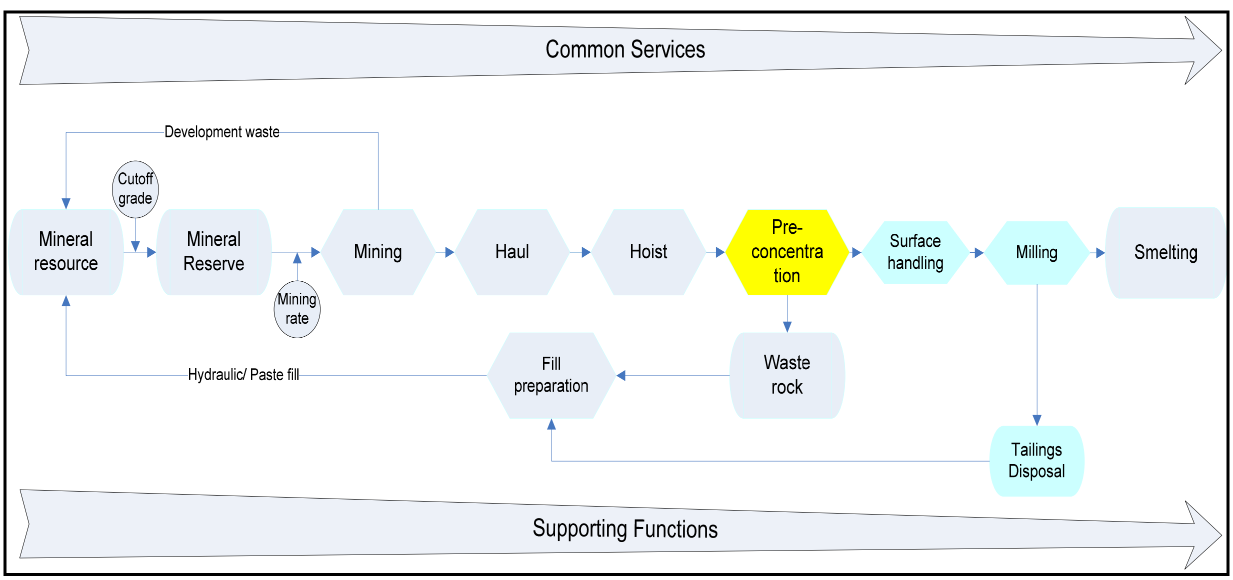

Figure 2.5 –Mining Value Chain with Surface Pre-concentration

Pre-concentration by dense media separation is introduced on surface prior to the transport of the ore by road to the mill. Additionally, the coarse waste rejects are returned via raise-bore to the underground stopes where they are mixed in a ratio of up to 25% by mass with conventional paste fill for use as composite fill (Kuganathan & Shepherd, 2001). The capacity and cost of processing in downstream processes is reduced proportional to the degree of waste rejection. An increase in mill capacity of 50% was achieved through reductions in the quantity of ore to be processed as well as an 18% reduction in the Bond Work Index of the ore (McCullough et al, 1999). This strategy is expected to benefit both open pit and shallower surface operations.

The impact of pre-concentration can be further enhanced if it is introduced earlier in the mining cycle, or value chain (Klein et al, 2002). There are two logical places in the value chain where pre-concentration can be introduced in the underground hard rock mining cycle – after blasting prior to underground haulage, or after haulage prior to hoisting (Figure 2.6);

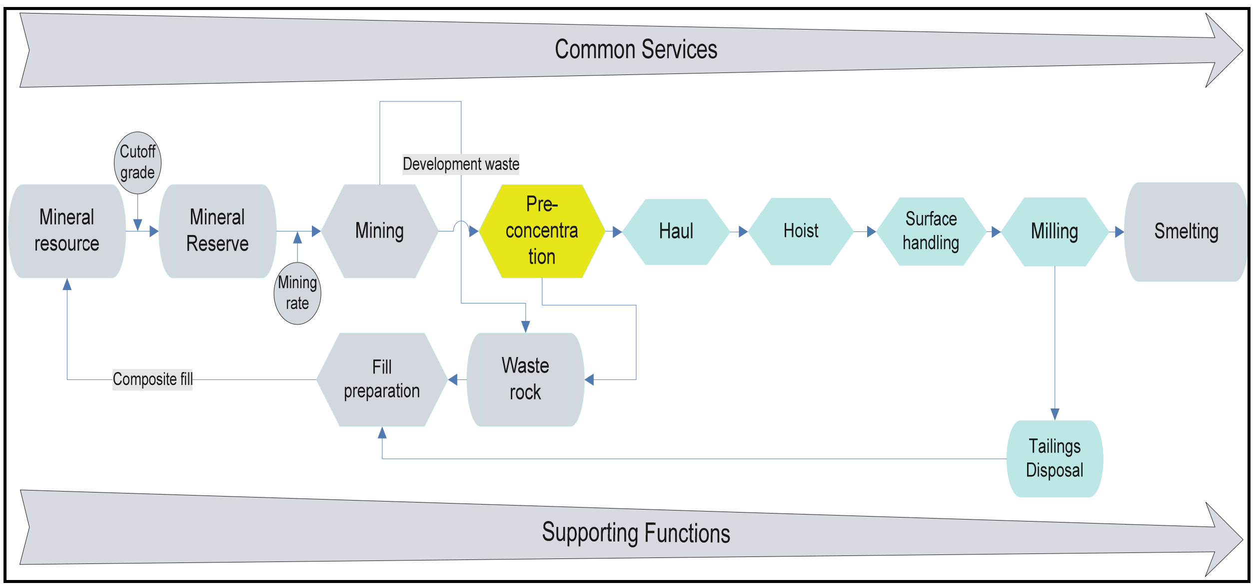

Figure 2.6 – Mining Value Chain with Underground Pre-concentration

Pre-concentration of the ore occurs prior to haulage or hoisting from underground. A solid waste recycle stream is now produced directly underground, and combined with tailings, water and cement as required form surface can be disposed of as fill. Capacity, quality and cost benefits are passed down all the steps in the value chain subsequent to the introduction of the process. As there are more steps in the process, benefits accruing in those processes are increased. Furthermore, the impacts and benefits are significantly enhanced for deep mines, or mines remote from the metallurgical operation. Overall impacts can be quantified parametrically using the model. A new step is introduced in the process, requiring additional capital, labour, power water and materials; however, the capacity, and thus the capital cost, operating cost and personnel requirements of all subsequent operations are reduced. Gross unit revenue is decreased due to additional metal losses incurred, however based on experience from previous studies, these are minimal and the overall impact is typically net positive (Scoble et al, 2006). Furthermore, the additional recovery penalty is often offset, if not overcompensated for, by increased recovery in the surface mill due to the higher feed grade and reduction in deleterious elements such as silica and talc in the ore (Stephenson, 2006; Blower & Kiernan, 2003). In such cases, the overall reduction in costs and increase in profitability decreases the economic cutoff grade for the conversion of the ore resource into a reserve; extraction of the ore is thus increased, further offsetting the negative revenue impacts (Bamber, 2006). Additionally, more cost-intensive processes can now be used on the resultant higher-grade ore, incurring further benefits. Finally, as in the case of operations such as Mt Isa, which employ the pre-concentration waste in combination with surface tailings as fill underground, there is a reduction in the total quantity of physical waste left on surface after operations have ceased, thus reducing closure requirements, which must be quantified.

2.4 'Lean' Case Study - Xstrata Nickel Ontario Operations

In the context of Mine-Mill Integration, the above concepts can be extended to groups of operations in order to fully realize the power of applying the Lean approach to the planning, development and operation of mines. A typical feature of mining is that the number and diversity of the steps in the value chain decrease at every step in the chain. Thus, many mines can typically feed a mill, many mills are required to feed a smelter and typically many smelters are required to justify a refinery. Many mining operations are standalone, others are logically integrated with a dedicated mill. It is common for a group of possibly open-pit or underground mines to feed a mill; however, not many individual mining and milling operations are large enough to justify a standalone smelter, thus a battery limit in terms of location (and possibly ownership) typically occurs at the point of feed to the smelter. Due to an increase in the degrees of freedom in the integrated production system, product quality variations and inherent losses are amplified in such a case. In order to address this variability, an integrated mining, pre-concentration and waste disposal strategy is proposed for Xstrata's Ontario Operations (Figure 2.7).

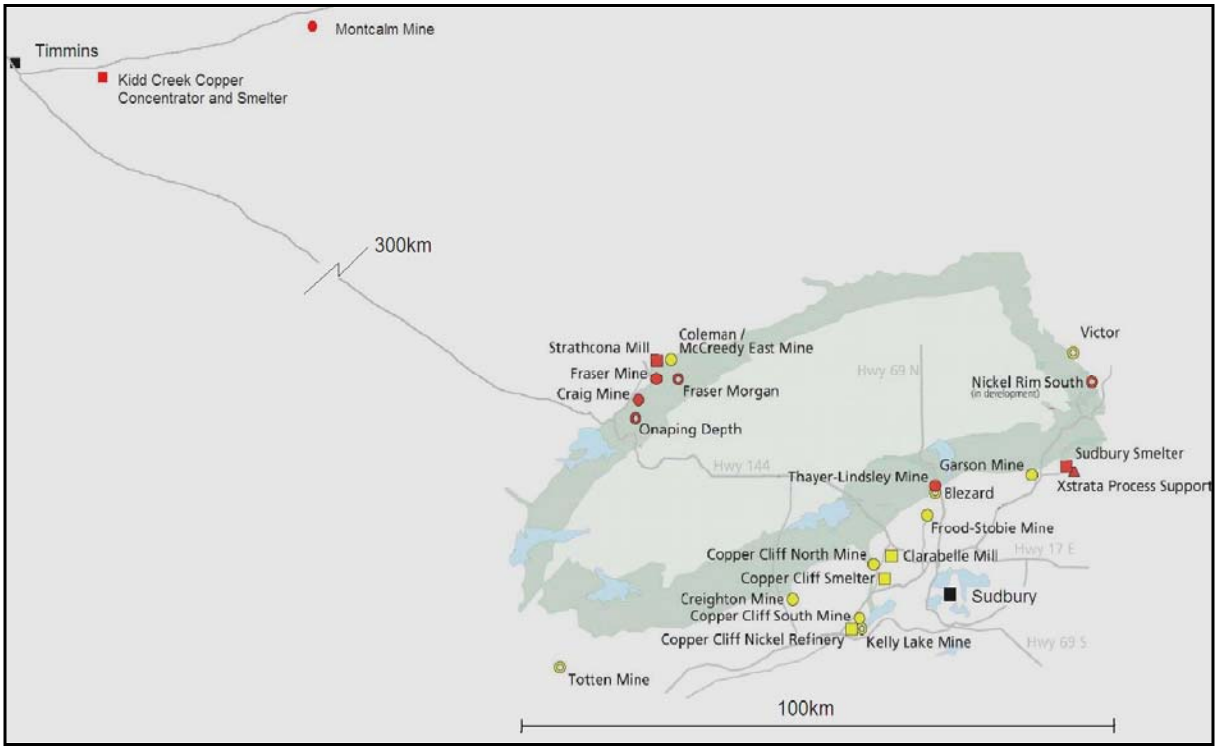

Figure 2.7 – Geographical Disposition of Xstrata's Ontario Operations (shown in red) (after Romaniuk, 2006)

Present mining and processing philosophy would indicate centralized milling for this group of operations in order to maximize economies of scale. At Xstrata Nickel, this is the case: mining operations are remote, however all ore is milled at Strathcona located in the North West of the Sudbury Basin. Nickel concentrate transported 75km SE to the smelter in Falconbridge Town; copper concentrate is trucked 300km NW to the Kidd Creek smelter. Ores is nominally stockpiled and blended at the mill, however significant variations in ore grade and quantity as delivered from the mine have been noted during studies. There are several reasons for this:

- Several different ore types are exploited simultaneously at Xstrata's Sudbury operations. These typically fall into 2 main categories, but up to 9 sub-ore types are identified (Proudfoot, 2006). Ores range from massive vein stringer type deposits, high in copper and precious metals and mined using highly selective mining equipment, to low grade nickel ores mined using bulk underground methods.
- Many mines produce both types of ore, and hoist them separately, however all ore types are mixed at the mill and milled simultaneously.
- The geographic diversity of Xstrata's operations is increasing as the company matures. Traditionally ores from the Strathcona locality only have been mined and processed, the exception being ores from the Thayer Lindsley Mine. A major expansion is currently

underway at the Nickel Rim mine, which is located 75km from the existing mill, through the town of Sudbury to the SE.

- Several operations require fill, and this is currently sourced from development waste at the mines, supplemented by classified tailings for those mines near Stratchona, and slag ballast trucked from Falconbridge smelter in the East to operations such as T-L which require ballast.

An integrated pre-concentration strategy for such diverse operations requires complex consideration. However, on closer examination of the range of ore types and geographic disposition of these operations, there are a number of obvious scenarios for the application of pre-concentration at Xstrata which are suggested for consideration in order to explore this concept:

- Customized milling at each mine by ore type
- Centralized pre-concentration of ore at Strathcona prior to milling
- Pre-concentration of ROM ore at each mine on surface
- Pre-concentration of ROM ore at each mine underground
- Creation of custom ore streams from each individual ore type through appropriate pre-concentration and/or classification of the ore underground

Of the nine principal ore types that were tested in the study, all displayed significant potential for pre-concentration (Bamber et al, 2005, 2006; Weatherwax, 2007). Footwall-type ores that were tested indicate a high degree of rejection of a barren waste, producing a high grade, possibly direct shipping pre-concentrate. Waste rejects were tested and found to be suitable for use as an aggregate for high-strength fill (Bamber et al, 2006). Contact-type ores presented a different scenario, where the degree of waste rejection was lower, however it was identified that a number of custom ore streams could be generated – low grade or barren waste for fill, low grade and high-grade ore streams. The results were used as inputs into the parametric model to evaluate the alternatives for applying the Lean approach to the mining and processing of these ores. At this stage of concept development, the consideration of these options through scoping level or pre-feasibility studies is obviously not practical, and the use of parametric methods as applied in the case study is suggested as a cost-effective means of defining, evaluating and comparing options for evaluating this as a Lean approach to mining. Through systematic evaluation of the options, a final strategy comprising the following was recommended:

- Pre-concentration of Onaping Depth ore underground
- Pre-concentration of Ni Rim ore on surface, with future potential for underground pre-concentration to be explored
- Pre-concentration of TL ore on surface
- Disposal of the solid waste underground at Fraser, Craig, Onaping, Thayer Lindsley and Nickel Rim

Applications at the Craig and Fraser Mines were not foreseen as life of mine is limited at these operations. Several major operational benefits are however expected to be enjoyed at future mines. Hoisting capacity at the two deepest mines, Onaping Depth (2500m) and Nickel Rim (1500m) is reduced by 30-50%. The quantity of ore to be trucked from Ni Rim to Strathcona would be reduced from 1.4Mtpa to 800 000 tpa, reducing the number of trucks driving through Sudbury town from 6 per hour to 4 per hour. The quantity of ore arriving at Strathcona Mill, and thus the size of the mill, could be reduced by 35% from 2.2 Mtpa to 1.4Mtpa, together with a resultant increase in feed grade of 32%. Based on the testwork, mill recoveries, especially for nickel and precious metals, are expected to increase; grade variability is projected to be reduced due to lower overall levels of dilution, and a resultant opportunity to shut down an entire grinding line at the mill has been identified. Further benefits would be enjoyed at the tailings dam. Present tailings strategy involves the deposition of a high-sulphur and low-sulphur product. The dam is presently topographically constrained, however, with pre-concentration the total quantity of tailings arriving at the pond is reduced, and an opportunity to generate a single high-sulphur tail on surface has been recognized. The generation of additional fill material from the waste streams at Onaping Depth, Thayer Lindsley and Nickel Rim would reduce the quantity of slag required to be trucked from Falconbridge smelter back to the mines from 800 000 tpa to 200 000 tpa. Total energy requirements for mining and beneficiation are projected to be reduced by 8.8% through testwork and modeling. Ontario is furthermore moving to an energy credit system, with Greenhouse Gas (GHG) credits valued at $15/tonne equivalent CO_2. Energy costs are projected to be reduced by 180MkWh/annum, thus GHG credit will be significant constituting a significant additional cost saving for the group (Bamber et al, 2007). These ores represent the majority of the present and future production at Xstrata Nickel's Ontario operations, and thus the potential impacts of adopting a Lean Approach as evaluated are essential for consideration for this group of operations.

2.5 Conclusions

The need for improvements in the effectiveness and efficiency of hard rock mining is dire, particularly in conditions of increasing costs and declining long term metal prices. Mine-mill integration, and specifically the integration of pre-concentration and waste disposal systems into the hard rock mining process, is seen as a strategy for improving the efficiency, cost effectiveness and environmental performance of mining operations. The case for evaluating this approach as an application of Lean Manufacturing philosophy to hard rock mining has been presented. The application of the concept is challenging, particularly as a structural change in the way mining and processing systems are designed and built is required. The model of the traditional mining system must be modified in order to evaluate the impacts and benefits. While the typical manufacturing system is discrete and discontinuous, hard rock mining is a constant and continuous process and thus requires special consideration in this context. An approach for adapting Lean Manufacturing principles to the typical hard rock mining system, as well as the modelling of the hard rock mining system as a discrete series of steps in a Value Chain is presented. The successful management of ore quantity and quality is seen as a major obstacle in achieving Lean goals in the industry; ore pre-concentration, whether on surface or underground is discussed as a means of achieving these objectives. The process of extracting and processing ore is shown to be less dependent on the given grade and physical presentation of the ore, grade control decisions are very much reduced in the system and ore grade and quantity variations are minimized through the introduction of process control on these variables. Mining selectivity is automatically reduced, and productivity increased (Bamber et al, 2005). Recycle streams are introduced where appropriate, capacity and thus cost and personnel are reduced in these areas. A significant improvement in product quality and quality control, efficiency and effectiveness coupled with a decrease in costs is projected for the cases studied. The operating scenario of a major nickel and copper producer Xstrata Nickel is discussed, and it can be seen from the results that a number of alternatives exist for the application of the concept, each of which brings a significant benefit to the operations. Thus it can be seen that the application of pre-concentration to the hard rock mining system facilitates improvements in all three of the key areas of Lean Thinking:

- reductions in overburden, in terms of the utilization of unnecessarily large equipment, and the additional operational and supervisory personnel required in transport and processing operations

- the ore production process is itself improved, and thus measures of the quality and quantity of ore produced are smoothed
- reduction in the quantity of waste material which traditionally has been unnecessarily processed in the mill, and the introduction of a new type of recycle stream which has the potential to reduce the quantity or alternately improve the quality of solid waste disposed of on surface
- increase in life of mine through increase in ore reserves at the same mining rate
- significant reductions in Greenhouse Gas (GHG) emissions due to a reduction in the energy intensity of the overall mining and beneficiation process

For the ideal case, a successful application of the concept would result in the maximum extraction of ore from the resource, the optimum disposal of solid wastes underground, reductions in the capacity required of unit operations downstream of the pre-concentrator, and a reduction in the overall surface and environmental footprint of mines. By enhancing the life of a particular deposit, the duration of the economic activity is also extended with further benefits to dependent communities. The triple bottom line of corporate environmental accounting for sustainability has been stated to be 'economy, environment and community'(Elkington,1994); it can be clearly seen that the integrated mining, processing and waste disposal approach impacts positively on all three of these areas, and thus the overall sustainability of hard rock metal mining.

3. Enabling Technologies

3.1. Introduction

Improved mine-mill integration, in the form of the strategic application of ore pre-concentration, either on surface or underground, and the subsequent disposal of the solid waste underground prior to delivery of the ore to the mill is suggested a as a means of improving the efficiency, economics and environmental performance, and thus sustainability of the sector. As has been previously discussed, the pre-concentration of ore is not a new approach. However, it is believed that the potential for the pre-concentration of hard rock metal ores is a general case and must be investigated in each instance in order to determine the feasibility of introducing this step into the mining system, whether on surface or underground. Several enabling technologies have been identified in the course of the research. Process technologies to be considered include optical sorting, conductivity sorting and dense media separation. Pre-concentration by comminution and size classification as well as coarse-particle flotation are also suggested as technologies with great potential. The application of these process technologies to the ore generates a coarse waste rock stream, and thus necessitates the further consideration of solid waste disposal technologies such as paste fill, cemented rock fill and in particular composite 'rocky paste' fills for the disposal of the reject material. Integration of these pre-concentration and waste disposal steps into the value chain also necessitates additional consideration of impacts on the mining methods available.

Two models for the application of integrated mining, processing and waste disposal technologies have been presented. In the first model, ore is pre-concentrated on surface prior to the transport activity. Waste rejected in this process may be classified by type and deposited on surface, as in open pit mines, or prepared as appropriate for disposal as unconsolidated rock fill in cases where no support is required, or alternately cemented rock fill or high-strength composite type fills in the case of mines which require the fill for additional geotechnical support (Figure 3.1).

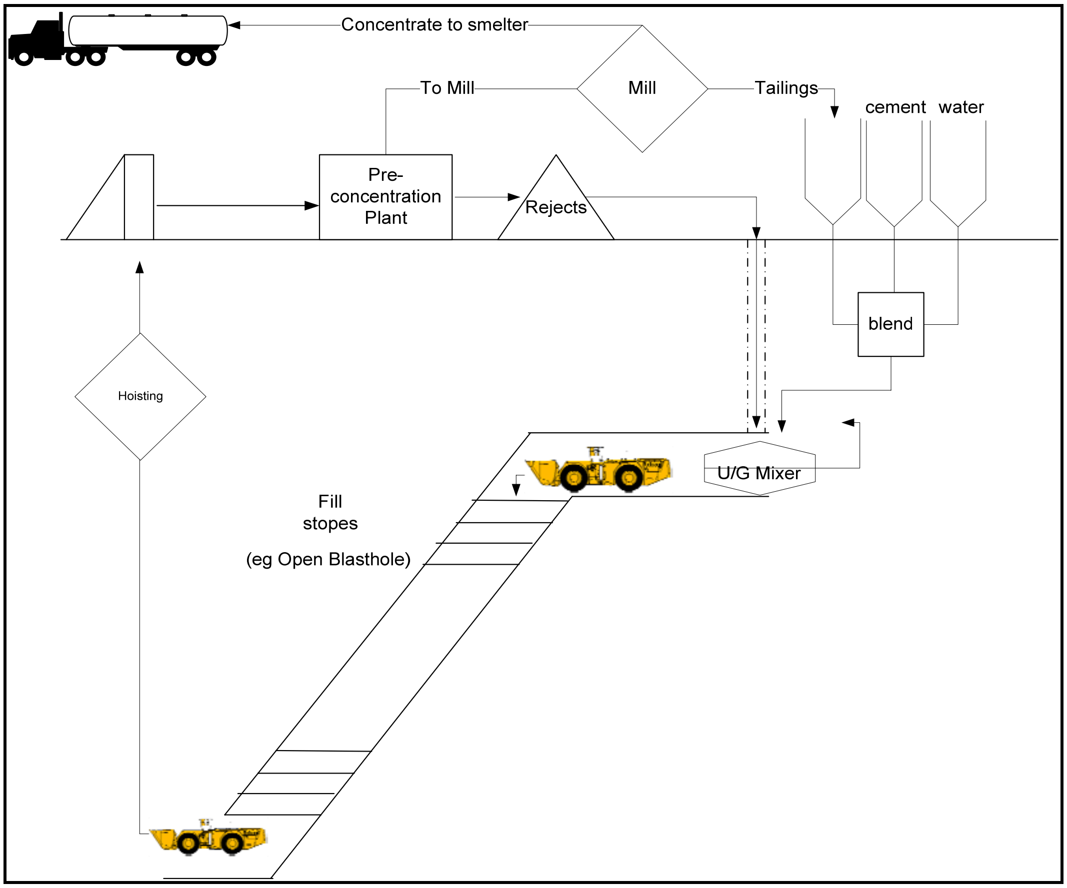

Figure 3.1 –Integrated Mining Processing and Waste Disposal Approach

For underground deposits characterized either by extreme depth, long underground haul distances, highly diluted ore, adverse ground control issues or perhaps with constraints on surface waste disposal, an integrated underground mining and processing approach is proposed, where ore is pre-concentrated underground, the pre-concentrate is hoisted to surface, and rejects are prepared in an underground fill plant together with tailings, cement and water from surface for disposal as backfill (Figure 3.2).

General consideration of pre-concentration of hard rock metal ores either on surface or underground, and the subsequent disposal of the waste rejects underground is proposed where that approach can be demonstrated to be economically and environmentally superior to the conventional approach. It is the integration of pre-concentration technologies simultaneously with waste disposal technologies in the mining of hard rock base metal ores which is considered both novel and highly original and is the focus of this thesis.

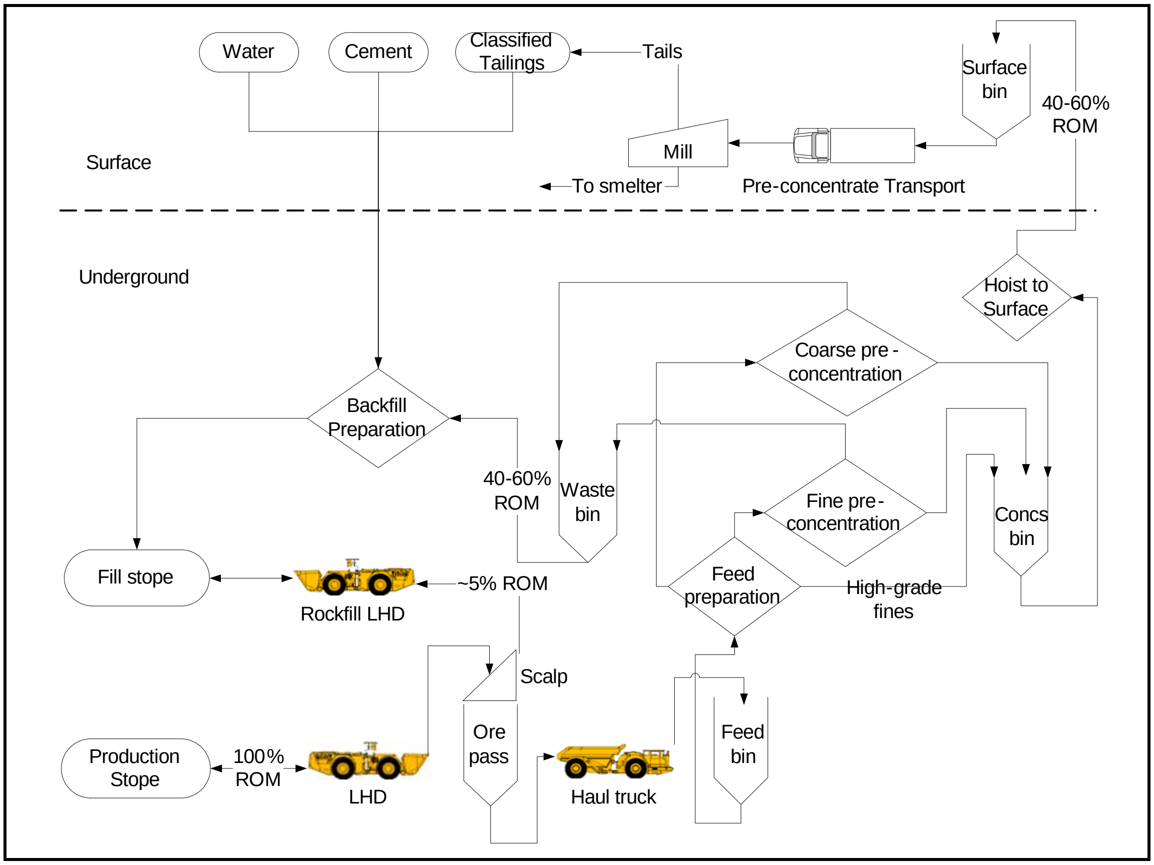

Figure 3.2 –Integrated Underground Mining, Processing and Waste Disposal System Design

This Chapter presents a review of the relevant technologies researched in the thesis and their application to the hard rock mining environment. Application of the system design to several of the available mining methods is also considered. Criteria and methods for measuring the potential for application of these particular technologies at selected operations have been established and tested through sampling and testwork in the laboratory during the course of the research, the methods for which are presented in Chapter 4.

3.2. Process Technologies

Coarse particle separation methods such as sorting or dense media separation are traditionally used on ores such as massive oxides, as well as coal, to produce a final product. However, the same processes are considered applicable in a pre-concentration role, and indications from the literature are that a high degree of waste rejection can be achieved from a wide range of ore types at a coarse particle size (Lloyd 1979; Ferrara & Guarascio, 1980; Schena et al, 1990; Mohanty et al, 2000). More recent additional examples supporting the use of pre-concentration on previously unconsidered ore types have also been noted in the literature (LionOre, 2006;

Collins & Bonney, 1995; McCullough et al 1999; Wilkinson, 1985; Wright 1985; Peters 1999; Sivamohan & Forrsberg 1991, Jones 2007, Salter & Wyatt, 1991; Kowalcyk 2002). In further support of this data, over 26 additional ore types have been tested at UBC with good results (Bamber, 2004, 2005; 2006; Mayne, 2005; Simonian, 2005; Stephenson, 2006; Weatherwax & Gillis, 2006; Weatherwax, 2007). Several technologies such as sorting by size, density, colour or conductivity have been emploted. All technologies that have been surveyed are compact, low-cost, high capacity mineral processes showing good metallurgical performance in the application. A combined table of results from the research combined with results from literature is presented in Table 3.1.

Table 3.1: Metallurgical Performance of Pre-concentration Technologies on Selected Ores

Pre-concentration method	**Ore Type**	**Feed size (mm)**	**%Wt reject**	**Metal Recovery**	**Reference**
HMS Cone separator	Pb/Zn	38	55.00	97.00	Wright, 1980
DMS cyclone	Cu porph	38	33.00	96.00	McCullough et al 1999
DMS	Cu porph	38	74.00	81.00	McCullough et al 1999
Shaking table/ Dynawhirlpool	Ni	3	90.00	89.00	Ferrara & Guarascio, 1980
DMS	Cu	13	75.00	95.00	McCullough et al 1999
Coarse flotation	Wits Au	3	60.00	98.00	Lloyd 1979
Coarse flotation	Wits Au	4	28.00	97.00	Lloyd 1979
Classification	Ni	1	27.00	89.00	Mohanty et al, 2000
Model 19 sorter	Pb/Zn	150	26.3	94	Collins & Bonney, 1995
Model 6 radiometric	U3O8	150	39	96.5	Collins & Bonney, 1995
Radiometric	U3O8	75	73	96	Collins & Bonney, 1995
Magnetic	Ni	100	40	96.7	Collins & Bonney, 1995
Radiometric	Au/U3O8	50	50	98	Sivamohan & Forrsberg 1991
Conductivity	Ni/Cu	25	80	80	
Conductivity	Ni/Cu	100	60	90	
Conductivity	Ni/Cu	25	87	61	
Conductivity	Ni/Cu	70	10.5	99	
Conductivity	Ni/Cu	70	27.83	82	
Model 16 optical	Cu	160	98	97	
Model 16 optical	Cu	100	32.7	96	
DMS	Pb/Zn	38	39.6	96.3	
DMS	Fe	38	31.4	98.3	
DMS	Pb/Zn	19	36.7	93.9	
Screening	Cu	100	20	99	
DMS	Cu	38	80	93.8	McCullough et al 99

Pre-concentration method	Ore Type	Feed size (mm)	%Wt reject	Metal Recovery	Reference
Optical sorting	Cu	100	44.8	94.8	Wilkinson, 1985
Conductivity sorting	Ni/Cu	100	54	82	Wilkinson, 1985
Optical + conductivity	Cu	100	50.2	97.7	Wilkinson, 1985
DMS	Cu	44	55	94.5	McCullough et al 99
DMS	Cu	38	35	97	McCullough et al 99
DMS	Cu	87.5	55	97	Wright 1971
DMS	Pb/Zn	19	37.4	98.7	Jones 2007
DMS	Pb/Zn	19	32.7	97.9	Jones 2007
DMS	Pb/Zn	19	23	99.4	Jones 2007
DMS	Cu/PGE	19	44.3	97.5	Bamber
DMS	Ni	6.7	14	99	Bamber
DMS	Ni	75	13	98	Weatherwax 2006
DMS	Ni/Cu/Co	75	32	89	Weatherwax 2006
DMS	Ni/Cu/Co	75	24.5	95	Weatherwax 2006
DMS	Ni/Cu/Co	75	54	98	Weatherwax 2006
DMS	Ni/Cu/PGE	75	36.6	97.7	Weatherwax 2006
DMS	Ni/Cu/Co	75	25.73	96.7	Weatherwax 2006
DMS	Ni/Cu/Co	75	19.52	94	Weatherwax 2006
DMS	Ni/Cu/Co	75	25.5	95.3	Weatherwax 2006
DMS	Ni/Cu/Co	75	32.37	96.4	Weatherwax 2006
DMS	Au	19	17.6	93.2	Bamber
DMS	Au	19	24	94.6	Bamber
Radiometric	Wits Au	250	44.1	87.9	Kowalcyk 2002
Radiometric	Wits Au	250	29.5	92.6	Kowalcyk 2002
DMS	Ni/Cu	300	22	95	Bamber
DMS	Cu/PGE	250	55	97	Bamber
Size	Cu/PGE	75	37	99	Bamber
Size	Cu/PGE	75	35	93.5	Bamber
Size	Cu porph	31.75	49	76.32	Burns & Grimes 1986
Size	Cu porph	31.75	54.37	75.75	Burns & Grimes 1986
Size		500	67.64	64.49	Burns & Grimes 1986

The results suggest that a high degree of waste rejection is possible at a coarse particle size with good metallurgical recoveries on a wide range of ores using the separation technologies selected for the research. Detailed results from testwork on specific approaches from the research are presented in the Chapter.

3.2.1 Ore pre-concentration by Comminution and Size Classification

3.2.1.1. Development of the Concept

Several citations from literature suggest that concentration by comminution and size classification alone would be effective for the concentration of some ores (Mohanty et al, 2000; Sivamohan & Forssberg, 1991; Jones, 2007; Burns & Grimes, 1986). Evaluation by the size-assay method is required to indicate an opportunity for this on a potential ore (Bamber et al, 2006). Using the size-assay data, similar results had been indicated in previous research with INCO (Buksa & Paventi, 2002; Bamber, 2004). The data indicated that Footwall ores of the Sudbury Basin exhibit the characteristic of both a coarse, barren fraction as well as significantly upgraded sulphide content in the fines fraction (Figure 3.3).

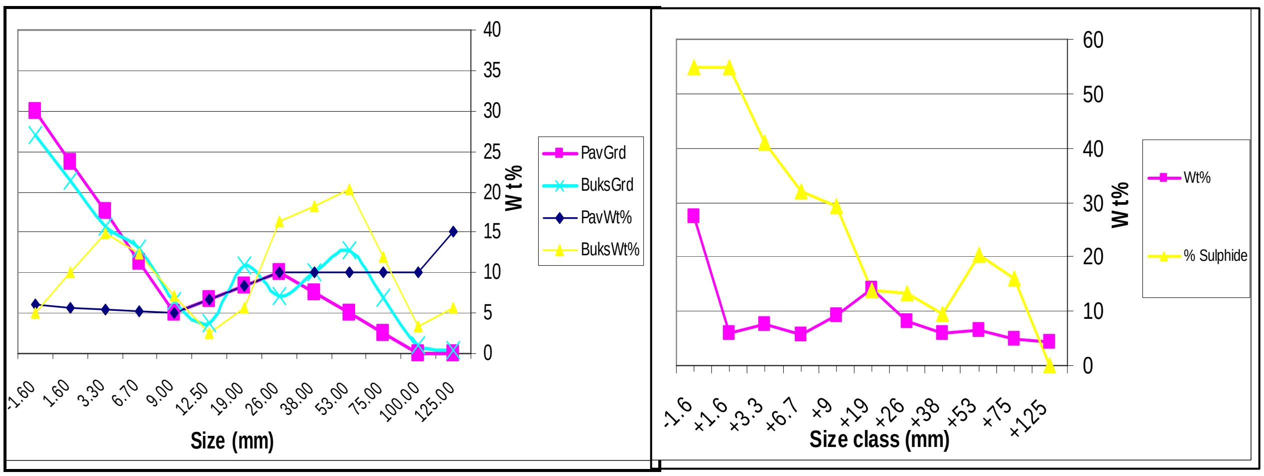

Figure 3.3 – Size-assay data for INCO 153 orebody (left after Buksa & Paventi, 2002)

It was suggested that this might indicate a case for the concentration of this ore simply on the basis of size alone. The literature suggests that this feature of variable grade distribution by size is common in a number of ores, including massive sulphide ores, base metal ores, as well as gold ores and some platinum ores, where, due to the friability of the high-grade ore, and the high density of the valuable mineral, the majority of metal values appear to be present in the fine fraction and thus this fraction is significantly upgraded compared to the average. Data presented from van der Berg & Cooke (2004) indicates that a characteristic upgrading in the fines of some Bushveld platinum ores could be a basis for pre-concentration of these ores prior to hydraulic hoisting (Figure 3.4).

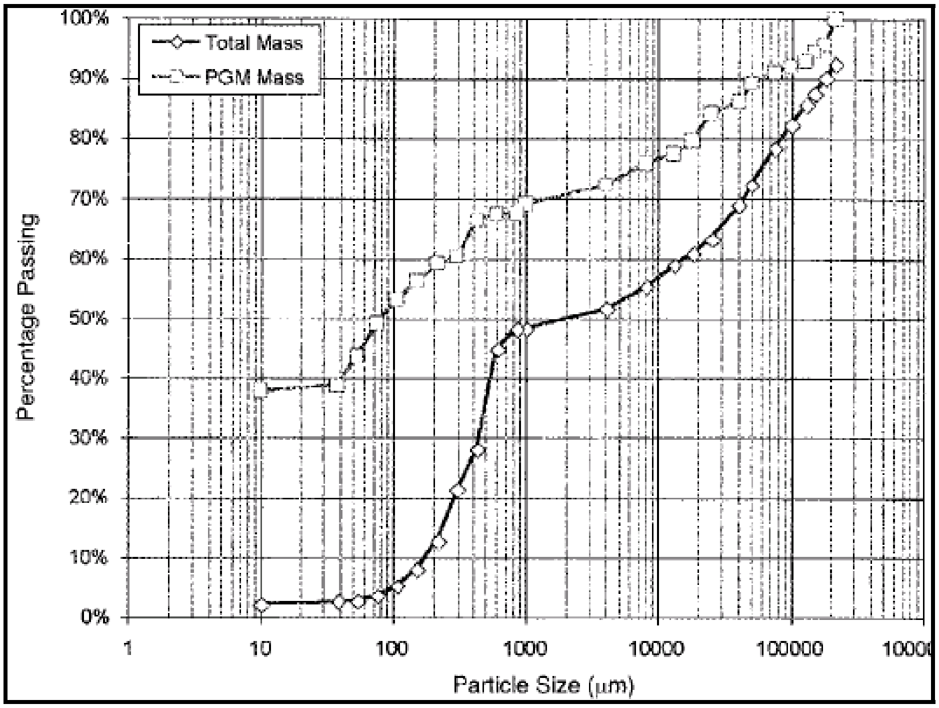

Figure 3.4 – Pt enrichment of fines in Bushveld Pt Ores (after van der Berg & Cooke, 2002)

This feature of ROM ore, if identified to be present, has great potential for the rejection of waste from ore at low cost. The characteristic size/grade relationship suggests one of two possible concentration opportunities:

- Rejection of a relatively barren size class
- Preferential acceptance of a relatively enriched size class

For maximum economic impact, waste rejection through pre-concentration is preferred at coarse particle sizes (Klein et al, 2002), either at the naturally arising ROM particle size distribution, or at a coarse crush size, as close as possible to the mining face. There are some examples of this concept applied in practice, as at Kroondal Platinum Mine in the South African Bushveld. Wide-reef mining methods are enabled in the mine as coarse barren pyroxenite is rejected simply by scalping in the stopes prior to the loading of the ore onto the underground strike conveyors. The ore is then further upgraded on surface by a combination of dense media separation and flash flotation[1]. A further example of ore pre-concentration by crushing and size classification only is referenced for Rio Tinto's Bougainville Mine in PNG where it was found that the -38.5mm material in the ROM porphyry ore was significantly upgraded in copper values compared to the average (Burns & Grimes, 1986). A crushing plant was installed in order to enhance this, and the enriched fraction was subsequently screened out and sent to the flotation

[1] http://www.aquariusplatinum.com/aquarius_db/pdfs/Kroondal.pdf

plant for processing. Similar characteristics were observed during testwork on ores of the Mississippi Bonaterre formation near Viburnum in Missouri (Jones, 2006). A significant upgrading of the fine fraction in the Buick and Viburnum ores can be seen from the size assay data. Concentration based purely on size is indicated in the preliminary assessment on both of these ores.

3.2.1.2 Testwork at UBC

During the research, it was observed that comminution may further improve the liberation of the ore as well as alter the distribution of metal values in the ore more favourably (Bamber et al 2007). Coarse particle sizes are desired thus comminution by gyratory crushing, jaw crushing or autogenous grinding was considered. 500kg stope samples were taken from the McCreedy East orebody and subjected to a series of tests. In the first phase of work, size assay of the contact-type ore sample indicated the presence of a coarse barren fraction, but little natural upgrading of the fines fraction. The +125mm fraction was weighed and assayed to assess the impact of removing this fraction by scalping. Waste rejection was 10% with a metal loss of 2% indicating some potential in this regard (Bamber et al, 2006). In the Footwall type ore, the high Wt%, as well as the high Cu grade of the fines indicated that a large portion of the ore value was represented in this fraction and the potential for mass upgrading of the Footwall ore through grinding and classification only was thus investigated (Mayne, 2005; Simonian, 2005). A 20kg full fraction sample of the 153-4550-2 sample, grading 13.2% Cu and 0.22% Ni was taken and dry ground autogenously at over 15min sequential intervals in a 3kW, 1m x 900mmØ tumbling mill. Products of each grinding campaign were screened at 19mm, weighed and assayed. Significant upgrading of the -19mm fines fraction was observed after the first grinding interval. Results are shown in Figure 3.5.

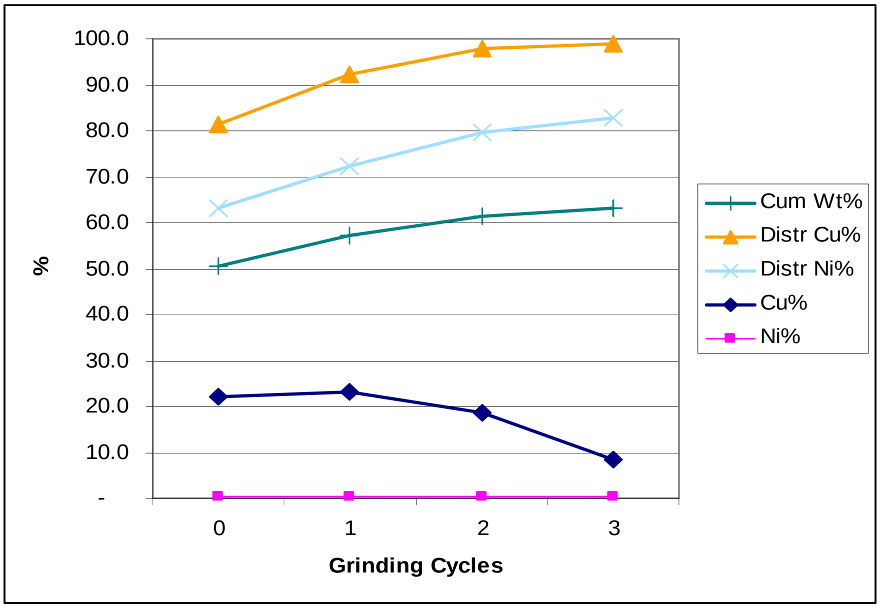

Figure 3.5 –Results from Autogenous Grinding and Classification of 153 ore -19mm Fraction

The -19mm undersize material was significantly upgraded compared to the feed sample over each stage of comminution, and was designated as concentrate in each case. A final concentrate comprising 63% of the mass of the sample grading 21.6% Cu and 0.28% Ni was produced at a metallurgical recovery of 99% for Cu and 83% for Ni. In this ore the nickel feed grade is low, and the pentlandite is transitional from the Contact ore down dip towards the Footwall (Naldrett, 1984), and typically remains mineralogically associated with silica in the Sudbury breccias and not the massive sulphides, and consequently metal recoveries are low. Overall the results indicate that 37% of the Footwall type ore could be rejected with a copper recovery of 99% by dry autogenous grinding in closed circuit with a 19mm screen. The results for the ore indicate that the ore is already well liberated at the ROM particle size; subsequent Bond Work Index testing on concentrate and tailings fraction of this ore type indicate that there is a significant discrepancy in the Work Index of the sulphide component compared to the gangue component of the ore (Altun, 2007), and that these two facts in combination indicate potential for this approach in upgrading this type of high-grade, massive-vein sulphide ore.

A second campaign of grinding and screening tests were performed on the 153-4550-3 sample grading 8.14% Cu and 0.35% Ni. In this test, assays were taken for all potentially valuable elements in the concentrate and tailings fractions of the ore in order to confirm metallurgical results across all metals. The results indicate a mass rejection of 35% overall to a concentrate grading 11.71% Cu, 0.29 Ni and 27.43 g/t total precious metals (TPM). Recoveries of copper, platinum and palladium to the -19mm concentrate are good, and indicate that these metals are all closely associated in the narrow vein sulphides. Nickel recoveries are also much improved over the first test with an overall recovery to concentrate of 92.3%. Ni and Au recovery is poorer indicating that these may not be associated with the massive vein Cu sulphides. The resultant grade distribution by size indicates a significant upgrading of metal values in the fines component of the ore after grinding and classification (Figure 3.6).

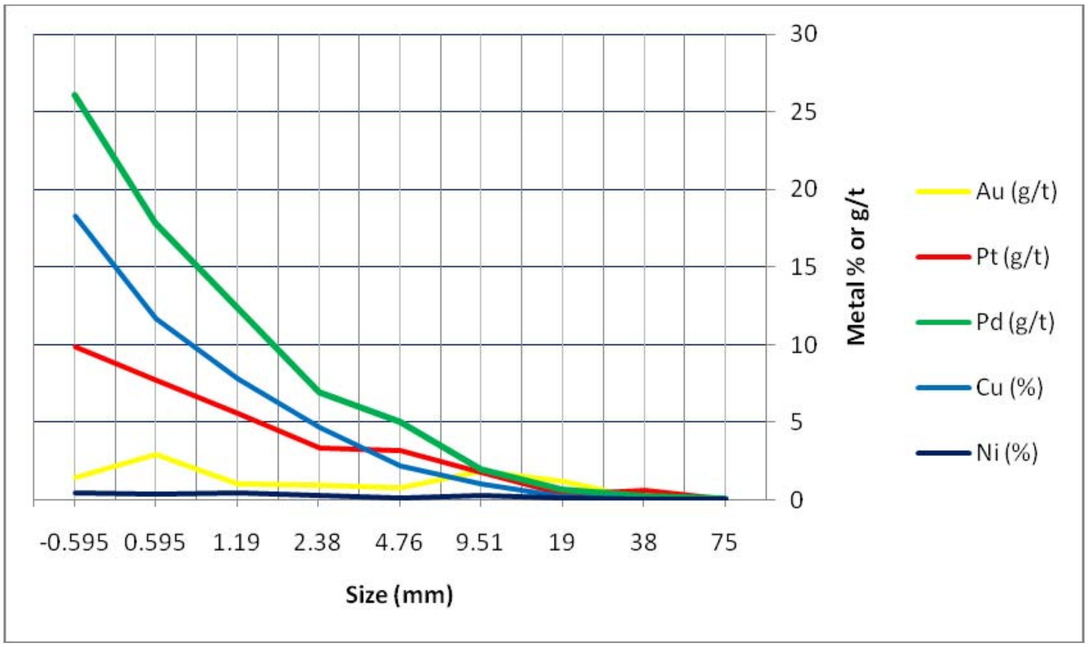

Figure 3.6 – Grade Distribution by Size in 153 ore after 60 minutes of grinding

Comminution and size classification is the most compact, and lowest cost, of all mineral separation processes, and the results indicate that this methods has great potential for the pre-concentration of this type of high grade massive sulphide ore. A proposed flowsheet for the adoption of this approach is shown in Figure 3.7.

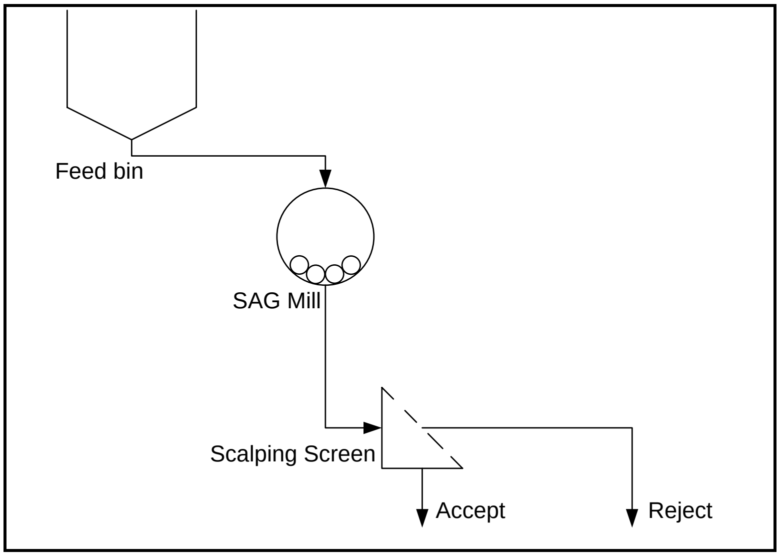

Figure 3.7 – Concentration by comminution and classification

3.2.2 Pre-concentration by Sorting

3.2.2.1. Sorting practices in the mineral industry

Sorting has been practiced in the mining industry for as long as ores have been mined. Native iron and copper ores have been mined and hand sorted since pre-Roman times. Cornish tin ores were traditionally hand sorted into various products, as were copper ores mined near Clausthal in Germany in the 1800's. Hand sorting as a means of pre-concentration the ore underground prior to hoisting has also been practiced historically – Agricola also noted the use of hand sorting in mines of Europe in the 16th century, encouraged by the mine owners in order to improve the economics of mining (Agricola, 1556). Automated sorting practices are more recent, with modern electronic sorters having been applied in the industry only since the Second World War. Pre-concentration by sorting can be used to improve the grade of large, low grade heterogeneous deposits by rejecting coarse, barren waste, thus significantly decreasing milling costs (de Jong, 2005). Radiometric sorting of uranium ores is the oldest known application of the technology at Port Radium and Port Hope in Ontario, Canada in 1958 (Collins & Bonney, 1995). Radiometric sorting was widely used in South Africa in the 1980s as a substitute for hand sorting of low-grade gold ores. Diamonds are probably the widest known application of sorting, in particular x-ray fluorescence, but literature on the technology is scarce. There are few examples of the application of sorting at a large scale in the metal mining industry, most recently the installation of an Ultrasort conductivity sorter at Kambalda in Western Australia (Goode, 2006), and the installation of a CommoDas Mikrosort Primary optical sorter at Amplats

Rustenburg UG2 Section (Kinver, 2002). Competing technologies to sorting are selective mining methods and gravity concentration; it has been suggested previously that pre-concentration is preferable both technically and economically to selective mining (Bamber et al, 2005) and that sorting as a simple low cost, dry coarse particle separation technology would be preferable to gravity methods as they typically require water, thus investigation and testing of selected sorting methods was considered crucial for this research.

The development of electronic sorting technology has largely been driven by the recycling industry and while sorting, and in particular photometric sorting, has found some application in the industrial minerals industry (Wotruba & Junsgt, 2000) the technology has not found as wide application in the hard rock mining industry due to several common misconceptions about the application and benefits of the technology (Salter & Wyatt, 1991). Despite the availability of a wide variety of sensing methods, and the low capital and operating cost of the technology, there are many challenges to the acceptance of sorting in the minerals industry, including a lack of understanding of the basic discrimination principles, opportunities for and applications of sorting; perceptions of high capital costs, high unit operating costs, and poor reliability for the technology; a perception that sorting incurs an unacceptable loss of metal; and the traditionally low capacity of sorters when compared to present throughputs of grinding and flotation plant. The need for specific feed preparation for sorting appears, wrongly, to be seen as a challenge, as other process technologies, for example DMS and heap leaching also require equally rigorous feed preparation in the form of feed sizing and fines removal. The main sorting technologies and their applications are presented in Table 3.2 (Salter & Wyatt, 1991).

Table 3.2 – Methods of Discrimination in Sorting

Method	**Application**
Photometric	Coal, sulphides, phosphates, oxides
Radiometric	Uranium, Witwatersrand gold ores
Conductivity	Metal sulphides, native metals
Fluorescence	Metal sulphides, limestone, iron ore
X-ray luminescence	Diamonds
X-ray transmission	Coal
Electrostatic	Salts, halite, slyvite
Magnetic	Iron ore, andalusite, quartz, kimberlites

Regardless of the principle on which the sorter is based, sorters share several common design features. Feed to the sorter must generally be closely sized, the particles are typically individually sensed and ejected, except in cases of whole-ore diversion, thus the feed from the hopper is typically accelerated by the sorting belt to between 3 – 6 m/s, and passed through the sensors. Belt widths vary between 1.2 to 2m for high capacity sorters. Under-belt sensors are used in conductivity and X-ray transmission applications, or in combined applications such as optical/conductivity sorting. The effectiveness of the process is often enhanced when used in combination, for example photometric + radiometric or photometric + conductivity. Sensor signals are passed to the micro-controller and compared to some previously determined threshold value in order to make a sorting decision. All sorters have one of several kinds of particle ejection mechanism, which can be physical (flappers), pneumatic (high pressure air valves), or even magnetic fields in the case of magnetic separators (Figure 3.8).

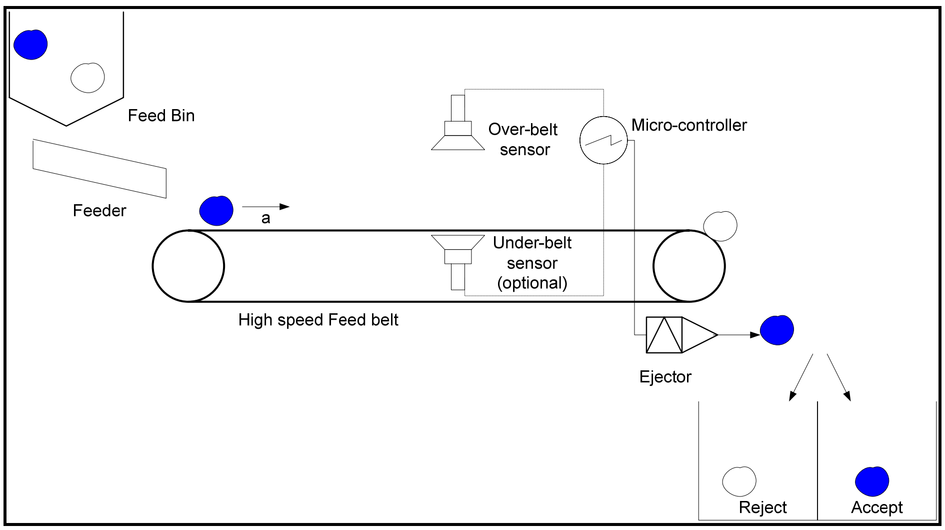

Figure 3.8 – Features of Sensor Based Sorters

The maximum throughput of the sorter is typically limited by the speed of signal processing, analysis and comparison that the microprocessor can achieve, thus the maximum number of particles the sorter can handle in a given time is a constant. This is limited to between 10 – 15000 particles per hour for modern sorters[2]. For optimum performance, the feed size to the sorter should be limited in range to approximately 4:1 (topsize : bottomsize) in order to control the particle count at the sensor within this range. Throughput is thus dependent on the nominal particle size and density of the feed to the sorter (Table 3.3). Recent developments with Ore Sorters Model 13 and 16 optical sorters indicate good sorting results on 20mm particles at throughputs up to 75tph and at belt speeds up to 4m/s (Arvidson & Reynolds, 1995).

[2] www.mogenson.com/mikrosort.htm

Table 3.3 – Typical Sorter Capacities (after Wotruba & Jungst, 2000)

Feed size (mm) **Min**	**Max**	**Capacity** **tph**
20	60	20
10	200	100
100	350	300

Sorters are also somewhat limited in application compared to gravity separation methods as a practical lower particle size limit of 10mm appears to exist for most discrimination methods. This, however, is perhaps not as limiting as it might first appear as it has been identified in the research that ores considered optimal for pre-concentration possess a significantly upgraded fines fraction, thus sorting of this fraction is mostly considered superfluous. The limitation on the sorter topsize is driven largely by limitations on the type of ejector mechanism used in sorting (air or mechanical diversion), as well as consideration of the typical decrease in the degree of liberation in the large ore particles. The overall limitation on sorter capacity is irrelevant at low tonnages, however this has been identified as a challenge to be overcome in designing a process plant for tonnages over 15 000 tpd due to the unfeasibly large number of units that would be required (Bamber, 2007). However, despite or because of these limitations, sorters are indicated for particular applications in the industry:

- Bulk treatment of low value commodities such as construction aggregates and industrial minerals
- Pre-concentration of low grade or highly diluted ore, in-pit or even underground
- Re-processing of waste dumps
- Recycling

It has also been identified in the course of the case studies as well as the literature that sorting has a particular capability to produce custom material streams. For sorting to be successful there must exist:

- Sufficiently large disparity in the physical characteristics of the valuable and non-valuable ore fractions
- Sufficient liberation of the non-valuable fraction at a coarse particle size
- The existence of a commercial sensor with the capacity to discriminate between the physical characteristics of the valuable and non-valuable fractions

The benefits of sorting when applied to base metal ores, as with all methods of pre-concentration, are manifold and significant and the potential for increased application of ore sorting in the industry is thus significant. Several of the ores sampled in the course of the research have demonstrated clear visual discrepancies between the ore and waste fractions (Bamber et al, 2006). Optical sorting had been previously investigated by INCO at a preliminary level on some of these ores with extremely good results (Buksa & Paventi, 2002). INCO Mines Technology conducted electronic sorting tests on synthetic McCreedy ore samples using a combination of optical and conductivity sensing in a Mogenson sorter, which gave results of up to 77% waste rejection at a recovery of 98% (Schindler, 2001). In the investigation of low-grade ultramafic hosted nickel ores of the Thompson nickel belt, the occurrence of chrysotile minerals present in the ore was visually discernible and identified as a basis for the preferential rejection of this fraction prior to grinding and flotation.

Several principal sorting methods are currently in use. By far the most common type of sorters are photometric models (Wotruba, 2006). Next most common are radiometric, fluorescent and conductivity sorters. A typical sorting flowsheet for base metal sulphides would comprise the following stages: feed preparation; bypass of the high grade fines; coarse sorting and fine sorting; product and waste handling. Possibly the biggest challenge for sorting is in the discrimination of optically indifferent low-grade sulphide ores as often there is a variety of colour and texture in the ore, thus photometric methods are ineffective, and the discrimination of metal values by conductivity-based methods below 1% is presently considered a challenge, thus a commercially available method for this class of ores has yet to be developed (Sivamohan & Forssberg, 1991; Wotruba, 2006).

3.2.2.2. Radiometric Methods

For naturally radioactive ores, the measurement of the radioactivity is possible using a basic scintillometer, and several sorters including the Ore Sorters Mk IVA and Model 17 are presently commercially available using this technology. For this type of sorting it is important to integrate an estimate of the size of the particle with the strength of the signal in order to obtain an accurate value for the ore grade, thus these sorters are often found in combination with photometric sensors in order to overcome this. Separation results are excellent and comparable to separation by gravity methods, however the application is limited as ores with natural radioactivity or associated with naturally radioactive elements, for example Witwatersrand gold ores, are not common. If ore is not naturally radioactive, radioactivity can be induced by neutron bombardment, however this approach is costly and no commercial machines based on this method are known to exist.

3.2.2.3. X-Ray and Laser Methods

X-ray sensing, where the particles to be sorted are subjected to X-ray excitation, and the subsequent decay radiation is then measured has the potential for much finer discrimination. Furthermore X-ray has the advantage of penetrating the particle, thus discrimination on a whole-particle basis is thus possible. Excitation can be by radioactive source or, more recently by laser methods, and the overall reading is a function of the permissivity (μ) and the diameter d of the particle, thus correlation of the reading with a visual estimation of the particle size as in radiometric or conductivity methods is not necessary. X-ray methods are widely used in the diamond industry, as diamonds have extremely low permissivity, while transmission methods are common in the measurement of calorific value in coal due to the low permissivity of coal compared to ash fractions (Sivamohan & Forrssberg, 1991).

Like X-rays, lasers can also be used to produce breakdown radiation, such as fluorescence or near visible spectra, in ores. Laser breakdown spectroscopy can be used for grade control on individual ore particles. However, the resolution of the laser is often too fine, ($<1mm^2$) and thus cannot accurately determine bulk properties of an ore. To overcome this, laser induced fluorescence can be used, in the evaluation of bulk ore properties, but this capability comes at the expense of accuracy (Kruuka & Briocher, 2002). Both methods do not give an absolute value and the sensor reading must be calibrated by back-assay for every ore to be measured.

3.2.2.4. Optical Sorting

Possibly the most common sorting technique in industry is optical sorting. Sensing is generally either by photocell or more recently by digital line-scan camera, and sorting decisions are typically based on differences in colour, reflectance, or transparency between the particles. Sensing frequencies are generally in the visible range, and in the manufacturing and recycling application, where particle sizes are typically larger than the present lower detection limit of 1mm for the technology, incident light is sufficient for most sorting applications. Optical sorters such as the Ore Sorters Model 16, Gunson's Sortex Model 612M and 811 operate in such principles and are common in these industries as well as the industrial minerals industry (Schapper 1977), and some application has been investigated in the metals mining sector. However, the method presents challenges as optical methods have no penetration and measure superficial ore features only; also, minerals are typically optically complex and variegated, and more sophisticated illumination sources as well as techniques are often required. Amplification of the light source is the first choice, as well as increasing the number of photo-receptors/ cameras. Alternative light sources and sensors such as UV, fluorescent, or near infra-red excitation and sensing may be used when discrepancies in the ore using the visible spectrum are not apparent. Infra red methods are promising and can be enhanced by differential heating of the ore for example via microwave. Fluorescence is promising, however, according to Wotruba (2006), only a small percentage of minerals are amenable to fluorescent detection and the use of high resolution colour cameras for this type of detection is only now in development. More sophisticated are shape and texture recognition methods, however there are currently no sensors available using this method, and while developments are ongoing (Sivamohan & Forssberg 1991), no commercial sorters are currently available for this. In order to evaluate ore potential, colorimetric analysis of the ore is required. In the course of the research, an optical ore evaluation system has been developed for this at UBC, the details of which are presented in Chapter 4. Mineralization in the samples is discriminated by means of Red Green Blue (RGB) / YES image processing and 'blob' analysis of the textures using neural network software. The system delivers useful data on correlations between the colorimetric characteristics and grade of the ore. No sensors are currently based on this principle, but applications for the method are foreseen.

3.2.2.5. Conductivity Sorting

Conductivity sensors are most commonly simple metal detectors - conductive copper coils excited by an AC current typically of a frequency between 2-200 kHz. Power and sensitivity are in inverse proportion to each other, and sensing power is also in inverse square proportion to the distance from the coil. Conductivity methods are applicable in the case where there is significant difference in the electrical properties for the valuable and non-valuable fractions in the ore (Table 3.4). Results are improved for discrepancies > $1x10^2$. Conductivity methods are thus generally applicable for discriminating native metal ores such as gold and copper, as well as massive sulphide ores grading between 2-3% metal with good results.

Table 3.4 – Conductive Properties of Minerals (from Ford, 1986)

Mineral	**Resistivity**
Silica	$3.8 \times 10^{10} – 1.2 \times 10^{12}$
Amphibole	10^{7}
Alumina	10^{7}
Wolframite	$10^{2} – 10^{5}$
Hematite	4×10^{4}
Sphalerite	10^{5}
Pyrite	10^{-3}
Chalcopyrite	1.2×10^{-3}
Pentlandite	10^{-4}
Galena	10^{-3}

Testwork reports on copper porphyry ores from a range of mines in the Montana and Upper Michigan area indicate conductivity sorting delivered up to 50% waste rejection by mass from the ores at recoveries from 85 – 92% (Miller et al, 1978). As in radiometric methods, conductivity is highly particle size dependent, thus a large low grade particle gives a similar reading to a small high grade particle. Due to a combination of these issues, conductivity sensors which can accurately and efficiently quantify the metal content of low grade base metal sulphide ores are currently not commercially available (Wotruba, 2006; Sivamohan & Forssberg, 1991). It is suggested that conductivity methods should be utilized in conjunction with optical sensors in order to compensate for some of the drawbacks of the method, with improved results. In previous ore sorting tests at INCO, tests of optical sorting or conductivity sorting alone on combined ROM ore from McCreedy East did not give acceptable results (Schindler, 2001). Sorting tests on the ore using a combination of optical and conductivity sensing in a Mogenson sorter, gave results of up to 77% waste rejection at a recovery of 98% confirming the potential of combined sorting.

A variation on the metal-detector type sensor which is indicated for use with lower grade base metal sulphide ores is the induction balance coil (Sivamohan & Forrsberg, 1991). This arrangement gives good results for native ores and is considered to have high potential for conductive sulphides such as such as sphalerite, galena and covellite in VMS orebodies where there is a significant conductivity differential between the ore and gangue minerals. In this method, an inductive coil is excited by a high frequency AC signal, generating an electromagnetic field around the coil. The coil has a natural frequency f due to the unique inductance and capacitance arising from its construction. Inserting a conductive lump of ore into the field changes the inductance of the system and the change in natural frequency is measured.

The method is appropriate for native and sulphide ores between 1-3%, however, is inaccurate for low grade disseminated sulphides < 1% and any particle < 1mm. The accuracy of the method is improved by introducing a second balancing coil, and measuring the difference in signal between the disturbed (sensing) and undisturbed (balancing) coil. A conductivity sensor based on this principal has been developed for the characterization of relevant ore properties at UBC, the details of which are presented in Chapter 4. No sensors are presently commercially available and it is believed there is significant future application for the technology.

3.2.3 Pre-concentration by Dense Media Separation

Dense media separation is the principal process technology used in surface pre-concentration plants. It is effective in removing coarse waste from a high-grade ore stream at low operating costs and high metallurgical recoveries. DMS can be used to produce a final product, as in coal washing, chromite and iron ore applications, or to prepare feed for downstream flotation or leaching such as on lead-zinc ores at Mt Isa Mines in Australia or for UG2 platinum ores at Kroondal Platinum Mines. DMS has also been used historically to pre-concentrate gold ores of the Witwatersrand prior to cyanidation (Adamson, 1972). DMS plants typically comprise a feed preparation section, the dense media vessels themselves (either HM drums or cyclones), together with related product and discard areas. DMS units are compact, low cost, and high capacity. A typical DMS cyclone arrangement for processing 100tph is shown in Figure 3.9. Note that the overall dimension of the processing module is 10m x 5m x 13m, which is of a size considered appropriate to be housed in an excavation in good rock, thus making DMS attractive in an underground pre-concentration application.

However, there are a number of challenges to integrating dense media separation into the underground environment. These include accommodating the required height of the plant, introducing and managing dense media in the underground environment and maintenance of the plant. DMS plants are, however, high capacity, and flexible in terms of feed tonnage and conditions, although variations in process efficiency across the vessel can be experienced when the feed tonnage varies.

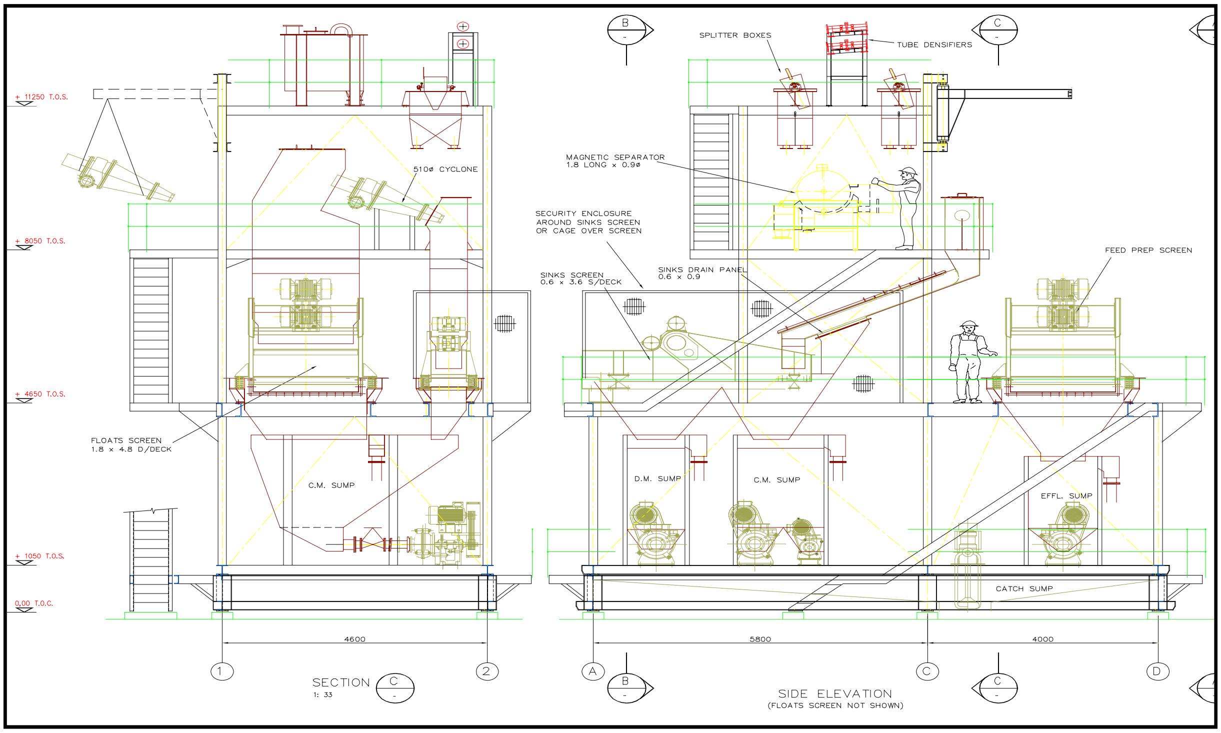

Figure 3.9 – 100tph DMS Cyclone Module (Drawing courtesy Bateman Projects, South Africa)

A focus of the design effort would be in simplifying the media preparation and recovery circuit, both to reduce the size of the plant, but also to minimize media losses to the underground environment, as this would make DMS cost-uncompetitive when evaluated against other technologies. However, several DMS operations, such as Sullivan Mine in BC, have used coarse galena recovered from the flotation section as heavy media (McCullough et al, 1999), which can substantially reduce media costs in the process. It is thus believed that these challenges can be overcome, and benefits in terms of process efficiency and improved mineral recovery would be enjoyed when compared to sorting or gravity concentration in the underground environment. As previously mentioned, there is good potential for this technology underground as evidenced by the many examples of surface pre-concentration plants as well as AMT's proposal for the installation of DMS underground on a copper porphyry orebody (AMT Annual Report, 1997).

Pre-concentration by DMS is presently employed widely in Bushveld precious metal operations such as Kroondal and Marikana Platinum, where due to the narrow (<800mm) and partitioned nature of the UG2 chrome- and platinum bearing seams, up to 30% waste can be included in the mining cut, which decreases the average ROM grade to below the economic cutoff grade in many cases. Pre-concentration typically occurs in three stages: coarse barren pyroxenite waste is removed by scalping underground prior to hoisting. Further oversize is removed on surface by scalping. Crushed UG2 feed reports to the pre-concentration plant where -1mm fines are treated by flash flotation and +1mm -38mm is treated via DMS[3]. The grade of the ore is increased to above the economic cutoff, and tonnage to the grinding and flotation stages is substantially reduced, leading to lower surface plant capital and improved PGM recoveries overall. Several other operations using pre-concentration have identified using DMS including Impala Platinum's Morula and some Lonmin operations such as Karee.

The most recent cited example of pre-concentration by DMS would be the plant built for Lion-Ore at Tati Nickel in Botswana: Phase I is a 1.7Mtpa DMS plant treating low grade base metal sulphide ore, and a Phase II study just completed is for the installation of a multi-stream, 12Mtpa DMS plant at a cost of ~US$70m (Lion Ore, 2006). Potential also exists for the pre-concentration of other ores such as copper porphyrys; although no known porphyry pre-concentration plants are in operation, testwork in this regard is ongoing. Bench and pilot scale metallurgical work has been done previously for AMT Copper in Arizona at Mountain States R&D, demonstrating that, in the case of relatively coarse copper porphyry mineralization, pre-concentration by DMS was effective in improving the grade of the copper ore at an acceptable recovery (AMT, 1990).

Separation is typically by static baths, such as the Wemco Drum, and although high capacity DMS cyclones are a possibility, drums are unlikely to be replaced entirely by cyclones in the near future. However, there are alternatives to DMS by drum or cyclone that have been developed that must be considered. These include the development of a new centrifugal separator by British Coal, the LARCODEMS , which has expanded the size range treated in dynamic DMS from 38mm (cyclone) too 100mm. A further alternative to the cyclone is the Tri-flo Separator which allows separation to occur at two densities allowing for either a scavenging or cleaning effect without the need for a second washing plant. However, the Tri-flo, like the Dynawhirlpool on which it is based, is prone to short circuiting of sinks to floats and further development is required for applications on high value ores. Feed size to the Dynawhirlpool/Tri-flo is similar to that of cyclone, thus careful evaluation of the performance of each option in the application is recommended.

Typical DMS process designs are compact, high-throughput processes operating on tonnages between 300 – 1000 tph. The application of DMS as a pre-concentration technology is not

[3] http://www.aquariusplatinum.com/aquarius_db/pdfs/Kroondal.pdf

considered research in itself. However, the technology has huge potential in the underground application, and several studies have been conducted in order to quantify and evaluate this potential (Bamber, 2004; 2006; 2007). The metallurgical and geotechnical results from these studies are excellent and indicate a wide application for the technology; however, the results do not preclude the use of any of the available DMS technologies, and the most appropriate separator must be chosen for the application in each case.

3.2.4 Coarse-particle flotation techniques for the pre-concentration of base-metal ores

3.2.4.1. Development of the Concept

Flotation is considered to have great potential for the pre-concentration of base as well as precious metal ores on surface as well as underground. Conventional flotation is typically undertaken at particle sizes between 10 and 200 μm in order to optimize recoveries, although conventional flotation has been utilized at particle sizes up to 600μm (Kallioinen & Niiti, 1997). Achieving these feed particle sizes can consume a substantial amount of energy as the particle size distribution of the ROM ore depends principally on the geotechnical characteristics of the rock, the blast design and the mining method employed (Laing, 2002), and can range from >1m for open pit or caving type operations, slightly finer in underground open stoping methods to approximately 300mm for mechanized underground methods. The particle size distribution is reduced in order to improve the liberation of the ore, creating sufficient exposed mineral surfaces to enable efficient flotation or leaching. As has been previously discussed, the dilution in the ROM ore is typically siliceous gangue, harder than the valuable metal bearing sulphides, and thus grinding of the ore to these fine particle sizes requires a significant amount of energy, up to 50% of the total milling energy required (Pitt & Wadsworth, 1989).

Fine particle flotation has been used in some Witwatersrand gold operations to pre-concentrate low grade ores prior to leaching, and in this light can be considered a pre-concentration technology in its own right (Adamson, 1972). However, conventional flotation of sulphide ores usually entails the grinding of the ore to extremely fine particle sizes, typically -149μm for primary flotation, -75μm for secondary floatation, and more recently even as fine as -30μm in tertiary regrind circuits, and is thus complex and costly, and most often is employed to produce a final concentrate and is thus not considered as a suitable approach for ore pre-concentration for the purposes of this thesis. The flotation of sulphide ores at a topsize of between 1 and 3mm is suggested as a method of pre-concentration, which, if commercially viable would be a massive improvement in terms of size and cost over the conventional approach, specifically in the underground application. Traditionally the term coarse particle flotation has been used to indicate approaches for improved recovery of coarser particles of between 300 – 500µm in conventional flotation. New developments in coarse-particle flotation have demonstrated that good recoveries can be obtained at particle sizes up to 3mm in the flotation of apatite (Hui & Achmed, 1998; Leppinen et al, 2003). Such coarse separation sizes are also common in flotation practice in the potash industry. Separation in Froth (SIF) is considered effective with coarse particles but is not an efficient process; conventional and flash flotation is efficient, but mass pull to concentrates is limited. However the literature indicates that even coarser particles can be separated by means of froth flotation - in the flotation of minerals such as potash, barites and phosphates feed topsize can be as high as 3-5mm, although unconventional low-capacity techniques such as Separation in Froth and Froth Flotation, where the feed is passed by gravity through a stable, previously established froth layer, are typically used in these instances. Furthermore, the grade and degree of liberation in these ores is typically high and the relative density is low, which increases the potential of successful separation by floatation at these particle sizes. However, the literature does indicate that coarse particle flotation of gold ores may be possible. The pre-concentration of Witwatersrand gold ores by means of coarse particle flotation followed by gravity concentration of the flotation tailings was researched by the South African Chamber of Mines Research Organization (COMRO) (Lloyd, 1978, 1979; Lloyd et al, 1986). Flotation of the ore at a topsize of 3mm, followed by gravity concentration of the flotation tails resulted in the production of a 40% by mass concentrate at an overall Au recovery of 98% (Lloyd, 1979). The testwork was undertaken in support of strategic research into underground pre-concentration and waste products were to be utilized as fine aggregate for backfill. Great potential for this technology in the underground application is thus indicated.

In designing flotation cells for the underground application, the liberation characteristics of the ore are critical in developing the machine, thus established mineralogical techniques must be used to determine the maximum particle size distribution at which flotation can occur for the ore thus determining the size requirements for preparation of the ore as feed by means of grinding (Morizot et al, 1991; McIvor & Finch, 1991). For the ores tested in the course of the research, good liberation of the sulphides at coarse particle sizes, and an α-log normal grade distribution has been observed, which appears to maximize the chances of success of pre-concentration. It is posited that the presence of these characteristics would also apply to improve the potential of a separation method based on surface chemistry as well, such as flotation, as opposed to physical means such as density or colour discrimination, in order to efficiently separate the gangue from the sulphides at coarse particle sizes. In addition to establishing good liberation and flotation conditions, appropriate hydrodynamic conditions must be established for the flotation of base metal ores at the particle sizes under consideration. A flotation arrangement comprising preparation of a coarse (-3mm) slurry feed, by HPGR or SAG milling, introduced high in the cell, with high specific energy and air inputs and a high degree of froth removal is envisaged as a starting point. Such an arrangement would be compact and high capacity and integrate well into the underground environment. It is suggested that if a more sophisticated flotation arrangement is required that the process should then be undertaken conventionally on surface.

3.2.4.2. Theoretical Basis

A significant increase in the feed particle size to the flotation cell would require a high energy density for the suspension of the slurry in the cell. As ore grades decrease and the mineralogy of the ore moves from massive through less massive to disseminated, grind sizes have decreased, tonnages have increased and the size of grinding and flotation plants have generally increased. Modern tank flotation cells have become continually larger; the maximum presently known cell sizes, as installed at the new Potgietersrust Platinum Concentrator in the Bushveld of South Africa, are of the order of 160m^3. Cells of 200m^3 and larger are also being considered for other future operations[4]. However, grinds in flotation are now typically finer, and the size of the drive has not increased in proportion with tank sizes, and these large tank cells, with energy densities of 1.2-1.6kW/m^3 are typically lower in power intensity than the preceding Wemco- or Denver type flotation cell designs with energy densities of approximately 3.6 kW/m^3. In order to overcome this limitation, flash flotation devices such as the Skimair, with smaller tank sizes, oversized drive units (thus increased energy density) and improved froth launder designs have recently been introduced to improve performance particularly in the coarse-particle flotation application (Table 3.5). However, the energy density of flash flotation cells is still low compared to the potential maximum, and the coarsest known application of flash flotation is for feed sizes of -1mm (Bushveld Platinum) or up to 2mm in cyclone underflow applications in gold milling circuits (Lloyd, 1981). Alternatives to flash flotation cells such as the Aeromix

[4] www.outotec.com

‘Flo-triter’ can be used in applications where the ore must be further cleaned during the flotation process[5], however, these are small machines with no rougher flotation applications.

Table 3.5 – Energy density comparison for commercially flotation cells

Machine	Size (m^3)	Installed Power kW	Power Draw kW	Installed Power kW/m^3	Power Draw kW/ m^3
SkimAir®-1200	49.00	132.00	74.58	2.69	1.52
SkimAir®-500	23.00	55.00	31.08	2.39	1.35
SkimAir®-240	8.00	22.00	12.43	2.75	1.55
SkimAir®-80	2.20	11.00	6.22	5.00	2.83
SkimAir®-40	1.30	5.50	3.11	4.23	2.39
SkimAir®-15	0.30	2.20	1.24	7.33	4.14
Wemco #6*	6.00	22.00	12.43	3.67	2.07
Wemco #30*	30.00	55.00	31.08	1.83	1.04
OK-50	50.00	110.00	62.15	2.20	1.24
OK-38	38.00	90.00	50.85	2.37	1.34
OK-16	16.00	45.00	25.43	2.81	1.59
OK-8	8.00	37.00	20.91	4.63	2.61
Denver D12	0.03	0.20	0.11	6.67	3.77

From: http://www.outotec.com/20846.epibrw
*Dworzanowski et al, 2007

Flotation of base metal ores with particle sizes of greater than 2mm presently appears to be out of the specification of present flotation equipment. The concept can, however, be tested at the laboratory scale, as laboratory and pilot cells are extremely energy intensive, with much higher installed power/unit volume than conventional field units, and higher energy intensity than even flash flotation units. However, there is a limit to the degree of agitation that can be successfully applied in flotation. For the successful recovery of the sulphide particles to the froth launder, a fine balance between the dynamic forces in the cell must be maintained, thus a good understanding and close control of the energy intensity and agitation speed in the cell is required. The hydrodynamics in the cell must be sufficiently vigorous to suspend the largest mineral particle, yet conditions must conversely be still sufficiently quiescent that the hydrodynamic forces do not overcome the electrostatic forces attaching the sulphide minerals to the air bubbles in the froth (Figure 3.10).

[5] www.aeromix.com/flo-triter.htm

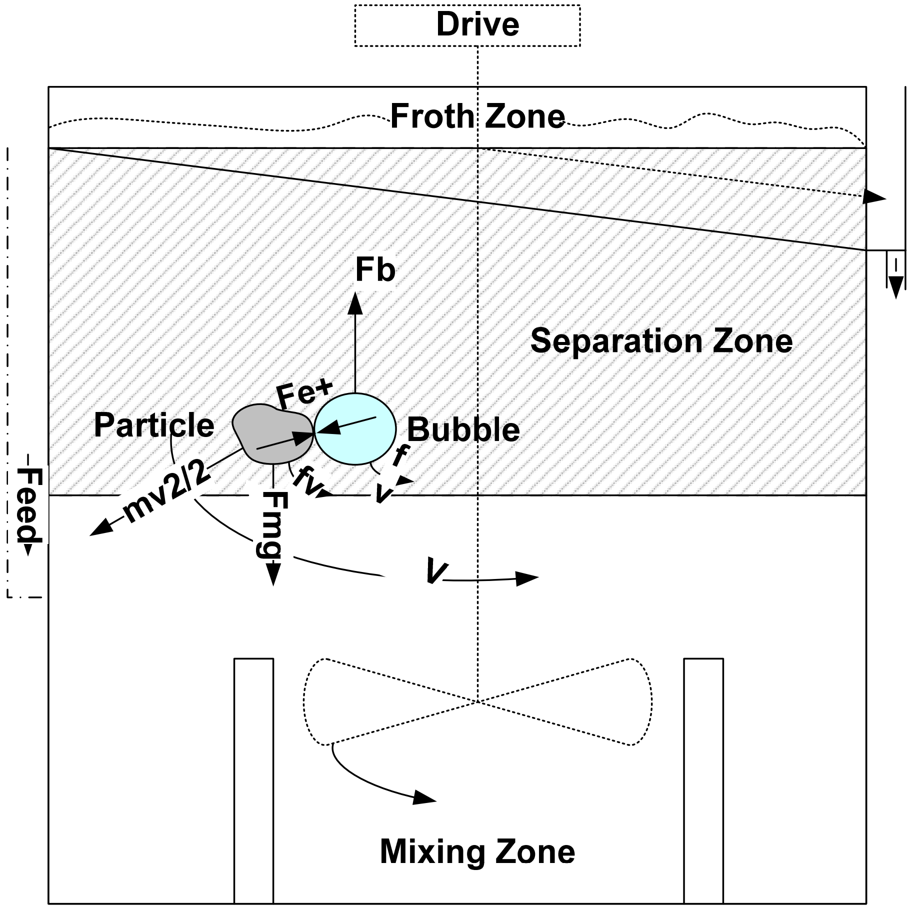

Figure 3.10 – Hydrodynamic Zones in a Flotation Tank Cell

For this, the effective agitation intensity must be evaluated, and the concept of Critical Speed is suggested (Equation 1).

$$N_{jsg} = K_S T^{-0.85} d_p^{0.20} B^{0.13} \nu^{0.10} g^{0.45} \left(\frac{\rho_S - \rho_L}{\rho_L} \right)^{0.45} + K_a Q_{GV}$$

- (1)

Where:

N_{jsg} – critical agitation speed

K_s, K_a - empirical suspension constant (dependent on tank diameter, impeller diameter and impeller height in the tank)

T – tank diameter

d – particle diameter in microns

B – solids content (%)

υ – kinematic viscosity

g – gravitational constant

ρ_s – solid SG

ρ_l – liquid SG

Q_{gv} – air volume addition in m3/min

At the critical agitation speed N_{jsg}, all slurry particles are considered to be in suspension off the bottom of the tank, particles are evenly suspended in the mixing zone, there is no settlement of particles in the separation zone, and the chance of exposure and attachment to air bubbles, and delivery to the froth zone maximized. Below the critical agitation speed, settlement occurs in the separation zone, insufficient particles are suspended in the mixing zone and thus the exposure rate of the mineral particles to the air bubbles is low. The minerals have a lower chance to become attached to the air bubbles. Above the critical speed, the solid particles in the slurry acquire velocities superfluous for suspension and thus the likelihood of remaining attached are substantially decreased. It is thus important in this situation to determine the critical agitation velocity and maintain conditions as close to the critical point as possible in order to achieve this, thus maximizing the potential to recover large, high-grade mineral particles. Using the parameters from the literature, a value for the empirical constant K_g can be determined. Through previous experimentation it was determined that for a flotation cell of diameter 0.5m, with a 0.15m Ø impeller, suspending silica at a particle size of 90μm,with air addition of 1.1m^3/min, the critical speed for solid suspension is 782 rpm (van der Westhuizen and Deglon, 2006). For the suspension of sulphide minerals with an SG of 3.1 and a topsize of 3mm, an agitation speed of 1500 – 1750 rpm is indicated from the equation. Subsequent observation of the hydrodynamics in the laboratory cell during the testwork has confirmed the functioning of this phenomenon in practice.

3.2.4.3 Scoping Testwork Results

Footwall Ores

The Footwall ores of the Sudbury igneous complex were an obvious candidate for evaluation in this context, as they are extremely high grade, with massive mineralization and good liberation. The footwall deposits consist of networks of massive sulphide veins associated with Sudbury breccia and felsic gneiss. On the basis of data acquired from both the Strathcona and Clarabelle mills, Footwall ores appear to be fast-floating ores (Kerr et al, 2003; Verdiel, 2006).

A series of trial rougher flotation tests were undertaken on a sample of the 153-4550 ore ground to 100% - 300μm and floated under standard conditions in order to determine the flotation kinetics, and grade recovery characteristics of this type of ore in a pre-concentration application

(Nunes & Malkuuz, 2005). Ore was prepared by crushing and grinding by rod mill prior to flotation. Flotation was undertaken in a 3dm^3 Denver D12 laboratory flotation cell with conditions for minimum suspension for this cell indicated at an agitation speed of 900rpm. These ores are typically treated at INCO's Clarabelle Mill in Sudbury, and conditions were selected to match flotation conditions at the mill. Flotation was undertaken under alkaline conditions at 1200 rpm and 5dm^3/min air addition, and thus conditions for full suspension of the particles were exceeded in this test. Collector and frother were potassium ethyl zanthate at 90g/t and Dowfroth 250 at 20 g/t respectively with 3 min. of collector conditioning plus 1 min. of frother conditioning. Concentrates were taken at 2, 7 and 12 minutes during flotation. Four rougher flotation tests were performed; in tests 3 and 4 a fourth concentrate sample was taken after 20 minutes of flotation time in order to evaluate the effect of additional residence time on the flotation kinetics. Flotation results indicate that on average mass pull to concentrates was 50%, with metal recoveries of 97.4% (Cu), 89.13% (Ni), 91.8% (Pt), 95% (Pd) and 82% (Au) after an average flotation time of 16 minutes. Nickel recovery was poor due to the association of the pentlandite with silica at fine particle sizes in this ore. Gold recovery is also lower than other minerals, however comparable recovery was achieved using gravity methods, indicating the possible presence of gold as native gold in the sulphides. The results indicate that rougher flotation at a feed size of 100% -300µm is possible with good metallurgical results on this ore. The results compare well to results from previous pre-concentration testwork on this type of ore using sorting as well as dense media separation methods (Bamber, 2004, Weatherwax, 2006). Furthermore recoveries are superior to the reported values for the mill (Kerr et al, 2006; Holmes et al, 2000), and it would be recommended to consider adopting a custom flotation circuit for the treatment of these ores at the mill. However, flotation of the ore even at these particle sizes still requires substantial feed preparation in terms of grinding, and the results do not appear conducive for consideration of this arrangement for the underground application, and thus it was determined to do further work at coarse particle sizes in order to investigate the minimum comminution/flotation arrangement which would give acceptable metallurgical results. As it has been determined that the 153 ore is unusual among the Sudbury ores in its extremely high grade and high degree of liberation, it was chosen to continue the work on low grade, and less well liberated ores.

Matrix/Breccia Ores

The Lake-Granite-Breccia (LGBX) ores of Xstrata's Craig Mine in Sudbury are pentlandite- and chalcopyrite rich transitional ores of the Sudbury Igneous Complex, situated down-dip between the principal pentlandite-rich Contact ores and the Footwall ores (Binney, 2007). Compared to other contact type ores, LGBX ores exhibit a high sulphide content and medium tenor, with massive- to matrix type sulphide mineralization, and were thus considered to be good candidates for coarse particle flotation trials. 12 flotation samples were prepared in order to investigate the performance of these ores during rougher flotation (Table 10). Flotation was performed at nominally 30% solids using a 0.2kW Denver D12 laboratory unit with a 6dm3 cell. In order to standardize flotation conditions as much as possible, excess reagent additions were planned, and kept constant over the course of the tests. KaX dosage was increased to 200g/t feed from the previous testwork, and frother addition was via Dowfroth 200 at an initial dosage of 300 g/t feed, with additional frother added towards the end of the test in order to maintain froth stability. Samples were crushed to nominally -8mm prior to testing. Sample feed size distribution was modified for selected samples by rod milling at 50% solids for specified intervals of 5, 10 and 15 minutes respectively. The various feed size distributions of the samples used in flotation testing are shown in Figure 10. The topsize of the samples varied between 8mm (CL1,2,10), 5.6mm (CL4,5,6,7,11,12) and 4.76mm (CL9,13) with a feed D50 of 2.4mm, 1.5 and 0.42mm respectively. A critical agitation speed of 1750 rpm was indicated by Equation 1, however this was found to be impractical in the D12 cell, and a standard of 1500 rpm was adopted. CL2 was attempted at an agitation speed of 1200 rpm, but results were poor due to complete saltation of the flotation sample on the bottom of the cell. Flotation tests were undertaken with 3 minutes of collector conditioning and 1 minute of frother conditioning prior to feeding the samples into the cell. Concentrate was collected continuously to avoid froth choking, and flotation of each sample was sustained up to 20 minutes or until the froth loading was negligible. Results ranged from the recovery of 6.26% by mass to concentrate with 36% metal recovery in sample CL4 to 12% recovery by mass with 55% and 58% metal recovery to concs for Cu and Ni respectively for sample CL11. Concentrate recovery in the coarse size fractions was excellent, with particles of up to 1mm being recovered in the CL13 test. The recovery of concentrate by size was also observed to be in inverse ratio to the feed size of the sample, thus the d80 of the concentrate increases from 0.12mm, to 0.15 and ultimately 0.18mm for feed samples with a d80 of 4, 3 and 2.5mm respectively (Figure 3.11).

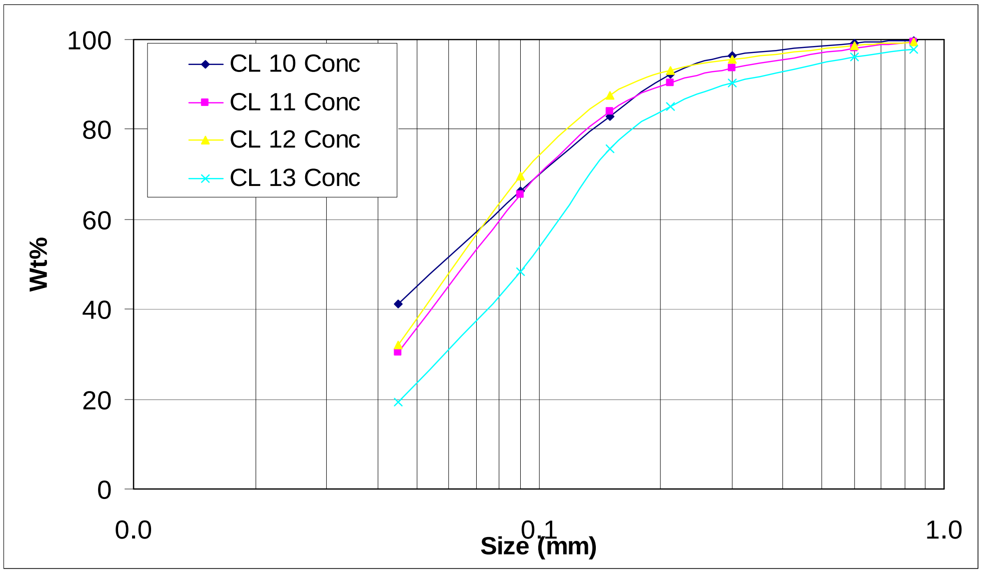

Figure 3.11 –Comparative Size Distributions – Flotation Concentrate

The results support the observation that recovery of coarse particles (between 0.6 and 1mm) to concentrate is improving in each test with finer grind. Good potential is thus indicated for coarse particle flotation of base metal ores, particularly in the pre-concentration application.

3.3. Waste Disposal Technologies

3.3.1 Background

A challenge in the consideration of this approach is the appropriate disposal of the waste products. The waste products from a number of case studies were available for this stage of the research, and it was decided to investigate several key aspects of waste disposal in order to identify a suitable technology by which to dispose of the waste. As it was desired to maximize the degree of waste rejected from the ROM ore and disposed of as fill, the use of classified tailings from the surface mill as a source of the fines component for the mix was also considered essential.

The science of backfill is largely empirical, and rigorous treatment of the topic has only recently been undertaken (Hassani et al (Eds) 1989; Potvin et al (Eds), 2005). For the purposes of this discussion the following principal types of backfill will be defined (after Grice, 1989):

- Rockfill (RF, CRF)
- Hydraulic (classified tailings) fill (CHF)
- Paste fill
- Composite fills (e.g. ‘rocky’ paste fill)

Fills as described above can be both cemented (using either cement or pozzolanic binders) or uncemented. Mines utilize such fill for several reasons:

- Localised roof support
- Long term regional geotechnical stability
- Limiting excavation exposure
- Disposal of mining waste underground

In addition to these benefits, there are a number of secondary benefits of fill that have been documented. These include a 45% reduction in post-mining closure, and a reduction in the Energy Release Rate (ERR) during rockbursts to below the critical value (Kamp, 1989). Kamp also notes increased ore extraction through a reduction in pillar size, a 45% increase in ventilation efficiency, improved fire control and improved safety and productivity leading to reduced unit working costs. The use of fill also leads to a reduction in the heat transferred to ventilation air through a reduction in the surface area available for conduction, as well as reduced air leakage past filled areas (Matthews, 1989). Cemented backfill has been successfully substituted entirely for engineered support in deep-level stopes at a competitive direct cost, and has been shown to substantially reduce rock stresses, plastic rock deformation and rock bursts when utilized at depths below 2000m (Patchett, 1977; Lloyd 1979).

Sources of backfill material are numerous. Coarse development waste as well as mining waste that can be clearly identified for resuing is utilized as uncemented rockfill, however this type of fill does not impart a significant degree of local or regional roof support. Cemented rockfill (CRF) comprises coarse fill which is consolidated by the application of a dilute sand/cement spray for the generation of additional compressive strength. Falconbridge's Strathcona Mine in Sudbury employs CRF comprising -125mm rockfill combined with 6.5% water and 6.5% Normal Portland Cement (NPC), although other fill systems have been considered in order to further alleviate rockbursts (Swan et al, 1993). Surface mill tailings is also used as a backfill material, and is typically combined with 2- 5% NPC by mass, generally to reduce slumping and improve pumpability rather than for any strength requirements; NPC can easily be replaced for this purpose by fly-ash, granulated blast furnace cement or some other pozzolanic material. This fill is typically delivered to the stope by means of gravity lines, or pumped to the stope, either from a surface facility, or less commonly from a backfill plant located underground. Mill tailings is not an ideal backfill material as the particle size distribution is too narrow, and contains too little fines (<25% < 10μm), in addition to a paucity of coarse material; fill strength development is thus severely limited and in order to generate compressive strengths > 1 MPa a substantially higher ratio of cement addition is required, which is usually uneconomical. In addition to this, the ratio of water in the fill can be as high as 55%, leading to slumping and drainage problems in the filled stopes (Blight, 1979). The size distribution of mill tailings is often modified through hydrocyclone classification to improve the fill characteristics (classified tailings fill). Water content in hydraulic tailings (classified or unclassified) is high, and drainage of excess water post fill is often a challenge.

These problems have been largely overcome through the development of high-density 'paste' backfill systems, where a superior fill is prepared with a broader particle size distribution and a decreased water content of between 10 and 25% (Brackebusch, 1994). A dense paste is produced from mill tailings and pumped or gravitated underground. Moderate quantities of coarser aggregates up to 25mm can also be added to the dense paste without significantly impairing its pumpability (Cooke et al, 1992). Mill tailings are dewatered in a conventional thickener in order to preserve the ultrafine fraction, and then filtered to approximately 13% moisture. The particle size distribution of the tailings can be modified by classification, and by the introduction of significant amounts of sand to the mix. The paste, typically comprising the tailings with 15% water, 2-4% cement and some additional aggregates, is made up on surface and delivered to the stopes by a combination of concrete pumps and gravity pipes of between 100 – 150mm diameter. The maximum realistic strength paste fills can achieve in situ is 1-3MPa regardless of the degree of cement addition; fill strength development is limited by the narrow size distribution of the tailings, and the addition of a high proportion of cement is generally uneconomic in most backfill situations.

Some movement towards the design of fill using more conventional concrete-type mixes have been made in order to maximize potential strength of the fill. Target Gold Mine in the Free State of South Africa has been experimenting with an underground concrete batching plant fed with aggregate prepared from crushed mining waste (SA Mining News, 1995). A high strength of fill is generated which is used for construction purposes underground. Typical concrete mixtures comprise a mixture of -19mm crushed aggregate, -1mm sand, cement and water in a pumpable mixture of 6:2:1:1 by mass, and can achieve compressive strengths of typically 20 – 40 MPa. Concrete strength varies with the overall particle size distribution in the specification, cement content and water:cement ratio in the mix (Talbot & Richart, 1927). Minefill is not generally required to achieve such strengths, however, and less competent mix ratios and lower cement contents should result in adequate backfill strength. More recent developments include the addition of aggregates to paste or hydraulic fills to create composite fills with higher strength. Composite fills include conventional cemented rock fill (CRF), rock fill with cemented sand fill, and more recently 'rocky' paste fills which are typically a blend of paste fill with up to 30% coarse aggregate by mass. While there is by necessity a focus on aggregate-based fills, the consideration of backfill in the context of pre-concentration requires consideration the complete range of fill materials currently in use in order to exploit the opportunity of combining coarse rejects from underground with fine tailings from surface in order to maximize the use of solid mine waste in the fill.

The use of high-strength backfills such as cemented aggregate fill or rocky paste fill can only further improve the benefits of utilizing backfill as documented above, and such benefits can outweigh the additional cement cost of these fills (Quesnel et al, 1989). This is borne out in observations by COMRO in South Africa (Adams et al, 1989): residual ore pillars were entirely substituted by 20MPa concrete at a depth of 3800m in an innovative mining system designed for use at extreme depth. Ore extraction was increased by 43% and the ERR was reduced from 50MJ/m^2 to below 40MJ/m^2. Benefits noted elsewhere include improvements in post-yield pillar stability and a 90% increase in residual pillar strength in sandstone (Yanaguki & Yamatomi, 1989). However, the cost of preparing and delivering these fills is prohibitive, especially for deep and more extensive mines, and it would be advantageous if the economics of producing such high strength fill could be improved. It is thus proposed to consider the use of the waste rejects from the proposed underground pre-concentration process to potentially increase the strength as well as decrease the cost of utilizing such fill.

3.3.2 Development of A 'Rocky' Paste Fill For Use With Underground Pre-Concentration Systems

Significant economic potential has been identified for pre-concentration underground simply arising from the rejection of large quantities of the barren fraction in the ROM ore directly underground. There is also potential for the application of pre-concentration simply on surface, as this does not preclude the disposal of the waste rejects fill underground. However, in either case it is critical to dispose of these waste products in an appropriate and environmentally sound manner. A suggested use is as an aggregate for fill, should the process deliver a suitably sized product. This seems possible as the naturally arising topsize of the waste products from the processes under consideration is expected to range between 3 and 200mm , which with further crushing and screening can be modified to produce a suitable size distribution for addition to fill as aggregate.

The addition of aggregate to hydraulic or paste fills to form a composite fill has several advantages. These include improving the strength of the fill, increasing the binder efficiency and reducing the quantity of water required in the fill mixture (McKinstry & Laukkanen, 1989; Annor et al, 2003). Optimum fill strength is indicated for the addition of 25% by mass of -10mm aggregate (Quesnel et al, 1989). A composite fill mixture with the optimal aggregate size distribution would be dilating by nature during curing (Kuganathan, 2005), which is desirable in a confined environment, and will lead to a higher fill strength in situ than indicated by the ASTM test procedure. Incorrectly graded aggregate results in a contracting fill which can be prone to failure of the fill mass even prior to loading.

Strength in composite fill mixtures derives either solely from the cement bond, interlocking of the aggregate particles or a combination of both mechanisms. Fill strength is determined principally by binder content of the fill, however, aggregate fills are expected to generate higher strength than hydraulic fills due to a higher maximum Proctor density due to the decreased coefficient of uniformity and thus higher cohesion of the mix. Aggregate fills are expected to generate higher strength than cemented hydraulic fills or paste fills due to the low coefficient of uniformity and increased natural cohesion of the mixture. Incorrectly graded aggregate results in a contracting fill which can be prone to failure of the fill mass even prior to loading. For hydraulic and paste fills:

$$\sigma_{max} = 27 \left[\frac{c}{v} \right]^{1.57}$$

and for aggregate fills:

$$\sigma_{max} = 63 \left[\frac{c}{v} \right]^{1.54} \qquad \text{(Henderson \& Lilly, 2001)}$$

Where: c = cement content in %

v = void ratio of the particle size distribution

Physical characteristics of the fill material such as UCS, mineralogy and particle geometry also have an additional influence on the final strength of the fill (Lamos & Clark, 1989). A comparison of fill mixtures comprising full spectrum plant tailings, a classified plant tailings and a mixture utilizing –9mm crushed waste are presented for comparison (Figure 3.12).

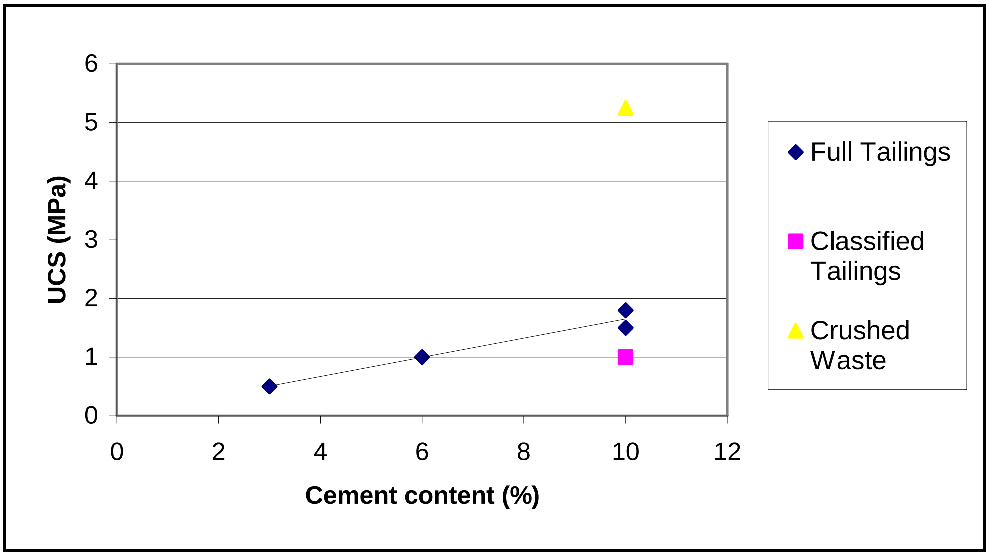

Figure 3.12 – Comparison of Backfill Strength by Aggregate Type

Note that full tailings in this case exhibited a higher UCS than classified tailings, and that comminuted waste generates a higher strength than either tailings product. A high-strength backfill mix design has also been pioneered at the Cannon Mine in the U.S. (Brechtel et al, 1989). Several aggregate mixes were designed and tested using a coarse (-38mm) and fine (-9mm) aggregate fraction. Fill strength of 9MPa at 6% cement addition was achieved for the coarse aggregate fill in the tests. Also, the confined strength of the resultant composite fill was found to be 25MPa at 27% porosity in situ.

Several potential mix designs for cemented backfill using rocky aggregate are thus suggested in the literature. Cemented hydraulic fill would typically be 100% -100um with < 10% -10μm fines combined with 3% OPC and 3% fly ash or similar (Grice, 1989). Various cement ratios for fill are suggested by Udd (1989) in order to optimize cement usage for different applications:

- 2.5% - 3.5% for cut-and-fill stopes
- 2.5% - 6.25% for bulk pours in blasthole stopes
- 12.5% for sill mats and containment structures

The use of aggregate in the fill mix is expected to reduce these binder requirements significantly. However, sourcing aggregates for mine fill is not usually economic as this requires special preparation of the non-valuable development waste, or in extreme cases, quarrying of a suitable aggregate on surface. A cost effective source of such aggregates would be through pre-concentration, and Mt Isa Mines in Australia have been a pioneer in the development and use of composite fills using DMS rejects. Pre-concentration by DMS is employed on surface, producing a coarse, rocky waste, which has been used to prepare fill for the underground blasthole stopes . A number of lessons can be drawn from their experience. Typical fill compositions in use at Mt Isa, and the typical fill strengths, are tabulated in 3.6 below (Kuganathan & Shepherd, 2001):

Table 3.6 – Mt Isa Backfill Mixes

Component	Mass in Mix (kg)		
	CHF	CRF	RF + CHF (3:1)
Aggregate	0	2116	1725
Tailings	1365	0	471
Cement	45	46	34.5
Pozzolan	90	138	69
Water	135	184	103.5
UCS (MPa)	~1	5.84	1.77

As can be seen, potential aggregate content varies widely, thus there is a definite potential to incorporate the waste rejects from all the processes under consideration in the study into an appropriately designed backfill for improved fill performance. Mt Isa has been innovative in the use of these waste rejects for the production of composite fills for difficult applications underground, such as steeply dipping blasthole stopes, as well as the replacement of sill mats and pillars. The open-stoping method at Mt Isa requires a high-performance fill with free-standing ability for rapid re-entry and high strength for good regional support. Fill designs based on these parameters are recommended for consideration in the context of the recommended approach in this thesis.

3.3.3 Fill Preparation and Delivery Systems

The fill preparation system at Mt Isa is a good model for an appropriate system in terms of the research, (McKinstry & Laukannen 1989) . Approximately 2Mtpa –70mm aggregate is sourced annually from the reject stockpile of the DMS plant and combined with cemented hydraulic fill in a typical ratio of 1:4. The composite fill is made up on surface and gravitated 220m underground via 300mmØ fill raises. A schematic of the 'rocky' pastefill preparation and delivery system at Mt.Isa is shown in Figure 3.13.

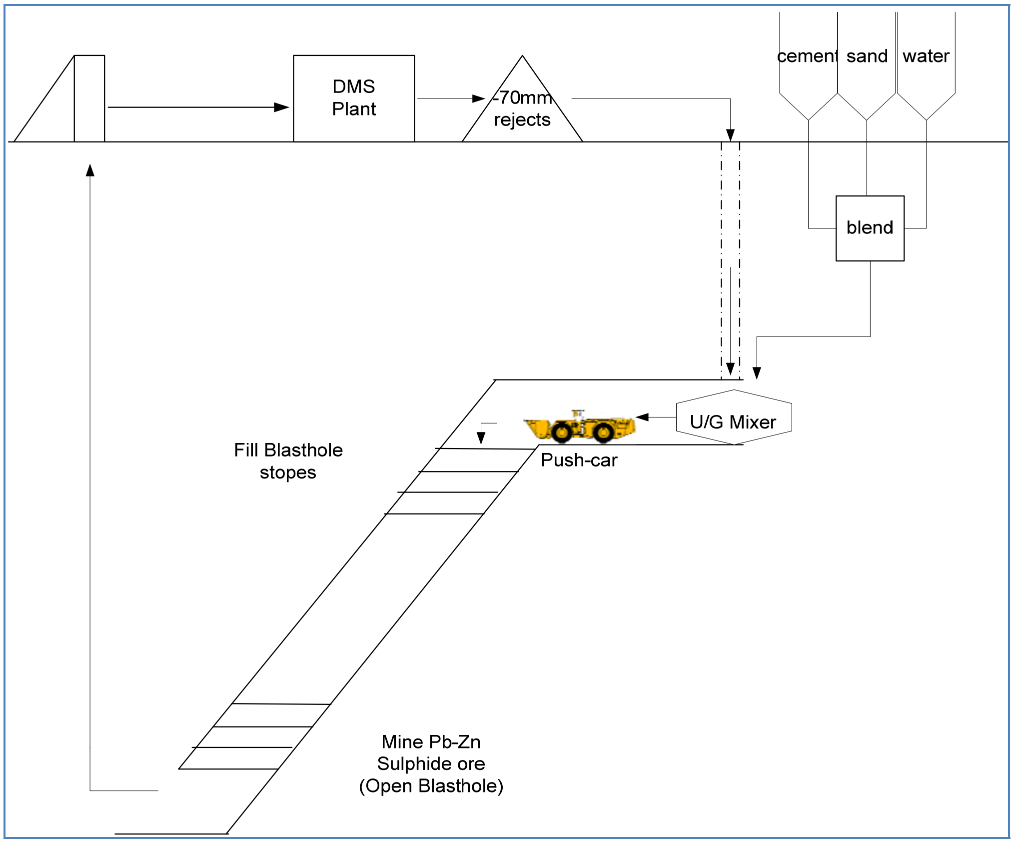

Figure 3.13 – Rocky Paste Fill Backfill System at Mount Isa Mines

Two methods of creating the composite 'rocky' paste fill have been researched in the course of the work. The first method requires the preparation of a conventional cemented paste fill, and the subsequent mixing of the rejects with the paste underground prior to placement. The second approach requires the simultaneous preparation of fill using all fill components, which is preferred for maximum dissemination of the binder onto the aggregates in the fill. The design of a conventional paste fill preparation system is presented in Reschke (2000). Metallurgical tailings are classified and the undersize fraction is delivered to the tailings dam. Suitably classified tailings are passed to the high rate, high density mechanical thickener through the addition of suitable flocculants. The thickened paste is passed to a high shear mixer where the paste is combined with typically 5% cement slurry. Cemented pastefill is pumped to the pug mixer where the aggregate is added and mixed. Fill can be pumped to the stopes or hauled and placed by means of push cars if not suitably viscous. However, this means of preparing the composite fill is considered ineffective as it is not possible to guarantee adequate coverage of the aggregate surface area with cemented paste, and more sophisticated batching and mixing methods are recommended in order to achieve a consistent, high strength composite fill. A suggested approach is to adopt batching and mixing technologies from the construction industry, where a range of concrete mixes can be efficiently batched and prepared using generic batching plant technology (Figure 3.14).

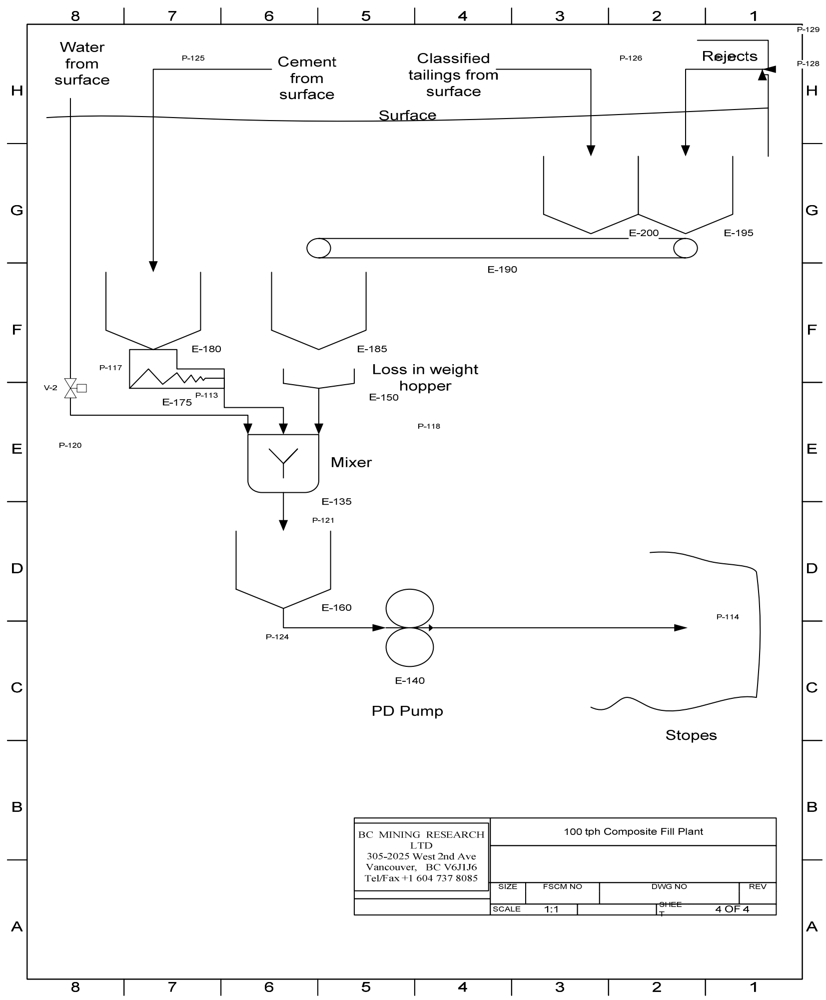

Figure 3.14 –Composite Fill Preparation System

In this method, the fill is prepared in the same manner as a ready mix concrete with positive impacts on fill quality and fill quality control. An infinite variation of mixes can be provided by such a plant depending on the application. Batching plants are compact and high capacity and would be easily accommodated in suitable excavations in the underground scenario.

Backfill is typically made up on surface and delivered to the stopes via a suitable delivery and emplacement system, although several mines have been experimenting with underground batching plants (SA Mining News, 1995; Kuganathan & Shepherd, 2001). Emplacement systems for backfill include gravity-based systems, pumped systems, conveying or batch haulage by LHD. Coarse rockfill systems may even require physical handling of the fill. Backfill emplacement can be direct to stopes via dedicated pipe ranges, or placed in intermediate storage dams underground prior to final placement (Kamp, 1989). Placement can be unconstrained if stope geometry permits, via constructed paddocks or in pre-filled geotextile bags, although the latter method is unwieldy and ineffective in delivering good post-fill support.

Pumped systems are common on hydraulic, paste- and sandfill systems. Pumped fill systems are less common on coarse backfill applications, although positive displacement pumps are used extensively in the concrete industry, and show good potential for use in the underground fill batching scenario. A pumpable composite fill mixture is preferred as this fill has superior material handling properties over rockfill. Composite fill thus batched can be pumped to the fill stopes using heavy-duty concrete pumps. A schematic of a typical positive displacement concrete pump is shown in Figure 3.15. Such pumps are capable of delivering -38mm aggregate concrete mix at 52 m^3/hr (100 tph @ 50% solids) over 400m horizontal, and 240m vertical distance.

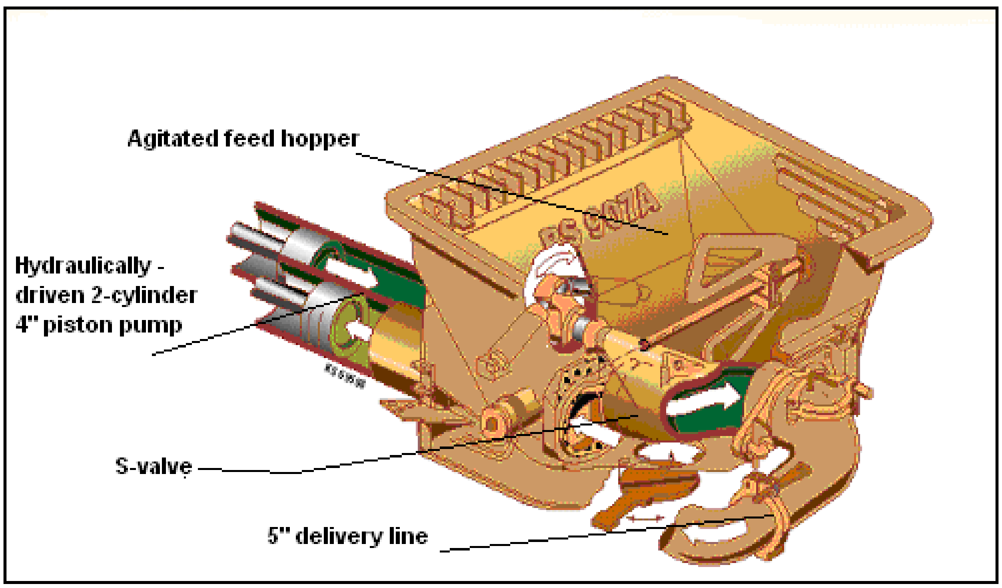

Figure 3.15 – Thomas Katts BS 907A Concrete pump schematic (Bamber, 2004)

There are thus several potential backfill strategies arising from the integration of a pre-concentration facility into the underground mining environment which are attractive.

3.4. Interfacing the Technologies with the Mining Activity

The integration of these pre-concentration and waste disposal technologies with the mining activity is expected to impact significantly on the economics of mining and processing and require some changes to the design, layout and planning of production stopes, backfill arrangements and other supporting infrastructure. The adoption of the integrated mining and processing approach is thus expected to impact significantly on the future design and planning of mines. The major interface with the mining method concerns the physical accommodation of the equipment required to perform the pre-concentration and waste disposal step, as well as the means of accommodating the waste material generated in pre-concentration in the stopes. There are an almost infinite number of variations of mining method presently employed in the hard rock mining industry. For the purposes of the thesis, methods will be split into 2 groups: mining methods which depend on backfill, non-backfilling methods which can accommodate fill. Methods which cannot accommodate fill at all such as sublevel caving are not considered. The integration of the envisaged processing and waste disposal steps with the mining activity will thus be discussed for the following methods:

- Resue Methods
- Mining methods which depend on fill
    - Overhand Cut & Fill (incl. drift & fill, cut & fill, post-pillar cut & fill)
    - Underhand Cut & Fill
- Mining methods which can accommodate fill
    - Open Pit methods (incl. open cast and strip mining)
    - Block Caving
    - Open Stoping methods (incl. shrinkage stoping / blasthole and longhole)
    - Room & Pillar

The integration of automated pre-concentration and waste disposal systems is thus considered for both surface and underground methods. For deep deposits, integration of the pre-concentration and waste disposal step underground is recommended, necessitating additional stable excavations for ore storage, the pre-concentration plant as well as the storage for waste rejects and the fill preparation and delivery system. Pre-concentration and waste disposal plant must thus be compact, efficient and high-capacity. Pre-concentration should be undertaken as early as possible in the mining cycle and at as coarse a particle size as possible (Klein et al, 2002). The process design, equipment selection and block layout for two of the potential pre-concentration processes has been previously undertaken for an 1800 tpd underground facility. This plant design has been used to develop an estimate of the excavation size required for the individual process areas making up the system (Figure 3.16). The plant design requires a series of excavations, linked by standard 5m x 5m haul drifts, each of which require designing for the specific setting of depth, stress and rock conditions. Surge requirements require additional excavations which would be typically found underground such as ore passes and access drifts. The maximum expected excavation size based on these layouts is 12m x 7m x 15m for the process module (Bamber, 2004).

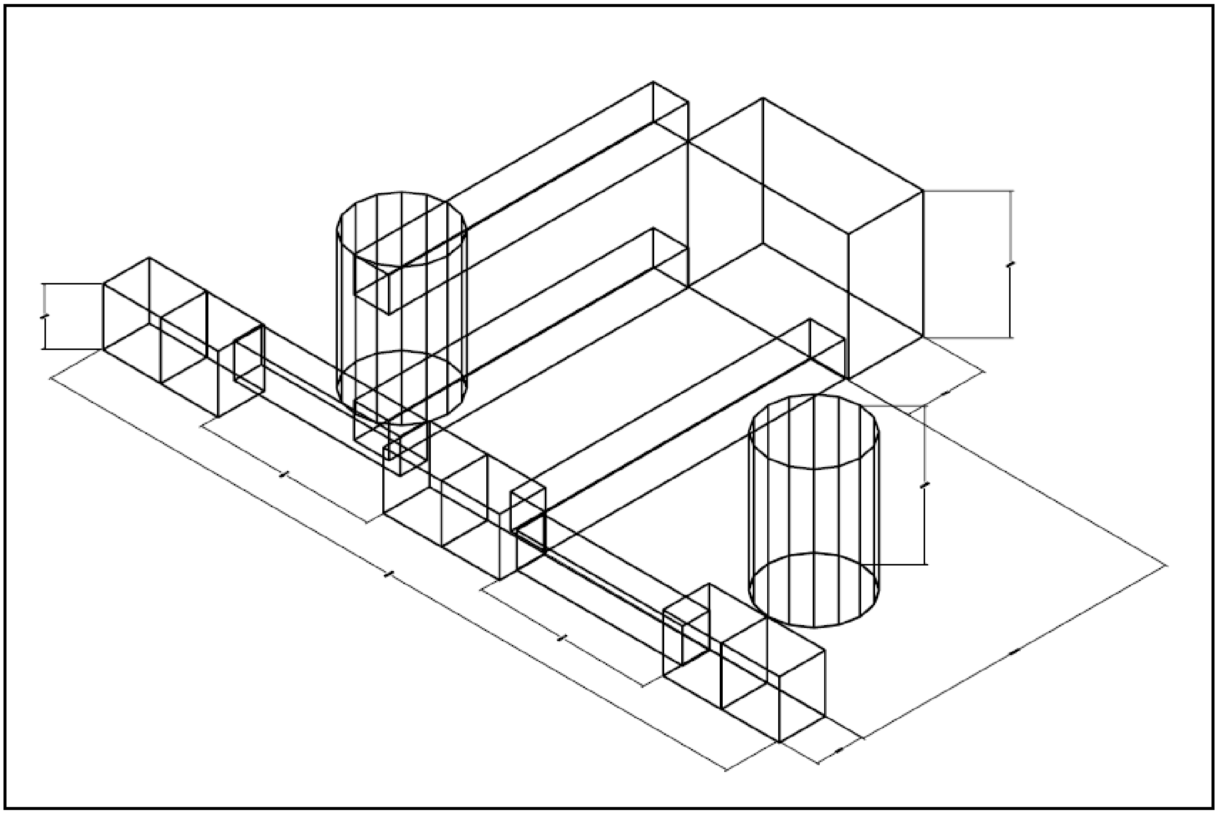

Figure 3.16 – 3D Layout of Excavations for DMS- and Sorting Based Pre-concentration System

The envisaged size and arrangement of excavations for the underground application is not considered impractical in the context of existing large and deep excavations. Hoek and Brown (1994) present a bibliography of large excavations which can be used for comparison (Table 3.7). From the table it can be seen that excavations of the order of that indicated are both possible and practical at depth.

Significant challenges in the consideration of the underground processing scenario lie in the areas of health, safety and environment, as well as in the sourcing of skilled operators for such a facility. Safety and environmental challenges for the underground installation include additional heat, noise and dust generated during ore and waste processing. While the processes considered are generally inert, or reagent free, significant additional services such as water, power and ventilation will be required to maintain process integrity and safety.

Table 3.7 – Data for Existing Large and Deep Excavations (from Hoek & Brown, 1994)

Location	**Date**	**Excavation size (m)**	**Depth (m)**	**Rock Type**
Aura Power Station, Norway	1953	18 x 17 x 123 + 17 x 15 x 95	250	Moderate Gneiss
Tarbela Dam, Pakistan	1972	19m Ø tunnels	270	Weak Gabbro
Ralu I Power Station, PNG	1975	15 x 24 x 51	200	Marble / diorite
Vaal Reefs sub shaft hoist, RSA	1966	10 x 12 x 13.5	1700	Strong Quartzite
Western Deep Levels sub shaft hoist, RSA	1974	16.6 x 12 x 32	2750	Strong Quartzite
Stonfors Power Station, Sweden	1958	18.5 x 24 x 124	200	Strong Gneiss
Hartebeestfontein 6# Refrigeration Plant*	1986	16 x 12 x 30	2000	Strong Quartzite
McCreedy East Crusher Station*	n/a	20 x 30 x 20	1100	Strong Norite
Nevada Test Site, USA	1965	24 x 43 x 37	400	Moderate Sandstone

Technology transfer and particularly the transfer of technology and skills to the underground scenario is seen as a serious challenge to be overcome. Evidence of chronic cultural resistance to the adoption of pre-concentration in general can be noted in the literature (Wotruba, 2006; Salter & Wyatt, 1991) and resistance to the concept of underground pre-concentration in particular has been continuously observed during the fieldwork stage of the research. Resistance of mine operators to the concept of introducing mineral processes into the underground environment was noted as well as an expressed reluctance on the part of mineral processors to consider transferring their operations to the underground environment (Verdiel, 2006). It is recognized that the underground environment can be highly aggressive towards both personnel and equipment and that substantial work is needed in these areas to address this if the approach is to be successful.

For shallower deposits, the pre-concentration and waste preparation step can be either on surface or underground, however for maximum positive impact, the rejects should be disposed of underground. For open pit methods, the pre-concentration and waste preparation step are situated on surface, obviating the need for excavations, and solid waste is by necessity disposed of either on surface or in the pit. In integration with surface methods, there is no requirement for process excavations, thus there is no limit on the size or location of the pre-concentration or waste disposal facility with these methods.

3.4.1 Cut-and-Fill Mining

Both overhand and underhand cut and fill methods will be considered in this section. In cut and fill methods a small tranche of the overall orebody is mined using mechanized methods and then backfilled with either cemented or uncemented fill prior to the removal of the next tranche of ore. The method is most commonly applied in narrow, moderate to steep dipping orebodies with weak wall rock where high recovery as well as high selectivity is required. Disadvantages of the method include the typical high cost of using fill, challenges in scheduling fill and typical high overall cost and low productivity. Also, in typical wall rock conditions, the method requires the retention of ore blocks in the form of sill pillars for stability. In these cases, underhand cut and fill is suggested as a means to eliminate sill pillars, although for extensive orebodies, the requirement for a sill pillar remains even in underhand cut and fill operations.

The integration of the ore pre-concentration and waste disposal step is expected to impact significantly on the design of the cut and fill based mine. A basic approach as suggested for the McCreedy East Mine (Bamber 2004) as well as for Xstrata's Onaping Depth Mine is a centralized underground pre-concentration and waste disposal facility located between the orebody and the shaft (Figure 3.17). Ore is mined as in conventional cut and fill and can be transported to the pre-concentration facility by haul vehicle or alternately for simpler layouts, by conveyor. Pre-concentrate continues up the shaft via the hoisting system or alternate methods such as hydraulic hoisting. Waste from the pre-concentration plant is prepared together with tailings from surface and gravitated back to the fill stopes on demand.

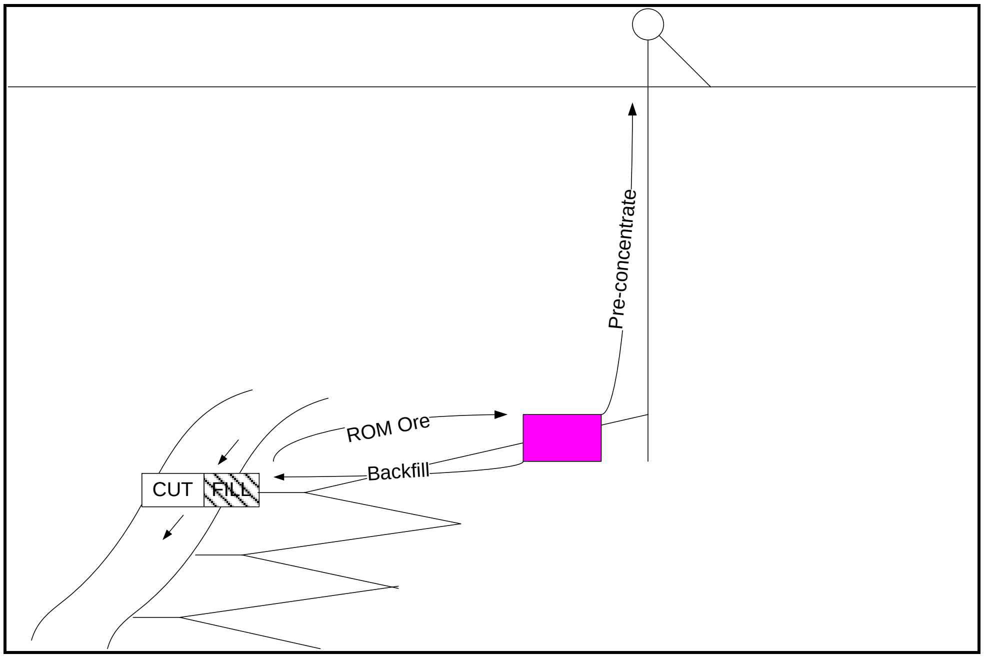

Figure 3.17 – Integration of Pre-concentration and Waste Disposal with Underhand Cut and Fill

An alternative approach would be to integrate a small pre-concentration and waste disposal facility designed for the production of one mining section only at the exit to the section; however, concerns about weak wall rock for orebodies requiring this method in the first place may preclude the creation of a stable excavation for this in the stope. Impacts on the method are significant (Bamber et al, 2005): selectivity is reduced, larger equipment can be employed, the costs of preparing and placing backfill, are reduced, thus the productivity of the method is improved and costs are reduced. Fill would typically be a composite-type fill, thus the total quantity of solid waste disposed of on surface is reduced for this method compared to the conventional approach. Composite fills are also generally higher strength than conventional cemented fills, and applications for the use of this fill to reduce or even eliminate sill-pillars are being considered.

Additional impacts are projected for underhand cut and fill operations. Pre-concentration facilitates the consideration of the use of composite fill, which is more efficient in the use of binder, more cohesive, higher strength and quicker setting than comparable conventional rock- or paste fills (Weatherwax et al, 2007). This is expected to greatly facilitate re-entry times in underhand cut-and fill operations, thus greatly improving productivity. As in overhand methods, there is potential to reduce or even eliminate sill pillars.

Challenges lie in the scheduling of what is now a continuous source of fill back into the stopes, although modeling and simulation on Extend^{TM} indicates that a cut and fill section comprising 4 to 5 stopes (3 working, 1 fill and 1 spare stope) possesses sufficient flexibility to accommodate a properly scheduled continuous fill sequence (Morin et al, 2005).

3.4.2 Open Pit Methods

The integration of pre-concentration with open pit methods is seen as a natural extension of this method of mining for increased extraction of the orebody. Several impacts and benefits with regard to open pit operations were identified in the course of the study for the Pipe II open pit in Thompson Manitoba. Firstly, pre-concentration impacts directly on operating costs, which impact directly on cutoff grade and the size of the ore reserve. Evidence from the literature, as well as from the outcome of the study indicates that in the case of applications to large, open pit operations, cutoff grade can be significantly reduced, if not eliminated for the pit (Lion-Ore 2006). This implies that, with the integration of pre-concentration as appropriate, the entire mineralized zone is now mined. A new definition of cutoff grade must be introduced as the cutoff grade of the deposit, to all intents and purposes, becomes the grade of the rejects from the pre-concentration plant. Grade control decisions in the pit are eliminated, and the stripping ratio is massively improved.

Secondly, the introduction of pre-concentration, particularly with the use of sorting methods (optical and conductivity), in addition to producing an upgraded ore stream, creates the opportunity to simultaneously analyze and classify the wastes into reactive (acid generating) or non-reactive waste prior to disposal. Both optical and conductivity methods can be used to distinguish between valuable ore components, barren rock, and mineralized waste of various grades. Thus the integration of pre-concentration technologies with open pit methods is expected to massively increase extraction of the deposit, as well as improve waste management at these operations (Figure 3.18).

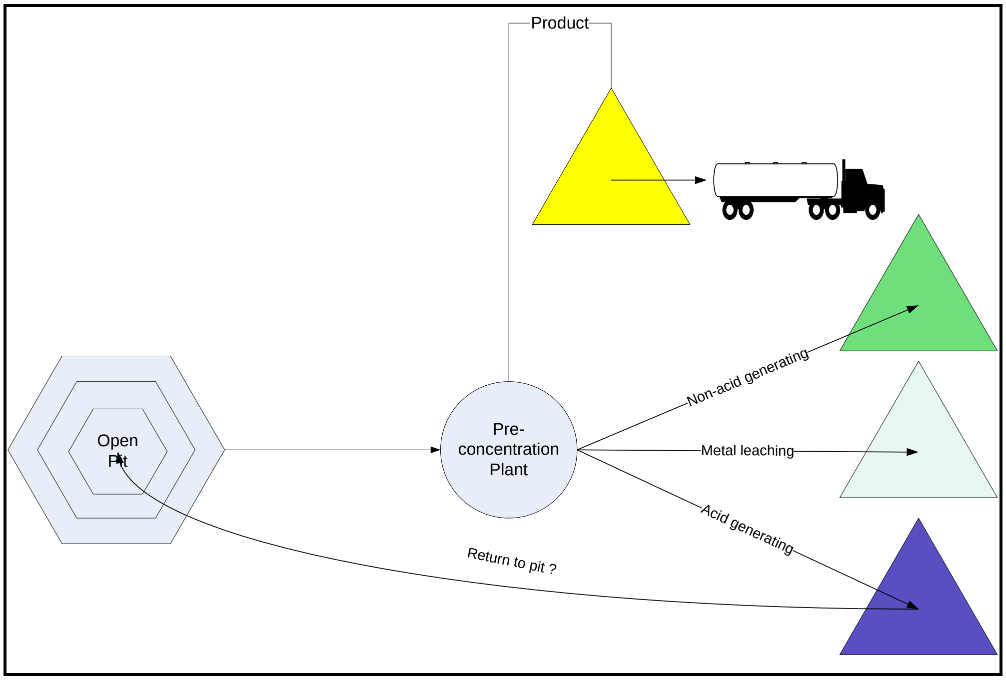

Figure 3.18 – Process Integration for Waste Management in Open Pits

3.4.3 Block Caving

Block-caving is a method gaining increased application and importance in the industry for deposits where open-pit methods are considered marginal due to increasing depth or where increased selectivity is required over the open pit method. Block caves are finding particular application in copper porphyrys, which by reason of their typically uniformly low grade must be mined at high mining rates, and according to Cross (2006) will be the principal source of primary copper by 2014. Block caving relies on the natural proclivity of mineralised rock to be fractured and weaker than the host rock, and hence its natural tendency to cave under gravity only when subjected to vertical stress. Ore blocks in block caving can be up to 200m high and extend the full width of the orebody. The block to be caved is prepared by undercutting the ore with a series of haul drifts accessing the orebody via a large number of regularly spaced drawpoints in a grid (Figure 3.19). Ore fractures and flows naturally under gravity and is delivered ultimately by gravity to the drawpoints where it is removed by large capacity LHDs and delivered to the ore handling system for hoisting.

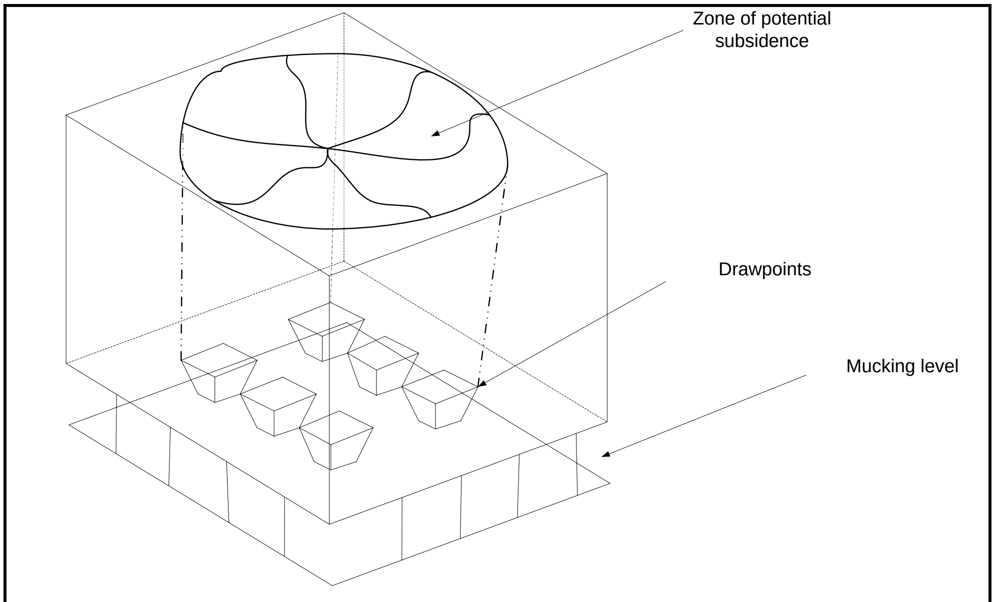

Figure 3.19 – Representation of Block Cave Stope

Block caving possesses a number of advantages and disadvantages which are presented for comparison in Table 3.7.

Table 3.7 – Advantages and Disadvantages of Block Caving

Advantages	Disadvantages
High capacity	Relies on presence of inherent geotechnical properties of ore for flow which may vary during the cave and stop production e.g. presence of clays
Low cost	Long pre-production lead times for development of drawpoint level
Low ventilation requirements	Unselective
Minimal support requirements underground	Management and maintenance of drawpoints is critical to maintain production
No lost blasts due to poor grade as grade control is executed directly at the drawpoint	Dilution can be unacceptably high at end of cave sequence
	Large block caves present a high risk of surface subsidence

There is some potential for pre-concentration at these mines and research has been undertaken into the pre-concentration of copper porphyry ores in particular (McCullough et al 1990; Burns & Grimes, 1986) with positive results. Pre-concentration is expected to facilitate the extraction of the valuable component of the porphyry while rejecting a large amount of barren material, the disposal of which on surface could be used to compensate for any expected surface subsidence during the cave, massively benefiting the operability as well as profitability of the block cave operation. There are several motivations for considering the integrated mining, processing and waste disposal model for copper porphyry ores. Costs per ton are low, however ROM tonnages are high, and costs per ton of copper produced for these methods are consequently high (Pitt & Wadsworth, 1980). Copper porphyry deposits, for example at Candelaria and North Parkes mines, are typically located in remote, arid regions where low water consumption is critical, and the application of pre-concentration at these mines would seek to reduce water consumption considerably. Studies are being considered for these mines, however, the integration of the appropriate processing and waste disposal technologies into a block caving scenario has not yet been examined in detail and is considered a significant challenge. It is obvious that waste rejects cannot be disposed of directly underground, and thus the option to pre-concentrate underground is not immediately attractive, although this option facilitates the opportunity to hoist valuable material only at peak times, while scheduling the hoisting of waste only from the cave at more opportune times. Pre-concentration and waste disposal on surface is a possibility, with rejects from the pre-concentration facility deposited in the area of potential subsidence as a means of managing the subsidence process. A typical block cave mining sequence would be thus discovery, design, development, preparation of the caving area, pre-concentration during operations and reclamation, landscaping and closure of the cave area post operations (Figure 3.20). It is recognized that significant further research is required to develop this concept.

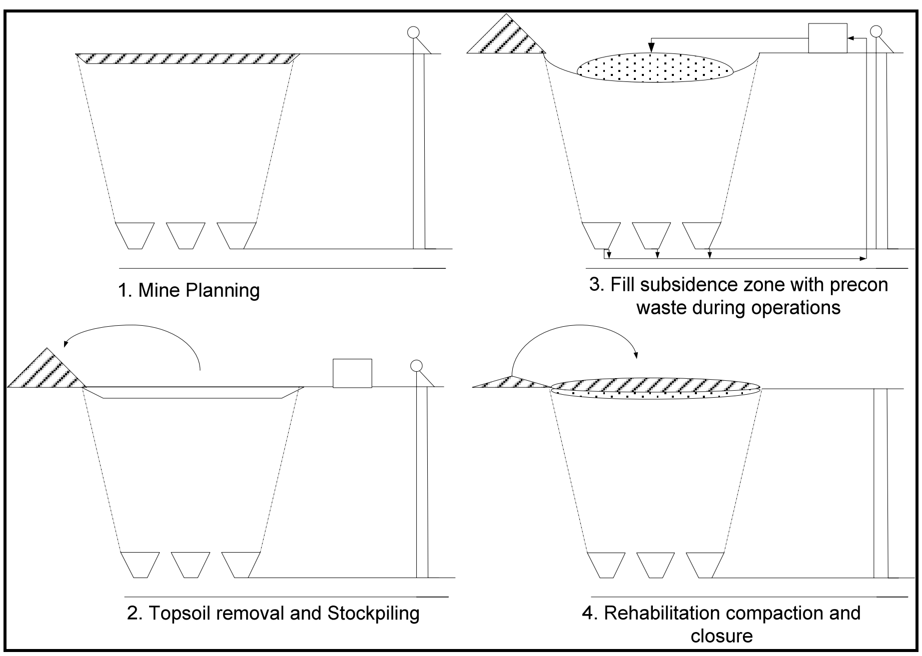

Figure 3.20 – Integrating Pre-concentration with Block Caving Operations

3.4.4 Open Stoping Methods

Open stoping is used in narrow, steeply dipping deposits where both ore and wall rock are relatively competent. Open stoping methods include shrinkage stoping, blasthole and longhole methods, and the following discussion is considered generically relevant for each of the methods. Open stoping is normally restricted to large, thick, steeply-dipping orebodies where both ore and wall rock are considered competent. Considerable development is required to prepare for the typical blasthole stopes, however much of this development can be in ore. Stopes are large, typically between 10 and 30 000 t, and barrier pillars are typically left at regular intervals within the orezone to enhance post mining stability (Figure 3.21). Dilution can also be high in more irregular orebodies, especially through wall sloughing when this rock is also less competent, thus the overall method, while high in productivity and low in cost, results in high levels of dilution and low overall extraction in unfavourable conditions (Canadian Mining Journal Sourcebook, 2007).

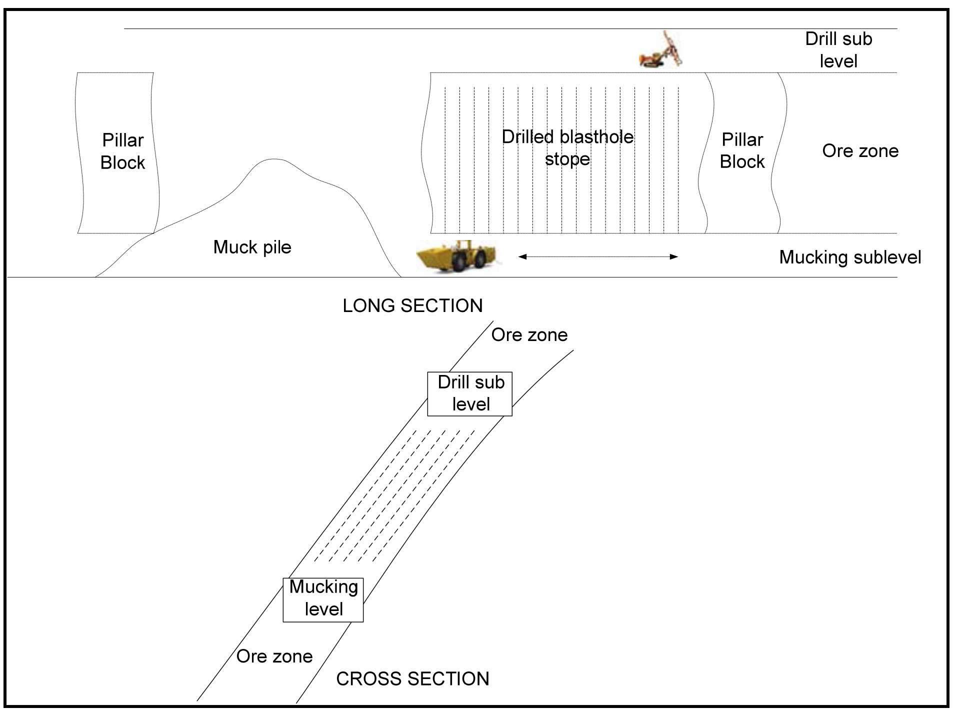

Figure 3.21 – Typical Open Stoping Method

As in the cut-and-fill scenario, integration of the pre-concentration and waste disposal facility into an open stoping environment will require the creation of excavations in competent rock, and located between the upper horizon of the orebody and the shaft. In shallower scenarios, the pre-concentration facility would be located on surface, as at Mount Isa, however the waste preparation and disposal facility should ideally still be located underground. For the successful integration of the pre-concentration and solid waste disposal steps with the generic open stoping method, the method would be adapted to the modified AVOCA type method, as already in use at one of the case study mines, Musselwhite Gold mine in Northern Ontario. The AVOCA method is a shrinkage, or open stoping method modified for retreat mining with fill, for situations where the wall rock is less competent and the stope must be filled prior to the removal of the following block to aid in ground control and limit dilution. The ore zone is typically mined from the bottom up, and from left to right or right to left as the case may be with the longhole/ blasthole stope face retreating sequentially from the fill face stope by stope (Figure 3.22).

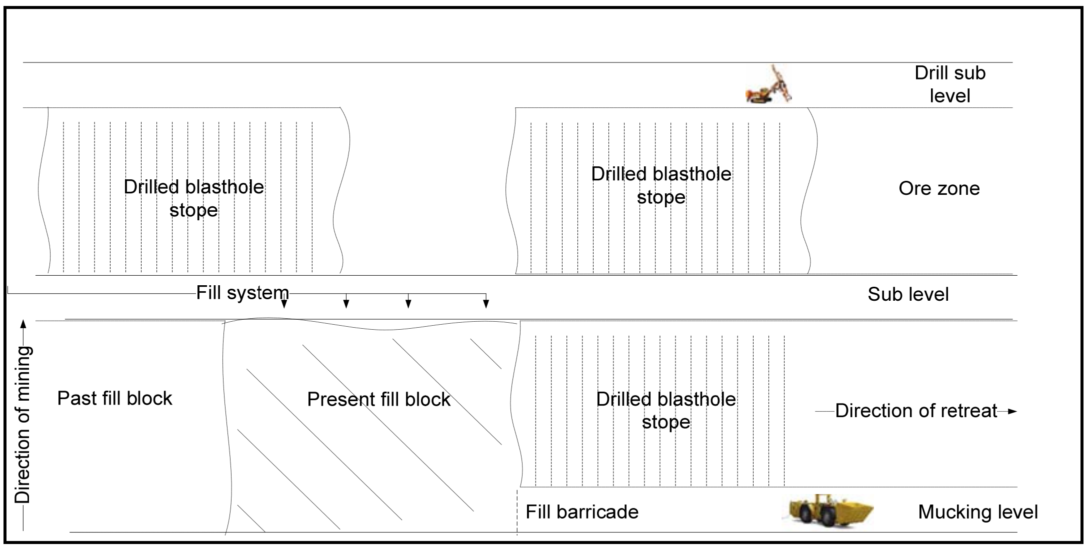

Figure 3.22 – Adaptation of Open Stoping to Modified AVOCA with Composite Fill

Several of the mines investigated during the research, such as Montcalm in Timmins ON, and the Thayer Lindsley Mine in Sudbury utilize open stoping methods, and conversion of these stopes to fill-based methods is recommended under the pre-concentration scenario.

3.4.5 Room and Pillar

Several adaptations of the room and pillar method are required to accommodate the concept of integrated mining and processing. Room and pillar mining is typically undertaken in extensive, flat to shallow dipping orebodies in shallow depth or where hangingwall strength is poor, for example as at the Doe Run Company in Missouri. Mining is highly mechanized and low unit cost. Dilution is low, however extraction is also low due to the requirement to leave remnant pillars of ore for roof support. Adaptation of the room and pillar method to accommodate fill is expected to convert this method to the post-pillar cut and fill method. Several additional impacts and benefits are expected. Dips are typically shallow, which is a barrier to simple filling operations. However, orebodies are typically thick, presenting several opportunities. Firstly the ore could be mined at a steeper apparent dip to facilitate the gravitation and confinement of fill. Also, in thick seam situations, rooms could be mined in a series of overhand passes, with previous passes to be filled in sequence, thus facilitating the fill cycle. Ore extraction in post-pillar cut and fill is higher than that of basic room and pillar mining, thus the economics of adapting to this method in the pre-concentration scenario are expected to be favourable, and has been recommended in for Doe Run in this instance.

3.5. Conclusion

The motivation for integrating pre-concentration and waste disposal systems into the mining cycle is maximized in situations of extreme depth, low grade, remote geography, or a combination of these issues. Underground pre-concentration has the potential to reduce the logistical costs of handling large quantities of low-grade ore over great distances and depths underground, as well as delivering an increased value product to surface. There are positive environmental and economic benefits to the adoption of this concept as well in the reduction of waste delivered to surface and reductions in the size of the mill required on surface.

The individual technologies for the development of integrated mining and processing, and waste disposal systems already exist at some level of implementation in the industry, and particular precedents for the practical application of these technologies also exist. Through the research they are projected to integrate well into the underground mining environment, and a successful application is envisaged to be found. However, it remains for these integrated technologies to be implemented in the field to create the envisaged system comprising mechanized mining, underground pre-concentration, automated preparation and placement of the waste as backfill.

4. Experimental Methods for the Geo-metallurgical Evaluation of Ores for Pre-concentration and Disposal of the Rejects

4.1 Introduction

For the successful design of the pre-concentration and waste disposal system, as well as an accurate quantification of the impacts and benefits thereof, a detailed characterisation of the Run of Mine (ROM) ore is required. This Chapter outlines the tools and procedures that have been developed to assess the mineralogical properties of an ore with respect to its amenability to pre-concentration, evaluations of the characteristics and potential uses of the valuable and waste products, as well as the quantification of additional impacts and benefits such as savings in transport and milling costs. Examples are presented for selected ore types. The methods presented in this Chapter have been shown to be applicable to a wide range of ores.

4.2 Mesotextural Evaluation, Fragmentation and Liberation

The first step in the design of the pre-concentration system is to assess the physical and mineralogical characteristics of the ore, including elementary visual observation of the presentation of the minerals in the rock. Models of mineral liberation often assume a random fracture pattern between particles, thus visual observation of the un-liberated ore is not typically considered; however it has been observed that for many ores that the response to breakage is largely controlled by the mineralogy and discrepancies in the comparative mechanical properties of the mineral components, such that preferential fractures tend to form along grain boundaries (McIvor & Finch, 1990; Olubambi et al, 2006). These grain boundaries can often be visually distinguished in the ore unaided, and have been shown in previous research to have a significant influence on the processability of the ore (Bocjevski et al, 1998). Furthermore, it has been noted in the literature that fragmentation and liberation can be successfully determined directly from this mineralogical texture, as behaviour between mineral phases in the rock are essentially structurally controlled (Schneider et al, 2003). Meaningful data can therefore be gleaned directly from visual and near-visual observation of the macro mineralogical texture, or mesotexture, in the unbroken rock. Thus in the research, study of the mesotextural characteristics of the ore is used to determine the potential for liberation of the valuable mineral, which in turn determines both the optimum feed size and performance of the selected coarse particle separation process. While microscopic examination as well as procedures such as QEM/SEM and Mineral Liberation Analyser (MLA) are useful for liberation analysis at finer particle sizes, this requires evaluation of fragmentation and liberation from ROM type particle sizes (~500mm) down to the lower limit of the separation processes under consideration (~1mm). 2 dimensional visual observations of the ore texture in situ are thus of primary importance; photographs can be taken prior to blasting and collection of the samples (Figure 4.1). Should an exposed face not be available, similar data can be obtained in one dimension only from observation and photographs of the core (Figure 4.2). Good pre-concentration potential is clearly indicated when there is a clear visual discrepancy between valuable and non-valuable components in the ore, and good liberation of the valuable components at a coarse particle size. In the visual evaluation of liberation and separability, several principal mesotextural classes, in order of increasing complexity, can be defined. Similarly the mineralogical micro-texture is noted, and the combination of mesotexture and microtextural data is used as a preliminary indication of the potential for pre-concentration of the ore (Table 4.1). For good pre-concentration potential, a combination of the simplest meso- and micro-textures (massive + massive, or massive + matrix) is expected to give the best separation results. The visual information can be correlated with actual measurement of physical separation potential by size fraction using one of the available bases for discrimination.

Table 4.1 – Mesotextural and Microtextural Classifications for Visual Evaluation of Ore Types (after Bocjevski et al, 1998)

Mesotextures	**Micro-textures**
Massive vein	Massive
Massive sulphide	Matrix
Massive discrete sulphide	Sieve textures
Banded sulphides	Simple intergrowth
Disseminated sulphides	Fine grained

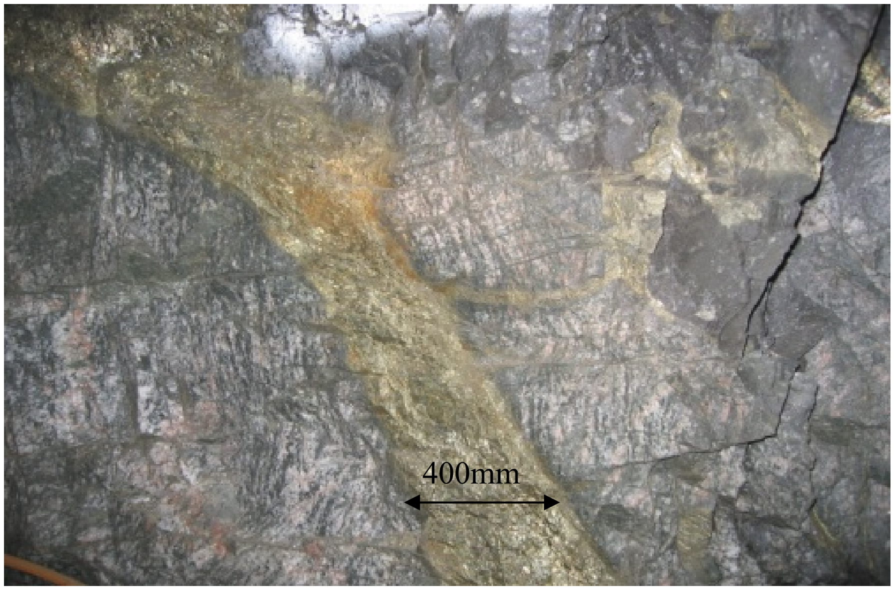

Figure 4.1 –In-situ observation of Massive-Vein Massive Sulphide Mesotexture at Fraser Mine

Figure 4.2 – Mesotextural Evaluation of Musselwhite Core

For the pre-concentration of coarse ore, as distinct from fine ore processing, the degree of liberation of coarse barren rock is as important as the liberation of metal bearing rock. The extent of liberation of barren rock will indicate the potential amount of waste that can be rejected without significant metal loss. For underground pre-concentration this criterion is practically limited to circa 60% waste rejection by mass arising from the volumetric bulking effect of broken vs. in-situ ore (Bamber et al, 2005). Thus an important target to be determined is the particle size distribution at which a coefficient of liberation of 60% with reference to the gangue mineral can be achieved. For this measure, the size-assay of a crushed sample of ore is recommended to determine the metal distribution and liberation data by size class in order to identify size classes with potential for direct rejection as well as the optimum liberation size for waste rejection.

Measurement of further visual data of value in the evaluation of the concept includes RQD data from the cores which is useful as an indication of the fragmentation behaviour of the ore; measurement of the Work Index of the various ore, pre-concentrate and waste fractions is required in order to quantify impact in the milling and flotation plant. Based on a particle top size of approximately 500mm a representative sample size would weigh several tons; due to practical constraints the actual samples are restricted to between 500 and 1500 kg each. Samples are weighed and split into 4 sub-samples by the cone-and-quarter method. Samples are wet-screened into a series of size fractions selected based on the indicated top size, and a screen series of $\sqrt{2}$ (Figure 4.3).

Figure 4.3 – Screened Sample Fractions for Size Assay

Screened sub-samples are washed and visually inspected for characteristics such as colour, lustre, texture, and degree of liberation, which are to be noted. A series of photographs are taken of each size fraction and particles representative of the ore and waste are identified and taken for mineralogical examination by polished section. Coarse fractions can be hand sorted into ore and waste products based on observed colour, texture, and density discrepancies. The densities of the ore and waste fractions identified are measured by the volumetric displacement method. A liberation evaluation is essential to be performed. Particles in each size fraction are sorted into groups according to an estimation of the degree of liberation. The Liberation Index, based on an evaluation of the proportion of fully liberated particles (particles with >80% sulphides) to the proportion of partially liberated particles, is evaluated according to the relation:

$$Li = \frac{n_{(fi=90)} * 0.9}{\Sigma n * fi} \qquad - (1)$$

Where: n = number of particles
fi = degree of liberation of particle

The overall liberation coefficient of the sample is then calculated based on the weighted average of the liberation indices of each size fraction.

A further characteristic indicator of the presence of pre-concentration potential is found through size assay of the fragmented ore. Size assay gives data on the particle size distribution of the ROM ore, as well as the metal distribution by size. Samples from each size fractions in the tests are split, crushed to –6mm, wet ground to –150 µm, oven dried and pulverised for assay. Representative 100g sub-samples of each size fraction are analysed typically for Cu, Ni and Co, as well as Pt, Pd and Au (TPM) and Mg content. In the course of the research, good liberation of the sulphides at coarse particle sizes, and an α-log normal grade distribution has been observed for ores which demonstrated good pre-concentration results (see Figure 4.4). It has been found that a normal distribution of grade by mass in the ore indicates poor pre-concentration potential.

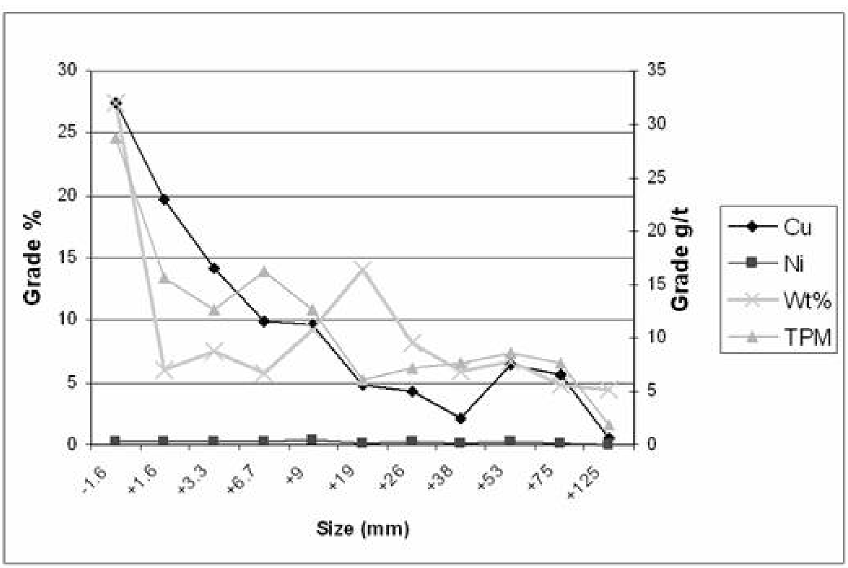

Figure 4.4 – Footwall Ore Size Assay Showing α-log Normal Grade Distribution

Should a stope sample not be available, meaningful data can still be determined from core samples. Mesotextural information from the core is directly comparable to the in-situ mesotexture. Size assay can then be performed on the core to determine a nominal particle size distribution as well as metal distribution by size. While the size distribution of the core sample will not be the same as the PSD of the ROM ore, core data can be used to determine the ROM size distribution by means of the Rock Quality Designation (RQD). RQD is determined for the core by measuring the total length of core pieces > 100mm and comparing to the total length of the core run:

$$RQD = \left[\frac{\sum_{0}^{i} li \geq 100mm}{\sum_{0}^{i} li} \right] \quad - (2)$$

The results of the RQD evaluation can be used together with data on the compressive strength of the rock, and information about the blast design, to correlate the size distribution of the core with the expected nominal size distribution of the ROM ore (Cunningham, 1983). See Figure 4.5.

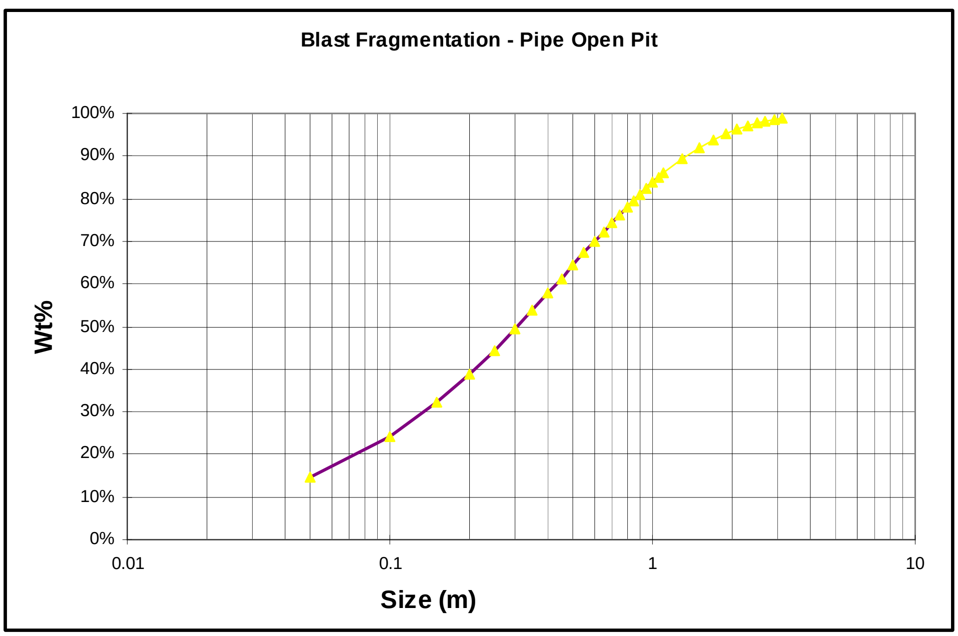

Figure 4.5 – Kuz-Ram Model Results for Pipe II ROM Ore

The metal distribution by size in the core can then be adjusted proportionally to the modelled size distribution of the ROM. Once mesotexture, fragmentation, liberation and metal distribution characteristics have been determined for the ore sample, several methods of discrimination can be evaluated based on measurement of the discrepancy between values for relevant physical properties such as colour, texture, lustre, density, conductivity or magnetic properties of the discrete ore and waste fractions.

4.3 Separability Testwork

4.3.1 Dense Media Separation

In the separability testwork, sink-float analysis is first investigated. Selected mineralised and non-mineralised particles are subjected to densimetric analysis by the volumetric displacement method. An indication of the separability of the ore can be determined by estimation of the gravity concentration criteria. The concentration criteria can be expressed as:

$$Cc = \frac{(\rho m - \rho l)}{(\rho l - \rho g)} \qquad - (3)$$

Where: ρm = mineral density
ρs = heavy liquid density
ρg = gangue density

The ore is then subjected to density separation tests at the range of SG's indicated in the densimetric analysis. In the course of the testwork, ores with good pre-concentration potential have demonstrated an enriched fines component, and for base metal sulphides this is typically below 9mm. A novel static-bath DMS unit has been built in the lab to perform these tests (Figure 4.6). The unit has an effective separation range of -53mm + 9mm, thus the -9mm fraction is typically screened to concentrate, with substantial positive impacts on the metallurgical balance for these ores. Samples can be tested in batches of up to 10kg at a time using the rig. Typical results are shown in Table 4.2.

Table 4.2 – Heavy Media Separation Results for 153 Ore (at SG 3.0)

		Cu		**Ni**		**TPM**	
	Wt %	%	Wt %	%	Wt %	g/t	Wt%
Feed	100	13.26	100	0.39	100	14.8	100
Sink	45	29	98	0.8	91	35	93.3
Float	55	0.5	2	0.06	9	1.4	6.7

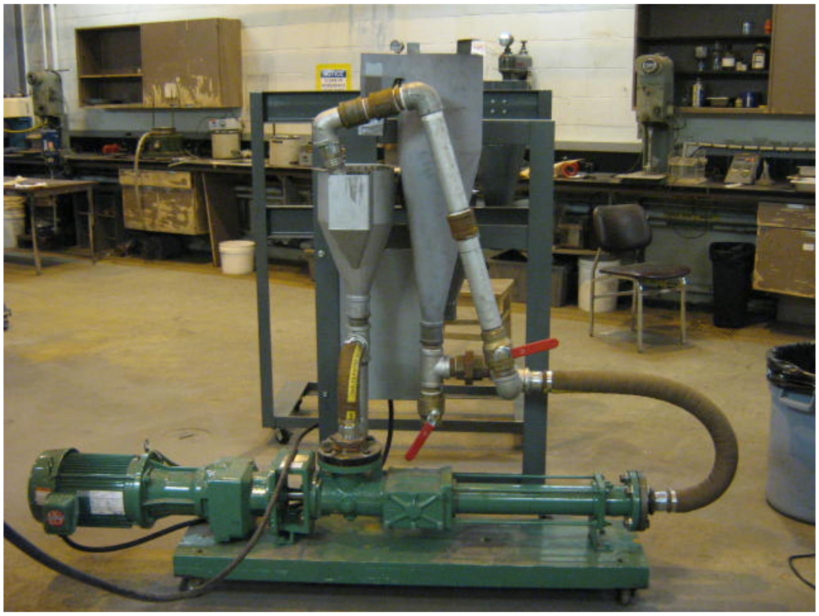

Figure 4.6 – UBC DMS pilot testing unit

The size distributions of the reject (float) and accept (sink) fractions can also be analysed (Figure 4.7). The presence of a bimodal size distribution in these samples reflects the different fragmentation behaviour of the sulphides and gangue minerals.

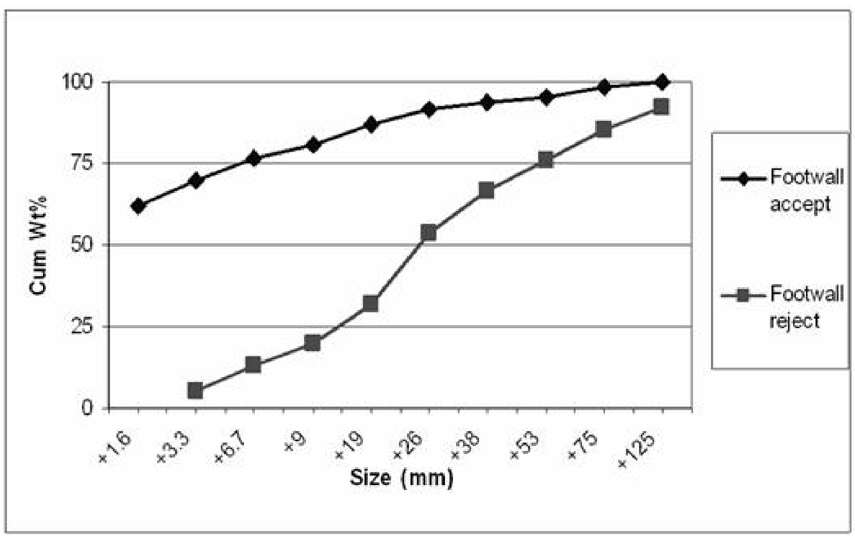

Figure 4.7 – Footwall ore sink and float fractions by size showing bimodal distribution of ore & waste fractions

Results can be been compared to current mine data to determine estimated dilutions for the samples.

4.3.2 Optical Characterization

Optical data in terms of RGB, reflectance, colour balance and YES balance are measured for selected ore and waste samples using the National Instruments Machine Vision Station that has been assembled at UBC. The NI Machine Vision station was specifically developed for the testwork due to the lack of commercially available optical sorting evaluation systems. Several mineralogical analysis packages are available, however these are typically for microscopic analysis and inappropriate for samples with particles larger than 10mm, and do not provide software for image comparison and discrimination, thus at present, investigation of this potential technology usually requires samples to be sent to one of the various sorter manufacturers, who are based either in Europe or Australia, which has been a financial barrier to the research. Therefore, in order to investigate optical sorting in more detail, an optical analyser has been developed and tested on range of ores at UBC.

System Design

Several options for this were available, and a functional specification based on previous observations was developed. The requirements for the system to be assembled were:

- System should be capable of analysis and discrimination
- Optical analysis was to be on the basis of colour, lustre, brightness, and texture
- Field of vision would be suitable for samples between 10mm – 100mm
- Images would be high resolution, digital colour images
- Colorimetric analysis would be by absolute Red-Green-Blue (RGB) value as well as colour balance ('gamma'), textural analysis and 'YES' colour comparison

Automated digital image analysis is a specialized field and only a few vendors worldwide offer hardware and software for this application. National Instruments is a leading vendor in the field, and offers educational versions of its Labview data acquisition and analysis suite to UBC, thus it was decided to investigate a solution using this avenue[1]. In digital imaging, colour, colour balance and overall intensity are automatic components of the image data set, thus analysis by these features is relatively elementary once a suitable image is captured.

Camera and lighting specification and setup are important in order to optimize this however, thus a specially designed image capture enclosure with standardized lighting was designed. Texture analysis was initially intended by means of the standard pattern recognition features offered by NI/Labview. Pattern recognition is a method for the accurate and efficient recognition and categorization of an object based on a characteristic subset of the object's

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[1] http://www.ni.com/analysis/vision.htm

features. Traditional pattern recognition requires comparison of the sample pattern area to every area in the image which is very computationally intensive. Advanced pattern recognition involves two steps – pattern learning and pattern matching. The learning phase involves identifying common characteristic features on a series of objects by repeated exposure of the image analyser to the object. If found, these characteristic features can be used for efficient and rapid pattern matching between new objects with similar features. Probably the most common application of pattern matching is in quality control for circuit boards for example the checking of capacitors on an amplification circuit board (Figure 4.8).

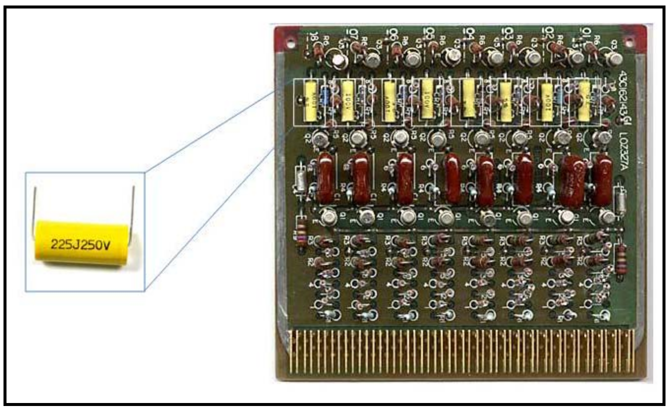


Figure 4.8 –Pattern matching application for electronic components

Labview features a standard pattern matching algorithm which incorporates pseudo-random sampling, neighbourhood checking, and pattern edge detection[2]. Pattern matching is independent of pattern orientation, which in conjunction with the above features suggested huge potential in mineral image analysis due to the irregularity of the typical mineral object. For the initial development of the system at UBC, Labview 7.1 was used and a National Instruments system integration specialist, Pishon Software of Surrey, Vancouver, was engaged to specify the final hardware setup and configure the Labview software. The final optical analysis setup is shown in Figure 4.9.

It soon became apparent that the pattern recognition method is limited in this application, as the routine requires an almost exact match for success, while the pattern of mineralization even within the same sample was found to be insufficiently consistent to generate an accurate result. Pishon Software was again engaged to overcome this, and it was decided to use fuzzy logic in the pattern recognition routine in order to obtain a distribution of matches instead of an exact pattern match.

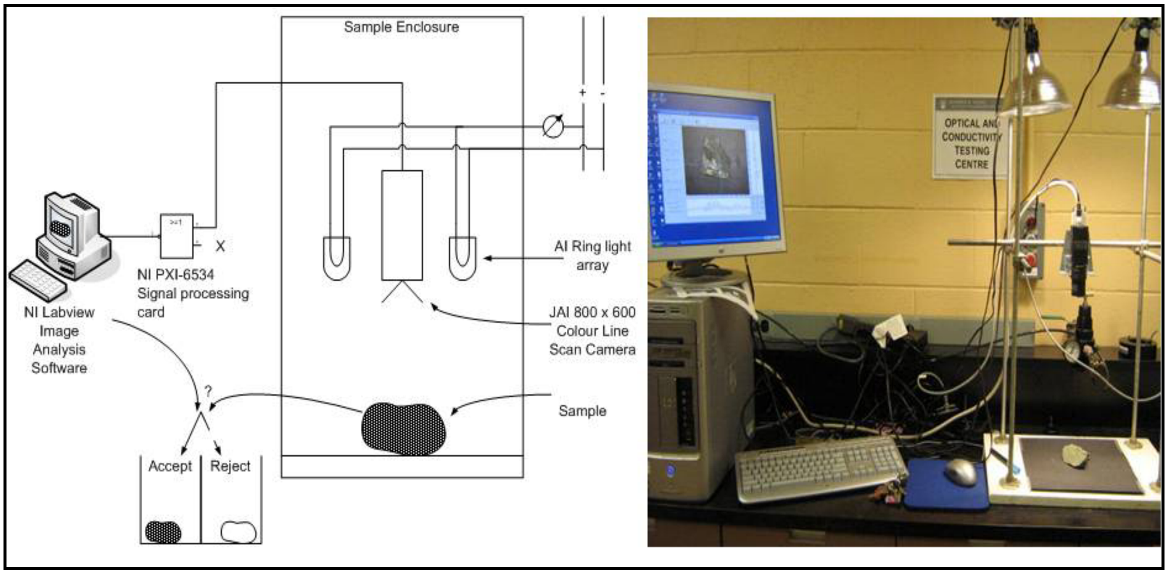


Figure 4.9 – UBC NI Machine Vision Optical Analysis System Schematic (left) and Actual System (right)

‘BrainCom’ is a freeware artificial neural network routine using back-propagation in the network to define the coefficients for the neural net (Wittnaum, 2001), and was obtained and configured for use with the UBC image analysis system. Results are encouraging and the system is now able to be trained to recognise the presence a range of typical sulphide minerals such as chalcopyrite, pentlandite and galena in a sample. The system calculates the total are of mineralisation in the sample and an approximate grade for each particle can be calculated.

Experimental Procedure

A typical test procedure involves the selection of several coarse rock samples for evaluation. It is important that the selection includes a full range of the styles of mineralization identified geologically. Rocks are weighed, numbered and split into ‘A’ and ‘B’ fractions. ‘A’ samples are subjected to image capture and analysis. ‘B’ fractions are prepared and sent for assay. Results of the optical evaluation are compared to the assay results and the basis for a sort using optical parameters can be established from the data. Selected samples of concentrate and rejects from the testing of the Xstrata ores were subjected to optical evaluation using the sensor setup. Reflectance, and colour measurements were taken and correlated with the grade of the samples. Results are presented in Table 4.3. RGB data from the images can also be analysed spatially to determine textural characteristics of the ore sample. Details of a complete photometric evaluation of ores from Xstrata nickel is presented in Appendix H.

[2] http://zone.ni.com/devzone/cda/tut/p/id/3763

Table 4.3 – Correlation of Photometric Measurements to Grade for Xstrata Ni Ores

Sample	Red	Green	Blue	Red	Green	Blue
Craig 8112 Ore: 1.26% Ni, 0.57% Cu				**Waste: 0.19% Ni, 0.12% Cu**		
Ave	66.94	66.94	108.28	107.68	100.95	105.32
Sdev	38.84	38.84	50.56	52.83	47.76	41.10
Craig LGBX Ore: 3.5% Ni, 0.38% Cu				**Waste: 0.21 % Ni, 0.18% Cu**		
Ave	109.41	104.76	98.22	104.27	101.15	87.44
Sdev	51.33	58.43	56.70	56.85	58.53	53.38
Fraser Cu Ore: 0.84% Ni, 22.01% Cu				**Waste: 0.03% Ni, 0.4% Cu**		
Ave	107.51	92.23	75.94	99.01	89.37	68.77
Sdev	64.10	61.10	47.79	37.49	42.16	21.20
Fraser Ni Ore: 1.97% Ni, 2.8% Cu				**Waste: 0.20% Ni, 0.12% Cu**		
Ave	95.73	97.45	105.38	107.68	100.95	105.32
Sdev	36.03	40.95	41.55	52.83	47.76	41.10
Montcalm H Ore: 2.12% Ni, 0.82% Cu				**Waste: 0.15% Ni, 0.18% Cu**		
Ave	116.27	120.28	117.55	95.062	111.05	125.78
Sdev	56.22	54.05	45.42	48.84	52.22	49.99
Montcalm L Ore: 1.4% Ni, 0.56% Cu				**Waste: 0.15% Ni, 0.1% Cu**		
Ave	88.63	83.23	76.34	86.82	81.76	75.52
Sdev	51.96	47.47	39.35	46.23	49.25	43.62
T-L 15 Ore: 1.83% Ni, 10.79% Cu				**Waste: 0.076% Ni, 0.4% Cu**		
Ave	92.33	104.66	109.00	129.74	121.60	104.08
Sdev	48.54	51.71	46.92	48.13	50.72	47.11
TL-80 Ore: 1.70% Ni, 1.11% Cu				**Waste: 0.114% Ni, 0.15% Cu**		
Ave	131.28	117.04	90.41	74.98	82.54	82.40
Sdev	47.12	48.35	43.33	34.41	38.10	32.62
T-L 670 Ore: 0.82% Ni, 0.45% Cu				**Waste: 0.16% Ni, 0.15% Cu**		
Ave	130.40	138.00	124.40	119.09	112.05	92.27
Sdev	44.65	47.55	44.22	46.10	49.66	33.40

For the Xstrata ores, the RGB data correlates well with the assay results and the basis for a colorimetric sort has been established for each ore. Based on the results, a synthetic grade-recovery curve can be constructed retrospectively from the data by correlation of the sensor response curve with the assayed sample grades. A synthetic sort can then be performed on the data to indicate potential sorting results through the selection of an appropriate cutoff grade.

4.3.3 Conductivity Evaluation

It was also desired to evaluate various samples for their amenability to conductivity based separation. Conductivity methods are generally applicable for native metal ores such as gold and copper, as well as massive sulphide ores grading between 2-3% metal with good results. Testwork reports on copper porphyry ores from a range of mines in the Montana and Upper

Michigan area indicate conductivity sorting delivered up to 50% waste rejection by mass from the ores at recoveries from 85 – 92% (Miller et al, 1978). It is suggested that conductivity methods should be utilized in conjunction with optical sensors in order to compensate for the variance in response by particle size for improved results. In previous ore sorting tests at INCO, tests of optical sorting or conductivity sorting alone on combined ROM ore from McCreedy East did not give acceptable results (Schindler, 2001). Sorting tests on the ore using a combination of optical and conductivity sensing in a Mogenson sorter, gave results of up to 77% waste rejection at a recovery of 98% confirming the potential of combined sorting. The method is appropriate for native and sulphide ores grading between 1-3%, however, is inaccurate for low grade disseminated sulphides < 1% and any particle < 1mm.

A variation on the metal-detector type sensor which is indicated for use with lower grade base metal sulphide ores is the induction balance coil (Sivamohan & Forrsberg, 1991). This arrangement gives good results for native ores and is considered to have high potential for sulphides such as such as sphalerite, chalcopyrite, millerite, galena and covellite in VMS and igneous intrusive orebodies where there is a significant conductivity differential between the ore and gangue minerals. In this arrangement a second balancing coil is introduced, and the difference in signal between the disturbed (sensing) and undisturbed (balancing) coil is measured. Laboratory equipment for the evaluation of an ore to this method is not commercially available, and as in the case of optical methods, testwork by the vendors is expensive and prohibitive to the research. On this basis, other avenues for the development of capabilities in ore characterization by conductivity methods were pursued.

4.3.3.1 Testing of the INCO 'B2'Sensor

A sensor based on these principles, and using a coil design taken from the field of geophysics, had previously been developed at Ecole Polytechnique (Boucher, 2005), and employed in testwork at INCO ITSL in Sudbury. Sensor design is of the 'pancake' induction coil type comprising a single-layer coil of signal wire (Figure 4.10).

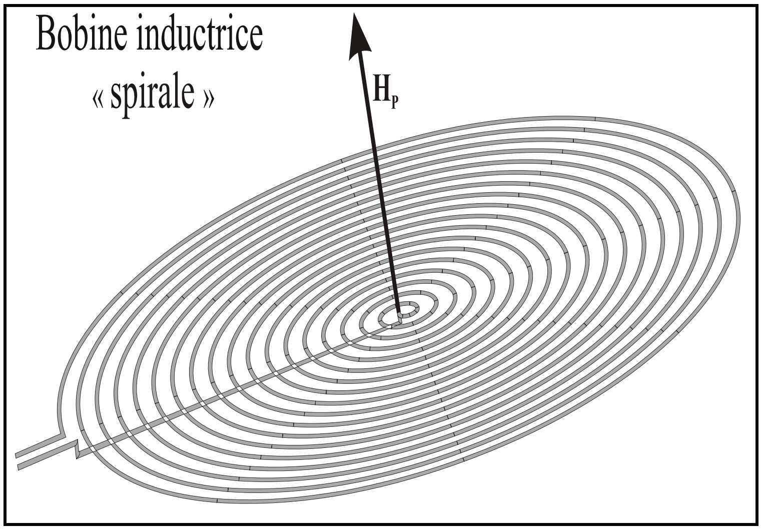


Figure 4.10 – 'Pancake' type inductance coil (after Boucher, 2003)

The sensor had been brought to INCO Technical Services Limited in Sudbury for further testing and development. This INCO 'B2' sensor was tested against a commercially available sensor from Applied Sorting Technologies of Sydney, Australia on selected Ni ore and slag samples at INCO Technical Services with good results (Boucher, 2003). The sensor was given by INCO to UBC for further testing versus the AST conductivity sensor on low grade ultramafic nickel ores from Thompson, Manitoba. The arrangement of the 'B2' sensor setup is shown in Figure 4.11.

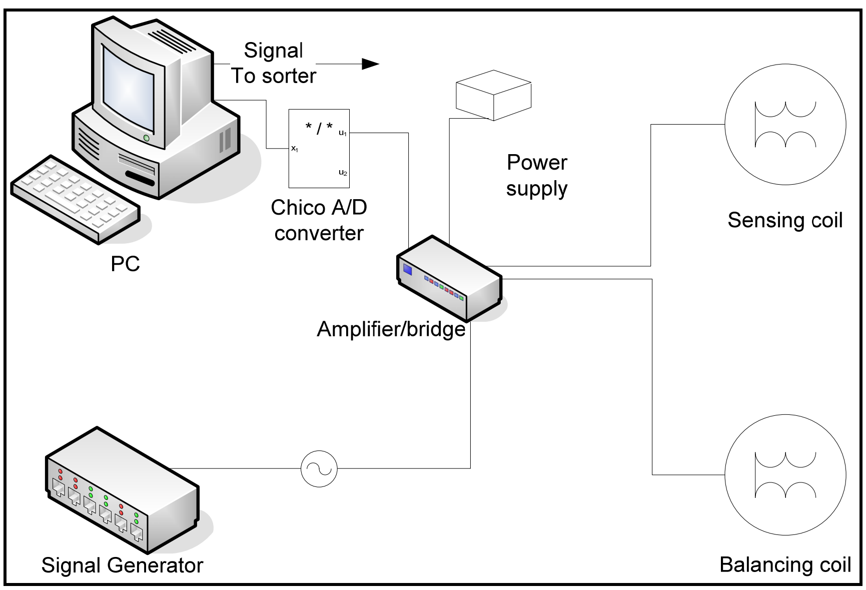


Figure 4.11 – INCO 'B2' Sensor setup

Prior to the testwork, the sensors were taken to Thompson and calibrated on selected core samples (Stephenson, 2006) using the half-rock assay procedure described under the section on optical sorting. Core conductivity tests were performed on NQ core halves selected from previously logged drill holes from various locations around the Pipe deposit in order to evaluate the conductivity response of the sensor to the core pieces, and compared to Ni assay data from INCO Sheridan Park in Mississuaga for each piece of core (Figure 4.12).

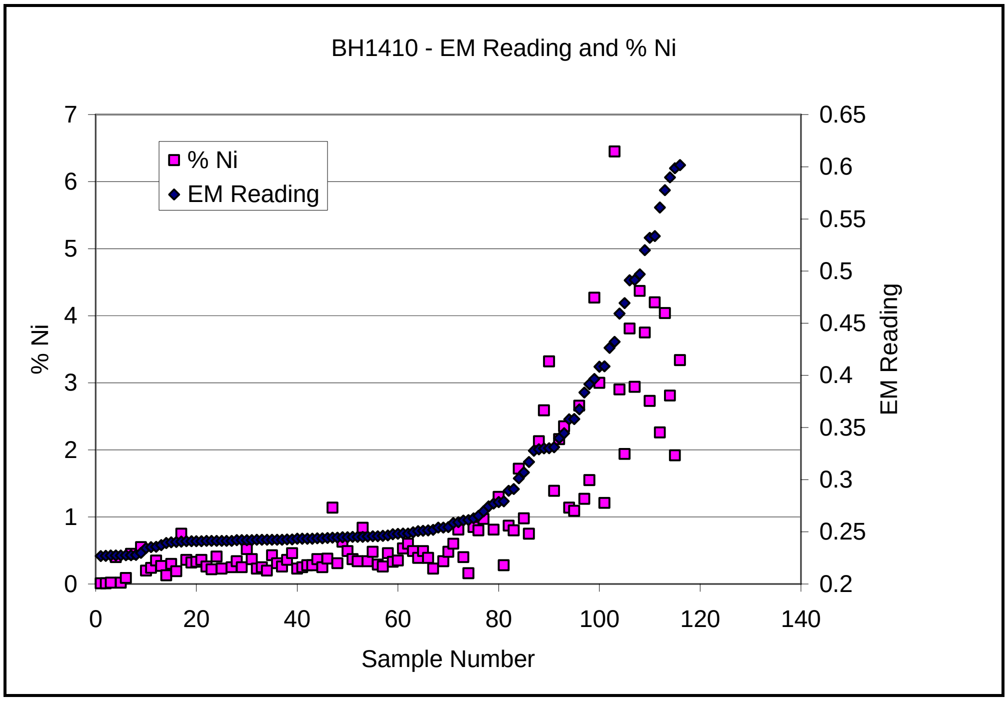


Figure 4.12 – Correlation of B2 reading with Ni Grade (from Stephenson, 2006)

A good correlation between the EM reading and % Ni was thus established for the samples using the sensor. The AST sensor was erratic and inaccurate at low Ni content, with accuracy improving with increasing Ni grade. The 'B2' sensor was shown to be unresponsive in the lower Ni grades, erratic in the cutoff range, but demonstrated an increasingly accurate response above this figure. From the tests, it was concluded that the current B2 sensor design had potential in this application, but embodied a number weaknesses. The sensor was:

- unresponsive in the lower ranges, and erratic around the cutoff grade
- unreliable and highly sensitive to ambient interference
- not of robust construction and unsuitable for an industrial environment
- the pancake sensors are large, and highly sensitive in the centre, but signal strength falls rapidly with increasing distance from the centre of the coil
- the setup comprises 10 individual system components, making the arrangement unwieldy and complicated to set up and use
- there were a number of redundant functions in some of the system components which would be eliminated with improved component selection

4.3.3.2 Development of the MineSense'B2' MkII Sensor

On further examination, it was found that the B2 sensor incorporated a number of further fundamental design and construction errors. Several components, such as the amplification and bridge circuits had been custom designed and possessed a number of flaws when compared to good practice in instrumentation design. The sensors were not robust, were imperfectly shielded, poorly grounded; furthermore the signal wires and connectors on the sensor supplied were damaged, possibly contributing to the poor sensor performance. Also, according to the software specification, it was intended to perform Fast Fourier analysis on the incoming signal, followed by calculation of the phase and amplitude components of the signal vector (Bamber & Houlahan, 2007). However, it was found that the vector analysis calculations in the software were incorrect and that the final sort decision was being made on a false basis.

After the testwork campaign, the 'B2' and AST sensor setup were returned to INCO for further testing at Sheridan Park. However, it was decided to proceed with the design and construction of an improved MkII unit at UBC in order to overcome the issues with the 'B2' that had been identified. Private sector funding was obtained from BC Mining Research Ltd and an electrical EIT was engaged. The sensors, bridge and amplification circuits were redesigned in accordance with good instrumentation and electrical engineering practice; additional design work involved integrating the power supply into the amplification and bridge box, and integrating the signal generation, signal acquisition, and sort signal processing into a single PCI data acquisition card. The primary improvement in the design is an improvement in the power and sensitivity of the sensing coils. The single layer pancake coil has been replaced by a smaller double layer coil incorporating a solenoid air gap which maintains the field strength across the majority of the coil diameter (Figure 4.13). The B2 sensor-bridge and signal amplifier was identified as an area for improvement, and the bridge and amplifier arrangement was redesigned according to conventional instrumentation principles. Principal design changes were the introduction of a balanced Wheatstone bridge, and the use of variable amplifiers on the input and output signals to and from the sensors. The modified sensors, bridge and amplifier network have been integrated with a new PCI data card and PC. The new PCI card is capable of generating the base signal as well as analyzing the attenuated signal from the sensors. The card allows up to 4 input channels, thus it is now possible to expand the sensor arrangement into a sensor array. Signal analysis software written in C++ was provided with the Chico PCI card and has been modified to analyze the attenuated signal and calculate the magnitude and phase of the sample impedance.

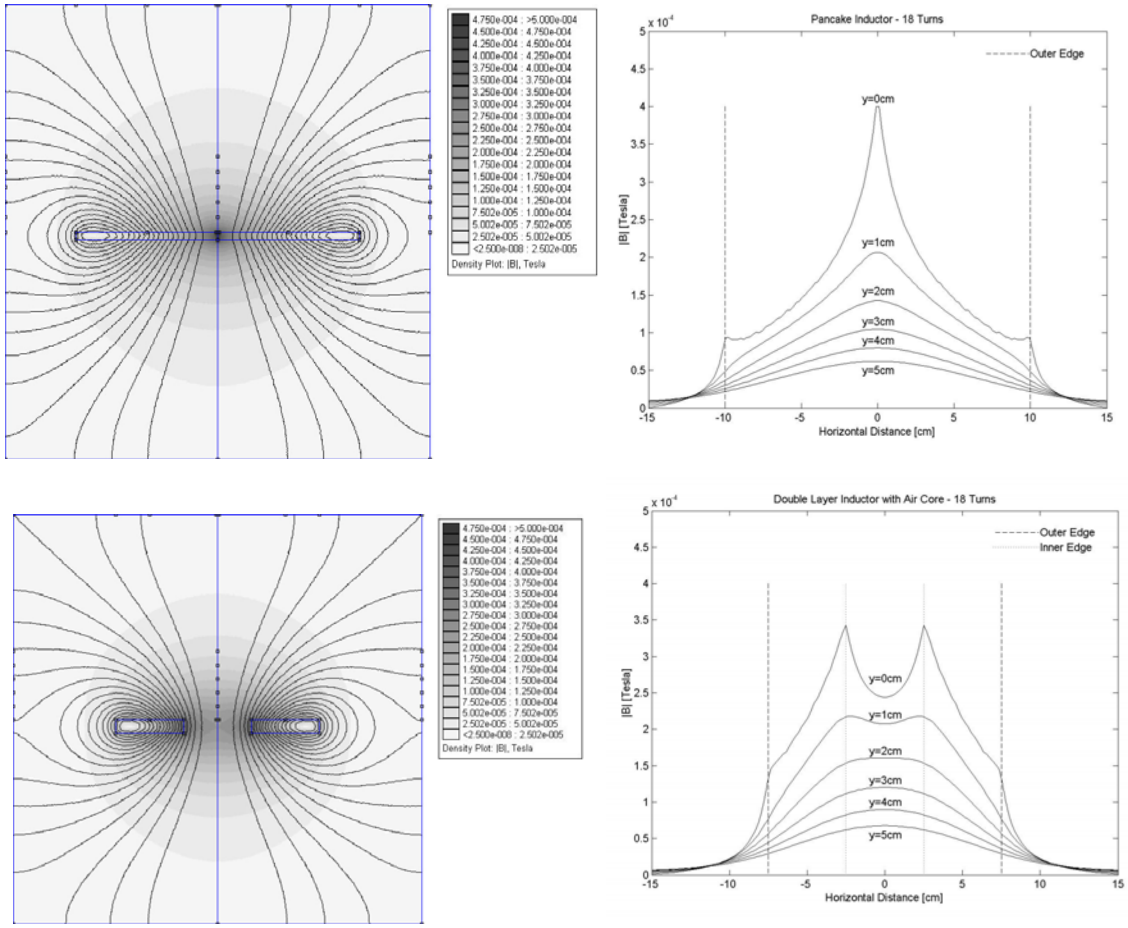


Figure 4.13 – Plan and Section Plot of Field Strength vs. Coil diameter (B2 Mk1 (top) vs B2 MkII (bottom) (Bamber & Houlahan, 2007)

Using these modified components, an integrated inductance-based ore evaluation station has been built and tested at UBC with significantly improved results over the ‘B2’ prototype. The final B2 MkII sensor setup is shown in Figure 4.14 & 4.15.

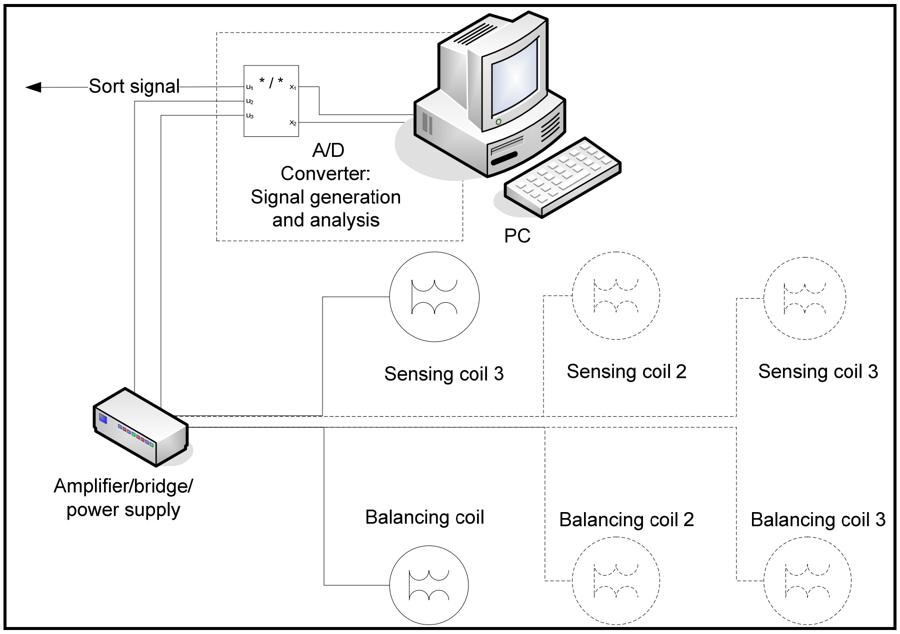


Figure 4.14 – MineSense 'B2' MkII Conductivity Sensor Schematic

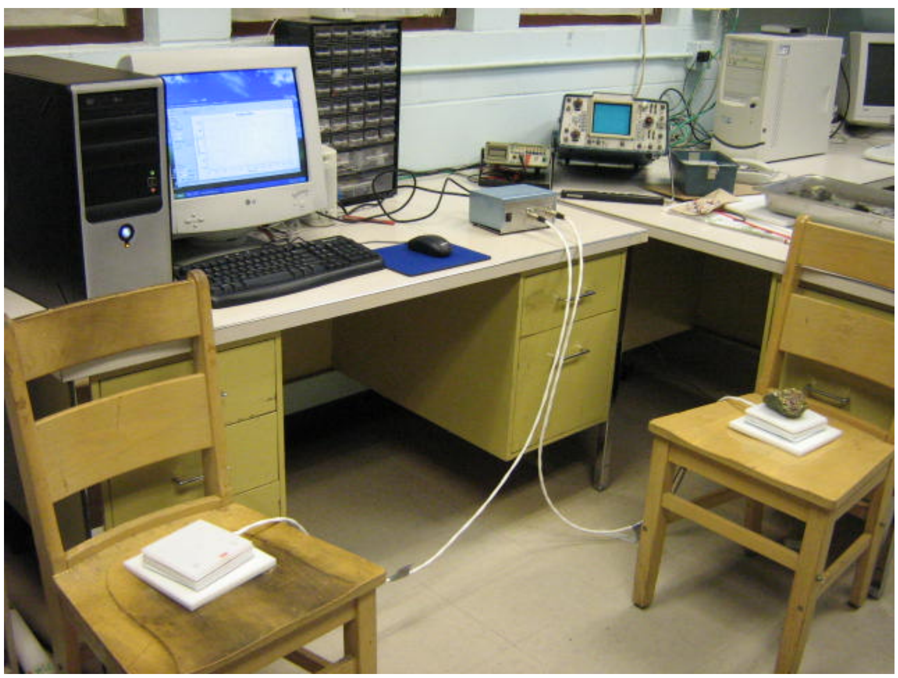


Figure 4.15 – MineSense 'B2' MkII Sensor setup showing data capture PC, bridge amplifier (centre) and sensing (right) and balancing coils (left)

The sensor gives a calibrated response for conductivity (in mV) as well as magnetic susceptibility (X_m). The sensor was tested on a sample set of Xstrata ores previously calibrated by half-rock assay for the optical characterization tests. A correlation of sensor response with metal grades is presented in Figure 4.16.

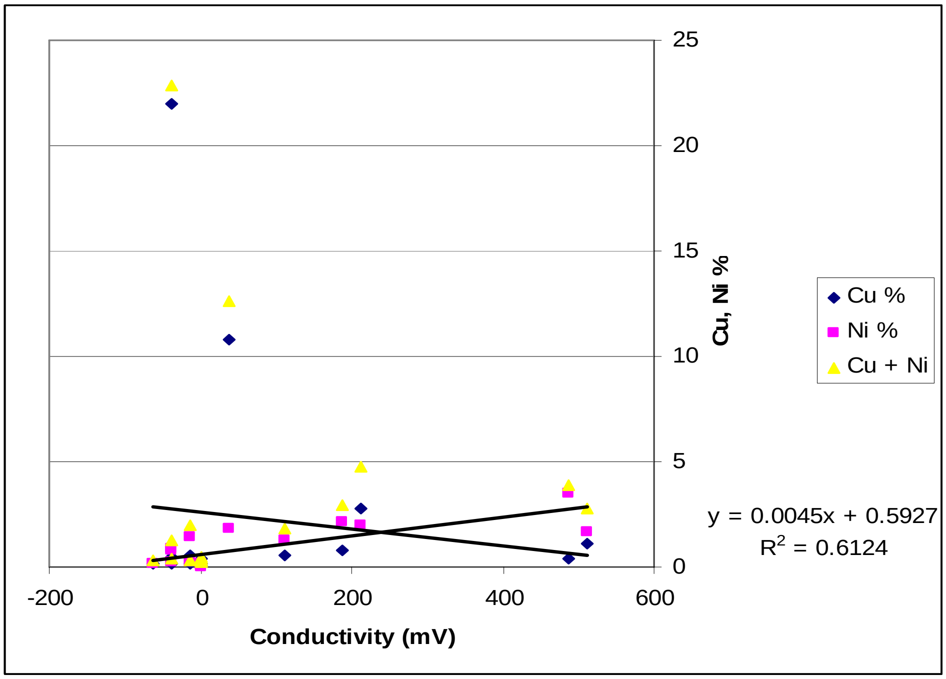


Figure 4.16 – Preliminary Correlation of 'B2' MkII Conductivity Readings with Grade for Xstrata Cu-Ni Ores

Initial results show a good correlation between the conductivity response and Ni grade. Correlations with Cu grade are not apparent, although an initial correlation between Cu grade and magnetic susceptibility has been observed. As in the optical evaluation procedure, data from the grade/sensor response curve can be used to generate a synthetic sort for evaluation of the potential for waste rejection prior to undertaking actual sorting testwork. The sensor is recommended for use as a laboratory tool for evaluating Ni sorting in this application. As with the original B2, it is felt that this sensor array, with increased power and sensitivity has a number of additional applications including core logging, grade analysis and down-the-hole sensing. However, based on initial tests, it appears that the new coil is also highly sensitive to changes in the dielectric characteristics of the coil environment, and due to the complexity of the attenuated signal arising from the coil it appears possible to discriminate more than just a scalar value of grade, and thus is potentially able to distinguish between the highly non-metallic samples passed across the sensor as well. This observation is supported by references from the literature (Kieba & Ziolkowski, 2002) and the application will be developed through further research and testwork using the MkII sensor. Further developments, including improved signal conditioning and the option to expand the sensor setup for belt sorting by adding additional sensors in an array are pending.

4.4 Grinding Work Index Testing

The pre-concentration of ores is expected to result in power savings at the mill in three principal areas. Firstly, the rejection of waste results in a reduction in tonnage reporting to the mill. Secondly, the waste is primarily hard siliceous rock; removing this material leaves a product enriched in relatively soft metal bearing sulphides which will have a lower grinding work index. Thirdly, the pre-concentration product will be crushed to a finer size than the present plant feed. This projected impact on grinding power requirements can be calculated from the Bond equation as follows.

$$\Delta P = \Delta t 10 \Delta Wi \,[1/\sqrt{P_{80}} - 1/\sqrt{F_{80}}] \qquad - (4)$$

Where: Δt = waste rejection (tph)
ΔWi = Change in Bond Work Index
P_{80} = 80% passing size of the product (µm)
F_{80} = 80% passing size of the mill feed (µm)

For the evaluation, the work index of a 2kg sample is determined through a full Bond Work Index test and used as a reference. The sample is crushed to nominally -6.7mm and subjected to grinding in a standard Bond Mill with a closing screen of 149µm. The Work Indices of the other ores and ore concentrate and waste products are then determined by comparing the product size distribution of the samples to the reference sample in comparative rod mill grinding tests under identical conditions:

$$WI_2 = WI_1 * \frac{\sqrt{p80_2}}{\sqrt{p80_1}} \qquad -(5)$$

4.5 Additional Geotechnical, Geo-metallurgical and Rheological Testwork

4.5.1 Geotechnical Characterization of Rejects

A range of waste products re produced during the pre-concentration testing. Waste products are to be characterized geotechnically in preparation for their use as potential underground fill material. Appropriate geotechnical properties were selected for evaluation and compared to values recommended in the appropriate ASTM standards (Talbot & Richart, 1923):

- Particle size distribution
- Particle shape
- Adsorption
- Specific gravity
- Void space
- Strength
- Chemical composition

Particle size distribution is evaluated by means of screening and weighing the screen oversize. Particle shape is evaluating by comparing the typical ratio of the longest and shortest particle dimensions respectively. Particle density is measured on a dry, weighed sample by the displacement method in a 1l volumetric flask. Absorption is also measured in the same test - aggregate is left in the volumetric flask for 15 minutes; the sample is then removed and dried on burlap. The difference in mass between the soaked and dry aggregate gives the absorption in % of the sample (Talbot & Richart, 1923).

The size distribution of the rejects is analysed and compared to the recommended ASTM aggregate size distribution curves (Talbot Curves) for fill aggregates. The coefficient of uniformity is determined by calculating the quotient of the d_{80} and d_{20} of the products. The coefficient of uniformity also determines the void ratio in the aggregate sample. Void ratio is a measure of the free space within a mass of broken material, and is thus used to estimate the bulk density of such a sample. This information is collectively used to determine the optimum density for the mix design. The size distribution of the rejects is ultimately controlled by the method of feed preparation and concentration and thus the coefficient of uniformity, void ratio, and ultimately the optimum mix design can vary widely. The ores tested typically demonstrate a characteristic high grade fines fraction and removal of this fraction prior to concentration increases the relative size distribution of the rejects when compared to the ROM ore, increasing their compatibility in terms of aggregate uses. Evaluation of the characteristics of the Xstrata rejects indicated the rejects measured generally coarser than a typical ASTM fill aggregate of similar topsize, but were generally within the envelope of the Talbot specification and are nevertheless considered acceptable for use as a fill aggregate. Rejects were further geochemically characterised by ICP whole rock assay in order to evaluate any potential acid or neutral rock drainage potential. Acid based accounting is derived from the chemistry of the minerals as evaluated in the ICP whole rock assay. Rejects can be classified as either non-acid generating, neutral or acid-generating depending on the stoichiometry of the sulphur assay in the samples. Results confirm that the rejects are generally non-acid generating, with a minority of potentially acid-generating rejects. The results confirm the potential shown in preliminary mix design and testing (Bamber et al, 2006), and it can be concluded that the rejects of the pre-concentration process under consideration can typically be considered acceptable for disposal on surface as non-acid generating waste or use as aggregate in fill underground.

4.5.2 Mix Design and Testing

Once the rejects are geotechnically and geochemically characterized, they can be evaluated and tested in terms of an appropriate fill mix design. Several mixes by mass are typically to be considered for the testing:

- A rockfill comprising typically 95% rejects, 5% OPC with 1:1 water cement ratio
- 'Maximum density' composite fill with typically 70% rejects
- '1:3' composite fill with typically 36.5% rejects
- '1:7' composite fill with typically 20% rejects

5% cement content by mass is suggested as a standard for the composite mixes in order to generate comparative results. Mixes are prepared using the data from the characterization and tested according to ASTM standards for backfills. Sample moulds were industry standard plastic cylinders typically 200mm long x 100mm Ø. Moulds are typically half filled with mix and left covered in polyurethane for 28 days to cure at room temperature. UCS testing was done at the UBC Geomechanics laboratory on an MTS 815 'stiff' UCS testing machine (Figure 4.17).

Figure 4.17 – MTS 815 UCS Testing Machine

The uniaxial stress/strain curves from the UCS Testing of the samples are then analysed to determine the peak stress at failure as well as evidence of any residual strength in the fill post-failure. The strength and efficiency of the mixes in terms of UCS : Wt% cement is also analysed (Table 4.8). The binder efficiency generally increases significantly with increasing reject content, thus indicating the positive benefits of utilizing these rejects as fill material. Ultimate cement content in the mixes as well as water cement ratio can vary widely, so care in preparing the mixes is recommended.

4.5.3 Rheological Tests

As it is the intention to use the rejects as a material for composite fills in underground backfill applications. Thus the transport of the fill becomes an issue and it is felt important to investigate the basic rheological properties of the fills for the purpose of determining an appropriate rheology for transportation by pumping. For this, the slump test method of Hu (1995) was chosen. The appropriate rheology for fill placement is determined by three characteristics, the shear stress τ_0, plastic viscosity μ and shear strain rate γ' (strain gradient) of the mix (Tattersall, 1991):

$$\tau = \tau_0 + \mu\gamma^{\cdot} \quad - \ (6)$$

The standard slump test returns a value for the slump of the concrete which can be used to determine the yield stress of the mixture in the following form:

$$\tau_0 = \frac{\rho}{270}(300 - s) \quad - (7)$$

Where ρ- density of the mix

s – standard slump in mm

For full determination of the parameters of the Bingham equation (5), use of the modified slump test to determine the plastic viscosity of the mix is proposed where the rate of slump in s is also measured (Chiara et al, 1998). Slump rate in s has been shown to empirically determine the plastic viscosity in mixes with low slump according to the relation

$$\mu = 25x10^{-3}\rho t \quad - \ (8)$$

Where t is the time of the slump in seconds

For the slump testing a cylinder with a diameter of 100 mm and height of 150 mm was utilized. The slump cylinders were filled as for the UCS testing, then drawn slowly over the slump sample. Slump was measured from the top plane of the mould to the upper surface of the extruded material. In combination with the physical observations, the slump test and the resultant rheological analysis allow for general discussion of the potential to pump the composite fill mixes. Figure 4.18 shows indicative slump testing results for selected fill samples composed of Fraser Copper rejects and full tailings.

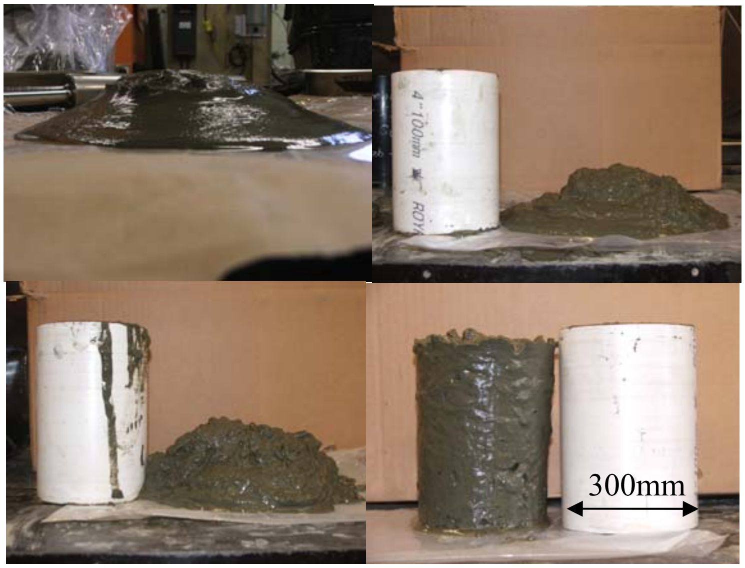


Figure 4.18 Slump test Photographs for Fraser Copper Fill Mixes showing qualitative differences in rheology of the mixes

For a pumpable fill mix, a τ value of between 0.15 – 0.5 is suggested; mix optimization to achieve maximize strength and binder efficiency while maintaining a low yield stress in the fresh mix is thus recommended.

4.6 Conclusions

A procedure for the evaluation of ores with reference to the mineralogical, metallurgical, as well as additional geotechnical and geometallurgical characteristics of the ore and waste fractions has been presented. Site visits for geological and mineralogical information gathering, examination of the in-situ meso- and microtextures have been shown to be important. Size-assay, liberation and fragmentation characteristics, and methods for potential particle separation must be determined. Geotechnical characteristics such as particle shape, strength, hardness and Work Index have also been determined to be important in the evaluation of the waste products. A range of fill mix designs have been developed and procedures for evaluating the relevant geotechnical and rheological properties of the mixes have been identified and used. The procedure generates data for the evaluation of the potential for pre-concentration of the ore, suggests potential separation methods and provides data for the evaluation of waste disposal strategies. The results from the evaluation also provide data for the parametric evaluation of the opportunity, impacts and benefits of the concept as discussed in Chapter 5.

5. A Parametric Impact Valuation and Evaluation Method for the Application of Pre-concentration and Waste Disposal Systems to Hard Rock Mining

5.1 Introduction

The results generated from experimental evaluation of the candidate ores for pre-concentration and waste disposal as described in Chapter 4 allows the subsequent evaluation of various impacts and benefits which have been identified in the course of the research. Pre-concentration reduces the quantity and increases the quality of ore arriving at all stages subsequent to the pre-concentration step by rejecting between 30-60% of the coarse, barren siliceous material from the ROM ore. Impacts and benefits arising from this that have been identified through the research include (Klein et al, 2002; 2003; Bamber et al 2005, 2006):

- Facilitation of lower unit cost mining methods through a decrease in selectivity
- Facilitating an increase in mining rate without increasing the capacity required of downstream facilities such as haulage, hoisting, transport and milling
- Increasing the mining rate without increasing the capacity required in the concentrator
- Increase in the grade of ore delivered to surface without significant loss of metal
- Reduction in the cost of engineered support through the generation of large quantities of fill material close to the mining face
- Improvements in geotechnical properties in the fill and thus ground control due to an improvement in post-mining rock mass stability
- A reduction in ventilation required through an reduction in the number and thus surface area of unfilled voids
- Reduction in overall mining, plant and infrastructure capital costs;
- Higher overall metallurgical recovery in the surface grinding and flotation plant compared to unsorted ore (specifically on well liberated, highly diluted low grade ores);
- Reduction in overall material handling costs subsequent to the pre-concentration step;
- Reduced energy consumption in transport and comminution;
- Reduced waste disposal costs of coarse dry waste compared to fine, saturated waste;
- Reduced water consumption in processing and waste disposal;

Negative impacts are also to be noted, and include in particular:

- Additional capital for the pre-concentration and waste disposal plant;
- Additional unit operating cost for pre-concentration
- Additional metallurgical penalty across the pre-concentrator;
- Additional excavations required for the underground situation
- Additional heat, noise and dust in the underground environment

Quantification of these impacts is not an elementary task, as it involves estimation and quantification of impacts along the entire mining value chain. The valuation and evaluation of impacts such as these typically requires a multi-disciplinary team of professionals, an option which is not available at this early stage of concept development. Using the expected impacts and benefits as a guideline, and data from fieldwork and laboratory testwork as a basis, a comprehensive framework for defining, valuing, and evaluating this approach in comparison to a more conventional exploitation approach has been developed and is presented in this Chapter.

5.2 Evaluation Methodology

Earlier studies conducted under this research programme utilized a spreadsheet-based approach incorporating fixed grade, tonnage and metal prices with cost savings arising from pre-concentration estimated by proportional adjustment of the activity-based costs (Bamber, 2005). However, this method did not include for the evaluation of capital cost impacts, or other indirect impacts of the suggested approach on the feasibility of mineral extraction. Parametric estimation and evaluation methods are suggested as a means of overcoming this limitation for the purposes of this thesis. The evaluation and valuation of the extraction and recovery of a mineral deposit is a special class of investment analysis. Mining projects are capital intensive and are characterized by high fixed costs, high risk (geological risk, political risk, market risk, operating risk), and an irreversible depletion in the major asset – the resource (Gentry & O'Neil, 1984; 12). Evaluation requires a methodology for quantifying and estimating impacts to these variables. Most variations in methodology relate to geostatistical methods used in determining the resource, cost estimating methodologies and the method by which the project value is determined. Such methods do exist, but are presently manual, paper based, of limited applicability and accuracy, and lack detail particularly in the estimation of capital costs for mine, surface plant and infrastructure. The economics of the project are largely determined by the price and cost regime at the time of evaluation, coupled with some forecast of how input costs and market prices will vary over time. Project evaluation can be by replacement cost, total sales value, earnings based (NPV) or by option pricing techniques such as CAPM (Barnett & Siorentino, 1994). The basic deposit evaluation methodology broadly incorporates the following features:

- Input of basic deposit parameters such as ore reserve, metal grades, geotechnical and geographic factors
- Automated calculation of capital costs and activity-based working costs for the pre-concentration scenario based on a user-input operations flowsheet
- Adjustment of the overall mineral extraction and metallurgical recovery based on the 2-product formula
- Re-calculation of the cutoff grade based on revised working costs and recoveries
- Re-calculation of the size and grade of the mineral reserve based on an idealized grade-tonnage model for the deposit
- Calculation of the Net Present Value
- Calculation of the comparative asset utilization efficiency between the proposed approach and the base case as a tool for the final evaluation of the concept

The inputs for the model are a data set which must be developed through site visits, sampling, and the results of the geo-metallurgical evaluation procedure described in Chapter 4. Additional data comprises the size and grade, and thus the value of the deposit; the selected mining rate, documented operating costs, data from existing milling operations; financial parameters such as metal prices, discount rates, and any smelter tolls or additional taxes which may be levied. The system cost estimate is factorized based on data from the literature and capital estimates from previous studies. Operating costs are activity based and are recalculated according to the degree of waste rejection achieved in pre-concentration and the location of the pre-concentration step in the overall value chain. Revenue losses are calculated according to the recovery data for the pre-concentration process and metallurgical performance at the existing mill. Recovery improvements due to the calculated increase in feed grade to the mill are also accounted for by means of the two-product formula. The result of the analysis is used to recalculate the cutoff grade for the deposit, which in turn is used to re-evaluate the size and grade of the economic mineral reserve to be exploited. The parametric project evaluation process for a given deposit is thus an iterative process as depicted in Figure 5.1 (after Gentry & O'Neil, 1984; 4).

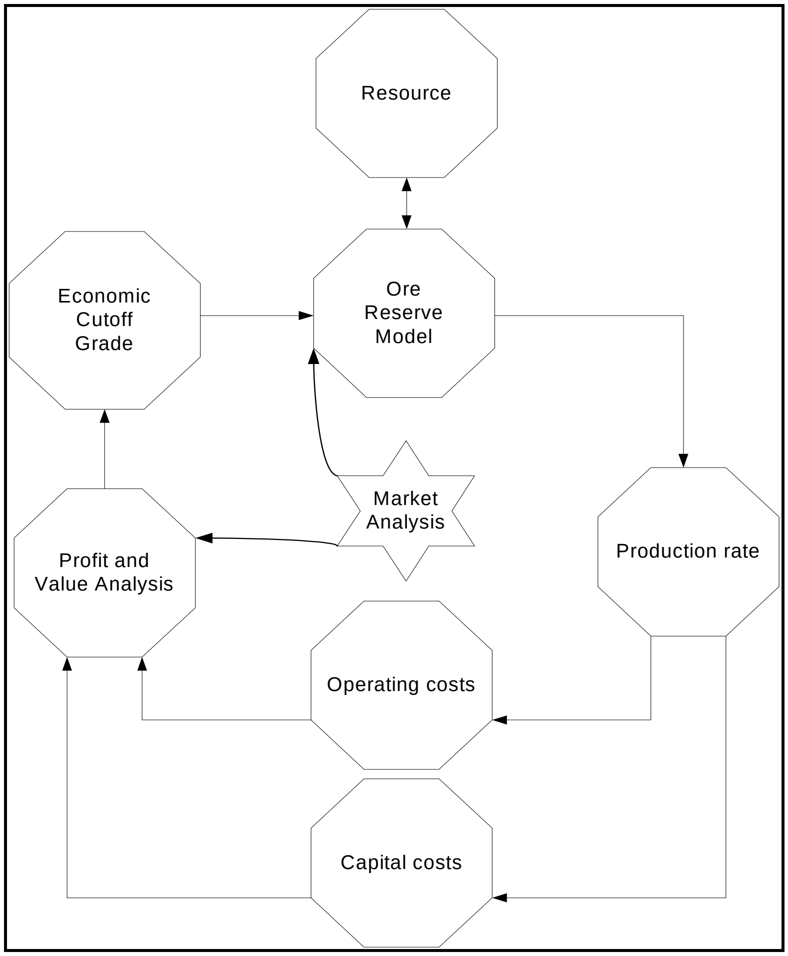


Figure 5.1: Mineral Deposit Evaluation Flowchart

A basic data set for mineral deposit evaluation is proposed for the purposes of the evaluation as follows:

1. **Ore reserve –** the size of the ore reserve to be mined in tonnes
2. **Mining rate –** the mining rate in tonnes per day and production profile if relevant
3. **Operating days/annum –** ($0 < x < 365$)
4. **Metals of Interest –** copper, nickel etc.
5. **Metal grades –** contained metal grades in % or in grammes/tonne as the case may be
6. **Mill recoveries –** existing or predicted % metal recovery in the surface mill
7. **Metal prices –** current metal prices in \$/tonne or \$/oz as the case may be
8. **Operating costs –** activity-based breakdown of the total (fixed + variable) operating costs of the mine (drill, blast, muck, haul, hoist, surface haul, mill and tailings)
9. **Capital costs:**
   - Mine development, mill development, infrastructure and pre-concentration capital
10. **Processing parameters**
    10.1. Throughput
    10.2. Recovery
    10.3. Concentrate specs

11. **Pre-concentration parameters** (projected or derived from metallurgical testwork)
    - **Waste rejection –** the percentage of total ROM ore (including planned and unplanned dilution) rejected in the pre-concentration step.
    - **Metal recoveries –** recovery in % feed content (by element) achieved in the pre-concentration process

12. External factors

- Geography – location, elevation, surface conditions, geotechnics, climate

13. **Financial parameters**
    - Smelting toll - % toll (including metal losses) levied on concentrate feed to the smelter
    - Discount rate – expected rate of return on investment / risk factor
    - Smelter tolls, penalties and recoveries
    - Technological factors
    - Market analysis – a price forecasting function is included
    - Source & cost of capital
    - Extent of pre-production design, development & construction
    - Shipping, Marketing, Freight & taxes
    - Expansion options
14. **Other factors**
    - Geographical parameters – location (urban, rural, remote, arctic)
    - Geotechnical parameters (impacts the evaluation of surface, hangingwall and footwall conditions) – good, fair or poor

Aspects such as environmental, legal & permitting, and exploration or production drilling costs are assumed to be sunk and are not accounted for. Data from the data set is used to configure the model, and evaluate a decision to invest or not to invest in the pre-concentration option. The model is sensitive to the date and relevant market conditions under which it is run, and evaluation results may change as these factors change over time.

The model evaluates the NPV of a particular ore reserve, and compares it to the projected NPV for the exploitation of a similar deposit with the implementation of pre-concentration on surface or underground. The model has been configured for copper, nickel, cobalt and three platinum group elements (3 PGE's) – platinum, palladium and gold - although other metals can be considered. The spreadsheet is protected for input of only the critical parameters. All variables are in S.I. units, and costs are in present-value US dollars.

5.3 Model Assumptions

It is suggested that the pre-concentration of ore, combined with the appropriate disposal of the waste rejects underground as backfill can address a number of the core areas impacting the operating and capital costs of a mine. A cursory examination of previous study results indicates that up to 55% of the ore can be rejected through sorting or dense media separation techniques at a metal recovery of 97%, with a resultant increase of 100% in the grade of the ore delivered to surface (Bamber et al, 2004). The rejects are suitable for disposal underground in a suitably designed rocky paste fill (Bamber, 2005; Kuganathan & Shepherd, 2002). The model assumes a degree of impact on based on the expected impacts of these physical aspects. Pre-concentration rejects a significant proportion of the ROM tonnes mined from the ore at high metallurgical recoveries, indicating a potential business case if savings can be derived therefrom. From previous scoping studies and evaluations, cost savings through underground pre-concentration are derived in a number of ways:

- potential savings in mining costs are derived through an increase in productivity at the mining face through a reduction in selectivity required, the facilitation of more productive mining equipment at the face and thus a potential increase in the minimum mining width allowable and a consequent decrease in mining costs (Bamber et al, 2005). For the model, mining productivity is assumed to improve by 10%.
- savings in fill costs are possible through the generation of a large quantity of cheap backfill material close to the stopes as a by-product of the pre-concentration process. The balance of the backfilling cost constitutes the batching of waste rejects with water, sand and cement, and costs to place the cemented backfill. Cost impacts such as a potential reduction in rockbursts or greater stope- and drift reliability through improved ground control are not accounted for here.
- savings in hoisting are directly proportional to the proportion of waste rejected. The total tonnage reporting to the shaft is reduced, shaft utilization and therefore operating time is reduced, possibly by up to 1 shift for waste rejection > 30%.
- hoisting and material handling constitutes more than 65% of all energy usage in underground mines. A reduction of 60% in the quantity of material to be hauled, hoisted to surface and delivered to the concentrator results in an estimated 30% savings in the overall power cost for the operation. Additional power savings are derived from the

elimination of a large quantity of hard siliceous waste in the feed, with a consequent reduction in grinding effort, as well as overall tons to be processed in the mill.

- up to 60% of the ore has been rejected prior to surface processing. This results in lower unit cost/ROM t for processing and tailings disposal. Savings due to a reduction in Bond Work index and thus grinding effort relating to the reduction in tons reporting to the mill, and the reduction in BWI are thus accounted for in the model.
- Additional operating costs – underground pre-concentration constitutes an additional processing cost, which must be accounted for. The metallurgical processes under consideration are typically cheap, high capacity coarse-particle processes such as sorting, dense media separation, and coarse particle grinding and flotation, for which good estimates of operating costs on surface exist. In previous studies, pre-concentration operating costs have been estimated at an additional 2.5% of the base cash cost. This is included in the evaluation. The cost of waste disposal is accounted for within the residual backfill cost as described above.

Previous investigation indicates that the increased volume of broken ore over in-situ material places a constraint on the volume of waste material that can be practically disposed of underground (Bamber, 2004). Bulking factor for broken ore can vary between 40 – 60%, thus it is calculated that the maximum quantity of ROM ore that could be returned to the mining void is 60% of the in-situ material and thus the model is only valid up to this limit.

5.4 Cost and Revenue Impacts

5.4.1 Operating cost Impacts

The operating cost of a mine and mill can be broken down into several categories of cost:

- Mine: drilling, blasting, loading, haulage, hoisting
- Surface transport
- Mill: ore receiving, crushing, grinding, concentration, tailings disposal

Costs such as labour, supervision, power, water, reagents, consumables and general services are accounted for but included proportionally into the activity-based costs. Raw operating costs included in the estimate generally include the following:

- Power
- Water
- Labour (incl supervision)

- Consumables
- Maintenance & spares

Power is a function of the total installed capacity of the plant in question, and can be estimated using load and diversity factors as appropriate. Water is generally drawn from groundwater and is accounted for in pumping and storage costs. Labour and supervision is generally proportional to the size of the plant and can be factorised from the throughput. Consumables and maintenance (incl spares) are allowed for at 3.5% of the estimated capital required for the plant (see section 4). Operating costs, while thus influenced by the value of the initial investment, are influenced principally by the following factors (Baxter & Parks, 1957:142):

- Orebody geometry and depth – increasing depth increases unit costs by the square of the depth
- Mining method and mining rate – affects the total cost of production
- Grade – affects the cost of production per unit metal
- Geotechnical factors
    - Improvement in hangingwall and footwall conditions decreases costs
    - Increase in hardness of ore increases labour, energy, steel & explosive costs
- Processing method required (ore dressing, flotation, leaching, SX/EW)
- Level of mechanization
- Infrastructural issues (power, water, roads)

It is generally accepted that increasing the mining rate decreases the unit cost of production by spreading fixed costs over a larger tonnage base. However, increasing the mining rate inappropriately for a given deposit can cause an increase in variable unit costs, which may be masked by the overall impact of increasing production. Several factors limit a simplistic increase in the mining rate:

- Financial constraints (development capital)
- Geological, geometric, geotechnical and geographical constraints
- Mining method
- Technological constraints
- Raw labour supply
- Supervision and logistical constraints on labour
- Shaft, hoist & haulage capacity
- Ventilation, heating, cooling and pumping capacity
- Concentrator and smelter capacity
- Market constraints

Over and above simply increasing the mining rate, mining companies have a number of other more immediate means to lower costs in the face of tougher competition or market conditions.

Such methods include improving productivities through increased mechanization, improving process efficiencies either through improved recoveries or waste management, temporary capitalization of operating costs, and limiting or even suspending maintenance activities. However, some of these techniques, particularly the capitalization of costs, and the suspension of maintenance activities can actually increase unit costs in the long term.

It is assumed that the impacts as described above impact proportionally on the operating cost for that process area. The total cost of an operation per ROM tonne can be defined in terms of the sum of the individual operating costs:

$$C = \frac{\sum_{1}^{n} c_i t_i}{T} \quad - \ (1)$$

Where

C= total operating cost in \$/ROM tonne and
T = ROM tonnage
c_i = unit cost of a single process
t = tonnage reporting to the process
n = number of processes in value chain

Thus, for the mining value chain selected for the model:

$$C_{Total} = c_{drill}t_{drill} + c_{blast}t_{blast} + c_{muck}t_{muck} + c_{haul}t_{haul} + c_{hoist}t_{hoist} + c_{haul}t_{haul} + c_{mill}t_{mill} + c_{tails}t_{tails}$$

And, for a given unit process:

$$c_1t_1 < c_1t_2$$

Where $t_1 < t_2$

and c_i does not vary significantly over variations in tonnage to the process. In such a relation, costs can be reduced in two ways. The unit cost of the process can be reduced, or alternately the tonnage reporting to the process can be reduced. If the tonnage reporting to a process is reduced in terms of ROM tonnes, the total cost in terms of \$/ROM tonne will be reduced. Conservatism is built into the model by the use of a cost adjustment factor, where:

$$c_2 = \left[\frac{t_2}{t_1}\right]^n c_1 \quad - \ (2)$$

And n is a value from 0-1. In this way, adjustment can be made for the impact of varying fixed operating costs, and small variations in the cost/t over variations in tonnage. For this model, n is set to 0.5 which accounts for the unit cost varying by the square root of the tonnage.

5.4.2 Estimation of Capital Costs for Hard Rock Mines

Capital costs for mining are dependent primarily on the size, depth, and grade of the deposit and estimation typically requires the establishment of a basic system design and manual and precise establishment of at least the mechanical cost component of the system. However, several parametric capital cost estimation methods are found in the literature for use in such cases. Two models that were identified for the estimation of mining and processing costs are the O'Hara method for underground deposits (O'Hara, 1980), and the model for small underground deposits developed by CANMET (Mular et al 1986). While these methods also include factors for the estimation of surface plant, including pre-concentration type plant, and surface infrastructure, they are considered inaccurate due to capacity and time issues. In the light of this, a more detailed surface plant and infrastructure capital estimation model was developed for use in the research using estimate factors derived from previous capital projects conducted by the author.

5.4.2.1 Estimation of Open Pit Mining Costs

The O'Hara method (1980), which is intended for open pit operations has been adapted for use in the open pit model. The estimated cost is dependent on the capacity of the mine and is of the general form:

$$C = \Sigma C_i Tp^n \qquad - (3)$$

Where

C = total capital cost

C_i = unit cost factor for item

Tp = Throughput in ore tons

n = scaling factor

The overall capital cost for the pit capital can be estimated using the basic O'Hara method as:

$$C = 600Tp^{0.7} + 5000Tp^{0.5} \qquad - (4)$$

However, this function is considered inaccurate, and more resolution on the costs was desired. O'Hara indicates that this overall cost factor can be broken down into:

Pre-production stripping of the pit:

$$C = 8500Tp^{0.5} \qquad - (5)$$

The cost of haul trucks:

$$C = \frac{9000t0.85(0.2Tp^{0.8})}{t} \qquad - (6)$$

Where C = cost of truck; t = capacity of haul truck

The cost of shovels/scoops is:

$$C = \frac{230000s^{0.73}(0.0525Tp^{0.8})}{S} \qquad - (7)$$

Where: S = capacity of scoop in m^3

Installation costs for overall surface infrastructure can be calculated as

$$C= 30000T^{0.6} \qquad - (8)$$

5.4.2.2 Estimation of Basic Underground Mining Costs

A more detailed method for the estimation of underground mining costs is presented by Mular et al (1986) in CANMET's 'Estimating the Pre-production and Production Costs for Small Underground Deposits'. The model can be used to predict the capital and operating costs of mining and processing based on the geological, geotechnical and geographical setting of the orebody. Similarly, the total cost of exploitation, thus cutoff grade and ore reserve can be calculated directly from this parametric estimating system. Historical data from a range of mining and processing projects undertaken by J.S. Redpath prior to 1986 have been used to develop the model. However, there are a few shortcomings - the estimation model is mining-focused and estimation parameters for plant and surface infrastructure are bulk numbers only, and more sophisticated methods are required. Furthermore, the model is manual and paper-based, comprising tables of values, functions and graphical data, base dated 1986, and is also only valid for mining rates up to 500tpd. The CANMET capital estimation model has been simplified and computerized and used as a basis for this parametric estimation model. The factors presented are for underground development only as a superior plant and infrastructure cost estimation model has been developed through previous work.

Table 5.1 Cost Estimating Factors for 500tpd Underground Mine (1986 US Dollars)

			Quantity	Unit	Cost US$
Feasibility, Reserve Estimation & Design					
	Feasibility & design		25%		$217,391
	Environmental & permitting		10%		$144,927
	Surface Drilling		15000	m	$706,521
	Cover Drilling		3600	m	$117,391
	Assay		150	ea	$ 1,304
Preliminary & General					
	Roads		20	km	$ 1,811,594
	Bridge		1	ea	$ 72,463
	Site clearance		20000	m2	$ 163,043
	Infrastructure & Services		1	ea	$ 57,971
	Power line (33kV; 2200kW)		50	km	$ 1,086,956
	HT substation		1	ea	$ 144,927
	Compressed air		2	m3/s	$ 181,159
	Main office & store		1	ea	$ 166,666
Mine access					
	Preliminary & general		1	ea	$ 192,028
	Shaft collar		1	ea	$ 235,507
	Shaft & equipping		1500	m	$ 8,152,173
	Shaft stations		4	ea	$ 217,391
	Loading pocket		1	ea	$ 65,217
	Shaft bottom		1	ea	$ 10,869
	Hoist (2.44m 1000kW)		1	ea	$ 554,347
	Headgear		1	ea	$ 253,623
	Loading bin		1	ea	$199,275
	Conveyances		4	ea	$ 202,898
U/G services					
	Ventilation		1	lot	$ 25,362
	Heating		1	lot	$ 32,608
	Vent raises (1.8m dia)		1500	m	$ 1,347,826
	Sumps & pumps		1	lot	$ 57,971
	Rockbreaker & grizzly		1	lot	$ 68,840
	U/G substation		1	ea	$ 26,811
	Miscellaneous		1	lot	$ 18,115
Pre-production development					
	4x3m drifts		308.6111	m	$ 322,028
	Orepasses (2.2m dia)		200	m	$ 191,304
	Ore pass controls		1	lot	$ 14,492
Mining equipment					
	Mechanised Mining Suite		1	lot	$ 688,405
Subtotal					$ 21,209,565
Contingency			10%		$ 2,120,956
Total					**$ 23,330,521**

5.4.2.3 Estimation of Plant & Infrastructure Capital Costs

Using the O'Hara method capital costs for the process plant can potentially be estimated from capacity factors:

Table 5.2: Capacity Based Factors for Process Plant Capital Cost

Crushing and screening	$45000T^{0.5}$	
DMS / Sorting	$45000T^{0.5}$*	
Grinding and Flotation	$72000T^{0.5}$*	*Estimated
Concentrator building	$30000T^{0.5}F_{cl}$	Where F_{cl} = 1 – mild climate/conditions 1.8 – poor conditions 2.5 severe / arctic conditions

Outputs of the O'Hara Method can be used directly to estimate costs, thus calculate the economic cutoff grade and ultimately the reserve or pit limits as the case may be (Akbari et al, 2005). In Mine Investment Analysis, Gentry & O'Neil (1984) present more detailed capital cost parameters for a range of operations where:

$$C = \Sigma ct \qquad - (10)$$

Where C = total capital cost
c = unit capital cost
t = throughput in tpd

And c is taken from a table of values from historical operations. Typically c depends on the nature of the ore and the metal extraction method:

Table 5.3– Process Plant Capacity Cost Factors (after Gentry & O'Neil, 1984)

Type of Operation	**C ($/tpd – 1984 terms)**
SX/EW	10 000
Cyanide leach	9 000
Bulk sulphide flotation	8 000
Dense Media Separation	5 000

However, these methods lack accuracy and more detail is required. In order to overcome these drawbacks, estimates for plant and infrastructure capital from recent studies by the author hae been used to both calibrate the model estimates as well as providemore detail in the estimation of surface plant and infrastructure. Previous estimates used in the calibration are presented in Table 5.4.

Table 5.4 – Previous Estimates of Mine & Mill Capital

Project	**Location**	**Base Date**	**Through-put**	**Mine Capital**	**Mill Capital**
Rhino Minerals	Richards Bay SA	1996	750		US$7m
Dwarsrivier	Steelpoort, SA	1999	3100	US$15m	US$20m
Kroondal	Rustenburg, SA	2000	5000		US$27m
Mimosa	Zimbabwe	2001	3000	US$35m	US$27m
Morula	Steelpoort, SA	2002	10000		US$100m
Voskhod	Khromtau, KZ	2006	3000	US$66m	US$36m

A detailed cost breakdown for plant & surface infrastructure for the typical 3000 tpd scenario is presented in Table 5.5.

Table 5.5 – Cost estimation factors, 3000 tpd pre-concentration, grinding and flotation plant 2006 US Dollars.

Area	**Estimate US$**	**% of Total**
Crushing & Screening	**$4,491,469**	8.5
DMS Plant	**$5,505,164**	10.4
Product Handling	**$2,893,412**	5.5
Tailings Handling	**$995,975**	1.89
Milling, flotation, reagents and concentrate handling	**$6,185,000**	11.7
Plant Conveyors	**$2,576,918**	4.9
Plant services	**$6,205,670**	1.18
Plant piping system	**$1,320,551**	2.50
Plant power system	**$5,087,907**	9.64
Freight and transport	**$1,256,102**	2.38
Site preparation, bulk earthworks, roads and terraces	**$2,484,116**	4.7
Infrastructure & Buildings	**$4,865,665**	9.22
Electrical buildings	**$693,295**	1.31
Engineering procurement and construction	**$1,1903,690**	22.5
First fill & Spares	**$654,080**	1.24
Contingency	**$1,256,848**	2.38
Total	**$52 791 398**	**100**

5.4.2.4 Adjustment of Capital Cost Estimates for Variations in Mine Capacity and Cost Escalation with Time

The above basic estimation parameters are either static in terms of the time value of money, or static in terms of the capacity of the asset for which costs are estimated. Hence the capital estimates produced by the model must be adjusted for 1) variations in throughput and 2) the time lapsed between the configuration of the model and the time of use of the model. For this, two methods are used. For capacity variations, the industry-standard exponential capacity adjustment method is used where the capital cost and date of estimation/construction of a previous operation is known in some detail:

$$\frac{C_a}{C_b} = \left(\frac{t_a}{t_b}\right)^n \qquad - (11)$$

Where C_a = cost of operation a
C_b = cost of operation b
T_a = capacity of operation a
T_b = capacity of operation b

And $0.5 < n < 0.9$ is an exponential scaling factor, typically 0.67 for most operations. This method has the advantage of discriminating between mining, plant and infrastructural capital, where the total capital required for an operation is:

$$C_m + C_p + C_i \qquad - (12)$$

And $C_{m2} = C_{m1}(t_2/t_1)^{0.67}$ etc.

The base date for the factors in the O'Hara pit model is 1980, thus estimates must be adjusted for accuracy using an average escalation factor of 3% for every year between 1980, i.e. O'Hara estimation factors have been adjusted by an overall factor of 2.22 to bring the estimate into 2006 terms. The base date for the CANMET underground estimate is 1986 and a similar adjustment is made for this base date. The revised plant and infrastructure parameters are based-dated 2003. The model then accounts for variations in the future value of money between 2006 and the present by means of a user-input escalation value.

Impacts to overall capital costs are in proportion to the change in tonnage reporting to that process area in accordance with equation (11).

5.4.3 Revenue Impacts

5.4.3.1 Impacts on Recovery

The impact on revenue of the pre-concentration step is derived from the impact on the recovery of metals in the entire process. Metallurgical recovery is defined as the percentage metal recovered from a particular metallurgical process. Non-recovered metal reports to tailings. Thus in a surface mill, for a given metal, the material balance according to the 2-product formula is as follows:

$$m_f g_f = m_c g_c - m_t g_t \qquad - (13)$$

And

$$m_f = m_c + m_t \qquad - (14)$$

Where

$m_{f,}\ g_f$ – mass and grade of feed
$m_{t,}\ g_t$ – mass and grade of concentrate
$m_{t,}\ g_t$ – mass and grade of tails

Recovery across the plant is the ratio of metal in the concentrate to metal in the feed, thus:

$$R = \frac{m_c g_c}{m_f g_f} \qquad -(15)$$

Pre-concentration introduces an additional recovery penalty on the beneficiation process in accordance with equation (15). There are also inherent metal losses to be accounted for in the performance of the existing mill. However, for a given mill, metallurgical recovery would improve with improving feed grade based on the assumption of a constant grade of tailings. This assumption is borne out in several places in the literature, and has been included in the model in order to evaluate the impact on overall metal recovery of the pre-concentration step. Thus actual overall recovery is $R_t = R_p R'_m$, where R'_m is the improved recovery in the mill based on the new feed grade (Figure 5.2):

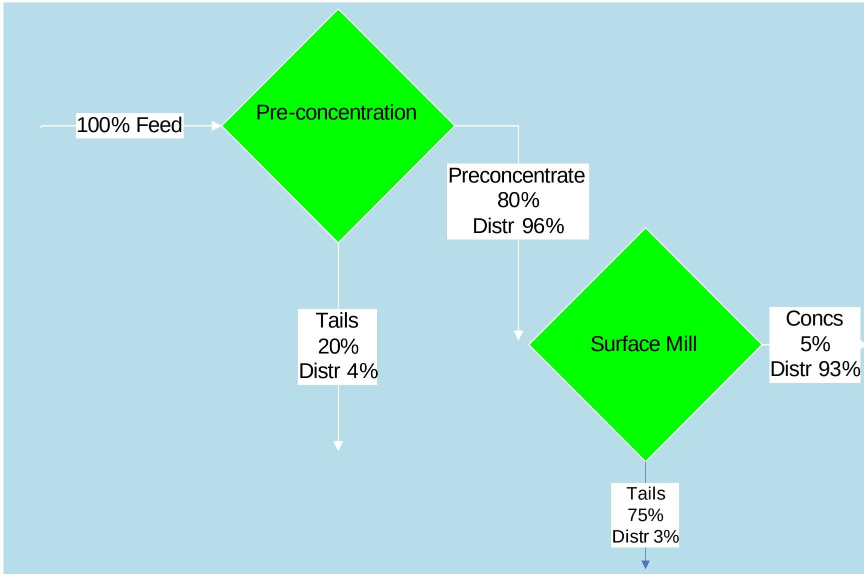


Figure 5.2 Synergistic Impact of Pre-concentration Step on Overall Metal Recovery

Other user-input models are found in the literature, for example the variable recovery model for Musselwhite Mine presented by Blower & Kiernan (2003), that can be substituted for the 2-product equation. The phenomenon can be represented in an equation of different form where:

$$R = X - \frac{(Y + 100*g_t)}{g_f} \qquad - (16)$$

Where

X = maximum achievable recovery in the plant
Y = recovery factor (shape of grade/recovery curve)

And

gt and gf are feed and tails grade in g/t as defined previously

Substituting for values in the equation gives the same general form as the 2-product equation as shown in Figure 5.3:

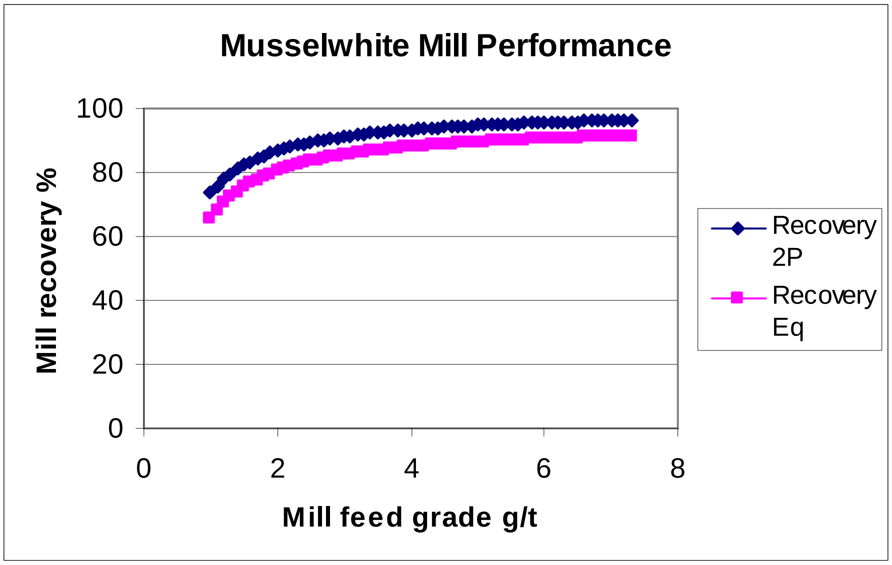


Figure 5.3 - Variations in plant recovery with varying feed grade

Different models are indicated where the ore is low grade, highly diluted, or contains additional deleterious elements to flotation or leaching such as talc or silica. CVRD-INCO's Thompson Mill in Thompson, Manitoba, accounts for mill recovery in terms of the feed grade according to the relation (Penswick, 2007):

$$R = 93\% - (16.1/g)*96.1\% \qquad - (17)$$

A variation in the feed grade of 0.7 – 1.1% Ni thus results in a recovery variation of 71% - 79% respectively. This relation also follows the general form of the 2-product formula presented in (13), however the variation of mill recovery with feed grade is greater than in the 2 previous models. The variation of mill recovery with feed grade has been built into the model as a general case.

Metal losses in smelting are to be accounted for; however, smelting tolls generally include for the cost of smelting and refining as well as metal losses (which are usually negligible in terms of ROM grades and tonnages), and are paid on the basis of contained metal in the concentrate. Thus, on the basis of a 20% toll smelting fee, the actual recovery penalty from pre-concentration is of the order of 80% of that calculated in terms of the raw recovery, which positively offsets the raw cost savings (as calculated in 5.2 above) against the value of lost metal. Values for smelting losses as well as toll fees and NSR can be input into the model.

5.4.3.2 Variation in Metal Price

Variations may also occur in the evaluation based on variations in the metal price. This can be accounted for by dynamic metal price functions which will allow the user to use variable metal pricing models throughout the life of mine. Generally, commodity markets in general, and base metal prices in particular show a long term decline in real price* (Krautkraemer, 1998; Slade, 1982). However, metal markets, and thus prices fluctuate continually in the short term based on market perceptions, demand cycles, inventory levels and producer responses. The market cycle has inherent delay and inertia resulting in often confusing trends, and difficulty in prediction. Precious metals, and certain base metals sometimes respond rapidly to external influences, and while present market conditions indicate an extended bull run for the general suite of commodities, present commodity price levels would be reasonably expected to decline at some point in the future. Short term mineral commodity cycles are thus a real and present issue at this time of historic high metal prices, and previous cyclicality indicates that a downturn at some stage in the near future would be expected over a life of mine of 20 years, and thus consideration of this factor could be considered important in the evaluation.

In order to illustrate this, Figure 5.4 (opposite page) shows a depiction of a general commodity market cycle with a description of salient features (after Baxter & Parks, 1957; 139). In the low cycle, the industry is characterized by several key behaviours – costs are often expensed to capital, management and labour effort is high, and innovation is required. In the high cycle, commodity prices are high, profits are maximized and management and labour becomes lax – the pressure to innovate is low; however, input costs are also high, thus unit profits are lower than at other times, thus innovation can still be required to maintain profit margins even in times of boom. Several simplistic models including price constant, constant decrease, constant increase can be considered. More complex models are fully dynamic models of a particular commodity market, involving interactions of commodity price, producer capacity and production cost inventory and market demand.

* http://tin.er.usgs.gov/mrds/

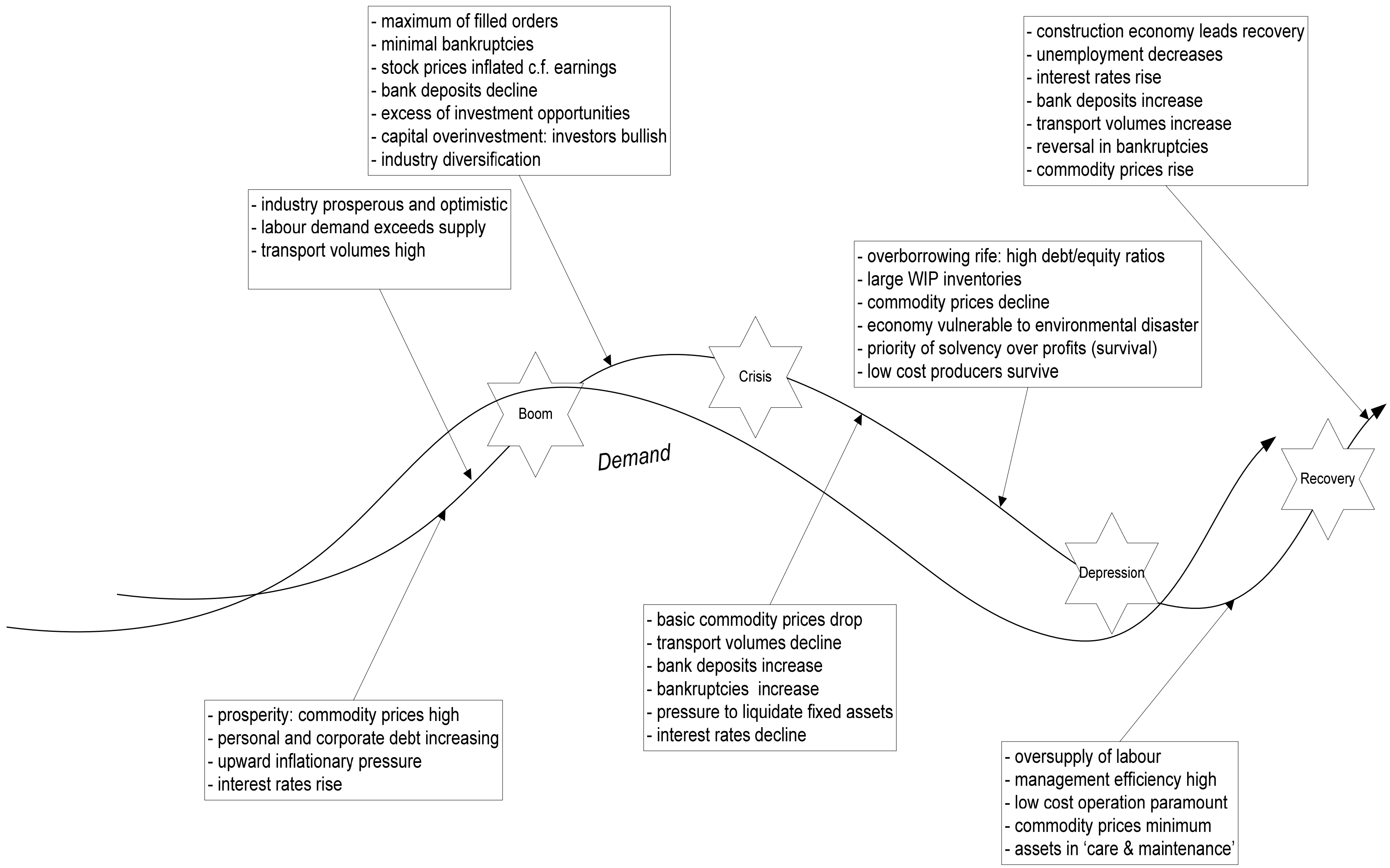


Figure 5.4 - Characteristics of the mineral commodity cycle (after Baxter & Parks 1957)

Such a model has been constructed using the Stella^{TM} Systems Analysis (Bamber & Dunbar, 2005; O'Regan & Moles, 2001) to evaluate these dynamics over the life of a typical mineral project. The model generally indicated sinusoidal price dynamics as indicated by the observations of Baxter & Parks (1957). Production costs appear to track increases in commodity price with a small delay, indicating a relatively constant long term cost-to-metal price ratio, thus in the parametric model the metal price and cost margin is assumed to remain constant over the evaluation period.

5.5 Impact of Cost and Revenue Variations on the Cutoff Grade, Mineral Reserve and Mineral Resource

The impact on the economic cutoff grade of a deposit with pre-concentration is assumed to vary in direct proportion to the impacts on working cost and the overall recovery of the valuable mineral. An important measure of cutoff grade is the measure of the quantity of the valuable mineral extracted to metal vs. the total amount of metal contained in the resource, the extraction. This is a very important measure of sustainability for mineral resource exploitation, and calculation of this is considered an important outcome of the model. Firstly the ore resource must be defined and quantified for a given cutoff grade. Secondly the cutoff grade for the calculation of the ore reserve is defined, and the ore reserve estimated. Estimation of the ore reserve allows for validation and/or recalculation of the relevant cutoff grades.

5.5.1 Estimation of the Cutoff Grade and Ore Reserve

Ore resource and reserve estimation are essentially estimates of the probability of occurrence of the ore, coupled with an estimate of the economic feasibility of extracting the ore, to increasing levels of confidence. In Canada, National Instrument 43-101 gives definitions for the physical and economic definition of mineral resource probabilities (Jensen & Bateman, 1981; 4). A mineral resource is the definition of probability of occurrence of the ore, and can be classified as proven or probable. The mineral reserve is the portion of the mineral resource which is considered economically feasible to extract, and can be classified (in order of decreasing confidence) measured, indicated or inferred. Inferred reserves are portions of the probable resource which may be economic to extract.

Assuming that a drilling programme has commenced, and that a resource has been defined, the first step in valuing and evaluating a mineral property is to establish parameters for the size and grade of the ore reserve. An ore is composed of several components which make up the total volume of material to be mined. The metal and associated elements such as sulphur make up the mineral fraction. The mineral fraction plus the gangue minerals such as silica make up the ore itself (Jensen & Bateman, 1981;41). A metal may be associated with many minerals (eg chalcopyrite, chalcocite), or conversely many metals can be present in a single mineral fraction e.g. stannite. The ore reserve is that fraction of the ore resource which is above cutoff grade and can be economically extracted. For the model, the cutoff grade relation selected follows the general form (Wheeler & Rodrigues, 2002):

$$gc = \frac{ci_{var} + (Ci_{fixed} + Copp)/T}{Price \times R} \qquad - (18)$$

Where

Ci_{var} - variable operating costs/t
Ci_{fixed} – total fixed costs/annum
C_{opp} – opportunity cost in \$
T – ROM tonnage/mill throughput
Price – metal price in \$ per unit mass
R – overall metal recovery

For an operation with a low proportion of variable to fixed cost, and a low opportunity to total cost ratio, this reduces to:

$$G_c = \frac{(F+V)}{PR} \qquad -(19)$$

Where

G - grade in g/t or %
F - the fixed cash cost /t
V - the variable cash cost/t
P - metal price in \$/g, \$/kg as appropriate
R - overall metal recovery

The cutoff grade thus varies in direct proportion with the magnitude of cost saving and the impact on overall metal recovery (Wheeler & Rodrigues, 2002). The tonnage factor is used to establish the total tons of rock which must be extracted to obtain the metal value in the ore. This is the mineral resource.

Table 5.6 – Ore Tonnage Estimation Factors

<table>
<tr><td>SG mineral</td><td rowspan="2">Ore SG ‘tight’</td><td rowspan="4">Rock SG</td><td rowspan="5">Ore bulk SG</td></tr>
<tr><td>SG gangue</td></tr>
<tr><td>+ porosity (%)</td><td rowspan="2"></td></tr>
<tr><td>+ moisture (%)</td></tr>
<tr><td>xBulking factor (%)</td><td></td><td></td></tr>
</table>

We must thus define the overall tonnage factor in order to define the mineral resource:

$$TF = \frac{\text{Ore SG}}{\text{Rock SG}} \qquad \text{- (20)}$$

The rock has a certain average metal content, which is the grade of the economic ore. The total metal contained is the mass of the resource multiplied by the grade of the deposit. There is a certain basic cost of extracting and treating one ton of the resource into a saleable product. This cost will determine the basic cutoff grade for determining the resource. The proportion of the resource above cutoff grade which can then be profitably mined under varying market conditions is the mineral reserve. It must be noted that not all ore in the reserve can be feasibly extracted. Ore is often sterilized by geological features such as faults, dykes and 'potholes' – pinching of the width of the ore to uneconomic widths between drill holes. Geological losses can be as high as 15% of the reserve tonnage. Ore can also be sterilized through the mining method chosen – ore is lost in pillars left in the stope for roof support, shaft and sill pillars, as well as losses arising from constraints in the layout of ramps and drifts. Mining losses can be as low as 10% for open stoping methods and as high as 20% and more for room-and-pillar methods. Once the mineral reserve is defined, the total amount of ore that can be extracted from the reserve must be estimated from geological and mining factors.

For the purposes of estimating the reserve, the level of dilution of the ore must also be defined. As previously mentioned, a certain proportion of gangue minerals are already included in the definition of ore. Additional gangue material may be inadvertently mined from sloughing of the hangingwall and the footwall, thus the grade of ore defined in the reserve model becomes diluted through actual mining of the ore. Thus the total amount of dilution in the deposit is:

$$\partial = \left[\frac{twaste}{twaste + tore}\right] \times 100\% \quad \text{- (21)}$$

However, dilution may simply be ore in the resource that is below the cutoff grade of the reserve, and thus still contains metal, the recovery of which must be accounted for in the economic analysis. The loss of metal in fines in the ore must also be accounted for. Often the percentage dilution is overestimated to compensate for this, which is erroneous. Alternately, companies account for this directly with factors such as the 'Mine Call Factor' (per Anglo American) or 'Mill Cuts' (per INCO) to account for this loss in metal, which is considered a more correct approach than discounting these losses. An example of a complete ore block definition for the South Deep Mine based on the above discussion is presented in Table 5.7 for clarity:

Table 5.7: Parameters for Hypothetical 30 000m^3 Gold Ore Reserve

Variable	**Unit**	**Value**
Volume of ore (from orebody model)	m3	30 000
Ore SG	t/m3	3.1
Tonnage factor	%	10
Rock SG	t/m3	2.9
Ore grade	g/t	7.7
Cutoff grade	g/t	3.6
Metal price	$/g	15
Mineral reserve	t	87 000
Total metal content	g	669 900
Total metal value	$	10 048 500
Geological losses	10%	
Mining losses	10%	
Total reserve tonnage		70 470
Value of ore reserve block	$	8 139 285
F/W + H/W Dilution	%	20
Waste grade	g/t	2.9
Total tons mined from block	t	84 564
Grade of block	g/t	6.9
Mine Call Factor	%	5
Head grade of ore	g/t	6.555
Value of ore to surface	$	8 314 755

Let us assume that 50% of the total ore mined can be rejected to waste at a recovery of 95% in the pre-concentration plant. The following assumptions pertain:

- The impact on economic cutoff grade is proportional to the decrease in operating costs
- The decrease in ore reserve grade arising from a reduction in cutoff grade is smaller than the decrease in cutoff grade
- The proportional increase in size of mineral reserve is expected to be greater than the decrease in cutoff grade
- More ore is thus extracted from the overall resource
- Gangue material included in the definition of ore, as well as gangue material included in the definition of dilution will be rejected
- Metal values contained in the dilution will be retained

- Loss of metal in fines usually lost through conventional mining and handling of the ore to surface are expected to decrease

With pre-concentration the tonnage of ore delivered to surface from the block is reduced to 42 282t, at an increased head grade of 13.66 g/t. Cutoff grade is now the grade of the waste rejects from the pre-concentrator. The size of the reserve block is increased to include material previously defined as waste and thus, the overall extraction of the mineral resource, as well as the recovery of metal values from the mineral reserve is expected to increase with the application of underground pre-concentration. Cutoff grade thus varies in direct proportion with the magnitude of cost saving and the impact on overall metal recovery. This value is calculated in the model and used to calculate the increase in ore reserve for the purpose of revision of the life-of-mine estimate based on an idealized grade-tonnage curve for massive sulphide deposits, although variations to the idealized grade-tonnage curve can also be accounted for in the model.

5.5.2 Cost-Cutoff Grade Interactions

It has been shown in the testwork that pre-concentration can typically reject up to 60% of the ROM ore by mass at recoveries in excess of 95% (Bamber et al, 2006; Munro et al 1999, Schena et al, 1999). This can be undertaken on surface with significant impacts, however undertaking such pre-concentration underground facilitates the use of higher productivity mining methods and also rejects a substantial amount of material prior to hauling and hoisting and milling with significantly increased impacts and benefits. Metal recoveries are maintained through high recoveries achieved in the pre-concentration step, as well as increases in the recovery at the mill arising from the increase in the feed grade. Based on previous research and testwork, the model indicates that costs can thus be reduced by between 20-30% through a pre-concentration of the ore 30-60% by mass. From the basic relationships presented previously in the Chapter, we can examine the impact of such a change in operating cost and recovery on the cutoff grade and size of a number of real deposits for which we have data. Mining is typically high in capital intensity, thus fixed operating costs are high and necessitate high production rates to sustain the high cost of fixed capital. The combination of fixed costs, variable costs and the unit value of the ore give an indication of the mining rate at which an orebody will be profitable (break-even):

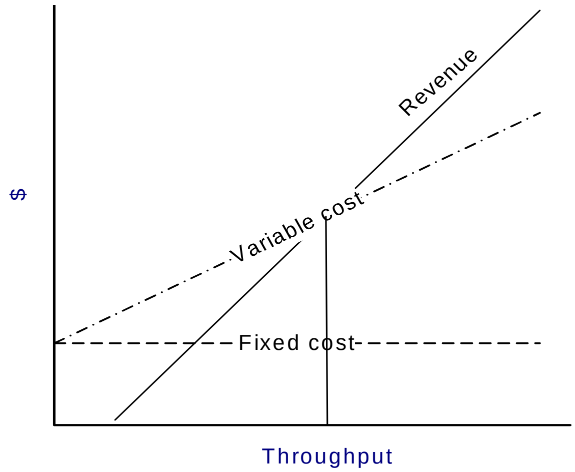


Figure 5.5 – Break-even Production Rate

An increase in fixed costs can penalize an operating mine by increasing the margin of loss for no production, and raising the minimum tonnage at which the mine breaks even. Similarly an increase in variable costs also raises the minimum tonnage at which a mine breaks even.

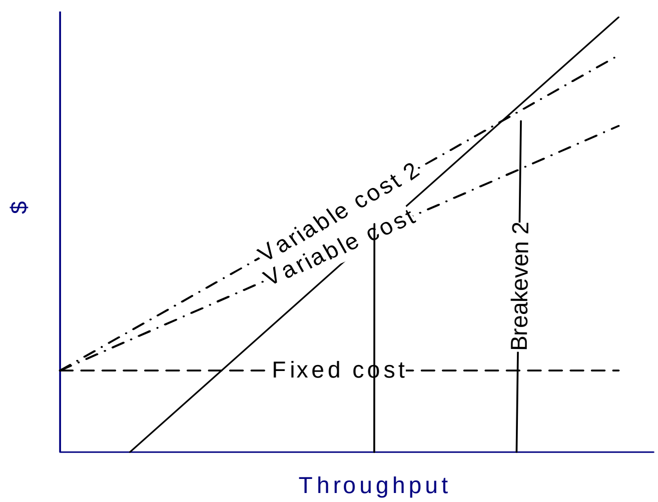


Figure 5.6 – Change in Breakeven Production Rate

A combined decrease in fixed and variable costs can result in substantial operating improvements at the mine, lowering the operating loss during periods of no production, lowering the tonnage at which the mine breaks even, and increasing the profit margin per ton of production. Pre-concentration of the ore underground has been shown to lower variable costs directly as well as reducing fixed costs through a reduction in shaft and surface capital requirements for a mine of the same production rate. Impacts on the cost profile of the mine are significant:

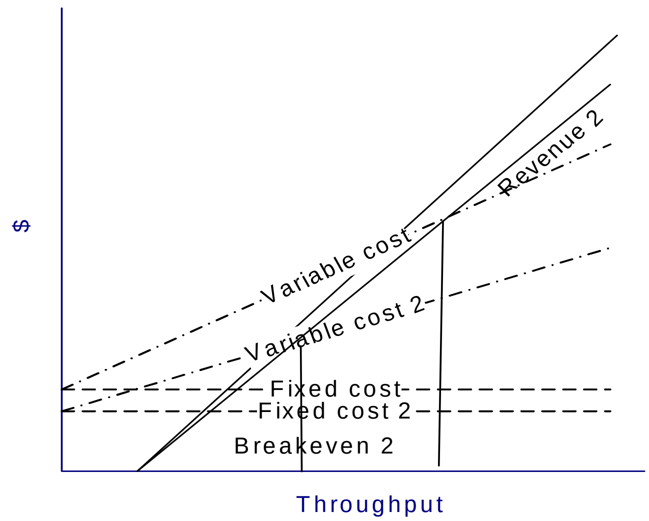


Figure 5.7 – Impact of change in Fixed and Variable costs on Breakeven Rate

The impact of lowering the cutoff grade reduces the break even mining rate as well as increasing the profit margin for all tons produced above the break even point, and is thus projected to have a massive impact on the exploitation of a mineral deposit.

5.5.3 Cutoff Grade and Grade-Tonnage Interactions for an Idealised Mineral Resource

Consider a generic mineral deposit of particular mineralogy, metal grade, grade distribution and arbitrary tonnage (after Lacy, 1969):

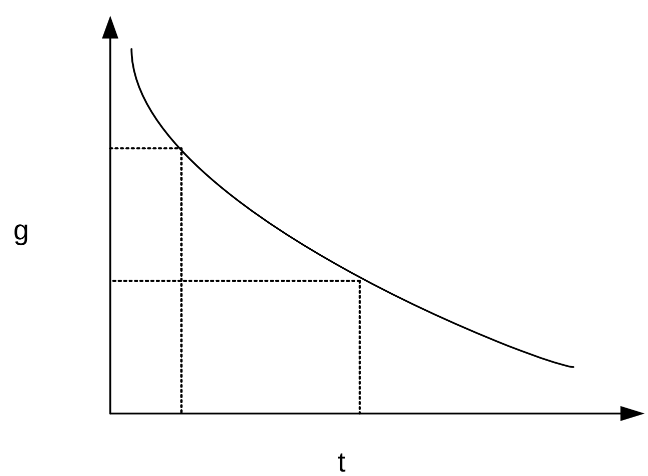


Figure 5.8 – Idealized Grade Tonnage Curve

The curve has several interesting features for analysis. It indicates that small resources are generally high grade and that the converse is also the case. An alternative indication, or co-observation is that there are a very low number of small, high grade deposits an increasing number of large, low grade deposits, and an almost infinite number of mineral deposits at zero economic cutoff. The curve is also exponential, indicating that a unit decrease in cutoff grade represents a disproportionate increase in the mineral resource:

$$\frac{t_a}{t_b} > \frac{g_a}{g_b} \qquad - (22)$$

Similar interactions would be seen for an economic mineral reserve. As cutoff grade is proportional to the operating costs for the mine, it can be seen that, all other things being equal, a unit decrease in operating costs will lead to a substantial increase in tonnage, and therefore mine life (Clark & Harper, 2000). Grade-tonnage data from the literature can be used to construct a generic grade tonnage curve in order to confirm this model. Figure 5.9 is a graph of grade-tonnage data published for the Navidad Hill Resource in Nevada as at March 2006 (Patterson, 2005):

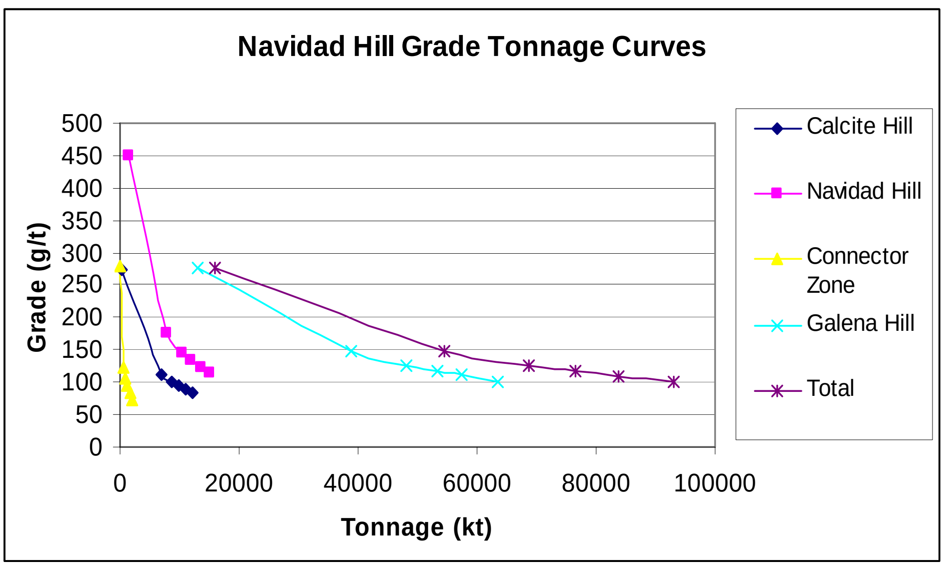


Figure 5.9 – Real Grade-Tonnage Relations for the Navidad Hill Deposit

Regression analysis of the data indicated that the grade tonnage curve can been fitted to an exponential equation of the form:

$$y = Ge^{-tx} \qquad - (23)$$

Where y = the grade of the deposit for a given tonnage; G = the maximum estimated grade within the resource; and t' = the slope of the grade/tonnage curve where t' ~ 1 x 10^{-4} for the smaller, high grade Connector deposit and t'~1x10^{-6} for the larger low grade Galena Hill deposit. A high correlation ($R^2 > 0.96$) between actual data and the model is indicated. Similarly, corroborating data from several global base- and precious metal deposits has been taken from the literature and statistically analyzed and is presented in figure 5.10 (Jensen & Bateman, 1989; Singer, 1995):

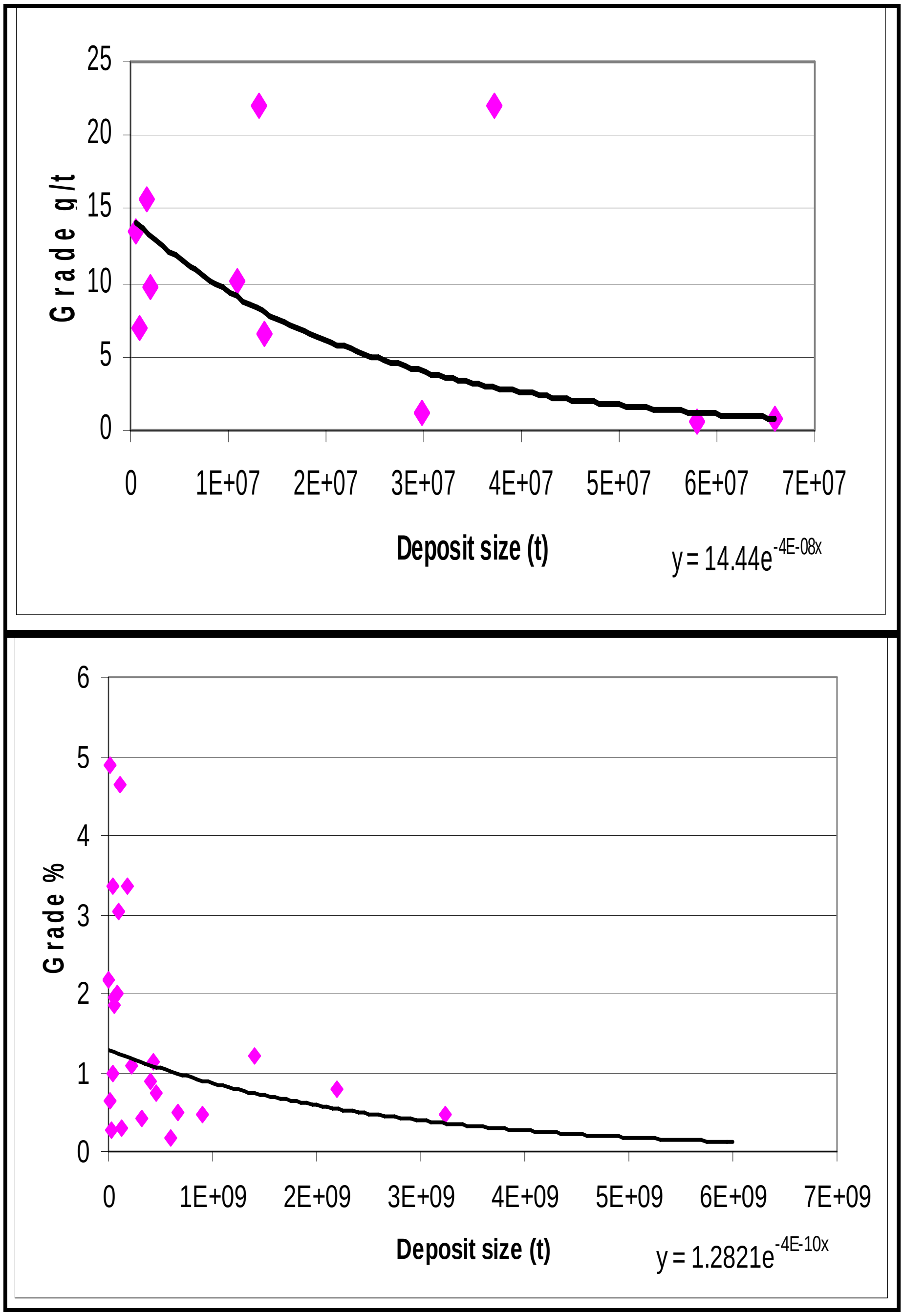


Figure 5.10 – Grade/Tonnage Curves for Selected Global Gold (above) and Base Metal Deposits (below)

Again, the data fits an exponential model of similar form, where t' ~ 10^{-7} for the gold deposits and t'~10^{-9} for base metal deposits. Using such a generic model, the impact of changing cutoff grade on reserve tonnages can be estimated for any given resource given a single data point for resource size and cutoff grade. At this stage of development, this simple grade-tonnage model for large low grade and small high grade deposits is used for the model. The combined impact of a projected operating cost reduction is shown on the generic grade-tonnage curves thus developed:

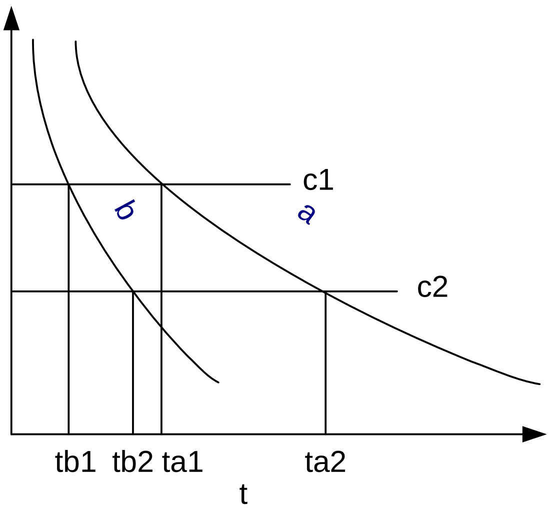


Figure 5.11 – Cost Impacts on Grade and Tonnage Curve

Curve *a* is for a large, low grade disseminated orebody with an exponent t of 10^{-9}. A decrease in cutoff grade resulting through a decrease in operating costs massively increases the mineral reserve. Curve b is a smaller, high grade deposit with an exponent of 10^{-7}. Increases in the mineral reserve are less significant; however, the operating margin is substantially increased, massively increasing the present value of the orebody. Using this mechanism the impact cutoff grade of the ore reserve by means of these curves is evaluated in the parametric model. Scaling factors and exponents are user-adjustable in order to explore different grade-tonnage scenarios for improved accuracy.

5.6 Impact Valuation and Evaluation Methodology

The flowchart for the parametric evaluation model developed for the research thus has several key elements. Basic data for the deposit to be evaluated is obtained and input into the model. Operating parameters for the present mine and mill if pertinent are also input. Results from the pre-concentration testwork are input to evaluate the impact of pre-concentration versus the base case. Activity–based operating costs are input by the user. Capital costs, and impacts to capital and operating costs for the base- and pre-concentration scenario based on the degree of waste rejection are automatically evaluated. Revenue impacts are calculated as described, and overall impacts on the size of the mineral reserve, life of mine, and Net Present Value of the deposit are automatically calculated. It is assumed that variables such as depth, grade, etc. of a given mine are catered for within the cost structure of a given mine i.e. extreme depth translates into a commensurately high hoisting cost component in the input model. The cost saving is translated into an equivalent cutoff grade which is used to calculate the increased ore reserve. Mining rate is considered constant for the purposes of the model, thus the life of mine is extended. An alternative model would increase the mining rate to accommodate the increased reserve, although this would impact negatively on the sustainability of exploiting the deposit. A flowsheet of the valuation and evaluation process incorporated in the model is presented in Figure 5.12.

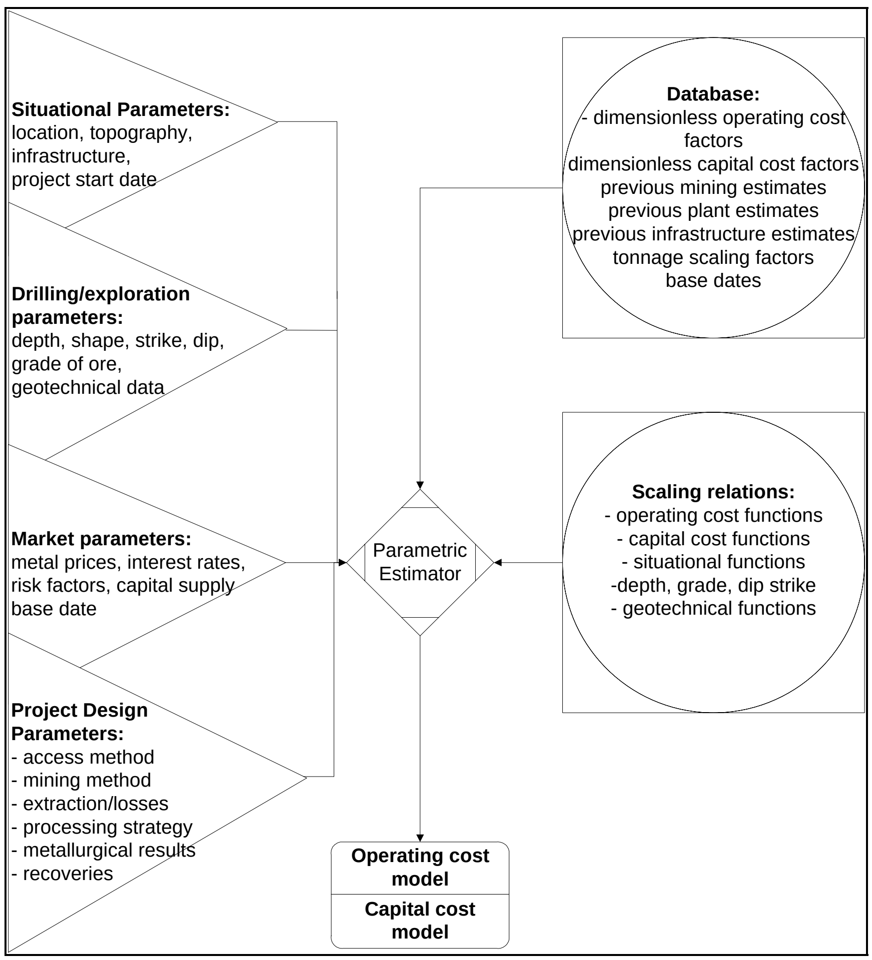


Figure 5.12 – Preliminary Parametric Valuation and Evaluation Flowchart

As has been indicated, the preliminary size, grade and thus value of the ore deposit has now been established, and the capital required to initiate mining and processing and the operating costs to continue mining and processing have been estimated. The value of the ore is typically determined in US$/t, and operating costs, which can be broken down in to fixed and variable plus distribution, sales and marketing costs and a contingency, typically also in US$/t. The overall cash flow for a mining operation, which is the basis of all the succeeding evaluations can be determined from basic accounting principles as shown in Table 5.8. As can be seen 8from the table, the net operating cash flow is influenced by every aspect of the operation, from ore value, to process efficiency, the nature of toll smelting agreements as well as initial and ongoing capital requirements for the mine.

Table 5.8 – Net Operating Cash Flow Breakdown

	Ore value/t
Less	Recovery/losses
Times	Production rate
Equals	Revenue
Less	Royalties
Less	Operating costs
Less	Smelting toll
Less	Depreciation & amortization
Equals	EBIT
Less	Tax
Plus	Depreciation & amortization
Equals	Operating cash flow
Less	Working capital reserve
Equals	**Net Annual Cashflow**

All parameters required for the cash flow evaluation are calculated by the model and the resulting cash flow analysis on an annual basis over the life of the mine is calculated giving the basic undiscounted cash flow profile of the mine. This is used as a basis for all subsequent evaluations. The Net Present Value of the ore reserve is calculated using operating costs and metal recoveries for the conventional mining scenario. The equivalent NPV for the deposit considering the integrated mining and pre-concentration approach is calculated using the revised operating cost (calculated on the basis of the tons rejected), and the revenue impacts calculated from the recovery data for the pre-concentration process, and a capital estimate for the pre-concentration facility as described above. The estimated increase in ore reserve and thus life-of-mine is also accounted for in the calculation. Capital estimates for the mine and or shaft development, surface mill and infrastructure as well as the pre-concentration facility is calculated using the methodology described above. The NPV of the scenarios are compared parametrically to evaluate the impact of pre-concentration on the deposit.

Once the size, location, orientation and value of the deposit is established, metallurgical performance as well as capital and operating costs have been determined, it is possible to make an economic valuation of the potential mining operation that is envisaged. Mining operations are characterized by high capital intensity, long pre-production lead times, high risks (e.g. geological, political, technological and market risk) as well as an irreversible depletion of the major asset, the mineral resource. Thus the valuation of mineral deposits is a specialized form of general investment analysis which must take these particular characteristics into account (Gentry & O'Neil, 1984). There are many techniques available to do this, both historical and current, including the simple replacement cost method, market (sales) value methods and earnings-based methods: Net Present Value (NPV), Internal Rate of Return (IRR), and payback. The model incorporates a probabilistic NPV estimation based on pre-programmed variations in capital requirements, operating cost, metal price (grade) and a range of discount rates in order to produce the NPV spreads. The asset efficiency of the comparative scenarios is also examined. The economic efficiency of an asset is a measure of the comparative return on investment on the asset (ROA) to the size and capacity of the asset (Russell, 2003):

$$ROA = \frac{Net\_Operating\_Income}{Revenue} \times \frac{Revenue}{Net\_Assets} \qquad - (23)$$

And

$$\frac{Net\_Operating\_Income}{Sales} = Revenue - Costs \qquad - (24)$$

Thus

$$ROA = \frac{(Revenue - Costs) \times Sales}{Net\_Assets} \qquad -(25)$$

The measure evaluates whether the asset is performing efficiently or not. It is expected that the pre-concentration of ore underground will increase the efficiency of the mining asset as the operating margin is vastly increased for a marginal increase in asset value. From analysis of the equation, there are a number of ways to increase ROI:

- Increase revenue
- Increase production volume and/or utilization of the asset
- Decrease costs
- Reduce the asset base

The impact of the pre-concentration and waste disposal scenario on the overall cashflow is thus determined in the model. The sum of these cashflows must then be evaluated in terms of their Net Present Value (NPV) in order to determine their present value and thus decide whether an investment in the concept is to be made. A suitable method of evaluation is the

NPV ‘profile’ method, where the robustness of the project cashflows are tested against scenarios of varying capital and operating cost, metal recovery and metal price. This method highlights the benefit of small, long-term cashflows with a small initial investment as well as large initial investments with large short-term cashflows, and exposes ‘vulnerable’ projects for which the NPV rapidly diminishes with a small increase in initial investment, capital costs or a decrease in metal price or recovery (Barnett & Sorentino, 1994). It is also considered a good method of accounting for project risk by use of these risk-adjusted values for recoveries, capital and operating costs. All other project variables appear to have a second order effect on the viability of the project. A screen plot of a typical NPV ‘profile’ is presented for the Pipe II deposit (Figure 5.13). This process is undertaken for the project base case as well as the pre-concentration case and the NPV and NPV profile of the two options is be compared. Further comparative analysis of the various project investment options, and the timing of these options can be subsequently undertaken by objective analysis or by some more sophisticated method such as option-exercise pricing, or real options theory (Winsen, 1994; Samis, 1993). Details of two previously developed parametric models for a surface and underground scenario can be found in Appendix 1a and 1b.

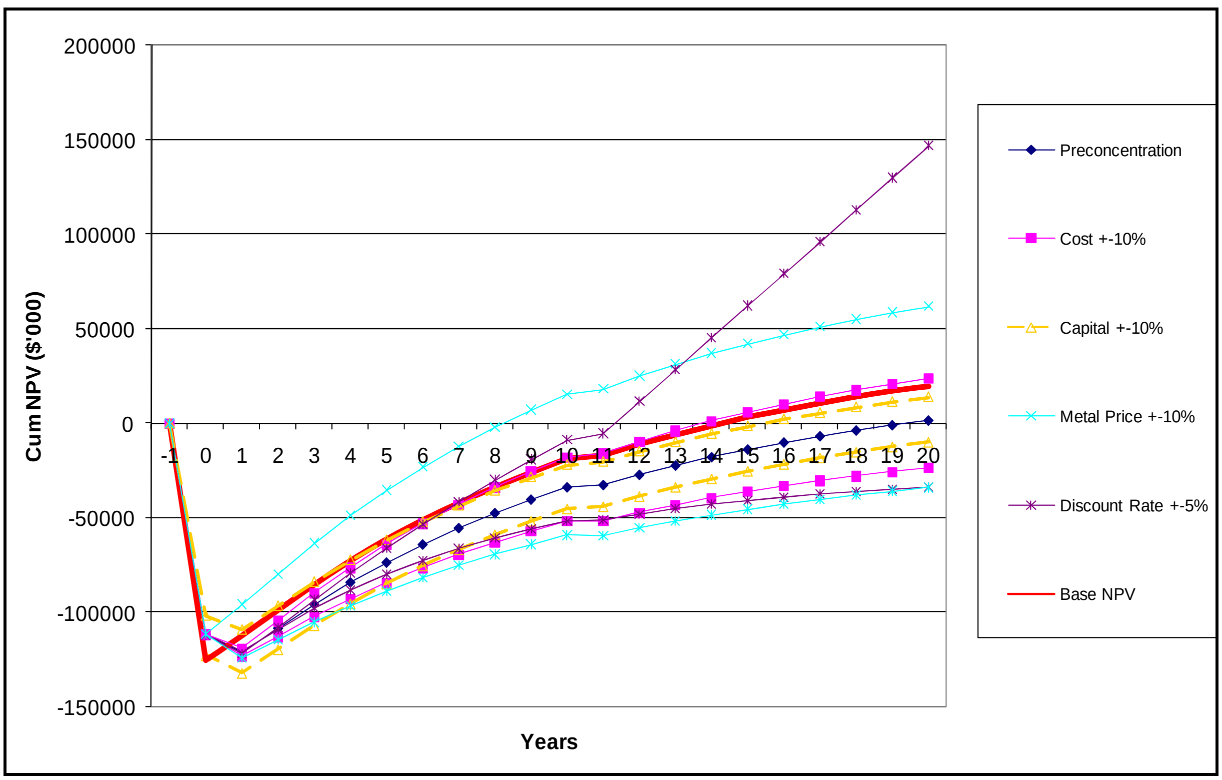


Figure 5.13 – Examination of the Robustness of the Project using the NPV Profile Method

5.7 Conclusion

A spreadsheet-based model for the financial evaluation of pre-concentration compared to a base case of no pre-concentration of the ore is presented. The size and grade of the deposit must be chosen, and a mining rate selected. Expected operational data for the deposit as well as any mill testwork and results from the pre-concentration testwork are entered as input variables on the sheet, and cost and revenue impacts are calculated in the 'Output' sheet. A capital sum for the initial mine and mill development as well as the pre-concentration facility is calculated. The net saving in operating cost due to pre-concentration is calculated, and impact on revenue through changes in metal recovery are also calculated. These outputs are used to calculate the overall impact on the cutoff grade and thus the size of the mineral reserve using a generic grade-tonnage model. The NPV and NPV profile of the two scenarios is calculated, with variations in basic parameters such as operating cost, capital requirements, metal price or grade as well as discount rate. Cost and revenue impacts are evaluated and compared to the baseline to establish a positive or negative impact overall. Different scenarios of grade, tonnage, mining rate, pre-concentrator performance, capital and operating cost structure can be explored using the model. For the case studies presented in the thesis, estimates and outcomes projected by the model have been calibrated against internal estimates by the sponsoring organizations and found to be accurate to within 30%, confirming the utility of the model in practice.

6. Integrated Mining, Pre-concentration and Waste Disposal Case Studies

6.1 Introduction

UBC has been involved with research into Mine-Mill Integration and underground pre-concentration since 2000. Fundamental and speculative papers highlighting the potential benefits of the approach were previously published (Peters et al 1999; Scoble et al 2000; Klein et al 2000, 2003). In 2001 UBC Mining entered into a research agreement with INCO Ltd with the objective of investigating appropriate technologies for pre-concentration, and considering the application of these technologies in a conceptual underground facility. Substantial work has since been completed, including an extensive literature review and the investigation of over 26 case studies for the application of pre-concentration, both on surface and underground for a variety of mining operations and mineral deposits.

Since the initial case study work was completed in 2004, research in the field has been ongoing with an expanded team of researchers: the scope of the research has been further expanded to include the development of advanced mineral processing techniques as well as the investigation of waste disposal aspects. Metallurgical and modeling results from earlier case studies at INCO's Sudbury operations have been previously published (Schindler 2001; Bamber, 2004; Bamber et al 2004, 2005, 2006; Scoble et al, 2006), however, selected results are presented in this Chapter for a more comprehensive analysis of the concept. In 2006, the initial underground pre-concentration work at INCO (now Vale) was followed up by a detailed ore sorting study for the Pipe II deposit in Thompson, Manitoba. Additional case studies have now been undertaken for Falconbridge (now Xstrata), and Placer Dome. Work at Placer Dome has not continued since their take over by Barrick Corp. However, following encouraging results, the Xstrata study was expanded to include investigation of pre-concentration and waste disposal strategies for 9 ore types. The study has now been continued with detailed investigations for two of Xstrata's mines, Onaping Depth and Nickel Rim. The scope of the research to date has thus included the sampling and testing of over 25 ores, including, inter alia: massive vein sulphides; massive and disseminated sulphides; banded sulphides; volcanogenic gold-pyrite deposits; paleo-placer gold deposits of the Witwatersrand Basin associated with both pyrite as well as uraninite; lead zinc ores of the Mississippi Valley type (MVT), as well as preliminary investigations into the pre-concentration of Cordilleran-type copper porphyry ores. Preliminary system design, configuration and layout of the generic integrated mining, processing and waste disposal system as presented in Chapter 3 has also been conducted for several of the cases, allowing a more detailed examination of the impacts and benefits of the concept for these deposits. This has enabled the evaluation of a complete suite of potential impacts and impacts and benefits of the concept in terms of mining method, capital costs, operating costs, cutoff grade, mineral reserve extraction and surface environmental footprint to be evaluated. A summary of the results and outcomes of these studies, as well as additional supporting data from the literature where relevant, are presented in this Chapter as further evidence in support of the proposed approach. Details of the case studies are presented in Appendices D-F. It is suggested from the extent of these results that the potential for the pre-concentration of ore and the subsequent disposal of the waste in the mining void is a general case and should be considered as an option for the exploitation of a deposit as early as possible in the development cycle.

6.2. Gold Ores

Gold-pyrite and Witwatersrand/Paleo-placer Deposits

An underground pre-concentration study was conducted for selected operations at Placer Dome Group in 2005 (Bamber, 2005). Operations studied were the South Deep Mine in Carletonville, South Africa, and the Musselwhite Mine in Ontario. Testwork was undertaken on selected ore samples. The initial evaluation model was developed for this project and used to estimate potential impacts and benefits at each operation based on the results of the testwork. Previous research had indicated that up to 60% of the Witwatersrand type gold ore could be rejected through coarse particle flotation and gravity concentration at a gold recovery of 98% (Lloyd, 1979). Results from testwork by Placer Dome during the project indicated that 50% of the ore by mass could be rejected at a metal recovery of 88% through radiometric sorting. A modification of the process design resulted in the ultimate rejection of 15% of the ore at 98.7% recovery (Kowalcyk, 2002). Capital cost for the sorting plant and excavations were estimated at US$90m (2005 terms). Using the evaluation model, these results translate into a decrease in operating cost from $78 to $71/t, and a commensurate lowering of cutoff grade at the mine and an increase in the ore reserve (Bamber, 2005), and an increase of $104m in the NPV of the mine over 20 years. For the Musselwhite Mine, the testwork indicates that 24% of the ore could be rejected by dense media separation at 94.6% Au recovery. An estimate for a dense media separation plant for the mine was made at US$30m (2005 terms). Modelling of the results indicate a decrease in operating cost from $61.50/t to $55.03/t; however, metal losses, at 5.4% are of the order of this cost saving, and the impact on NPV predicted by the model was marginally negative overall: Musselwhite Mine is essentially a shallow, high-grade, low cost producer, and the impact of pre-concentration was marginal for the mine, indicating a very close dependency between ore value and operating cost structure for the success of the concept (Bamber, 2005).

Meta-sedimentary gold

Testwork was been conducted on samples of gold ore from International Wayside's operations near Barkerville, BC. The material tested was near-surface, sedimentary-hosted, high talc gold-pyrite ore at nominally 14 g/t, and was not initially considered an obvious candidate for pre-concentration. However, the mine intended direct shipping the ore to an existing mill some 50km away in Quesnel and dense media separation testing on the ore at -19mm indicate that up to 48% of the ore by mass could be rejected at a gold recovery of 97.4%. A preliminary estimate of the feasibility of implementing ore pre-concentration at the mine on this type of ore was positive (Weatherwax & Gillis, 2006).

6.3 Mississippi Valley Type Lead Zinc Ores

In 2007, a preliminary study into the pre-concentration of Mississippi Valley type (MVT) lead zinc ores was conducted for the Doe Run Company in Viburnum, Missouri. A combination of hand samples, core samples and ROM samples were taken for analysis. The results of previous Dense Media Separation and basic size assay data were also obtained for the evaluation. Doe Run is situated on the Viburnum trend in SE Missouri, which is a Pb-Zn deposit hosted in the Mississippi Valley 'Bonaterre' formation. Doe Run currently operates a complex of 6 underground mines and 4 mills along the trend, with a future planned mine at the Higden deposit near Farmington, MO. Total present reserves are estimated at 60Mt @ 6% Pb, 1% Zn and 0.15% Cu at a cutoff of 5% Pb equivalent. Mining is generally by sequential room and pillar mining with the balance of production by planned extraction of the remnant pillars.

Mineralogy is generally massive galena, sphalerite and lesser occurrences of disseminated chalcopyrite in flat-lying seams hosted in dolomitic sediments of the Bonaterre formation. Pb is associated either with Zn or Cu but rarely both. Gangue minerals are $CaCO_3$, pyrite and marcasite, with a typical combined density of 2.86 (Paradis & Hannigan, 2007). Two major ore types are identified, high grade 'banded' ore comprising perhaps 20% of reserves and lower grade 'breccia' type ore comprising the balance. Higden ore is situated lower in stratigraphy than Doe Run's present operations and is hosted in weak sandstone and clays (reported 5-15%

clay). Doe Run Co. is currently forced to mine high-grade reserves including high-grade remnant pillars in order profitably meet the requirement of historical Pb supply contracts. Doe Run thus desires to increase production of reserves below the present 5% Pb eq cutoff. Options for the Doe Run expansion are:

- to change the mining method to a more selective method on lower grade sections,
- increase hoisting and milling capacity
- increase ore production by means of pre-concentration.

Furthermore, as Doe Run is situated in the Mark Twain National forest, the operations are also constrained in terms of coarse and fine tailings disposal, as well as any further surface infrastructure, hence the option of underground pre-concentration was considered attractive. Hand samples of both breccia and banded ore types were taken during the underground visit. Photographs of several stope panels, as well as close up photographs of the ore were taken for mesotextural evaluation. From these images, it appears that the galena and sphalerite mineralization is clearly visually distinguishable at the meso-textural as well as micro-textural scale (Figure 6.1, 6.2 & 6.3).

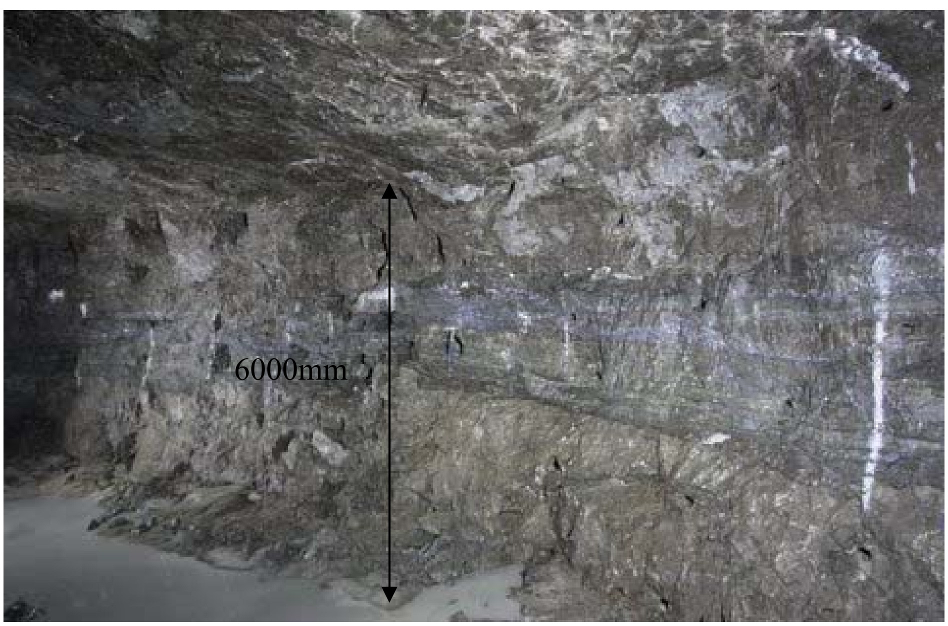


Figure 6.1 – Banded ore of the MVT type at Doe Run Company showing horizontal orientation and clear mesotextural characteristics

Figure 6.2 – Close up of breccia type ore in-situ showing sharp contact with the dolomites

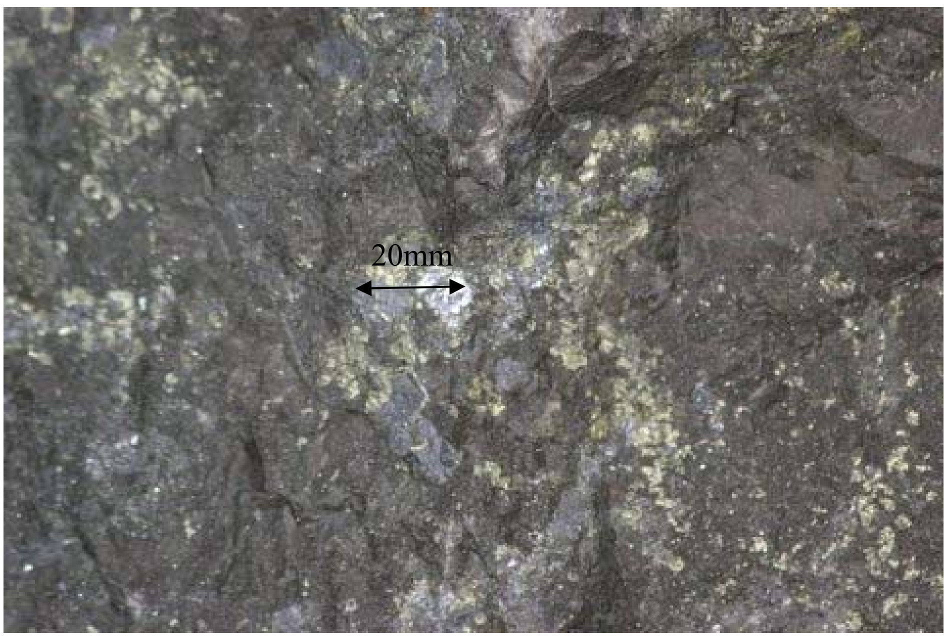


Figure 6.3 – Close-up of Doe Run breccia-type ore showing both massive and disseminated sulphides

Initial indications from previous sink-float testwork results on ore from the Buick and Viburnum properties (Jones, 2006) include a clear upgrading effect in the finer fractions, as well as good metal recovery to the sinks fraction in each case (Table 6.1). Preliminary work on a 5kg hand sample of Brushy Creek ore confirms the pre-concentration potential of this ore. Concentration by means of dense media separation as well as concentration based purely on size is indicated in the preliminary assessment, with potential rejects of between 32.7 – 64.6% of the ore by mass on average with an expected metallurgical recovery of 98% Pb.

Table 6.1 – Results of Doe Run Ore Evaluation (after Jones, 2006)

Product	Wt%	Pb %	Zn %	Cu %	Pb Wt%	Zn Wt%	Cu Wt%
Viburnum Ore	**100.00**	**3.31**	**0.50**	**0.17**	**100.00**	**100.00**	**100.00**
+9mm	72.90	1.77	0.31	0.11	38.92	45.26	47.39
-9mm	27.10	7.46	1.01	0.33	61.08	54.74	52.61
2.81 sink	64.60	5.05	0.72	0.26	98.56	93.02	98.80
Buick ore I	**100.00**	**3.89**	**2.09**	**0.01**	**100.00**	**100.00**	**100.00**
+6.7mm	88.80	2.73	1.99	0.01	62.40	84.41	66.40
-6.7mm	11.20	13.06	2.91	0.03	37.60	15.59	33.60
2.87 sink	35.40	9.84	4.62	0.02	89.55	78.25	70.80
Buick ore II	**100.00**	**2.19**	**0.40**	**0.02**	**100.00**	**100.00**	**100.00**
+13mm	65.10	1.36	0.28	0.03	40.56	45.91	82.55
-13mm	34.90	3.73	0.62	0.01	59.44	54.10	17.45
2.85 sink	67.30	3.19	0.57	0.02	98.03	95.90	67.30

Other separation options are to be examined as both sphalerite and galena are conductive minerals, with distinct photometric properties with respect to the dolomite gangue, indicating good potential for pre-concentration by both optical and conductivity methods which will be confirmed during further testwork.

6.4 Cordilleran Copper Porphyry Ores

Porphyry deposits are the world's most important source of Cu and Mo; they account for about 60 to 70% of world Cu production and more than 95% of world Mo production. Porphyry deposits are also major sources of Au, Ag, and Sn; significant by-product metals include Re, W, In, Pt, Pd, and Se (Sinclair, 2007). Copper porphyrys are a major economic deposit type in the Canada and South American Cordilleran region as well as Australia. The deposits are predominantly mined by open pit methods, however many of these mines are mature and production at many mines is scheduled to move to the block caving method; Rio Tinto predicts a predominance of Cu production by this method by 2014 (Cross, 2006), thus this is envisaged as a significant application for the integrated mining, processing and waste disposal approach. There are currently no pre-concentration plants operating on copper porphyrys, although there is some suggestion in the literature that potential for the pre-concentration of copper porphyry ores exists (Burns & Grimes, 1986). Copper porphyry ores are typically 1-2% copper and 98% gangue minerals, therefore the likelihood of a coarse, barren fraction seems high. Furthermore, while the mesotexture of a typical copper porphyry is highly disseminated by definition, the typical microtexture of the ores indicates preferential deposition of the sulphides along interstitial cracks and fractures within the ore zone (Figure 6.4).

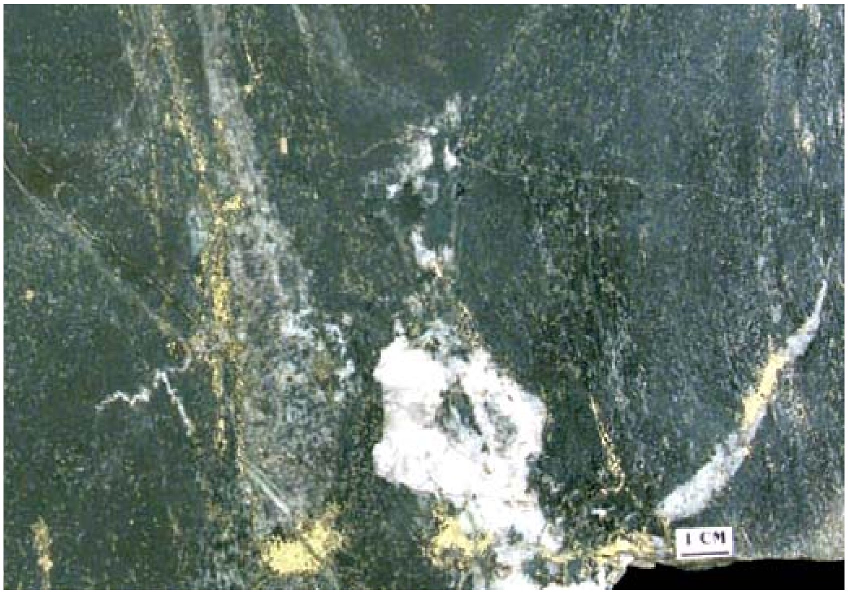
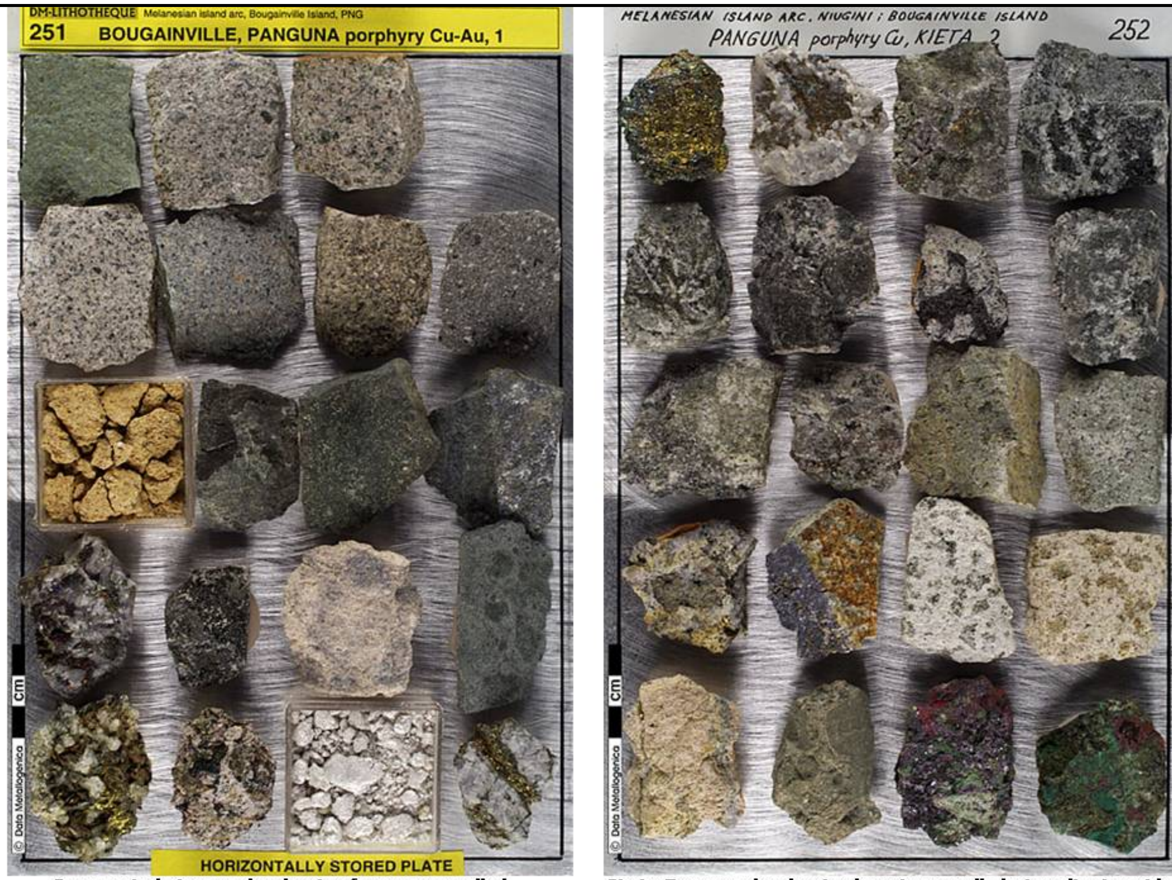


Figure 6.4 – Massive to disseminated chalcopyrite mineralization along fractures and foliations in biotite porphyry, QC, Ca (after Sinclair, 2007).

Further supergene enrichment may also occur in these ores, maximizing the potential for variable grade distributions which have been identified as characteristic of good pre-concentration potential. Possibly the best cited example in terms of this potential is from AMT Copper in Arizona (McCullough et al, 1999): bench and pilot scale metallurgical testwork was conducted at Mountain states R&D on a range of ores from the Copper Creek property in Arizona, demonstrating that, in the case of relatively coarse copper porphyry mineralization, pre-concentration by DMS was effective in improving the grade of the copper ore at a high metal recovery. Mineralization was principally porphyritic chalcopyrite, with some massive chalcopyrite, and minor mineralization of bornite and chalcopyrite in sericitic dolerite, with a nominal liberation size of 13mm. Metallurgical testwork indicated a good potential for DMS and the flowsheet was piloted. Rejects from the DMS pre-concentration step were to be crushed and prepared as backfill for the proposed mechanized cut-and-fill mining method.

Further pre-concentration potential for copper porphyry ores has been demonstrated in ores from Rio Tinto's Bougainville Mine in Papua New Guinea. It was observed during early operation of the mine that mineralization in the Panguna zone of the mine occurred mainly on fracture planes in the ore, whereas other porphyritic mineralization was highly disseminated. Sampling and testwork on 3 major ore types at the mine, Panguna Andesite, Kawerong Diorite and the biotite/diorite/granodiorite zone was undertaken in 1984 in order to assess this potential

(Burns & Grimes, 1986). Photographs of Panguna Andesite samples and biotite/granodiorite samples are shown in Figure 6.5 (after Laznicka, 1973, photos courtesy AMIRA Data Metallogenica). A significant increase in the grade of the fines fraction was noted in both the Panguna and Kawerong ores (Figure 6.5). The lower grade biotite and granodiorite zones, comprising a small portion of the mineral reserve, were significantly lower grade and did not indicate any increase in metal content in the fines fraction (Figure 6.6).



Panguna Andesite samples showing fracture-controlled mineralization (bottom left & right)

Biotite Zone samples showing breccia-controlled mineralization with variable amounts of chalcopyrite (top right)

Figure 6.5 –Panguna Andesite (L) and Biotite Zone (R) Ore Textures from Bougainville Mine (after Laznicka 1973)

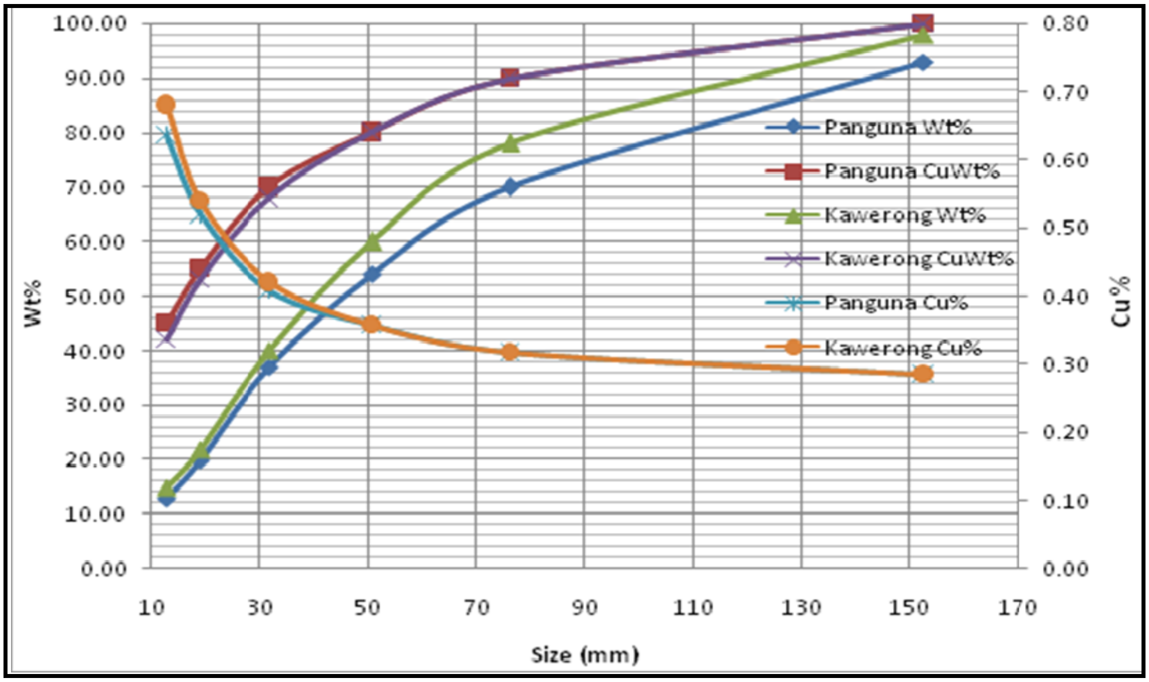


Figure 6.6 - Size / assay relationationship in Bougainville copper porphyry ores (after Burns & Grimes, 1986)

Historical cutoff grade at the mine was 0.3% copper. However, a significant upgrading effect was achieved for lower grade ore zones through crushing to nominally -150mm and screening of the ROM ore at -31.5mm (Table 6.2). An increase in metal recovery in flotation of 1% over that of the ROM ore was observed during testwork on pre-concentrated ore samples.

Table 6.2 – Bougainville Ore Pre-concentration Results

	Feed	**-31.5mm Fraction**		
Sample	**Cu%***	**Wt%**	**Cu%**	**Cu Wt%**
DC1	0.19	34.46	0.33	59.50
BS	0.29	32.46	0.58	64.49
1A	0.29	36.99	0.51	64.70
1B	0.53	50.09	0.80	76.32
2	0.52	45.63	0.85	75.75
3	0.70	68.91	0.86	84.88
4	0.31	47.01	0.43	65.36
5	0.82	63.10	1.19	91.08
PS	0.37	37.30	0.69	68.70
PN	0.39	47.40	0.57	69.50
PE	0.10	29.10	0.16	47.40
KD	0.13	36.30	0.21	56.70
Average	**0.39**	**44.06**	**0.60**	**68.70**

Based on the strength of these results, a crushing and screening plant was designed and built at Bougainville by Minenco Ltd to treat 35 Mt of ROM ore per annum at the mine. The impact on the mine was to decrease the cutoff grade to 0% Cu, and then reject uneconomic material through crushing and screening of the ROM ore at 38mm, thus increasing the extraction of the mineral reserve by 58% through the ability to mine and process previously uneconomic material (Burns & Grimes, 1986).

6.5 Footwall and Contact Type Ores of INCO's Sudbury Operations

In 2004, a preliminary study was conducted at INCO's McCreedy East Mine, in the Sudbury Basin, Ontario to determine the potential for underground pre-concentration at the mine (Bamber, 2004). McCreedy East is a mature, medium depth base metal mine with increasing haul distances, poor ground control and resultant high unit mining costs. Sudbury ores fall into three principal categories: contact ores, footwall ores and offset-dyke deposits (Peredery & Morrison, 1984). Two of the principal ore types, Footwall and Contact ores were represented at the mine and were thus sampled and tested. Contact ores are generally hosted in the ultramafic zone between the igneous complex and the transitional Sudbury breccias, and are typically rich in Ni, but poor in other metals. Contact ores are pentlandite-rich, thick and shallow dipping, with a high Ni:Cu ratio and containing complex pentlandite / pyrrhotite with chalcopyrite occurring as massive sulphides (Coats & Snajdir, 1984). Footwall ores are narrow-vein, massive sulphide stringers, situated in the Sudbury breccias down-dip of the Contact ore. Footwall stringers are rich in Cu, and total precious metals (TPMs), but poor in Ni. Stringers vary in width from 0–6m, dip variably between 20-60˚, and are mined typically using mechanized drift-and-fill methods. The veins are mostly chalcopyrite, grading about 30% copper, with secondary veins of pentlandite, millerite and occasionally bornite (Coats & Snajdir, 1984). The highly variable thickness of the veins leads to mining dilutions of up to 70%, and thus the underground pre-concentration of this ore in particular is expected to generate significant benefits. Offset dyke deposits have not been included in this evaluation.

Both ores are mined by cut and fill methods, and high levels of dilution of the narrow vein copper ore in particular was a problem at the mine. DMS testwork on the ore indicated that 55% by mass of waste could be rejected from the Footwall type ore at 97% Cu recovery. Optical sorting on core crushed to -19mm indicated that on average 44% of the ore could be rejected at 97% Cu recovery. Rejection by DMS from the Contact ore was 22% at 95% Ni recovery. Conductivity sorting was also shown to be effective on this ore (Buksa & Paventi, 2002). Waste rejects were coarse, competent breccia aggregates sized between 9mm and 75mm and were potentially identified excellent material for inclusion in a suitably designed backfill mix. An economic evaluation was undertaken based on these results, indicating a potential 20% operating cost saving at the mine with the production of direct shipping ore (>20% Cu + Ni) from the NV copper through pre-concentration. The study also indicated that the Footwall ore concentrate could meet the feed grade of a typical copper smelter, and if accepted in the smelter, a higher overall PGM recovery of 95% vs. presently 80% in the mill would be achieved. A system cost estimate was developed, and based on the estimated cost of CDN$30m, the increase in NPV was CDN$134m over the life of mine.

6.6 Polymetallic Base Metal Sulphide Ores of Xstrata Nickel's Ontario Operations

Based on the successful conclusion of previous work on the pre-concentration of Footwall- and Contact-type Sudbury deposits at INCO, Falconbridge Limited was approached in August 2004 to see if there was potential for the application of the concept at Falconbridge's Sudbury operations on similar ores. Previous discussions with Falconbridge had indicated possible applications at the Craig and Fraser Mines, however, two further sites, Thayer Lindsley Mine and the Fraser Copper Zone were identified where the technology might be applicable. The Fraser Mine is presently near the end of its ore reserves in the Fraser Copper Zone, which is a massive-stringer sulphide orebody, and thus there is little potential to implement underground pre-concentration here. However, the ore is considered representative of other ore types on the North Range, particularly the Footwall ore to be mined at Nickel Rim, and thus was to be evaluated for amenability to pre-concentration in this light. At Thayer Lindsley, the shaft was originally an exploration shaft and is currently the bottleneck to increased production. T-L mines a combination of classic contact and footwall ores as well as a low-grade area of banded pentlandites in Zone 1. Currently the mining of low grade ore is a bottleneck to revenue, and the pre-concentration of this ore in particular was identified as a means to hoist more metal to surface at Thayer Lindsley. Increased production could increase the earnings contribution of TL to Falconbridge operations and thus generate cash flow for other projects such as Nickel Rim, Fraser Morgan and Onaping Depth. Site visits were conducted over November 2004, with scoping tests for the mines undertaken on selected core samples. Results of preliminary metallurgical testwork were positive and it was decided to expand the scope of the study to include the 9 major ore types of Falconbridge's Ontario Operations.

The expanded Phase II study comprised site visits, ore sampling, metallurgical and geotechnical testwork, as well as the evaluation of the pre-concentration rejects as a source of material for backfill. Further laboratory work involved evaluation of the impact of systemic pre-concentration on mining, ore transport and centralized milling activities at the Strathcona Mill.

6.6.1 Preliminary Core Evaluations on 2 Mines

Core samples from two preliminary Xstrata mines were selected for evaluation. The Fraser Mine comprises two distinct ore bodies, Fraser Ni and Fraser Cu. Fraser Ni is a pentlandite rich 'contact' orezone typical of the Sudbury Complex, while the Fraser Copper Zone is a 'footwall' type ore comprising typically narrow-vein chalcopyrite in Sudbury breccia (Naldrett, 1984). The orebody grades at 0.53% Ni and 5.77% Cu with additional PGMs. Mineralization is typically narrow-vein stringers of chalcopyrite with secondary pentlandite, bornite and occasionally millerite (Coats & Snajidr, 1984). Veins vary greatly in orientation, and vary typically in width between 0-6m, thus dilution is high in panels with narrower veins (Figure 6.7). There is a very sharp contact between the orezone and the wall rock, and ore typically breaks off at the contact after fragmentation.

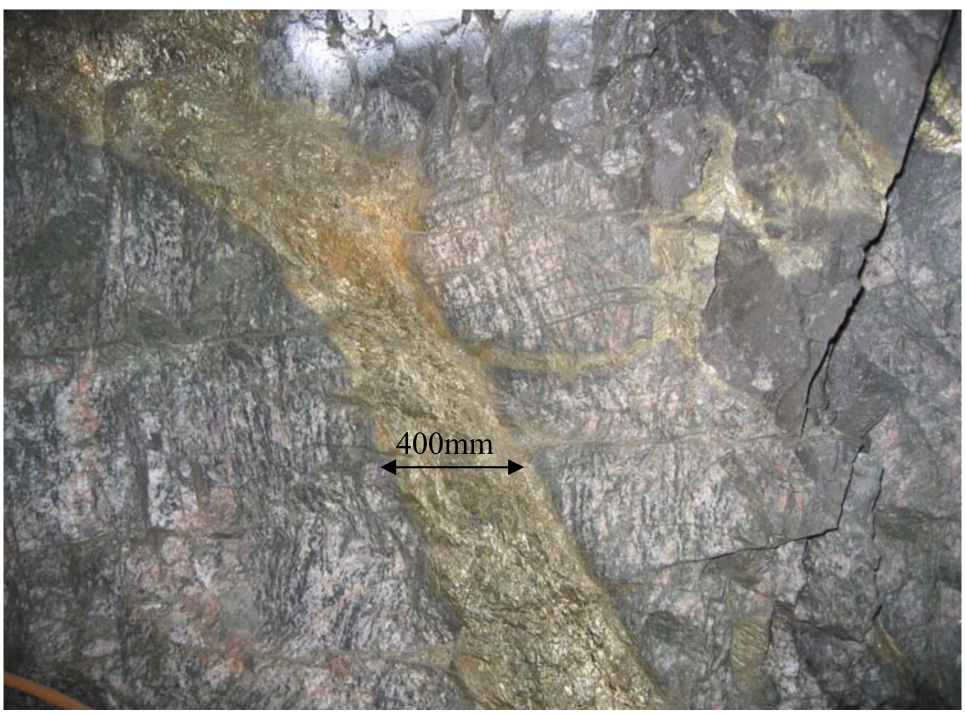


Figure 6.7 – Fraser Copper Massive Vein Sulphides

Thayer Lindsley is a copper-nickel mine situated on the Southern Rim of the Sudbury Igneous Complex, adjacent to the Murray Pluton. Present production at the mine is approximately 600,000 tpa, which is planned to be increased to 635,000 tpa through efficiency improvements

(Private communication with Falconbridge, 2005). It is planned to further increase the tonnage to a maximum of 660,000 tpa in 2007 through improved hoisting efficiencies and increased hoisting times at the shaft. It is presently felt that increasing production beyond 660 000 tpa is not possible due to constraints in terms of drilling, mucking and hauling equipment, as well as hoisting constraints. The orebody is a low-grade contact-type orebody, comprising principally banded and disseminated pentlandite in an ultramafic complex (Coats & Snajidr, 1984). Mineral reserves are estimated at 5 400 000t at an overall grade of 1.2% Cu and 1.1% Ni. Life of mine at the present mining rate of 635 000 tpa is thus currently planned to 2009. Ore zones vary in width from 4 to 30m, and thus mining presently occurs through a combination of blasthole stoping (±50%) and cut and fill methods (±50). Planned dilution is typically 24%, with unplanned sloughing sometimes contributing an additional 7% dilution in the longhole stopes. A photograph of a typical T-L cut-and-fill panel is shown in Figure 6.8. Note the orientation of the orezone and the angle of the contact in the cut and fill panel.

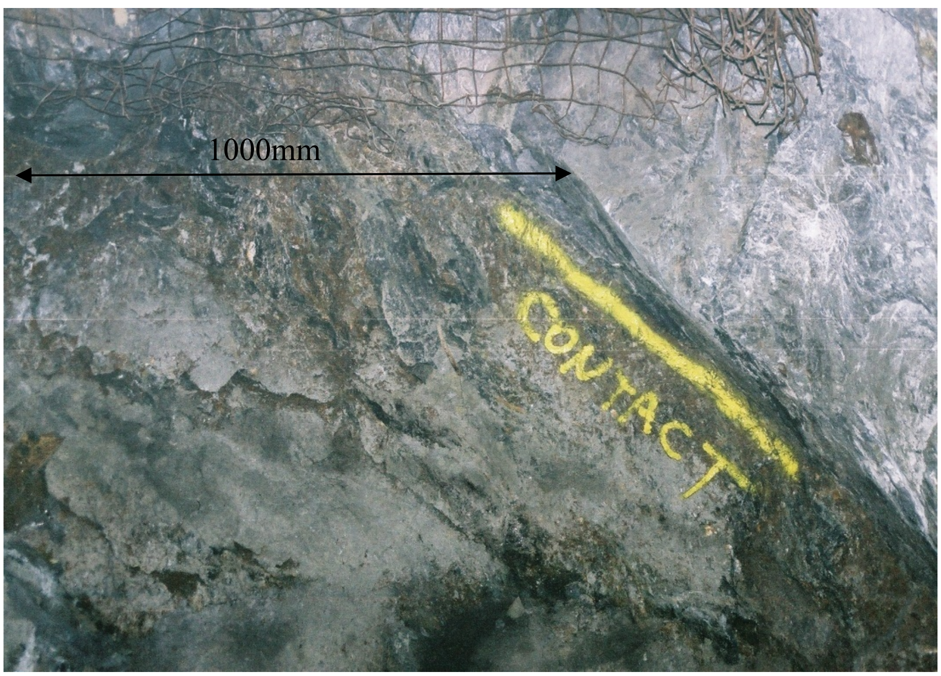


Figure 6.8 – TL Cut and Fill Panel Showing Banded Pentlandite Ore Contact

The respective core samples were photographed, weighed, evaluated visually and crushed to -19mm to provide feed for the size-assay and heavy liquid separation testwork. SGs for the ore and waste fractions were taken and washability testwork was undertaken to determine a final cut SG. -3.4mm fines were removed for inclusion with the concentrate. Initial results for the TL ore were poor, and the core was re-crushed to nominally -6.7mm to evaluate the effect of increased fragmentation of the core. -1.7mm fines were taken to concentrate. In the testwork, 44.27% by mass of the Fraser Footwall ore was be rejected at a metallurgical recovery of 97.5% Cu and 89% Ni. Ni recoveries were poor due to some dissemination of the pentlandite into the breccias, an aspect which requires further mineralogical and metallurgical evaluation. The grade of the final concentrate (sinks plus -3.4mm fines) was 20.28% Cu and 6.5% Ni which, with suitable preparation, indicates potential for feed of this ore directly to a suitable smelter. For the banded pentlandite ore form T-L, mass rejection was 14% at a metallurgical recovery of 99% Ni and 97% Cu. The –1.7mm fines represented 36% by weight and were assayed at 0.81% Cu and 0.78% Ni. The final concentrate (sinks plus -1.7mm fines) grade was 0.72% Cu and 0.72% Ni, an increase of 16.5%. It must be noted that the grade of the core sample, at 0.63% Cu and 0.61% Ni, is below the planned cutoff grade for the mine, and cannot be considered representative of the optimum waste rejection potential.

Pre-concentration of the Fraser ore was not practically considered due to the limited life of mine. However, for the TL ores, the results indicate that several strategies for increasing the metal producing capacity at the shaft are made possible. Based on the results, the mining rate could be increased to 770,000 tpa, from which 107,440 tpa of waste could be rejected through underground pre-concentration, delivering 660,000 tpa of high-grade ore to surface via the existing hoist. The fill factor would be reduced from 0.55 to 0.41 by the additional fill generated through underground processing, which would result in further unit cost savings versus the present scenario. Using such a system, the metal carrying capacity of the shaft would be increased from 7920t Cu and 7260t Ni to 8993t Cu and 8357t Ni per annum respectively. At the ruling 3-month average metal prices of US\$1.47/lb Cu and US\$7.30/lb Ni, and a US:CDN exchange rate of 1.2:1 this equates to an increase in revenue from Thayer Lindsley of approximately CDN\$25m/annum (2006 terms). Results for both mines are encouraging. However, due to the limited life of mine at these operations, it is not feasible to undertake a project for the pre-concentration of these ores. However, the results were applicable to a number of Xstrata’s present and future operations, and additional ore types were recommended to be tested. Based on the results, two flowsheets were suggested. For the high grade narrow vein copper ores, a flowsheet based on size classification and optical sorting was suggested (Figure 6.9). For the contact type ores, a process of size classification followed by dense media separation of the middlings fraction was suggested (Figure 6.10). Further metallurgical work was indicated as required in order to determine the exact flowsheet and metallurgical balance for each envisaged application for the process at Xstrata Nickel.

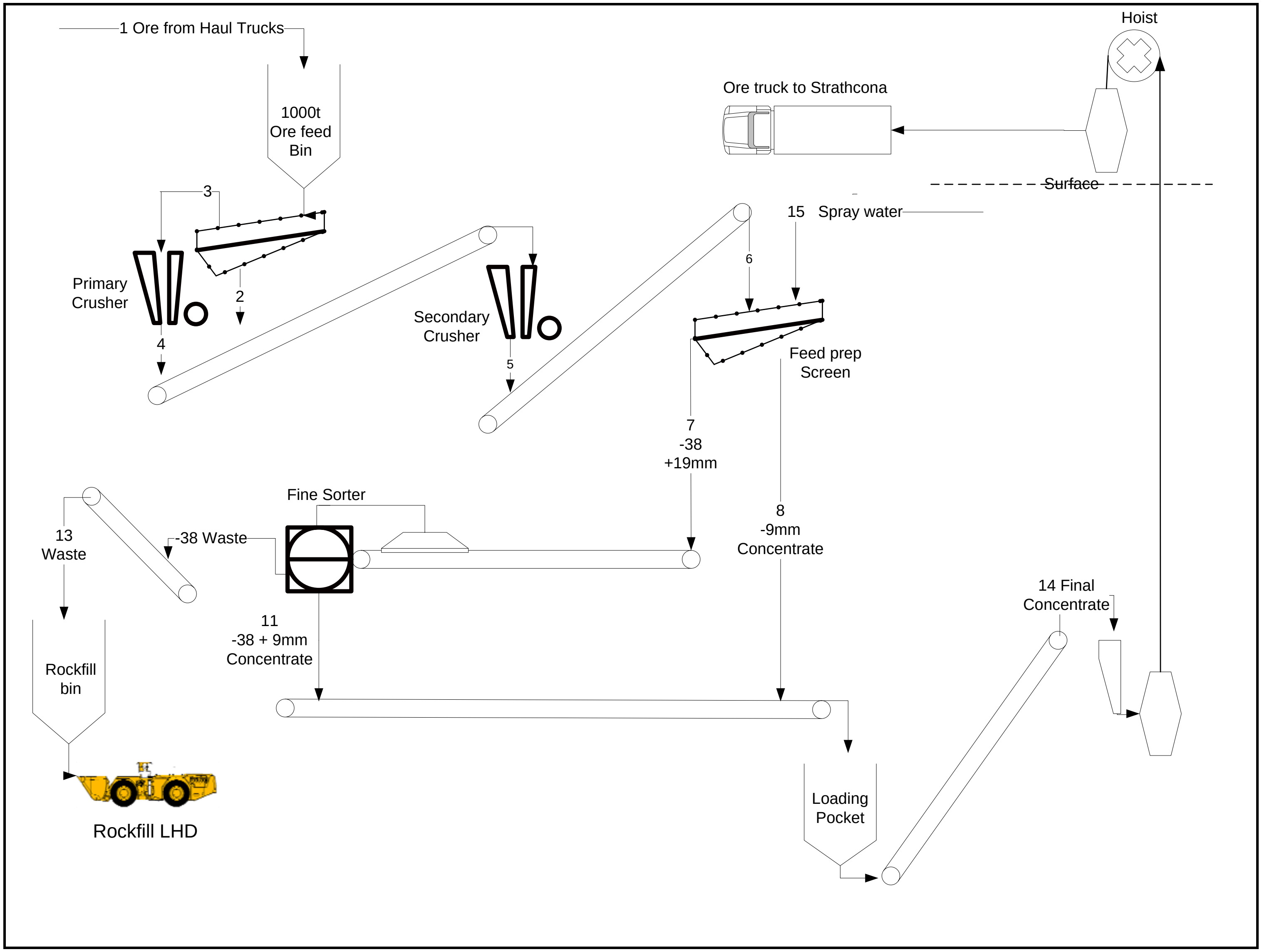


Figure 6.9 – Size Classification and Optical Sorting for Narrow Vein Footwall Copper Ores

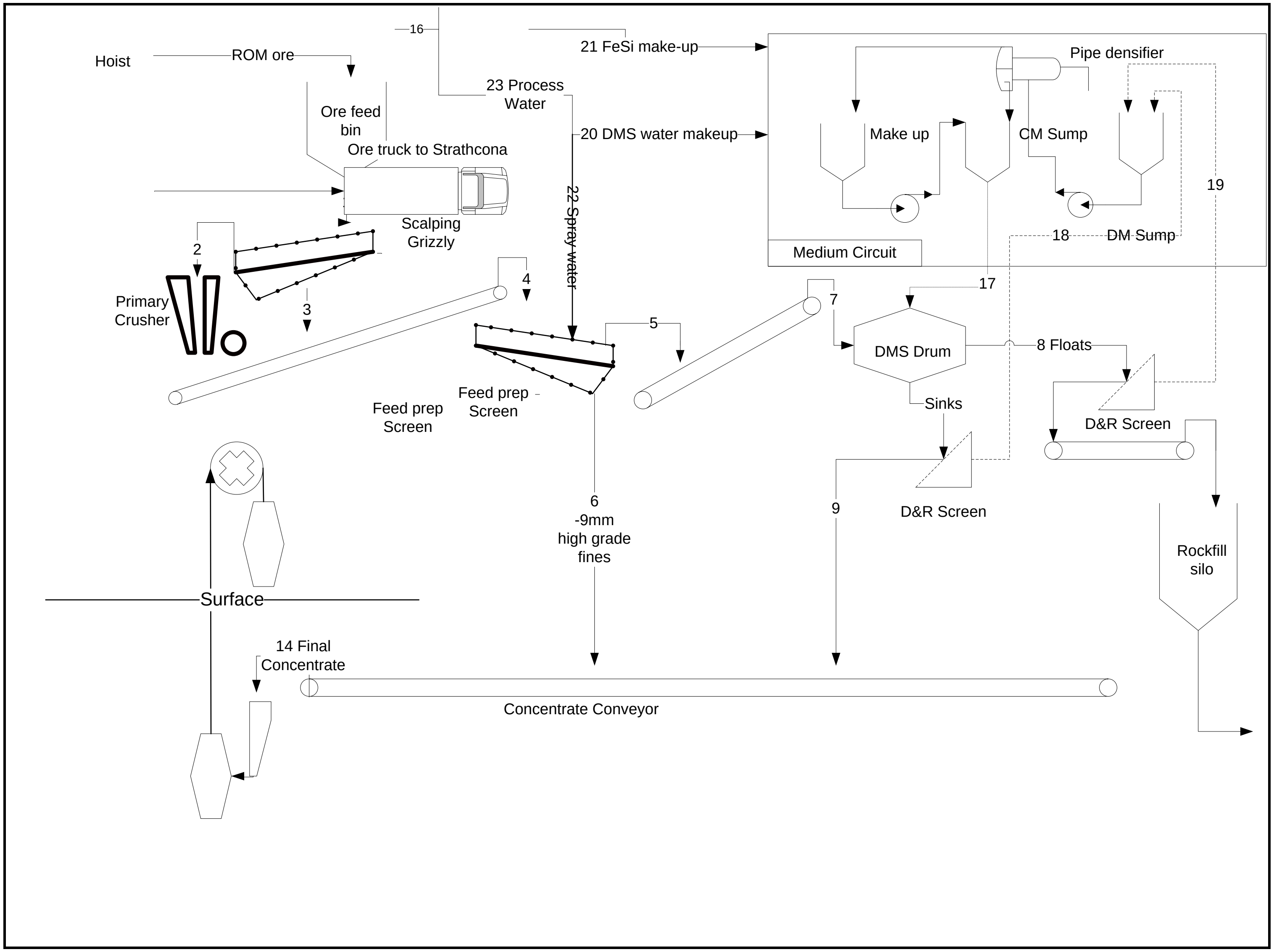


Figure 6.10 – Size Classification and Dense Media Separation for Contact-type Nickel Ores

6.6.2 Phase II Study for 9 Ores of Xstrata Nickel's Ontario Operations

Based on the successful conclusion of previous work at the Fraser and T-L Mines, a comprehensive study of the benefits of pre-concentration, and potential strategies for the disposal of the pre-concentration rejects was initiated at Xstrata Nickel's Ontario operations in 2005. Xstrata Nickel presently has three principal producing mines located around the Sudbury Basin in Ontario Canada: the Fraser, Craig and Thayer Lindsley Mines. A further mine, the Montcalm Mine was acquired from Outokumpu in 2003 and is located some 100km East of Xstrata's Kidd Creek metallurgical complex in Timmins, Ontario. Raglan mine in Quebec was not considered in the study. Ores from the Fraser and Craig Mines are considered analogous to the ores of two significant future operations in the Group: Craig LGBX for Onaping Depth and Fraser copper and Fraser nickel for Nickel Rim footwall and contact sones respectively. The study comprised sampling, metallurgical and geotechnical testwork, as well as backfill mix design and evaluation for 9 principal ore types from the four operations. Results from Fraser and Craig were used to extrapolate results for the two future mines described. A systems engineering approach was used to model the impacts to mining, material transport, and milling activities at these operations. The decrease in overall energy usage arising from the decrease in mass, and increase in metal grade of the ores was thus evaluated for the combined present and future operations using energy-based methods. The results have significance in terms of improvements in the efficiency of existing operations as well as the operational management of future mines. A map showing the spatial distribution of Xstrata's Ontario Operations is shown in Figure 6.11.

Stope samples weighing about 300 kg each, with a topsize of approximately 300mm were collected from nine different ore zones at all four of the mines. Several of these ore types, including massive vein, massive- and banded sulphides have been examined in previous studies. However, the Craig LGBX and Montcalm type ores were new ore types to the programme and had not previously been tested. LGBX type ores are Lake-Granite Breccia ores with a matrix-type mesotexture (Figure 6.12). Montcalm ores are highly disseminated to matrix sulphides from the Timmins igneous belt East of Timmins (Figure 6.13).

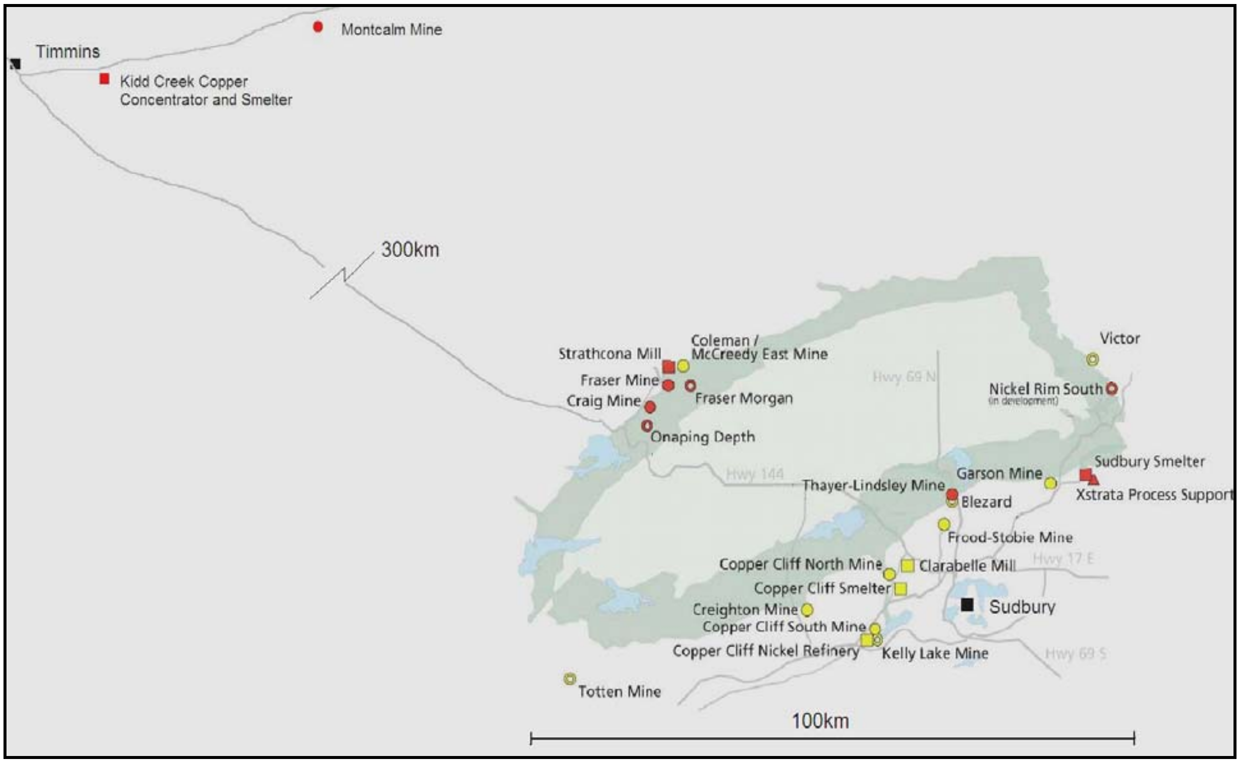


Figure 6.11 – Xstrata Nickel Ontario Operations (operations shown in red)

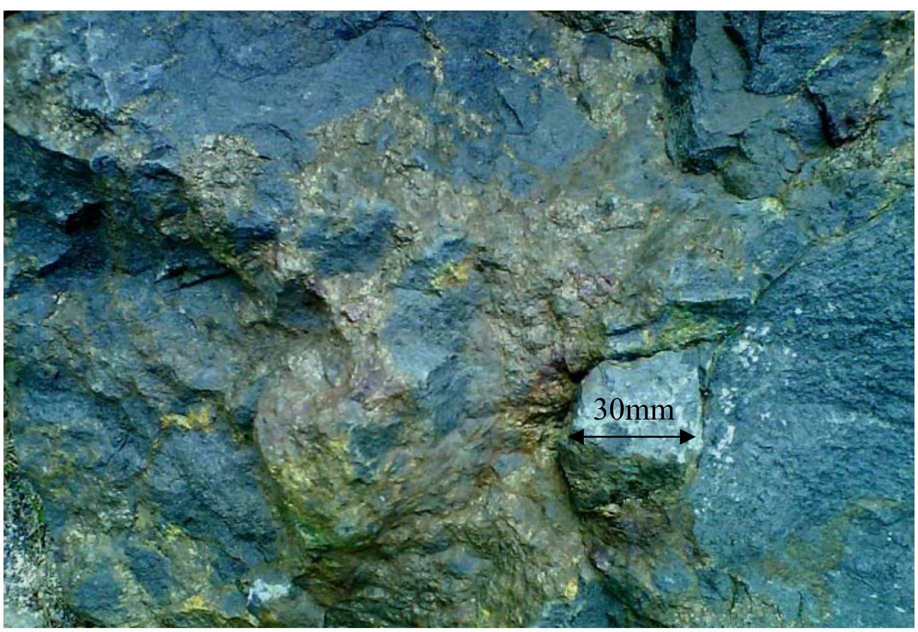


Figure 6.12– Sudbury breccia /matrix type ore texture

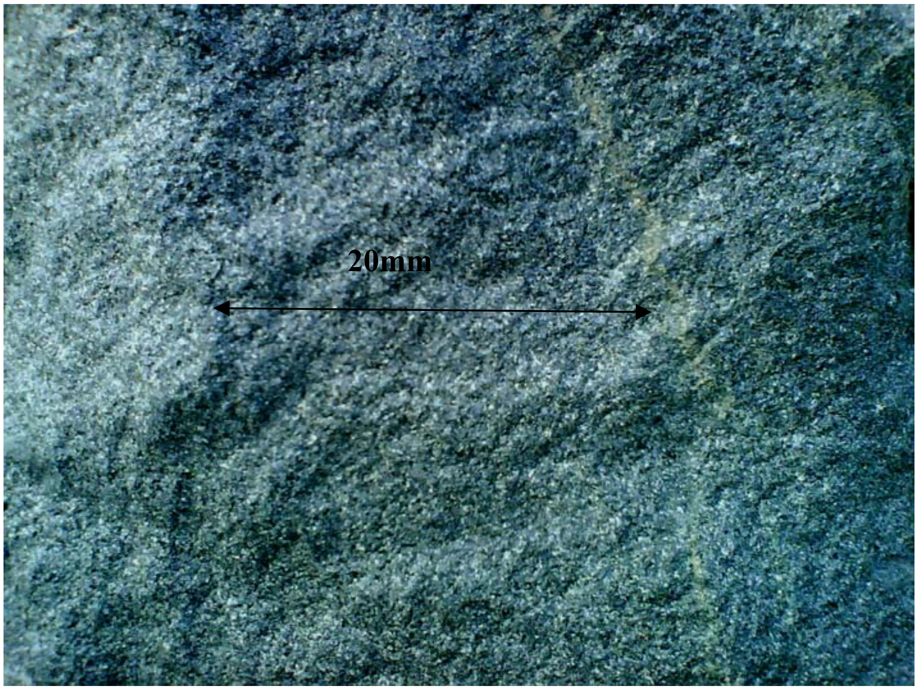


Figure 6.13 – Disseminated Ore Texture (from Montcalm, Timmins)

6.6.2.1 Metallurgical Results

A significant density differential was observed between the mineralized sulphides and the siliceous gangue material in all of the ores tested. Liberation of the gangue material was also observed to be generally high. DMS testing was conducted on the samples at a separation S.G. of between 2.7 and 2.95 at SG intervals of 0.5. Final cut SG was selected depending on the results. -6.7mm fines were generally considered high-grade and were recombined with the DMS concentrate. The combined test products were dried, weighed and assayed to produce a metallurgical balance. The results of the DMS test program are presented in Table 6.3 (after Weatherwax, 2007).

All nine ores exhibited high metal recoveries accompanied by significant mass rejection. The massive-vein Fraser Copper ore yielded the best results in the DMS study, with nickel and copper recoveries in excess of 96% and mass rejection in excess of 53%. Thayer Lindsay Footwall ore is of the massive-vein type and yielded excellent results, with nickel and copper recoveries greater than 97% and a mass rejection of 37%. Footwall ores in general present an excellent opportunity for pre-concentration as the pre-concentrate grades obtained are of the order of a typical smelter feed grade, which presents particular advantages in terms of the transport and processing of these ores. Metallurgical results for other ores are also generally acceptable; however, the Craig LGBX ore was more refractory and showed a significantly lower copper recovery than nickel, indicating that the copper and nickel are not associated mineralogically. Further metallurgical work is required in this area.

A preliminary conductivity test on ore and waste samples for each ore was conducted using the MineSense 'B2' MkII conductivity sensor. A clear discrepancy between the conductivity response of the selected ore and waste fractions was determined (see Appendix G).
A sample of each ore type was then subjected to sorting using the INCO conductivity sorting test rig. Samples were typically crushed to -75mm, and the arising -10mm fines were removed. The -75 mm + 10mm fraction was sorted, and the sorter concentrate re-combined with the fines for a final concentrate. The results of the preliminary sorting tests are presented in Table 6.4. Waste rejections of between 19 – 64% were achieved, with metal recoveries varying between 50 – 96%. Montcalm Low ore showed excellent waste rejection, but poor metal recoveries. However, ores with moderate nickel contents such as Montcalm High, Craig LGBX, Craig 8112 and Fraser Ni all showed fair responses to the technology and further work is recommended on these ores. The TL670 and Fraser Cu showed the poorest results which was expected from the initial sensor responses of these ores.

Optical Sorting

Selected gangue and mineralized particles were selected for optical evaluation for sorting purposes. Samples were analyzed using the UBC NI Machine Vision Station. RGB data, as well as textural analyses were undertaken on the samples, with significant discrepancies between the valuable and non-valuable fractions in each case, particularly in terms of texture. Detailed results for the evaluation of Xstrata ores are presented in Appendix H. Results indicate a preliminary potential for the optical sorting of these ores, although preliminary sorting tests have not yet been completed.

Table 6.3. Summary of DMS testwork results on 9 Xstrata Nickel ores (after Weatherwax, 2007)

Deposit	**Mineralogy**	**Product**	**Cut SG**	**Wt%**	**Grade (%)**			**Distribution (%)**		
					Ni	**Cu**	**Mg**	**Ni**	**Cu**	**Mg**
Craig 8112	Massive/disseminated sulphide	DMS Concentrate	+2.95	85.19	1.22	0.57	5.80	97.34	97.34	86.12
		Final Concentrate		86.19	1.26	0.57	5.71	97.63	97.63	86.90
Craig LGBX	Breccia/matrix sulphide	DMS Concentrate	+2.95	65.60	3.53	0.37	2.42	96.90	79.75	66.00
		Final Concentrate		67.88	3.52	0.38	2.37	97.18	81.55	67.83
Fraser Ni	Massive/ disseminated sulphide	DMS Concentrate	+2.8	66.33	0.82	0.40	3.34	80.04	80.04	62.97
		Final Concentrate		75.45	0.82	0.48	4.00	91.35	91.50	42.70
Fraser Cu	Massive vein sulphide	DMS Concentrate	+2.9	35.79	0.70	23.62	0.54	92.74	97.03	9.60
		Final Concentrate		46.63	0.84	22.01	0.69	96.01	97.95	17.49
TL Footwall	Massive vein sulphide	DMS Concentrate	+2.9	60.70	1.90	11.11	0.95	97.47	97.70	38.94
		Final Concentrate		63.40	1.83	10.79	1.03	97.65	97.88	43.68
TL Zone 2	Massive sulphide	DMS Concentrate	+2.9	71.61	1.71	1.08	3.64	97.42	94.93	68.62
		Final Concentrate		74.27	1.70	1.11	3.58	97.73	95.65	71.14
TL 670	Disseminated/ banded sulphide	DMS Concentrate	+2.9	78.21	0.71	0.41	6.45	94.02	90.91	78.60
		Final Concentrate		80.48	0.82	0.45	6.20	95.40	92.60	80.21
Montcalm East	Disseminated sulphide	DMS Concentrate	+2.95	72.67	2.14	0.84	4.38	97.36	92.62	65.25
		Final Concentrate		74.50	2.12	0.82	4.39	97.56	93.11	67.417
Montcalm West	Disseminated sulphide	DMS Concentrate	+2.8	57.78	0.56	0.24	6.09	84.64	82.49	70.97
		Final Concentrate		67.63	0.55	0.24	6.04	97.57	95.25	82.36

Table 6.4 – Summary of Conductivity Sorting Testwork on Xstrata Nickel Ores (after Weatherwax, 2007)

	Product	Wt%	Assay (% or g/t for PGMs))								Distribution (Wt%)							
			Co	Cu	Mg	Ni	Au	Ag	Pd	Pt	Co	Cu	Mg	Ni	Au	Ag	Pd	Pt
Craig 8112	Reject	25.7	0.01	0.22	6.54	0.27	0.03	0.64	0.07	0.06	8.4	11.5	30.6	5.8	16.5	11.1	15.3	15.7
	Final Conc	74.3	0.04	0.58	5.12	1.52	0.06	1.76	0.13	0.12	91.6	88.5	69.4	94.2	83.5	88.9	84.7	84.3
	Total	100.0	0.04	0.48	5.48	1.20	0.05	1.47	0.11	0.10	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Craig LGBX	Reject	15.8	0.02	0.28	3.47	0.51	0.03	0.61	0.05	0.04	4.6	12.2	21.7	3.7	20.9	10.6	7.1	7.0
	Final conc	84.2	0.06	0.37	2.35	2.51	0.02	0.97	0.13	0.11	95.4	87.8	78.3	96.3	79.1	89.4	92.9	93.0
	Total	100.0	0.06	0.36	2.53	2.19	0.02	0.91	0.11	0.10	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Fraser Ni	Reject	19	0.18	6.08	0.29	0.03	0.66	0.02	0.04	0.19	9.9	9.4	27.1	6.6	17.5	9.4	5.7	6.8
	Final conc	81	0.41	3.73	0.93	0.04	1.45	0.09	0.14	0.81	90.1	90.6	72.0	93.4	82.5	90.6	94.3	93.2
	Total	100	0.37	4.17	0.81	0.04	1.30	0.08	0.12	1.00	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Fraser Cu	Reject	50.2	0.01	4.85	2.59	0.27	0.08	23.22	0.70	0.77	28.3	19.4	77.5	15.3	26.0	22.5	19.1	17.0
	Final Conc	49.8	0.01	20.39	0.76	1.48	0.24	80.94	2.98	3.78	71.7	80.6	22.5	84.7	74.0	77.5	80.9	83.0
	Total	100.0	0.01	12.58	1.68	0.87	0.16	51.94	1.83	2.27	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
TL-15-2	Reject	32.0	0.01	3.25	3.51	0.20	0.38	6.93	0.41	0.32	7.2	11.5	59.5	5.1	32.2	11.3	5.8	10.1
	Final conc	68.0	0.07	11.74	1.12	1.80	0.38	25.73	3.14	1.33	92.8	88.5	40.5	94.9	67.8	88.7	94.2	89.9
	Total	100.0	0.05	9.03	1.89	1.29	0.38	19.72	2.27	1.00	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
TL670	Reject	50.9	0.02	0.39	6.33	0.44	0.17	1.22	0.09	0.12	31.7	43.9	55.1	29.6	59.1	43.0	33.5	50.8
	Final Conc	49.1	0.04	0.52	5.36	1.09	0.12	1.68	0.18	0.12	68.3	56.1	44.9	70.4	40.9	57.0	66.5	49.2
	Total	100.0	0.03	0.46	5.86	0.76	0.15	1.45	0.13	0.12	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
TL80	Reject	34.5	0.02	0.28	3.91	0.36	0.03	1.08	0.12	0.21	11.9%	13.7	37.8	8.7	5.6	10.8	12.6	13.3
	Final Conc	65.5	0.06	0.93	3.38	1.99	0.26	4.69	0.43	0.72	88.1%	86.3	62.2	91.3	94.4	89.2	87.4	86.7
	Total	100.0	0.05	0.71	3.56	1.43	0.18	3.45	0.32	0.55	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
MH	Reject	23.0	0.02	0.33	5.94	0.43	0.05	1.13	n/a	n/a	7.9	13.4	29.7	5.9	11.1	15.0	n/a	n/a
	Final Conc	77.0	0.07	0.63	4.20	2.04	0.11	1.91	n/a	n/a	92.1	86.6	70.3	94.1	88.9	85.0	n/a	n/a
	Total	100.0	0.05	0.56	4.60	1.67	0.10	1.73	n/a	n/a	100.0	100.0	100.0	100.0	100.0	100.0	n/a	n/a
ML	Reject	64.24	0.01	0.09	5.94	0.19	0.01	0.37	n/a	n/a	43.2	37.4	64.1	35.4	28.3	34.3	n/a	n/a
	Final Conc	35.76	0.02	0.28	5.97	0.61	0.06	1.26	n/a	n/a	56.8	62.6	35.9	64.6	71.7	65.7	n/a	n/a
	Total	100.00	0.01	0.16	5.95	0.34	0.03	0.69	n/a	n/a	100.0	100.0	100.0	100.0	100.0	100.0	n/a	n/a

6.6.2.2 Investigation of Waste Disposal Aspects

As an extension of the study into pre-concentration at Xstrata's Ontario operations, it was decided to evaluate the capabilities of the various types of pre-concentration rejects as a material for geotechnical support in the underground context. Of the mines investigated, several mining methods were employed, ranging from drift and fill at Fraser Cu to open stoping methods at Montcalm. What is of benefit to the consideration of the concept is the fact while only the mines using drift and fill or cut and fill methods depend on fill for their success, it was found that the mining methods at all the operations were able to accommodate fill in the mining cycle, and thus the potential to dispose of the rejects underground in each case has been demonstrated. For the present and future mines that require fill for the success of the operation, it is expected that there would be additional benefits in terms of utilizing the material rejected through the pre-concentration of ore to create high strength backfills.

A preliminary study into the impact of using pre-concentration rejects was conducted in 2005 and 2006 (Bamber et al, 2006). It was decided to use the rejects from some of the DMS testwork on a Ni-Cu sulphide or from the Sudbury basin as a starting point. Rejects from the concentration process were typically coarse, low-sulphide and considered suitable for use in an aggregate fill. As it is desired to maximize the degree of waste rejected from the ROM ore and disposed of as fill, classified tailings from the surface mill were used as a source of the fines component for the mix in order to indicate the overall potential for solids disposal underground. A typical composite fill would thus comprise up to 90% ROM material, consisting of the DMS rejects, classified tailings, plus cement and water for maximum disposal of the solid waste arising from the ROM ore underground. Results were positive and an extended programme was initiated for Xstrata Nickel's Ontario division mines. Ores were typically pre-concentrated using dense media separation, although optical and conductivity methods were also considered, and the results presented in this Chapter are expected to be consistent for all pre-concentration methods considered in the thesis. A range of waste products were produced during the pre-concentration testing on the Xstrata ores, these products were physically and geochemically characterized in terms of the ASTM standards for use as aggregates; fill mixes were designed based on these results, and tested for uniaxial compressive strength (UCS) as well as tested to determine the rheology of the fresh fill mix for pumping purposes.

Results

Approximately 50kg of the reject material was typically available from each sample (Figure 6.14).

Figure 6.14 – Typical Dense Medium Separation Reject fraction (Fraser Ni)

The size distribution of the waste rejects was typically -75+19mm due to the rejection of the -19mm fines to concentrate in the process. -75 +19mm is generally considered too coarse for suitable aggregate so it was decided to produce two mix designs for testing, firstly with the rejects as-is, and a second mix with the rejects crushed to nominally 100% - 50mm in order to improve the rheology of the mix in the light of the desire to pump the fill. Results of the geotechnical characterization are shown in Table 6.5 (Weatherwax et al, 2008).

Table 6.5: Measured Geotechnical Properties of Pre-concentration Rejects

Ore Body	Hardness	P80	P60	P10	Cu	% Flat and Elongated	Adsorption	ABA	SG	Void Ratio
	moh's	mm	mm	mm		+9.5mm	% Dry Wt.			
Craig LGBX	5-7	15	7	1.5	4.7	14.3	0.58	0.54	2.77	0.41
Craig 8112	>7	8.5	5.4	0.8	6.8	34.0	0.39	0.40	2.77	0.40
TL Zone 1	>7	6.5	5	1.3	3.8	82.8	0.82	0.47	2.98	0.42
TL Zone 2	>7	22	15	2.5	6.0	21.7	0.33	1.01	2.78	0.43
TL Footwall	>7	23	17	2.5	6.8	25.4	0.29	0.63	2.86	0.39
Fraser Cu	5-7	22	17	3	5.7	24.0	0.29	0.85	2.86	0.51
Fraser Ni	5-7	20	15	2.5	6.0	27.3	0.19	0.27	2.94	0.44
Mont. East	>7	22	17	3.5	4.9	18.3	0.24	0.99	2.95	0.51
Mont. West	5-7	22	18	5	3.6	10.3	0.66	1.19	2.93	0.49

Rejects arising from the pre-concentration of the Xstrata ores were found to be generally competent, with a high coefficient of uniformity, denser as well as generally more flat and

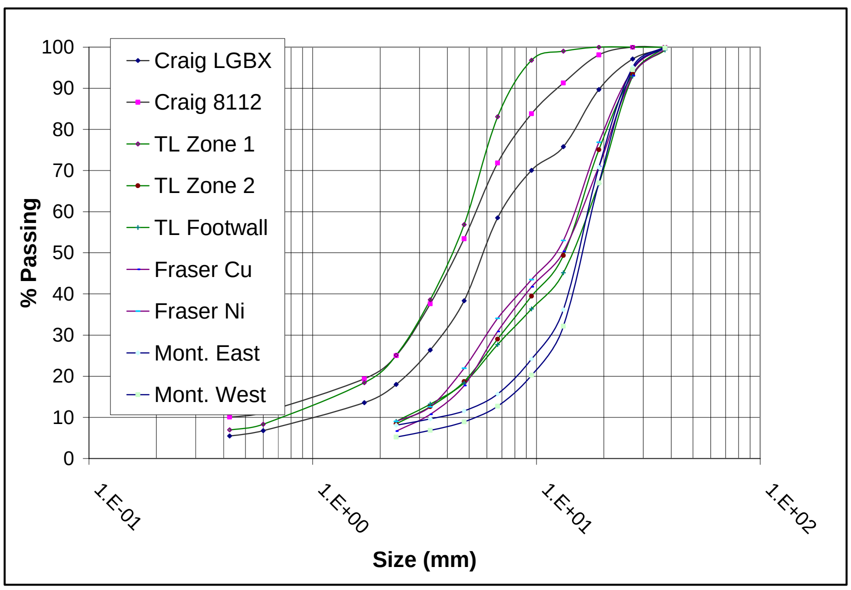


Figure 6.15: Size Distribution of Pre-concentration Rejects

elongated than standard recommended ASTM aggregates. The size distribution of the rejects arising from the pre-concentration testwork was analysed and compared to the recommended aggregate size distribution for fill aggregates (Figure 6.15). Size distribution of the rejects is ultimately determined by their feed preparation for concentration and thus this was not considered a variable. As noted in preliminary work for INCO at McCreedy East (Bamber, 2004), the ores demonstrated a characteristic high grade fines fraction, and removal of this fraction prior to concentration increases the relative size distribution of the rejects when compared to the ROM ore, increasing their compatibility in terms of aggregate uses. The rejects were measured generally coarser than a typical ASTM fill aggregate of similar topsize, but are nevertheless considered acceptable for use in the fill. Void ratios of the aggregates were also calculated from the size distribution data, indicating a high overall bulk density of the rejects. Results confirm the potential shown in preliminary mix design and testing (Bamber et al, 2006), and it can be concluded that the rejects of the pre-concentration process can typically be considered acceptable for use as aggregate in fill.

Four standardized mixes were designed and tested for further evaluation of the concept (Table 6.6). Classified tailings obtained from Strathcona mill was typically 80% - 149um (Verdiel, 2006) and was used as the fine aggregate for the preparation of the paste fraction of the mix. Binder was 5% by mass Lafarge CSA A5 Type 10 NPC. Mixes were prepared and tested according to ASTM standards for backfills using an M-Test 841 testing machine.

Table 6.6 –Mix Ratios for Xstrata Fill Testwork – Average Mix (dry weight basis)

Sample	Rejects Wt%	Tailings Wt%	Cement Wt%	Water: Cement
RockFill	95.0		5.0	1:1
CT Max	69.1	28.8	1.4	6.5:1
CT 1:3	36.5	59.3	3.0	6:1
CT 1:7	19.9	74.7	3.7	5.6:1
Full Max	69.7	29.1	1.5	6.5:1
Full 1:3	37.0	58.7	2.9	5.4:1
Full 1:7	20.1	74.5	3.7	5.2:1

Results for Rockfills

Results of the UCS testing showed that the rejects could be used as aggregate to produce a cemented rockfill with a high fill strength / unit cement and high stiffness. 28-day strength of the samples was measured between 1.28 - 3.41 MPa with an average of 1.89MPa. Stress strain curves and observations during the test indicate that cement content is the primary mechanism in the strength of the fill, and that strength falls rapidly in these fills post-failure. Failure is generally stiff, yet plastic, with low residual strength. Figure 6.16 shows the UCS results for these fills. Some evidence of binder segregation by gravity was observed in the samples (Weatherwax, 2007). Failure in the reject particles was observed in only Thayer Lindsley Zone 2, where an individual reject particle showed signs of failure.

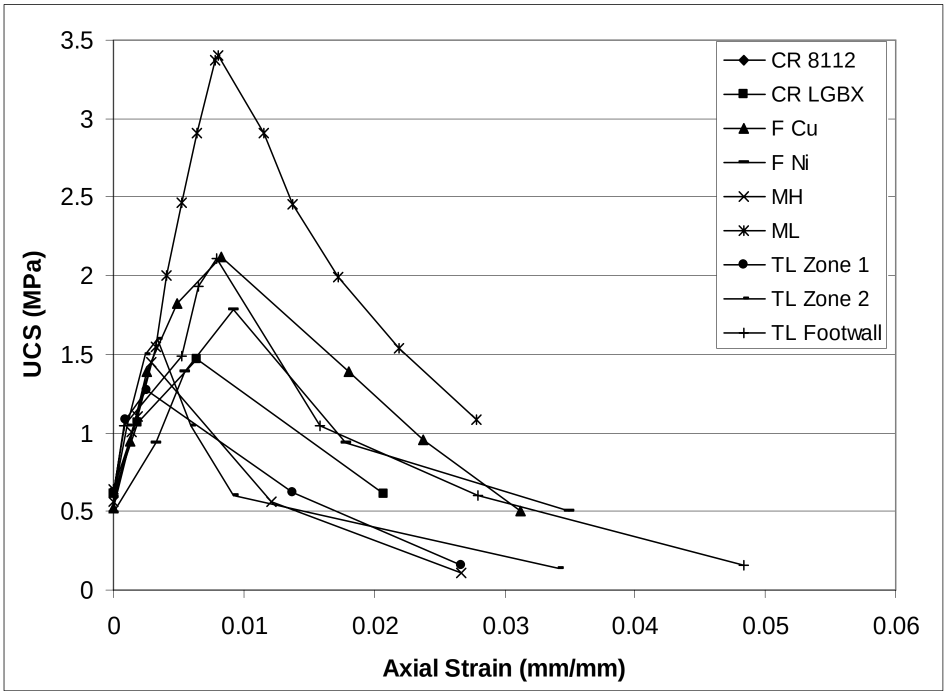


Figure 6.16: UCS vs Axial Strain for Rockfills

Results for Composite Fills

A range of composite fills comprising various ratios of pre-concentration rejects, mill tailings and cement were designed, prepared and tested. Results are summarized in table 5. Strength generally increased with increasing reject content, with the 'Maximum density' samples showing the highest average strengths for the composite fills. The efficiency of the mixes was also analysed, and results showed that the efficiency of the fills in terms of UCS : Wt% cement increased significantly with increasing reject content, thus indicating the positive benefits of utilizing these rejects as fill material. Absolute compressive strengths achieved were generally low, albeit at a low cement content; ultimate cement content in the mixes as well as water cement ratio was highly variable and thus results varied substantially across the mixes (Figure 6.17). This can be attributed primarily to excessive water in several of the mixes which should be addressed in subsequent phases of testing (Talbot & Richart, 1923).

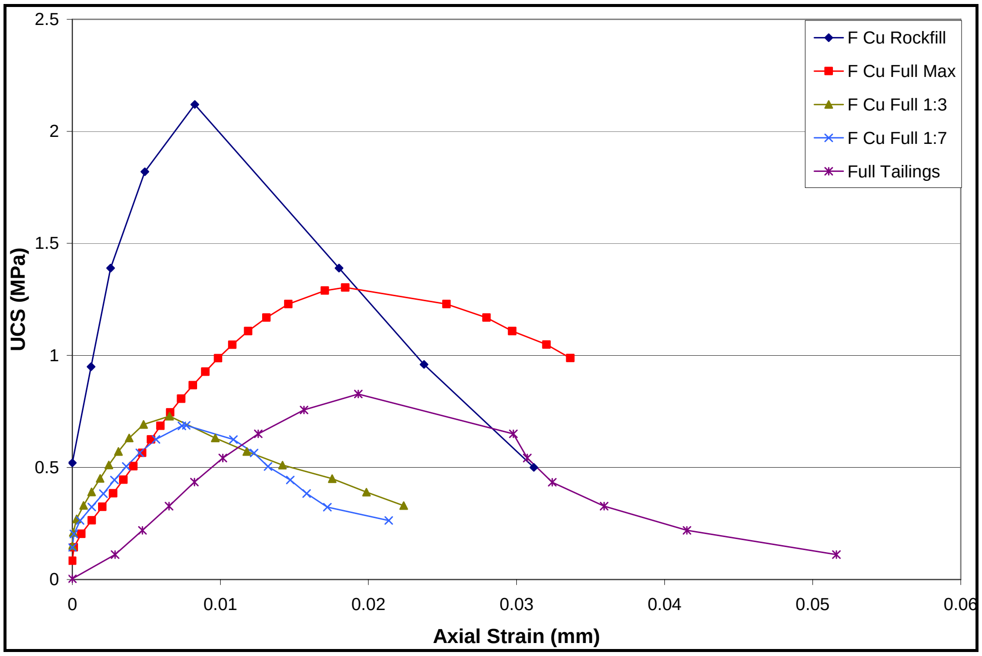


Figure 6.17: UCS vs Axial Strain for Composite Fills

Overall Results

From the results it can be concluded that the addition of rejects has an increasing effect on the overall performance of the mixes in these samples, until maximum density is reached and point-to-point contact is made among the aggregates (Weatherwax, 2007). Absolute strength is a maximum for the rockfills. However, the maximum binder efficiency was achieved with the maximum density composite fills, with a resultant UCS:% cement ratio of 0.73 – 0.86, indicating that a fill strength of between 4 – 5 MPa would be possible at a binder content of 5% (Table 6.7). The confined strength of the fill, and strength post failure is of interest in high stress, plastic conditions in underground stopes, and the composite fills indicate good potential in this regard based on the high residual strength demonstrated in the testing.

Table 6.7 – Overall UCS Results for Xstrata Fill Mixes

Sample	Rejects%	Tailings%	Cement%	W:C	UCS	UCS: %Cement
RockFill	95.0		5.0	1.0	1.89	0.38
CT Max	69.1	28.8	1.4	6.5	1.24	0.86
CT 1:3	36.5	59.3	3.0	5.9	0.8	0.27
CT 1:7	19.9	74.7	3.7	5.6	0.77	0.21
Full Max	69.7	29.1	1.5	6.5	1.07	0.74
Full 1:3	37.0	58.7	2.9	5.4	0.82	0.28
Full 1:7	20.1	74.5	3.7	5.2	0.7	0.19

6.6.2.3 Rheological Evaluation of Fill Mixes

It is the intention to use the rejects as a material for composite fills in underground backfill applications, thus the transport of the fill becomes an issue and it was felt important to investigate the basic rheological properties of the fills for the purpose of determining an appropriate rheology for transportation by pumping. For the evaluation of this, the modified 'slump test' method was used to determine a dimensionless yield stress (τ') which allowed for a comparison of the different mixes (Hu et al, 1995). The results show that an increasing the amount of rejects increases the value of τ'. Results of the testwork are presented in Table 6.8 (Weatherwax et al, 2008).

Table 6.8 – Average of Slump Test Results for Xstrata Fill Mixes

Mix	Rejects	Tailings	Cement	Water:Cement	τ (von Mises)
RockFill	95	0	5	1.0	0.58
CT Max	69.13	28.77	1.4385	6.5	0.58
CT 1:3	36.52	59.28	2.964	5.9	0.15
CT 1:7	19.87	74.66	3.733	5.6	0.09
Full Max	69.74	29.05	1.4525	6.5	0.58
Full 1:3	37.04	58.66	2.933	5.4	0.13
Full 1:7	20.08	74.46	3.723	5.2	0.05

The shear stress in the mixes varies from 0.05 for the low density composite fills to 0.58 for the maximum density and rockfill mixes. For a pumpable fill mix, a shear stress of between 0.15 – 0.5 is suggested as reasonable, and further optimization to maximize strength while maintaining pumpability of the mixes is required. At this stage of testing, slump rate was not measured and a further phase of testing on fresh mixes is recommended to explore aspects of plastic viscosity as well as shear stress in order to fully characterize the fill mixes.

6.6.2.4 Grinding Index Testwork

The majority of ores are presently delivered to the centralized Strathcona Mill near Levack village in the North East of the Basin for milling. Montcalm ores by exception are delivered to the Kidd Creek Metallurgical Complex in Timmins. The pre-concentration of ores is expected to result in power savings at the mills in three principal areas. Firstly, the rejection of waste results in a reduction in tonnage reporting to the mill. Secondly, the waste is primarily hard siliceous rock; removing this material leaves a product enriched in relatively soft, metal bearing sulphides which will have a lower grinding work index. Thirdly, the pre-concentration product will be crushed to a finer size than the present plant feed. This projected impact on grinding power requirements was calculated from measurements of the Work Index using the Bond method. Feed, concentrate and reject samples from the 9 ores in the test programme were subjected to a series of tests in order to evaluate this impact. For the evaluation, the work index of the Fraser Nickel ore was determined through a complete Bond Work Index test and used as a reference. The Work Indices of the other ores were then determined through comparative work index tests on whole ore and the pre-concentration products. Results of the Work Index tests are presented in Table 6.9 (Altun, 2007). In the evaluation, it was also considered to evaluate the influence of pre-concentrate grade on the Work Index of the pre-concentrates. For the Xstrata ores, the lowest work indices were obtained for the highest grade Cu ores, Fraser Cu and TL 15. Also, the largest decrease in the work index occurred with the Fraser Cu ore, which had the highest waste rejection, and thus greatest increase in grade after pre-concentration. Overall Work Index of the ores was reduced by 8.8% through the rejection of between 30-54% waste from the ores.

Table 6.9 - P_{80}, F_{80} and Work Indices for the raw ores and their concentrates

Sample		**F_{80} (microns)**	**P_{80} (microns)**	**Work Index (kWh/tonne)**	**% Reduction in WI**
Fraser Ni	Feed	2930.26	86.93	10.63*	10.17
	Concentrate	2849.25	76.00	9.83	
Fraser Cu	Feed	2504.50	53.98	8.13	19.10
	Concentrate	2685.67	37.72	6.58	
Montcalm West (ML)	Feed	2784.50	87.48	10.73	6.99
	Concentrate	2739.12	77.34	9.98	
Montcalm East (MH)	Feed	2791.72	73.76	9.68	11.78
	Concentrate	2787.28	59.68	8.54	
TL 15	Feed	2693.67	56.43	8.29	9.65
	Concentrate	2649.76	47.23	7.49	
TL 80	Feed	2777.29	66.78	9.13	12.16
	Concentrate	2773.00	53.49	8.02	
TL 670	Feed	2778.21	84.95	10.54	2.84
	Concentrate	2771.57	80.90	10.24	
Craig 8112	Feed	2801.86	74.28	9.72	-
	Concentrate	-	-	-	
Craig LGBX	Feed	2791.57	70.02	9.38	-
	Concentrate	-	-	-	

A direct relation between the grades and work indexes cannot be generally expected, as work index is affected by a number of additional mineralogical and petrographic characteristics of the host rock and the valuable mineral associations. However, for this group of Xstrata ores, it was demonstrated that ore grade and grindability were somewhat related (Figure 6.18). The work index vs Cu grade correlation and in particular the relationship between Work Index and total Cu+Ni grades, thus total sulphides, shows an extremely high correlation (R^2 = 0.85) and thus a relationship between the raw metal content and the Work Index of these ores was demonstrated.

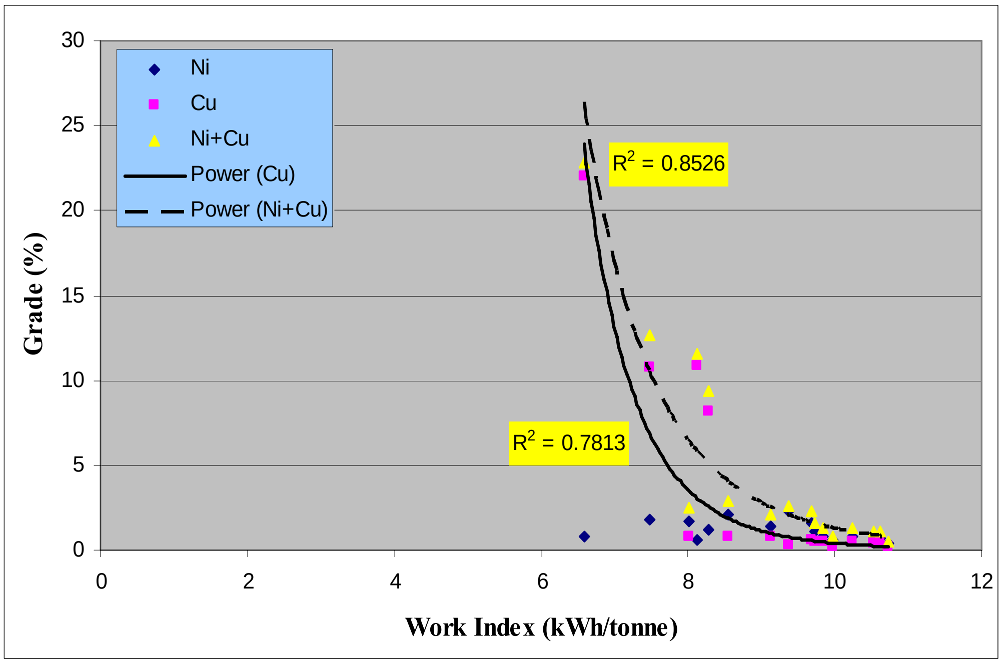


Figure 6.18 - Work Index-Grade Correlation

6.6.3 Evaluation of Impacts on Energy Usage at Xstrata's Ontario Operations

All of Xstrata's ores in the Sudbury region are transported by truck to the Strathcona Mill located at the North West edge of the Basin. Nickel concentrates from Strathcona are transported back to the Ni smelter near Falconbridge Town; copper concentrates are shipped 300km north to Xstrata Copper's Kidd Creek metallurgical complex in Timmins. Ore from Montcalm is hauled 100km on surface to the metallurgical complex at Kidd Creek. Montcalm and Thayer Lindsley are shallow mines, while Craig and Fraser are medium depth. However, these mines are mature and have limited mine life, and the future operations, Onaping Depth and Nickel Rim South, are significantly deeper, and more distant from the existing metallurgical complex. Ore from Nickel Rim is planned to be trucked 75km through the town of Sudbury for processing at Strathcona. Furthermore, continued processing of these ores at Strathcona places enormous pressure on existing tailings disposal facilities at this location.

Any savings in terms of hoisting, ore transport and tailings disposal requirements for these future ores will be significant. Xstrata is presently evaluating various options in terms of the strategic processing of these ores as well as additional synergies arising from the fact that CVRD-INCO mines and processes similar ores at different locations (Romaniuk, 2006). Significant benefits arising from these synergies have been identified, which are expected to be further enhanced through the selected application of an appropriate pre-concentration strategy at each organization. Figure 6.17 shows the spatial relationship of the operations from both Xstrata Nickel as well as CVRD INCO who also operates significant mines in the Basin.

Operational parameters for each mine were investigated, discussed and agreed with Xstrata personnel prior to the evaluation (Proudfoot, 2007). Pre-concentration is principally anticipated to impact material handling activities such as haulage and hoisting, with additional impacts expected in the mill through the reduction in tonnage, particle size distribution and increase in metal grade of the pre-concentrates. Material transport energy utilization relates primarily to production rate, and deposit depth for hoisting to surface, and distance from the Strathcona Mill for trucking. At existing operations, deposit depths range from 150 m at Montcalm to 1400 m at Craig. Future operations Onaping Depth and Nickel Rim are deeper at 2500 and 1500m respectively. While the present Fraser and Craig Mines are close to the mill, the haulage distance for the other mines are significant. Montcalm transports ore 100 km to the Kidd Creek Mill while trucking distances for the Thayer Lindsley Mine and Nickel Rim Mine are 54.4 km and 75 km, respectively. Evaluation model parameters are summarized in Figure 6.19.

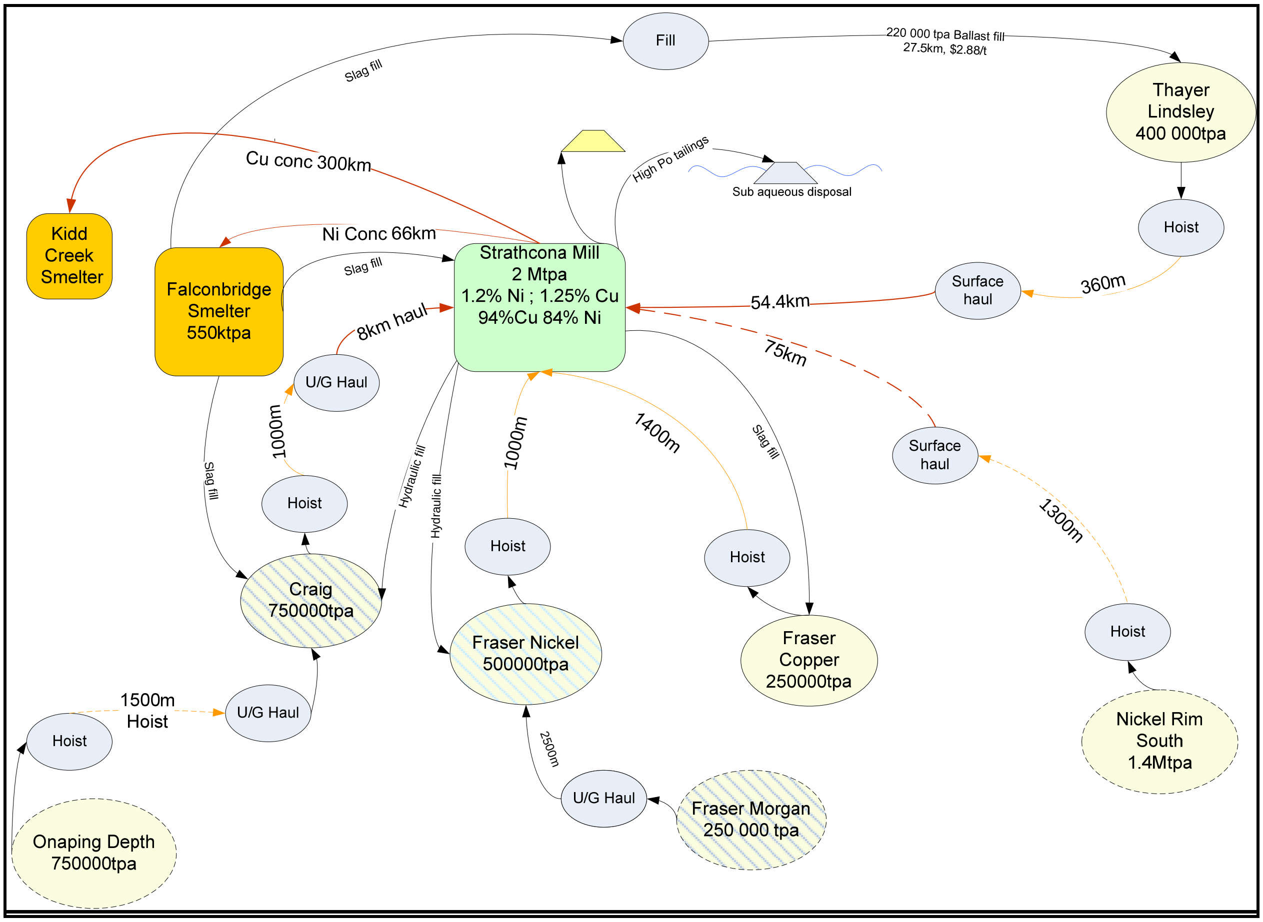


Figure 6.19 – Xstrata System Evaluation parameters

An evaluation model was developed to estimate energy usage and savings for each mine operating with- and without pre-concentration (Pitt & Wadsworth, 1980). The key areas of energy usage relate to hoisting, transportation by road to the mill, and beneficiation of the ore. The additional energy requirement of the pre-concentration plant is also included in the model. An energy cost of \$0.057/kWh plus the expected maximum demand charge of \$5000/MW was used to calculate the power cost and savings at each stage (Delphi Group, 2004). Additional credits due to potential overall savings in greenhouse gas emissions (GHG) of \$15/t CO_2 and 1720 kWh per tonne CO_2 equivalent and are also accounted for at the end of the evaluation[1]. Energy usage for processing derives primarily from the feed size distribution, the grinding work index of the ore and the tonnage processed. The pre-concentration test results indicate the amount of waste that can be rejected and therefore s not processed at the mill. Further energy benefits result from the lower grinding work index of the relatively soft sulphide rich pre-concentration product as compared to

[1] http://www.nrcan.gc.ca/es/etb/ctfca/PDFs/electrical-markets/en/3-1.html

the whole ore, and the reduction in feed particle size to the mill. In the case of the footwall ores, additional potential to bypass the mill and deliver a smelter feed directly is also considered.

6.6.3.1 Impact of the Pre-concentration Step

Table 6.10 summarizes the projected energy usage and power costs for the pre-concentration plants. The energy value was obtained from the estimated power requirements for crushing, screening and dense media separation of the tonnage indicated based on flowsheets and equipment lists developed in previous metallurgical testwork (Bamber et al, 2005).

Table 6.10. Estimated Annual Energy Usage and Costs for the Pre-concentration Plant

Operation	Montcalm	Thayer Lindsley	Fraser Copper	Fraser Nickel	Craig	Onaping Depth	Ni Rim S Contact	Ni Rim S F/W
Mining rate tpa	800000	400000	250000	500000	750000	750000	650000	650000
Power (kW)	846	532	388	618	810	810	736	736
Energy kWh/a	2354701	1479945	1080153	1718602	2255052	2255052	2048884	2048884

6.6.3.2 Impacts on Hoisting Energy

The estimated energy usage for hoisting was obtained from information on the hoisting depth, cycle time, production rate, number of trips per hour, plus the calculated payload-, skip- and rope mass. Input parameters were a hoisting acceleration of 2 m/s^2, 15 m/s max hoist velocity, a wait time of 20 s and hoist efficiency 90% (Figure 6.20). Skip payloads and skip mass are derived from Edwards (1990). Savings are enjoyed in terms of reduced payloads, reduced skip and hoisting rope masses and savings in overall rotational inertia in the hoisting system. Table 6.11 presents the estimated savings arising from pre-concentration in hoisting energy requirements and costs. Energy savings vary between \$84 200/annum for T-L (shallow and low tonnage), to \$640 000/annum for Nickel Rim (deep and high tonnage).

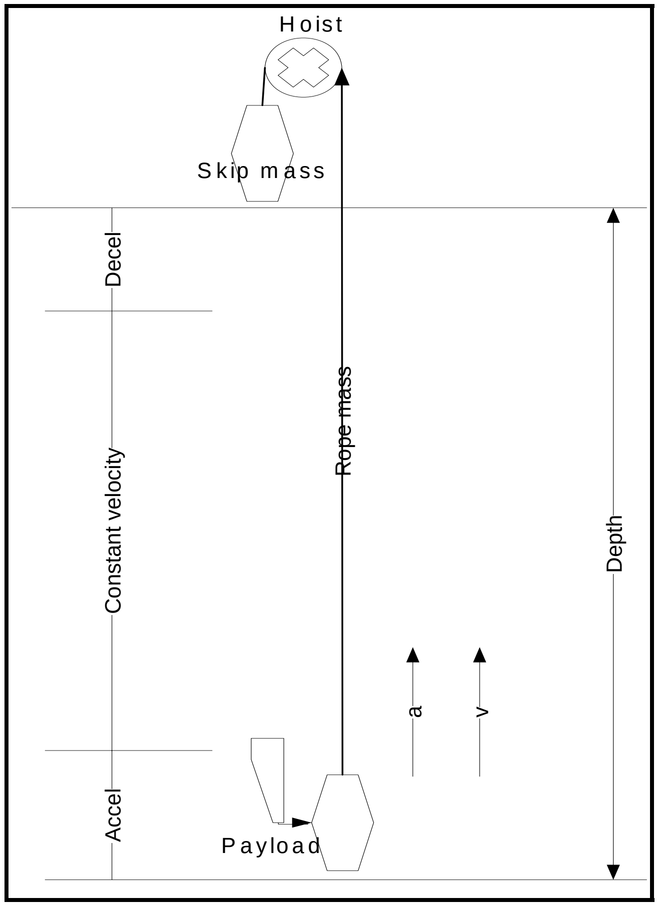


Figure 6.20 – Hoisting Impact Evaluation Model

Table 6.11 Summary of Annual Hoisting Energy and Cost Savings.

Operation	Thayer Lindsley	Fraser Copper	Fraser Nickel	Craig	Onaping Depth	Ni Rim S	Ni Rim S F/W
Depth	900	1400	1000	1000	2000	1100	1300
Energy (kWh/a)	6514	16552	8769	12469	42315	13025	38574
Cost saving ($)/annum	$84,209	$277,816	$106,316	$144,077	$503,526	$154,400	$641,879

6.6.3.3 Impacts on Surface Haulage

Of the current operating mines, only Montcalm and Thayer Lindsley have significant haul distances to the mill of 100 km and 54 km, respectively. Fraser and Craig Mines are proximal to Strathcona mill and thus haul savings are considered minimal. However, ore from the future Nickel Rim operation will be hauled some 75 km from the mine on the east side of the Basin, through the town of Sudbury to Strathcona mill in the northeast. Energy savings were calculated for transport of the pre-concentrated ore (based on the degree of waste rejection as shown in Figure 6.8) versus the transport of the whole ore to the mill (Maxim, 2001; Kodjak, 2004). Additional energy savings are enjoyed at Thayer Lindsley due to a reduction in the transportation of fill from Falconbridge to the mine. For the evaluation, a calorific value of 50337 kJ/l and cost of $1.10/l for the fuel was used. Figure 6.21 shows the energy impact in each case and the total savings resulting simply from transporting less ore. Cost savings vary from about $300,000 per annum for the Thayer-Lindsley Mine to almost $900,000 per annum for the Nickel Rim S Footwall.

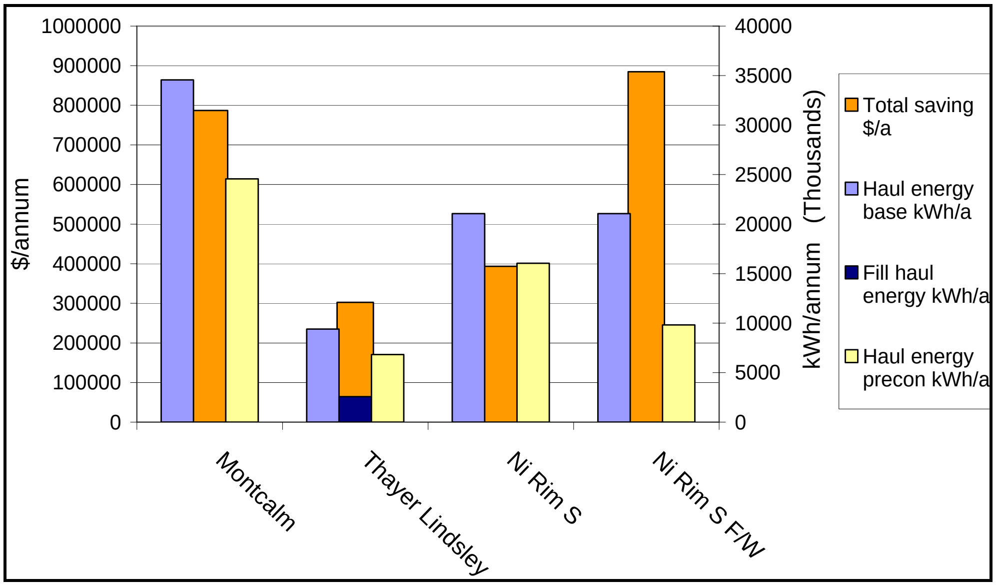


Figure 6.21 - Haul Energy Cost Savings from pre-concentration of the ore

6.6.3.4 Impacts on Grinding and Overall Beneficiation of the Ore

For both the Strathcona and Kidd Creek mills, pre-concentration results in power savings in three principal areas. Firstly, the rejection of waste results in a reduction in tonnage reporting to the mill. Secondly, the waste is primarily hard siliceous rock; removing this material leaves a product enriched in relatively soft metal bearing sulphides which will have a lower grinding work index. Thirdly, the pre-concentration product will be crushed to a finer size than typical plant feed. The testwork indicates that the ore feed size is reduced from typically 300mm topsize to nominally -75mm, and reductions in Bond Work Index through the rejection of between 20 and 54% coarse, siliceous waste can vary from 6 -13% (Altun, 2007). Results are presented in Figure 6.22.

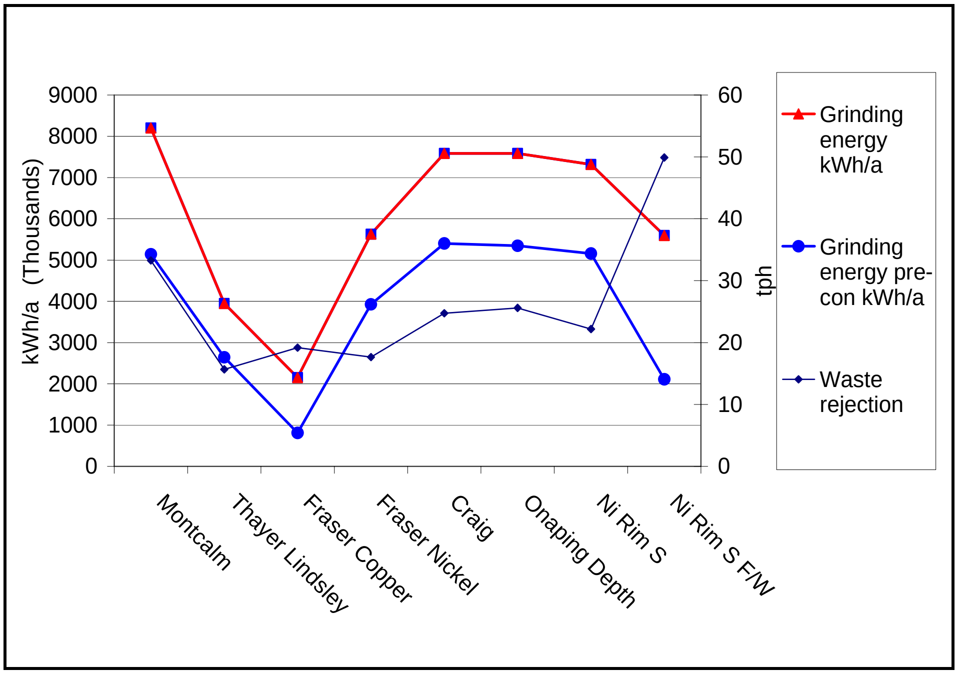


Figure 6.22 - Grinding energy requirements with- and without pre-concentration

Impacts on the overall energy requirements for the beneficiation of the ore were also analyzed. Savings in crushing and screening at the existing mill were accounted for at 50% of present requirements, and grinding energy savings are as calculated above. These savings were offset against the additional energy required for the pre-concentration step shown in section 2.1. The overall estimated impact of pre-concentration on.. milling energy requirements for these ores is shown in Figure 6.23. A significant further opportunity to use the pre-concentration step to produce ore feeds that can be customized for processing at a particular mill has been identified. The results indicate that it would be possible to produce low nickel, low-copper, high copper and high nickel feeds from various mines, as well as barren waste streams with potential for use as fill. Custom feeds thus produced could be directed to either the Clarabelle or Strathcona Mill in a possible integration scenario between Xstrata and CVRD-INCO. Pre-concentrates from Fraser Copper, the T-L Footwall ore as well as Nickel Rim South are indicated as potentially meeting a smelter feed grade (Bamber et al, 2006; Weatherwax, 2007), thus creating the opportunity to deliver ore directly to either the Falconbridge smelter, Kidd Creek or potentially INCO's smelter at Copper Cliff. Additional research is required to evaluate this opportunity; however, major positive benefits would accrue particularly at T-L and Ni Rim should some potential for synergies with INCO be established.

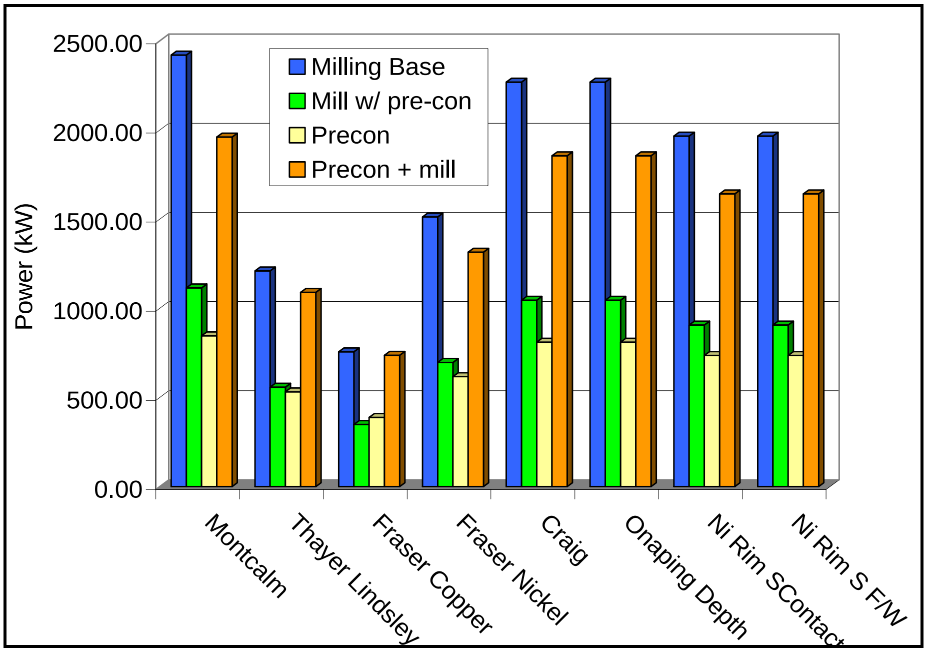


Figure 6.23 – Impacts on Power Requirements for Full Ore Beneficiation

6.6.3.5 Impacts on Overall Energy Usage

The individual energy impacts evaluated in the previous sections were combined in order to evaluate the potential overall energy savings achievable at each operation through the application of pre-concentration. The degree of savings varies principally with the overall throughput, depth, haul distance and ultimately the degree of waste rejection. Individual energy savings have been converted into cost savings at the applicable energy cost of $0.057/kWh and $5000/MW demand, and offset against the additional energy cost for the pre-concentration step in order to present a balanced picture. The additional GHG credit for the overall energy saving has also been calculated at $0.01/kWh; the combined result is presented in Table 6.12.

Table 6.12 – Overall Projected Impact on Energy Costs at Xstrata Nickel's Ontario Operations

Operation	**Montcalm**	**Thayer Lindsley**	**Fraser Copper**	**Fraser Nickel**	**Craig**	**Onaping Depth**	**Ni Rim S**
Hoisting		$84,209	$277,816	$106,316	$144,077	$503,526	$398,140
Haul	$786,583	$302,422					$884,600
Pre-con	-$282,564	-$177,593	-$129,618	-$206,232	-$270,606	-$270,606	-$245,866
Grinding	$367,053	$156,510	$160,848	$204,137	$261,724	$268,455	$258,883
Milling	$1,030,760	$542,396	$275,969	$669,496	$1,048,726	$1,041,994	$876,840
GHG Credit	$138,054	$65,907	$42,466	$56,164	$85,940	$112,033	$157,708
Total	$2,039,885	$973,852	$627,480	$829,880	$1,269,860	$1,655,402	$2,330,305

Direct savings vary from $585 000 per annum at Fraser Copper to $2 173 000 per annum at Nickel Rim. Savings at Nickel Rim are maximized due to the depth of the deposit, a high degree of waste rejection obtained and the extreme haul distance planned for the operation. Accounting for the potential additional greenhouse gas credits, total savings range from $627 480 at Fraser Copper to $2,330,305 at Nickel Rim.

6.6.4 Conclusions

Nine principal ore types of Xstrata Nickel's Ontario operations have been tested to evaluate the potential for the application of the integrated mining, pre-cocnentration and waste disposal strategy for these deposits. The ores represent the majority of the present and future production Xstrata Nickel. Metallurgical results are good overall, with further work being required to maximize waste rejection and metal recovery for LGBX-type ores. Several of the Footwall ores show potential for the production of a direct shipping ore through pre-concentration. Due to geospatial constraints and the short mine life, pre-concentration is not expected to be utilized to benefit present operations. However, the two future operations, Onaping Depth and Nickel Rim, by nature of their increasing contribution to metal production, as well as their increased depth, and distance from Strathcona Mill present ideal candidates for the application of the proposed approach as a means of reducing waste and improving the efficiency, economics as well as environmental performance and sustainability of these operations. Geotechnical evaluation, backfill mix design and testing has shown that the rejects would be suitable for use in as conventional rockfills or as composite fill, with major benefits to be accrued in terms of fill strength and fill economics. The results of the testwork and subsequent evaluations have been used as inputs into a model to evaluate the overall impact on energy utilization for the mining and processing of these ores of Xstrata Nickel's Ontario operations. The savings in terms of energy are significant at each operation that was evaluated. It has been shown that hoisting, haulage and milling energy can be reduced by 9% – 15% overall, resulting in energy cost savings (including additional GHG credits) of between $0.6 - $2.3m per annum. Savings are maximized for Montcalm and Thayer Lindsley due to the significant saving in ore- and fill- haulage, and at Nickel Rim due to the high degree of waste rejection achieved, the depth of the operation, and the distance of the planned ore haulage on surface. Additional savings opportunities exist through the potential production of direct shipping ores, as well as the opportunity, with some modifications to the existing flowsheets, to produce custom feeds for delivery to the Strathcona Mill, Clarabelle Mill, or in the case of direct shipping concentrates, the Falconbridge or Copper Cliff smelters. These results must be further evaluated in terms of the synergies that have identified between Xstrata's Nickel and CVRD INCO.

The strategic application of pre-concentration to Xstrata Nickel's Ontario operations has thus been shown to have great potential in terms of reducing the energy requirements, and thus the Greenhouse Gas emissions, of a group of major Canadian mining operations. Further work in evaluating the impact of these results for both Xstrata's and CVRD-INCO's Ontario operations is strongly recommended.

6.7 Low Grade Ultramafic Ores of the Thompson Nickel Belt

6.7.1 Introduction

Following the successful conclusion of the work at INCO's Sudbury Operations, INCO was approached to initiate an expanded study for pre-concentration into the pre-concentration of low-grade ultramafic nickel deposits at INCO Thompson in Northern Manitoba as well as additional work on VMS nickel deposits at Voisey's Bay. Results of the INCO Thompson study are presented in this section. Work on the Voisey's Bay study will start in January 2008 and is not considered a part of the scope of this thesis.

The Thompson Nickel Belt (TNB) is a 10-35 km wide geologically complex igneous intrusive belt located on the boundary between the North West Superior Province and the Churchill Province of the Canadian Shield (Bleeker, 1990; Layton & Matthews, 2007). Since its discovery in 1961, two deposits in the region have been principally exploited, at the Thompson and Birchtree Mines. These operations are mature and have limited mineral reserves with which to keep the integrated Thompson milling and smelting complex operating. However, several other so called 'ultramafic' Ni deposits have been identified throughout the TNB, including the Pipe II, Kipper and Opswagan Deposits, but these are generally low-grade, low Ni tenor and high MgO deposits which have been found to be difficult to process using the present mill configuration. The presence of chrysotile MgO is a further challenge at the mill due to respiratory safety concerns and full respiratory protection is required when handling these ores. Ore has been mined from the Pipe deposit previously, but operations were ceased in 1986 due to the incumbent low Ni prices in conjunction with the problems mentioned above. Pipe is the southernmost of the Northern sequence of the ultramafic deposits in the Thompson

Nickel Belt, which include the Moak, Manasan, and Opswagan deposits. Ore mineralization can be classified into 4 main categories (Bleeker, 1990; Rubingh, 2006):

- Ni enriched metasedimentary sulphides (minor, not considered at Pipe)
- Extraparental massive sulphides external to the ultramafic sills
- Intraparental sulphides occurring as massive, vein, net-textured and disseminated sulphides within the ultramafic sill (see Figure 6.22)
- Breccia sulphides – massive matrix sulphides incorporating small zenoliths of schist wall rock (see Figure 6.23)

Pipe is thus representative of a class of large, highly ultramafic Ni deposits in the TNB which have been previously mined and processed, but are typically low in Ni:S ratio and high in MgO which makes them unattractive from a technical and economic perspective. However, INCO presently has resources of over 200 Mt of this type of ore, and due to the diminishing reserves of the high tenor, low magnesium ores at Thompson and Birchtree, is interested in assessing the present economic potential of mining and processing these ores, starting with an assessment of the Pipe II deposit. INCO had previously developed a type conductivity sensor for low grade ores, and it was decided to investigate the impact of pre-concentration, either by conductivity sorting or dense media separation (DMS), on the mining, transport and downstream processing of the Pipe II ore. UBC were approached to undertake pre-concentration research with INCO Thompson on the deposit for the purposes of assessing the potential for application of these technologies to Thompson ultramafic ores. Site visits for fieldwork and sampling were undertaken in April, June and August 2006. A prototype of the INCO sensor was made available to UBC and subsequently laboratory evaluation of the response of the core to conductivity sensing was undertaken in 2006, with a further planned phase of laboratory testwork on conductivity based sorting in 2007 should the ore prove amenable to the technology. Design criteria were developed from the laboratory testwork and a preliminary system design was undertaken at the scoping study level and evaluated with results for inclusion into the scoping study by INCO ITSL. Mining is to be by open pit, and options for the project included a 'small mine' scenario operating at 13 000 tpd delivering feed 54km to the existing Thompson mill, and a 'large mine' scenario delivering 45 000 tpd of feed to a custom designed mill located at Pipe (Penswick, 2007). Both scenarios were to be evaluated with- and without pre-concentration in order to develop comparative economics for the options. An order of magnitude study including metallurgical testwork and process development for the pre-concentration option was conducted in order to assess this, the results of which are presented in this section.

6.7.2 Methodology, Sampling and Metallurgical Testwork

The main purpose of the study was to assess the response of selected Pipe ore samples to conductivity-based sorting methods, and in particular the INCO 'B2' Conductivity sensor. The results of preliminary testing would be generalized to produce the basis of a metallurgical flowsheet for ore pre-concentration at Pipe, as well as the basis for the succeeding impact evaluation. Two systems were selected for the testwork: a commercial sensor supplied by Applied Sorting Technologies (AST) out of Sydney, Australia, and a conductivity sensor, the 'B2' sensor developed in-house by INCO through a previous project (Boucher, 2003). The purpose of the testwork was twofold: to determine the amenability of low grade nickel ores from the Pipe deposit to sorting; and comparison of the INCO B2 sensor to the commercial sensor in this application. Both sensors were installed on a commercially available conveyor belt fitted with a pneumaticly activated ejector system below the head pulley in order to undertake the actual sorting. The intention of the testwork was to deliver a technical result for the pre-concentration of the Pipe ore using the technology as well as a preliminary business case analysis for pre-concentration at Pipe using the parametric evaluation model previously developed at UBC. Major impacts to be quantified included any impacts to NPV as well as mine life for Thompson as the project could potentially add significant life to the Thompson mining camp.

Grab samples were obtained in May 2006 for preliminary evaluation. Core logging and mineralogical evaluation were undertaken by INCO ITSL in Thompson. The massive sulphides are of moderate tenor and have been determined to be the most economical ore to mine at Pipe. Mineralogically, the ore consists of pyrrhotite, pentlandite, chalcopyrite, cubanite, shear-hosted magnetite and a large proportion of chrysotile magnesium oxide spinel (Figures 6.24 & 6.25). However, mining of the intraparental and brecciated sulphides is considered challenging due to variable grade, high levels of internal dilution, and the presence of magnetite and chrysotile magnesium which negatively affects metallurgical performance in the Thompson mill (Drapak, 2006).

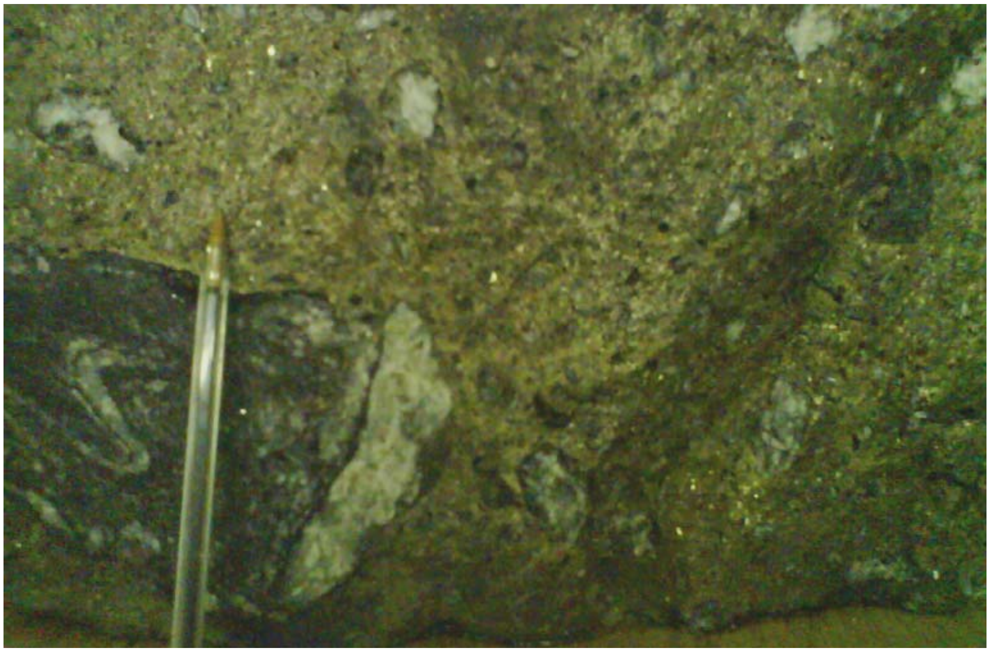


Figure 6.24 –Extraparental massive sulphide sample at Pipe. Note highly foliated nature of breccia inclusions that mimic the overall structural geology of the TNB

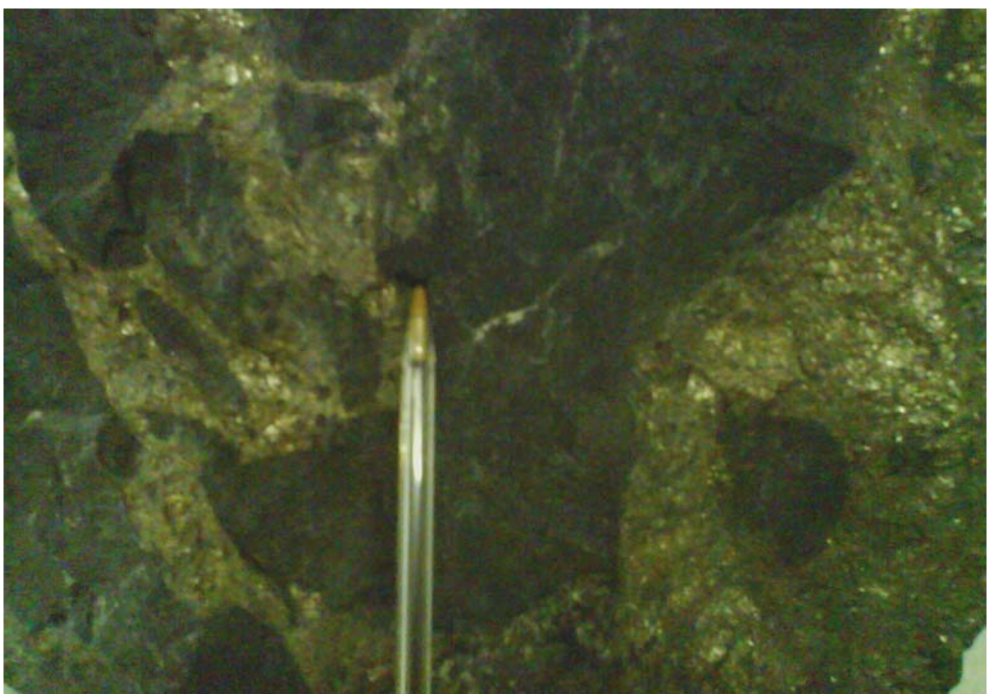


Figure 6.25 – Photograph of intraparental brecciated sulphide sample from Pipe. Note massive / net textured sulphides with incorporated schist inclusions

Preliminary metallurgical testwork was conducted to establish the amenability of the ultramafic Pipe type ores to conductivity sorting. Dense media separation testwork was not performed, although testwork on similar ultramafic ores indicates that comparable metallurgical results could be achieved using either process. During core logging and sensor calibration tests (see Chapter 4) a good correlation between static EM reading and % Ni was established for all four holes using both sensors. The performance varied between holes with variation in mineralization, and the presence of magnetite in BH115140 was identified as a potential problem. The MgO content measured showed a reverse correlation with the EM reading, therefore the potential for preferential rejection of MgO was indicated (Stephenson, 2006). Data from the calibration testwork was utilized in a synthetic sort in order to provide an indication of the performance of the AST and ‘B2’ sensors on these samples. Results are shown in Table 6.13.

Table 6.13 – Preliminary Sorting Results for Pipe II.

	B-2 Sensor	AST Sensor
Feed Grade	1.87 % Ni	1.87 % Ni
Pre-conc Grade	2.43 % Ni	2.08 % Ni
Ni Recovery	94.9 %	94.1 %
Mass Rejection	27 %	15.5 %

Both sensors indicated a moderate degree of waste rejection with high Ni recoveries. Based on the results of the amenability evaluation, a further test was performed on the mineralized zone of BH 115141 in order to provide a sample of sorter concentrate for downstream flotation testwork at Sheridan Park. An 80 kg sample of the core was crushed and screened into -50 + 26.9mm, -26.9 +10mm and -10mm fractions, then split in preparation for sorting using the sorting conveyor equipped with the AST and ‘B2’ sensors. The -10mm fraction was not sorted. Products from the sort were then sent to Sheridan Park for standard flotation tests.

Table 6.14 – Sorted Products for use in Comparative Flotation Testing

	INCO B-2 Sensor	**AST Sensor**
Feed Grade	1.03 % Ni	0.68 % Ni
Pre-conc Grade	3.21 % Ni	0.98 % Ni
Ni Recovery	68.4 %	79.9 %
Mass Rejection	78.2 %	45.2 %

Mass rejection using the ‘B2’ sensor was 78.2% at a recovery of 68.4% Ni. Mass rejection with the AST sensor was lower at 45.2% but higher in metal recovery. The results indicate potential for this technology to sort low grade ore with moderate mass rejection and high recovery. Flotation of sorter concentrate after grinding to -149um produced a concentrate grading 23% Ni and 2.5% MgO at a recovery of 70%. The product is considered acceptable smelter feed, with high Ni grade and low MgO content. However, overall Ni losses on the sample were considered too high. A further 1910kg sample of mineralized rock was selected from the old low-grade stockpile at Pipe and sorted (Figure 6.26).

Figure 6.26 – Low grade stockpile at Pipe. Note the relatively consistent particle size distribution of the rocks in the sample.

The sort was performed in order to provide a basis for conceptual design criteria, based on which an economic evaluation of the pre-concentration process for the ore in the 'small' and 'large' mine scenarios could be made. The sample was crushed and prepared into -300 +80mm, -80 +9.5mm and -9.5mm fractions, which were then sorted, sampled and assayed. The -9.5mm fine fraction was not sorted. Individual results of the sort vary widely with the type of mineralization. Sorter performance on the Massive and Brecciated sulphides is almost 100%, with lower results for the disseminated sulphides (Table 6.15, after Stephenson, 2006). Improving the sorting result for the most ultramafic of the disseminated-type ore proved difficult, however there is still potential to scalp residual Ni sulphides from this material if mined using this method. Ni grades are greatly increased in the sorting, which will massively benefit the Thompson mill in processing this ore. The preferential rejection of MgO is also indicated in the results with possible major positive impacts at the mill and the smelter.

Table 6.15 – Pipe II: Sorting results by Ore Type

Mineral Type	Feed Grade (% Ni)	Conc. Grade (% Ni)	Ni Recovery (%)
Brecciated Massive Sulphide	2.97	2.98	99.1
Extraparental Massive Sulphide	3.51	3.51	99.3
Ultramafic / intraparental	0.40	0.49	10.6
Ultramafic / extraparental	0.32	0.95	29.4
Disseminated matrix sulphide	1.86	2.07	87.2
Disseminated matrix sulphide	2.05	3.11	85.9

Based on these results, the pre-concentration of the Pipe ore by conductivity sorting on the -300mm +80mm and -80mm + 10mm fractions, either using the INCO 'B2' or AST sensors is strongly indicated as possible. Due to the coarse liberation size of the massive sulphides, fine crushing appears not to be required.

6.7.3 Design of a Surface Pre-concentration Facility for the Pipe II Deposit

The initial resource at Pipe was indicated by INCO to be 27Mt at a nickel grade of 0.76%. A preliminary mining and processing scenario was evaluated for this deposit at a mining rate of 3250 tpd. Results were marginally positive at a relatively conservative long-term estimate for the Ni price of $4.50/lb, due to low Ni grades and low expected metal recoveries from this ore in the present Thompson mill. In consideration of the pre-concentration results, it was decided to lower the resource cutoff grade and evaluate the exploitation of a potential 150Mt, 0.46% Ni deposit, to be mined at planned rates of 13 000 tpd and 45 000 tpd respectively (Penswick, 2007). Stripping ratio was to be 9:1, with a planned bench height of 15m. The PSD of the ROM feed was not known at this stage of the study, so this was modeled using the 'Kuz-Ram' blast fragmentation model (Cunningham, 1983). Basic open pit mining parameters were agreed with INCO in order to provide a blast design for input into the Kuzram model. A topsize of ~3m was indicated with a nominal particle size of 800mm. While the bench layout and blast design were considered by INCO to be optimal for a pit of this size, this feed size was considered unsuitable for economical crushing. Scalping of the +1.5m material in the pit for secondary blasting was indicated in order to prepare adequate feed for a sorting plant. The topsize of the feed to the pre-concentrator after secondary blasting was thus modeled at 1.5m, which is considered reasonable for which purpose a Metso 30-60 Gyratory was selected.

Process plant design was to be for a conductivity-based sorting plant processing either 13000 tpd (small mine scenario) preparing feed for introduction into the present Thompson mill, or 45 000 tpd (large mine scenario) preparing feed for a dedicated mill to be built at the Pipe II site. Plant operation was to be 355 days/annum, 7 days/week, 24 h/day with 85% design availability, resulting in a nominal feed rate to the crusher of 637tph (small mine) and 2200 tph (large mine). The fundamental plant design for the two scenarios was essentially the same, with primary differences being in the number of units selected for each duty. Sorter capacity is limited according to the topsize of material fed to the sorter, thus a relatively coarse crush size of 300mm was targeted for the design. Furthermore, the size range of ore reporting to the sorter is to be limited to a maximum of 4:1, thus coarse sorters operating on -300 +100mm and fine sorters operating on -200 + 100 mm for the small mine were required. For the small mine scenario, 2 Mikrosort® Primary sorters operating at 200 tph were required for the coarse sort, and 3 Mikrosort® Secondary sorters operating at 100 tph were selected for the fine sort.

In sorting, as in Dense Media Separation, good feed preparation, with the minimum of adhering fines is essential, thus feed to the sorters was to be wet screened at 10mm prior to sorting. The fine- and coarse sorter product is then to be combined with the bypassed fines and delivered to the concentrate stockpile for transport to the Thompson Mill. Waste products from the sorters are characterized during sorting into acid- or non-acid generating wastes and disposed of to the appropriate surface stockpile. The metallurgical performance of the sort was determined using the suite of results obtained from the scoping testwork as a basis. Significant waste rejection yet acceptable metal recovery was desired, thus from the data 30% waste rejection at 93% Ni recovery was chosen as the sorter operating point. The number of sorters required increases dramatically for the large mine scenario, and the proposed sorting plant design is presently considered unfeasible at this tonnage. A proposed flowsheet for the 13000 tpd option is shown in Figure 6.27 (courtesy BC Mining Research Ltd). The projected metallurgical performance of the plant is indicated in the Table 6.16.

Table 6.16 – Projected Metallurgical Performance of the Pipe II Sorting Plant

Metallurgical performance:	**Pre-concentrate Products:**	**Waste products:**
Waste rejection: 30%	-200 + 0mm	-200 + 10mm
Mass pull to concs: 70%	0.728% Ni	0.013% Ni
Ni Recovery: 92%	0.058% Cu	0.001% Cu
MgO recovery: 30%	16% MgO	

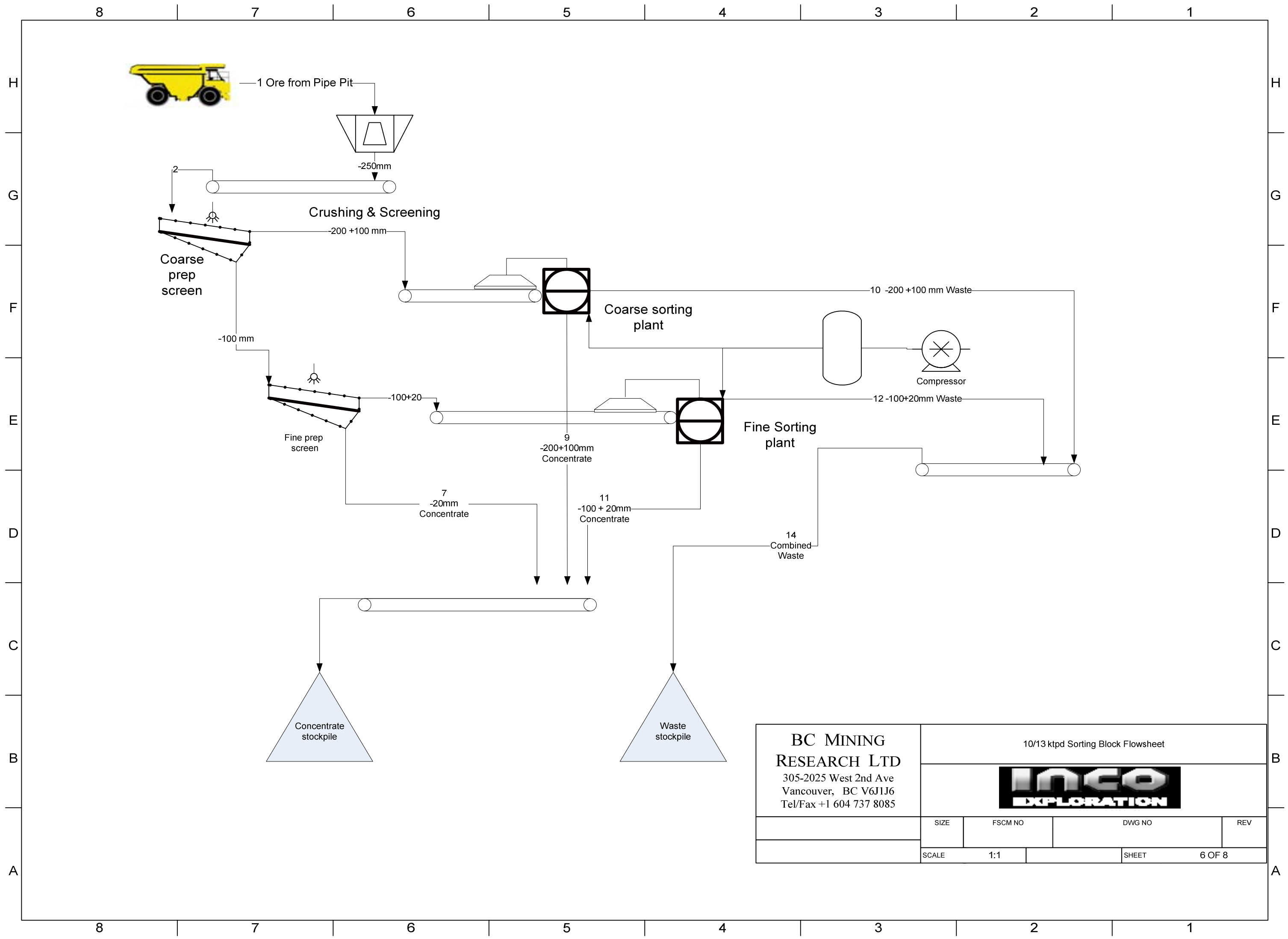


Figure 6.27 – Pipe II – 13 000 tpd Conductivity Sorting Flowsheet

Significant waste rejection is projected for the plant, with a 7% loss in metal values. Ordinarily this would constitute an unacceptable financial penalty as metal prices are high. However, as indicated by the testwork, there is preferential rejection of MgO during sorting with significant benefits in the performance of the mill. The Thompson Mill was designed with a capacity of 15 000 tpd, but is presently operating at nominally 10 000 tpd, and in the absence of new feed from Pipe or other local deposits will be reduced in future to 6000 tpd. The integrated metallurgical complex at INCO Thompson includes a conventional grinding and flotation mill, and a dedicated Ni smelter and refinery. The Thompson mill circuit is a simple grinding and flotation (MF1) circuit, with secondary scrubbing of the rougher tails prior to scavenger flotation and secondary recovery of copper concentrates by differential flotation. Primary grind is 149 μm. Flotation is generally by Denver-type flotation cells; however the mill was recently upgraded with a new MgO rejection circuit comprising Outokumpu OK80 tank cells with enhanced CMC depression. Combined Ni recovery on the Thompson and Birchtree ores presently mined is approximately 85% (Drapak, 2006).

Copper concentrate is shipped from Thompson mill to the Copper Cliff smelting and refining complex in Sudbury.

Ni recovery in the mill is highly sensitive to feed grade and magnesium content (Penswick, 2006):

$$R = 93 - \frac{(16.1*96.3\ \%)}{g} \qquad - (1)$$

Where g is the feed grade to the mill in %Ni (Penswick, 2007).

At an average feed of between 0.46 – 0.76% Ni, it is thus expected that the recovery of Ni from Pipe ores in the Thompson mill would range from 69 – 76%. The high levels of magnesium assayed in the ore also present a number of problems at the mill. Present levels of MgO minerals in the feed is found to decrease the pulp density and viscosity in the grinding mills, thus decreasing grinding efficiency. The MgO also grinds preferentially and reports to the flotation circuit as fine, hydrophobic talc slimes which tend to report to the flotation concentrate, lowering concentrate grade, and increasing filtration times as well as final moisture content in the cake. MgO in the final concentrate is a problem as it requires a higher temperature in the furnace to melt it to slag, which reduces the life of furnace linings. This can be countered by increasing the depression of MgO, however this also negatively affects Ni recovery. Pipe II ore at 0.76% Ni and 34% MgO contains over 100% more MgO than the

Birchtree ore, thus the planned milling of this ore involves severely exacerbates these problems (Table 6.17).

Table 6.17 –INCO Thompson Mill Operating Parameters - 2006

Feed	**Thompson**	**Birchtree**
Throughput	5300	4500
Ni	2.07	1.81
MgO	~9	17
BWI	14	13
Ni Concentrate		
Ni	14.4%	
Cu	0.29%	
MgO	3%	
Cu Concentrate		
Ni	4.1%	
Cu	16.4%	
MgO	3%	
Tailings		
Ni	0.28%	
Recovery	90%	80%

Recoveries on unsorted samples of Pipe ore indicate standard recoveries at the Thompson mill of 66.5% However, with sorting, the feed grade increases by 27%, MgO levels are reduced from 34% to 16%, and thus the mill recovery model indicates that mill recovery would improve to 71.9% which is in close agreement with the value from the flotation testwork (Stephenson, 2006). Thus the overall Ni recovery to concentrate on Pipe ore is unchanged with sorting, however the full operating cost saving is realized, thus the expected NPV of the savings in mining, transport and processing costs is maximized through the implementation of this technology. The impact of this result for the small- and large mine scenario has been evaluated using the parametric model that has been developed.

6.7.4 Parametric Evaluation of the Exploitation of the Pipe II Deposit with and without pre-concentration

Capital costs for the open pit mining fleet, surface plant and infrastructure are estimated automatically in the parametric model using modified O'Hara indices. The base case estimate for a 13000 tpd mine and surface mill located at Pipe is presented in Table 6.18. Capital costs were developed in using the parametric model and in consultation with CVRD-INCO and are considered accurate to ±30%. The estimate includes for the design and construction of a winterized plant, including a 50% provision for supporting infrastructure such as offices, laboratories, workshops, stores, HT power transformation & power distribution as well as in-plant roads. Access roads, overhead power lines, tailings facilities and closure costs are not included in the costs. The model indicates a capital requirement of $508.5m for the development of the small mine (Table 6.18).

Table 6.18 – Pipe II Model – 13000 tpd Base Case Capital Estimate

	Cost (2006 US$)
Mining	
Scoops/trams	$20,658,192
Haul trucks	$25,930,606
Pre-strip & boxcut	$7,205,848
Preproduction development	$149,935,165
Infrastructure	$25,432,404
Subtotal	**$229,162,217**
Plant	
Crushing & Screening	$38,148,607
Process Plant	$61,037,771
Concentrator building	$25,432,404
Subtotal	**$124,618,783**
Feasibility Engineering & Design	$88,445,250
Contingency	$66,333,937
Total	**$508,560,188**

Capital costs are automatically adjusted for the pre-concentration scenario in the model. Mining capital remains unchanged, and an additional capital amount of $54m is allowed for the construction of the sorting plant. Surface plant and infrastructure requirements are reduced to $98.3m for a flotation plant now treating 9000 tpd compared to $124.6m for the base case of 13 000tpd. Operating costs were also examined. Costs for the sorting facility include for power, labour, consumables and maintenance and are estimated from operating cost estimates for similar plants using scaling factors for the tonnage and are considered accurate to ±30%. Projected plant operating costs were $1.80/t for the small mine.

The impact of pre-concentration on the economics of the Pipe operation are thus substantial: capital costs are reduced overall by a small percentage; operating costs are reduced by 7% overall due to the 30% reduction in tonnage after pre-concentration. Both extraction of the resource and metal recovery are improved over the base case, with extremely favourable economic impacts. NPV of the base case without pre-concentration is estimated at $610.9m at a discount rate of 10%. The NPV of the small mine scenario with pre-concentration is increased over this figure by $371.6m to $982.5m.

After completion of the small mine design, the scenario was extended to the 45000 tpd scenario. The number of coarse- and fine sorters increases from 2 and 3 respectively to 6 and 6 respectively, which was considered presently unfeasible due to the increased cost of feed preparation and delivery as well as the raw cost of the sorters themselves. A 45000 tpd option for the pre-concentration of Pipe ore was thus not examined. Further research is required for the consideration of sorting over 15000 tpd. In the final analysis, the NPV of the small mine scenario with pre-concentration was compared to the base case NPV of the large mine without pre-concentration and found to be comparable. For a similar NPV and IRR, capital outlay and operating risk is lower which is an excellent indication of the extreme potential of this approach. Further development of the technology in terms of increasing, the power, sensitivity and accuracy of the sensors will also improve this comparison. Environmental and permitting impacts were also found to be significant as it was determined that same value can be extracted from the resource with a significantly reduced total surface impact. Waste management is also much improved for the small mine scenario with pre-concentration vs. all other scenarios. Further research and development on the sorting technology is required to improve the power, sensitivity and accuracy of the sensors. Further development is required also in the general field of sorting technology in order to develop sorters appropriate for the large mine scenario. Preliminary design work indicates that a sorter with a capacity of 400 – 600 tph would be appropriate for such a duty. For the project in question, a bulk sample is recommended to confirm and improve confidence in the metallurgical parameters. Sampling and testwork is recommended to be followed by a full pre-feasibility study to compare in detail the outcome of the pre-concentration scenario to the base case.

6.8 Conclusion & Recommendations

From the selected results of the research reviewed in this Chapter it appears that the integrated mining, processing and waste disposal approach can be demonstrated to engender significant benefits over the conventional approach in a number of cases. A wide range of case studies, including inter alia massive and disseminated sulphides, volcanogenic, meta sedimentary and paleo-placer gold deposits, MVT type lead zinc and copper porphyry ores have been investigated. Significant levels of waste rejection were identified for each ore using a range of pre-concentration technologies. Impacts and benefits were quantified for each ore studied. While the impacts of the proposed approach are marginal for shallow, high grade, low cost operations, impacts and benefits for most operations studied were significant due to a number of reasons, ranging from adverse geospatial factors, environmental constraints or the presence of deleterious minerals in the ROM material subsequently rejected in pre-concentration. Most cases studied were able to accommodate the generation of significant quantities of waste rejects, either as solid waste disposed of on surface, or as high performance cemented fill underground. From combined suite of results obtained, it appears that the successful implementation of the integrated pre-concentration and waste disposal approach relies on several factors:

- The occurrence of appropriate meso- and microtextural features in the ore which support the potential for pre-concentration
- Sufficient liberation of the barren gangue mineral at coarse particle sizes coupled with preferential fragmentation of the valuable mineral
- The existence of an appropriate basis of discrimination (colour, density, conductivity, surface chemistry) between the valuable and barren components in the ore
- Resultant high degree of waste rejection at high metal recoveries leading to high overall combined metal recovery to final concentrate
- Resultant decreases in Work Index of the pre-concentrate c.f. the feed ore
- Suitability of the waste product for disposal (physical, chemical, geotechnical)
- High strength and high binder efficiency of the resultant fill mix design for underground operations
- High workability of the optimum fill mix for optimum distribution to the fill stopes
- Existence/potential for a low-cost, high capacity unit process to perform the separation of ore and waste on the selected discrimination basis
- A high operating cost / revenue ratio at the operation

Based on experience from the studies, and the supporting data from the literature, there is a high probability of these factors occurring in combination for a wide range of deposits, thus it is believed that there is a general case for the consideration of this approach for deposits yet to be investigated.

7. Discussion & Conclusions

7.1 Grounds for the Generalized Application of Ore Pre-concentration and Waste Disposal in Hard Rock Metal Mining

The integrated underground mining and processing approach, through the integration of systems for the automated identification, rejection and disposal of solid waste from the ROM ore stream into the conventional hard rock mining system, is suggested as a realistic and practical alternative for the development of new mines, or alternately as an option for the expansion of existing surface or underground hard rock mines. The results from a wide range of case studies indicate that the exploitation of a deposit with ore pre-concentration and waste disposal technologies integrated into the mining process prior to beneficiation on surface is superior to the conventional approach. Preliminary work was undertaken at operations of INCO (now CVRD-INCO) in the Sudbury Basin, Ontario, Canada. Further case studies have focused on polymetallic base-metal sulphide ores, particularly those of Xstrata Nickel's operations in Ontario, Canada, although studies have also been conducted additional ore types from several locations worldwide. Geo-metallurgical aspects of waste rejection, as well as geotechnical issues relating to the disposal of the rejects as fill were examined. Waste rejections ranging from 14% to 66% with Ni, Cu, Pb, Zn and PGM recoveries averaging over 95% were achieved. Waste rejection and metal recovery on the copper ores in particular were high. The Xstrata Nickel study has recently been extended to examine the impact of underground pre-concentration at 2 future mines at the Ontario Operations division. Subsequent studies for CVRD-INCO focused on conductivity-based sorting methods for waste rejection from low-grade ultramafic Ni deposits of Northern Manitoba, with substantial success. Preliminary testing was conducted on drill core to assess the textural heterogeneity and thus the potential amenability of the ore to pre-concentration. Preliminary assessments were encouraging and good results were achieved during metallurgical testing, rejecting on average 30% waste with a Ni recovery of 93%, leading to significant projected savings on transportation costs and improved recoveries in the mill with major positive impacts resulting on the mine life and economics of the project. An important outcome of this study was that sorting of the ore preferentially rejected chrysotile MgO present in the rock, which has a deleterious effect on Ni recovery during conventional flotation. This preliminary characterization and metallurgical testing was followed by pilot sorting of a 2 ton sample that indicated similar metallurgical performance. A large-scale pilot programme is now planned on a 300 ton bulk sample, which will contribute results towards a pre-feasibility study for the open pit mining of the deposit. Further studies have included geo-metallurgical testing and impact assessments at Placer Dome's Musselwhite and South Deep Gold Mines, as well as additional research into the pre-concentration of MVT-type ore at the Doe Run Company in Missouri, and preliminary investigations into the pre-concentration of Cordilleran copper porphyry ores. Maintaining a high degree of metal recovery together with a high degree of waste rejection is paramount to achieving a positive overall operational benefit.

A large amount of supporting data has also been collected from the literature on historical testwork detailing pre-concentration parameters leading to positive benefits at the mines studied. Ores covered in the literature include platinum-bearing chrome ores of the Bushveld Complex, low grade nickel ores from the ultramafic deposits of Eastern Botswana, lead-zinc ores from a wide range of mineral districts, iron, copper and nickel deposits of the Scandinavian Peninsula, gold ores of the Witwatersrand as well as copper porphyry ores from the Cordilleran region of the Americas and Papua New Guinea. From these results it appears that ore heterogeneity as well as high levels of dilution is more common than not and that there appears to be a general case for the pre-concentration of ore prior to conventional fine particle processing.

This data from the literature has been collated and augmented with additional data from the case studies to create a data set of over 100 samples in order to investigate this concept. Results are presented in Figures 7.1, 7.2 7.3 and 7.4.

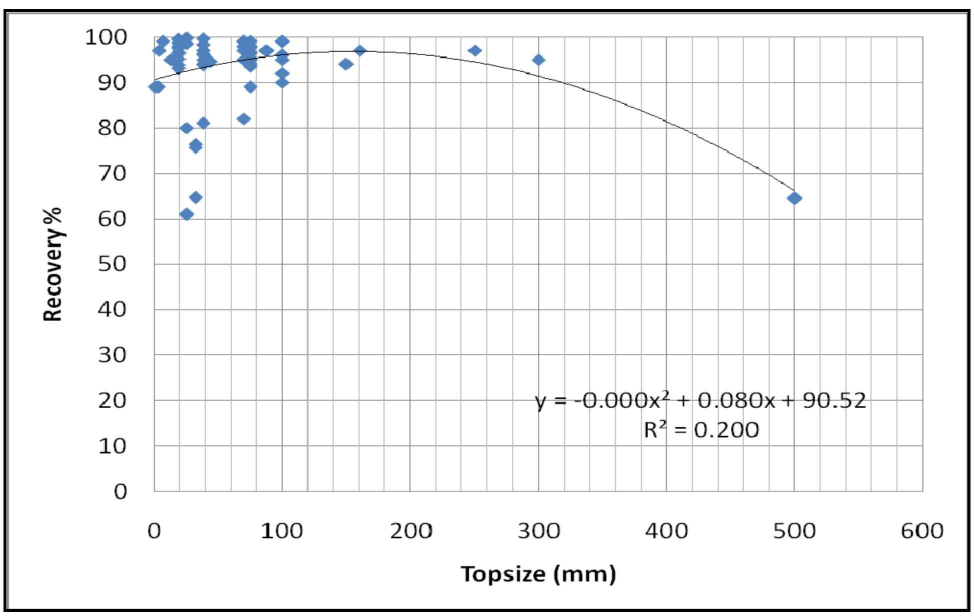


Figure 7.1 – Metal Recovery by Topsize (n = 106)

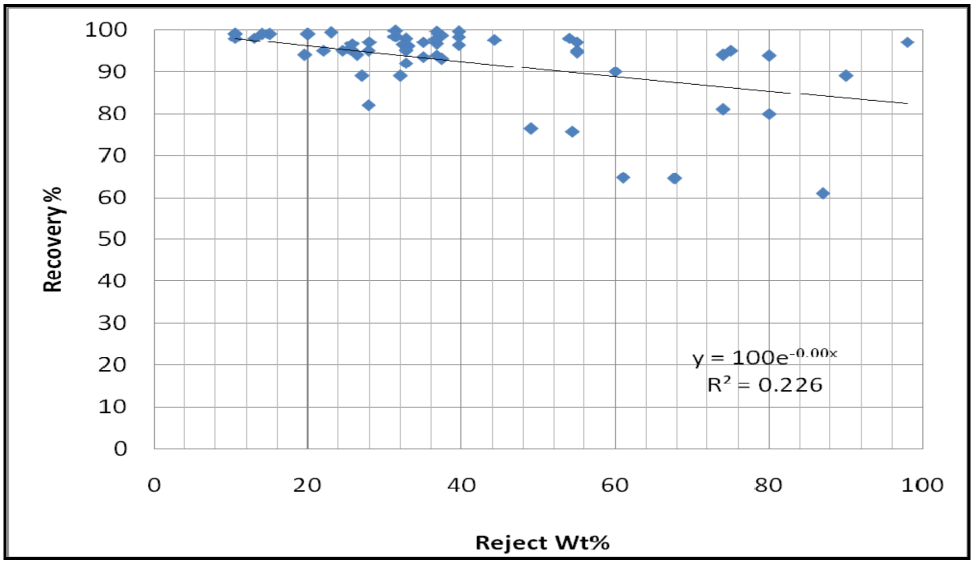


Figure 7.2 – Metal Recovery by Reject Wt% (n = 106)

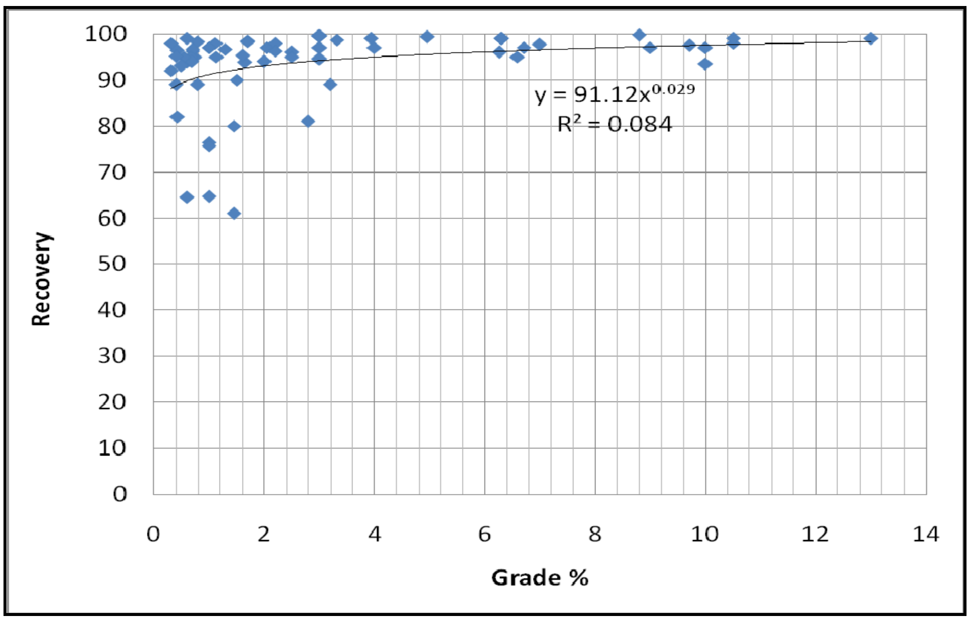


Figure 7.3 – Metal Recovery by Feed Grade (n = 106)

From the graphs it can be seen that the degree of metal recovery is consistently high across the full range of results. Recovery falls considerably as the topsize of material processed increases above 200mm; however, the cost of processing material at these sizes is considerably lower which must be taken into consideration in the evaluation. Metal recovery can also be seen to fall below 90% as the data moves into the realm of true fine particle processing, where the degree of waste rejection increases above 60% and feed sizes to the treatment process fall below 1mm. A potential performance envelope of 30-50% mass rejection at metal recoveries between 95-99% appears to be reasonable for a well designed separation process on most ores that have been investigated.

It is important to examine the results of the general case in terms of the quantification of benefits. The data from the general waste rejection model has been input into the evaluation model for an idealized case study of a medium depth Cu-Ni mine producing nominally 3000 tpd ROM ore. Results are presented in Figure 7.4.

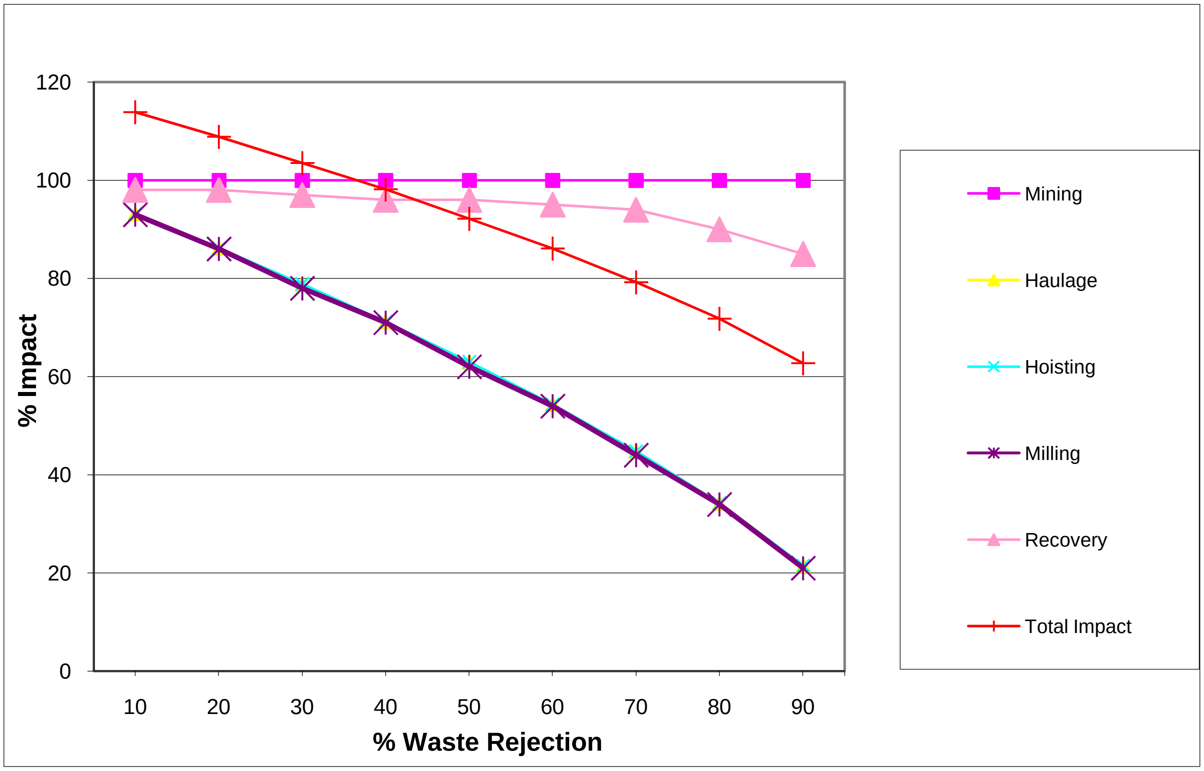


Figure 7.4 – Projected Collective Capital, Operating, and Revenue Impacts from the Model

It can be seen from the graph that savings are enjoyed in direct proportion to the degree of waste rejection. The projected operating cost savings margin is between 40 – 60% of the waste rejection value. Evaluation of capital impacts indicate that the cost of the pre-concentration facility is fully offset by projected capital savings in the surface mill and infrastructure for waste rejection over 30% by weight. For metal recoveries in pre-concentration of between 97-99%, the impact of pre-concentration on overall metal recovery, and thus revenues is negligible. For pre-concentration losses over 5% the model predicts overall metal losses to accrue. For several of the case studies, in particular the Pipe II study, metal recovery is predicted to increase overall due to the preferential rejection of talc prior to final treatment in the mill with a consequent increase in the positive impacts enjoyed.

7.2 Integration of the Enabling Technologies

Integrated mining, processing and waste disposal involves the rejection of barren waste at a coarse particle size, and preparation and delivery of this waste as a high performance fill material in the mining void. The application of the concept is possible with a number of mining methods: simple waste disposal for methods such as open stoping, in which it is possible to accommodate fill, and more importantly as a means of improving the quality and cost of support in methods that rely on backfill for success, such as cut-and-fill or AVOCA methods. Particular benefits have been identified for open pit operations where pre-concentration, and in particular sorting, can be used to classify waste into AG and NAG wastes, as well as delivering a high grade product to the mill. Great benefits are also envisaged for underhand cut-and-fill operations at deep deposits where rapid development of high strength in the fill is important to enable quick re-entry into the stopes. Several enabling technologies have been identified, including photometric sensing and sorting, conductivity sensing and sorting, dense media separation, as well as novel separation techniques such as comminution and size classification and coarse particle flotation. The processes that have been researched are all low-cost, high-capacity mineral separation processes with good potential for this application. Applicable waste disposal technologies investigated include conventional paste fill systems as well as novel composite fills and related preparation and delivery systems. A detailed geo-metallurgical characterization procedure has been developed which incorporates site work and sampling, visual mineral texture evaluation, size-assay, optical liberation analysis, and detailed physical evaluation of the ore including determination of ore density, conductivity, grinding Work Indices, and characterization of additional geophysical and geochemical properties. The procedure enables both the characterization of the ore for pre-concentration, characterization of the rejects for disposal as fill, an evaluation of the degree of pre-concentration achievable by the various separation methods, as well as a preliminary evaluation of the potential benefits of pre-concentrating the ore to be made.

Several tools, including an improved laboratory scale dense media unit, a photometric ore evaluation station, and an improved inductance-based ore evaluation station have been developed and utilized in the course of the research. Further to this a parametric impact valuation and evaluation tool has also been developed, incorporating the automated estimation of cost and revenue impacts, modeling and calculation of impacts on cutoff grade and mineral reserves, and calculation of the overall impact on deposit exploitation in terms of cashflow, life of mine and NPV. It has been concluded that the feasibility of ore pre-concentration is maximized with the existence of several of the following factors in combination:

- The occurrence of appropriate meso- and microtextural features in the ore which support the potential for pre-concentration
- Sufficient liberation of the barren gangue mineral at coarse particle sizes coupled with preferential fragmentation of the valuable mineral
- Existence of an appropriate basis of discrimination (colour, density, conductivity, surface chemistry) between the valuable and barren components in the ore
- Resultant high degree of waste rejection at high metal recoveries leading to high overall combined metal recovery to final concentrate
- Existence/potential for a low-cost, high capacity unit process to perform the separation of ore and waste on the selected discrimination basis
- Resultant decreases in Work Index of the pre-concentrate c.f. the feed ore
- Suitability of the waste product for disposal (physical, chemical, geotechnical (UCS))
- High strength and high binder efficiency of the resultant fill mix design for underground operations
- High workability of the optimum fill mix for optimum distribution to the fill stopes
- A high operating cost : revenue ratio at the operation

7.3 Benefits of the Approach

Based on the outcomes of previous work, the benefits of the successful application of pre-concentration would be significant and manifold. Impacts and benefits predicted through sampling, testwork, and parametric modelling are expected to include, inter alia:

- Rejection of between 30 -60% of the Run-of-Mine ore as waste
- Metal recoveries in pre-concentration of between 93 – 99%
- Operating cost savings between 15-30%
- Marginal impacts to capital costs for pre-concentration, grinding and flotation compared to grinding and flotation alone
- Resultant increase in profit margins of between 10 – 20%
- Potentially higher overall metallurgical recovery compared to unsorted ore (specifically on well liberated, highly diluted low grade ores)
- A 30 – 60% increase in process plant capacity

- Improved quality and consistency of ore as a product of pre-concentration
- A reduction in transport costs of between 15-30%
- A reduction in energy consumption for mining and beneficiation
- Reduced water consumption
- Disposal of the waste rejects as a competent fill material underground
- 30 – 60% reduction in disposal requirements for the coarse dry waste compared to fine, saturated waste
- A 10 – 20% increase in ore reserves due to the collective lowering of costs and the facilitation of the mining of ore reserves presently below cutoff grade

Further benefits in terms of the scientific classification of solid wastes into acid- and non acid generating types prior to disposal on surface, as well as the opportunity for improving the performance of underground backfill with the use of pre-concentration rejects further supports the argument for consideration of this approach. However, several additional unforseen benefits which are often out of proportion to the expected result have been quantified through the course of the research. Specific impacts and benefits quantified include:

- The potential to produce a direct-shipping concentrate from 'Footwall' type ores of the Sudbury Basin through the rejection of up to 55% by mass of non-sulphide material
- The opportunity to hoist up to 20% more material at the Thayer Lindsley mine in Sudbury, while reducing fill requirements by 50%
- The opportunity to reject up to 60% of the ROM material from low grade disseminated ores at Montcalm prior to a 100km surface haul to the mill
- The opportunity to reduce trucking requirements through Sudbury Town for the future Nickel Rim operation by 40%
- The opportunity to reduce overall energy consumption at Xstrata's Ontario operations by 8.8% through a strategic application of pre-concentration at the individual operations
- The opportunity to either increase the strength of cemented fills by 280% over conventional paste fills or alternately reduce the binder consumption in the fills by 70% while maintaining fill strength
- The opportunity to reduce costs at Pipe by 15%, while simultaneously increasing Ni recovery in the mill by 3%, enabling exploitation of the deposit at a significantly lower mining rate and lower environmental impact with the same or greater NPV

- The opportunity to increase operator safety in the Thompson mill by reducing respiratory risk through the preferential rejection of chrysotile MgO at coarse particle sizes prior to grinding and flotation at the mill
- The opportunity to transform an operation such as South Deep from a loss-making operation into a profitable operation through the rejection of the non-valuable material prior to hoisting the ore 3000m to surface
- The opportunity to increase production, decrease costs, lower the cutoff grade and increase the mineral reserve at the Doe Run Company's SEMO Operations in Missouri by up to 50%, while minimizing the disposal of tailings on surface through the rejection of barren dolomite from the ore underground
- Reduction of the cutoff grade to 0% Cu at Bougainville Mine in PNG through the rejection of up to 60% of the ore at a metal recovery of 80% by simple crushing and scalping of the upgraded undersize material
- Reduction of the cutoff grade to 0% Ni at low grade open pit operations such as Pipe, Bougainville and Tati Nickel through the pre-concentration of low grade ores

7.4 Significance of the Research

Research and testwork has been conducted for a wide range of ores from mineral districts in Canada as well as globally (Figure 7.5). The results of the research are significant for the enhanced exploitation of a number of types of mineral deposit in Canada and throughout the globe which are presently marginal, or difficult to exploit through technical challenges such as depth, remote geography or poor ground conditions. A wide range of ores have been shown to be amenable to pre-concentration and an appropriate envelope of operating cost to ore value ratios has been defined within which a significant degree of waste rejection at acceptable metal recoveries can deliver a significant economic and environmental benefit to the operation.

Figure 7.5 –Sites for Research Data from the Literature and the Testwork

There are several mineral districts which are expected to benefit from the application of the research outcomes. These include:

- low grade ultramafic Ni ores of the Thompson Nickel Belt type
- deep massive sulphide ores of the Sudbury Igneous Complex
- deep, narrow-vein UG2 ores of the Bushveld Igneous Complex which are presently uneconomical to exploit through conventional methods
- deep Elsberg and VCR-type ores of the Witwatersrand basin which are presently uneconomical to exploit
- Cordilleran type copper porphyry ores at mines such as Candelaria and North Parkes mine which are deep and in remote or arid areas

7.5 Conclusions

Major aspects in the consideration of this approach as a realistic and viable alternative to the conventional exploitation of hard rock metal deposits have been addressed in the thesis:

- The establishment of a significant case for the introduction of a new approach for the enhanced exploitation of marginal hard rock metal deposits for increased sustainability
- The identification of technologies, which, when integrated into the hard rock mining system, engender significant technical, economic and environmental benefits
- The establishment of a comprehensive system definition and context for the approach

- The development of bespoke mineralogical, metallurgical and geo-metallurgical procedures for the evaluation of the potential, designs for and evaluation of the benefits of ore pre-concentration and waste disposal systems
- The development of a parametric tool for the valuation and evaluation of the proposed approach by comparison to conventional methods of extraction
- Quantification of results from a comprehensive set of case studies comprising extensive sampling, testwork and evaluations indicating potential applications and favourable impacts and benefits of the approach in several cases
- A rigorous and objective environmental assessment of the impacts of the concept from the perspective of energy usage and surface impacts

A range of potential pre-concentration technologies have been tested at UBC, and tools for the evaluation of characterization of the ores, in particular optical and conductivity methods, have been developed through the research. However, pre-concentration of the ore is not always indicated and the results obtained do not always translate into a positive impact on the exploitation of the deposit. Economic feasibility for the concept has been found to principally depend on the degree of waste rejection, the combined metal recovery of the pre-concentration and final processing step, and the operating cost:ore value ratio for the particular operation. The success of integrated pre-concentration and waste disposal systems as an application of Lean Manufacturing in hard rock mining relies on the successful combination of several factors which have been clearly identified and quantified in the research. The approach impacts the triple bottom line of ‘economics, environment and community’, and thus its impact on the sustainability of hard rock mining has now been established beyond the proof-of-concept stage. The approach is thus proposed to mining companies for automatic consideration as a logical and viable alternative to conventional exploitation methods, either at the design stage of future underground hard rock mines, or as an option for the expansion of existing mines.

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Appendix A - Publications List

Bamber, A.S. Klein, B., Morin, M., Scoble, M.J., 2004, Reducing Selectivity in Narrow-Vein Mining through the Integration of Underground Pre-concentration, *Proceedings IV CANMET International Symposium on Narrow Vein Mining Techniques*, Val d' Or, Quebec, October 2004

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Appendix B – Parametric Valuation and Evaluation Models

Appendix B1 – Musselwhite Evaluation Model (attached in separate file)

Appendix B2 – Pipe Evaluation Model (attached in separate file)

Appendix C - Variations to the Idealised Grade Tonnage Model

The idealized grade tonnage model used in the parametric model is not expected to represent all ore types equally. In previous papers (Bamber et al, 2004; 2005) an amenability to pre-concentration has been demonstrated for a diverse range of ore types, including massive stringer sulphides, massive sulphides, banded-sulphides, as well as gold-pyrite and gold-conglomerate deposits. The results as well as the literature indicates that similar degrees of pre-concentration can be achieved on a wide range of ores, including massive sulphides, laterites, copper porphyrys, Pb-Zn sulphides as well as Witwatersrand gold ores, and thus consideration of the impacts on deposits of these types is considered important. The approach is less applicable to extremely low grade or highly disseminated ores due to the indistinct difference in properties for discrimination between ore and waste. The impact of changing cutoff grade on tonnage will be different for different ore types. For illustration of this point, a number of significant ore types from previous studies have been selected for discussion.

a) Massive hydrothermal deposits

Deposits of this type include significant base and precious metal producers such as Eskay Creek in British Columbia and isolated Footwall deposits of the Sudbury Igneous Complex, such as the original Thayer Lindsley Mine and the present Nickel Rim. There are a decreasing number of this type of deposit, although the potential for significant new discoveries does exist:

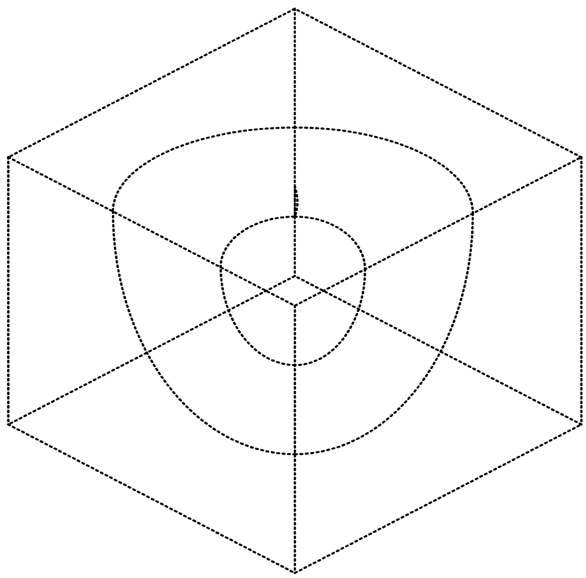


Figure 5.13 – Schematic of massive hydrothermal deposit

In this case, the distinction between ore and waste is significant and distinct. Pre-concentration of this ore is generally not necessary, and the production of direct shipping ores is possible. The typical grade tonnage curve for such deposits would look as in Fig 4 (after Lacy, 1969).

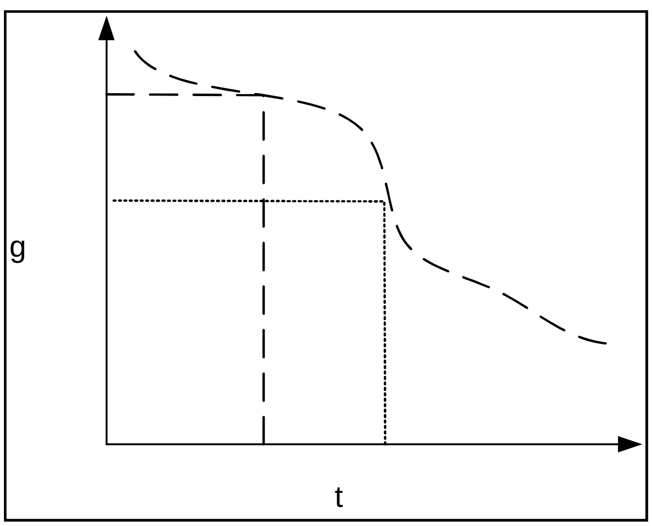


Figure 5.14 – Massive hydrothermal grade tonnage curve

A small change in cutoff grade does not significantly change the tonnage profile, however beyond the lower grade limit, added tonnages can be brought into the reserve through a drop in operating costs.

b) Massive-stringer sulphide deposits

These are more regularly mineralized massive sulphide deposits with the valuable minerals constrained by massive interleavings of gangue. This ore type is typical of footwall deposits of the Sudbury Igneous Complex, some Bushveld UG2 platinum ores, as well as the paleao-placer Elsburg type reefs of the Witwatersrand basin.

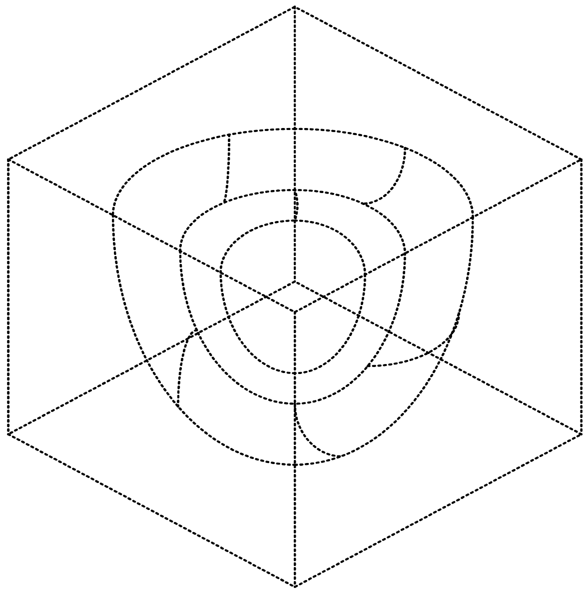


Figure x – Massive Stringer type deposits

Mining methods are typically selective, and thus high cost, and cutoff grades are consequently high and large proportion of mineralized rock is typically left in the ground post-mining, with obvious negative economic and environmental (ARD) consequences. However, a high proportion of low grade waste material is included within the defined ore blocks, most of which would be targeted for rejection by the pre-concentration plant. Undertaken at the correct liberation size, pre-concentration of this type of ore is possible with a high degree of waste rejection at extremely high metallurgical recovery due to the extreme physical discrepancy between the ore and waste. The grade tonnage curve for such deposits is shown below (Gentry & Neil, 1984:177).

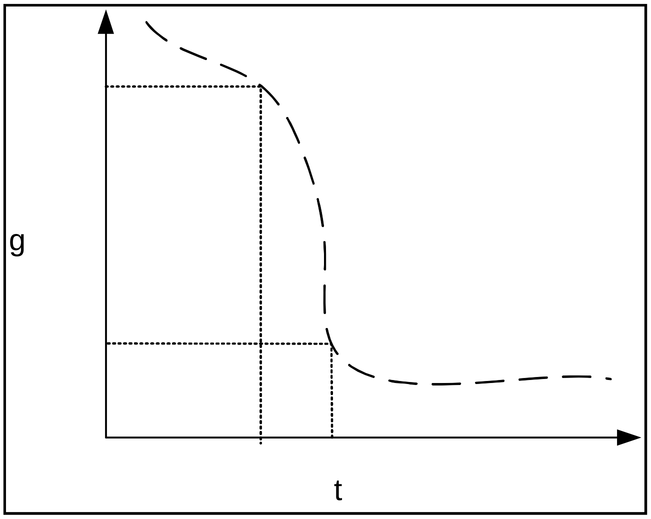


Figure x – Grade tonnage for massive stringer

As in the idealised curve, a small change in cutoff grade does not significantly change the tonnage profile. However, the margin of profitability is greatly enhanced and also lowering the cutoff grade to below the lower waste limit can bring large tonnages into the reserve.

c) Massive sulphide deposits

An important class of mineral deposits, representing a large proportion of existing mineral inventories, as well as potential future discoveries, is massive sulphide ores. Ore values are more dispersed with localized zones of higher concentration (Figure x).

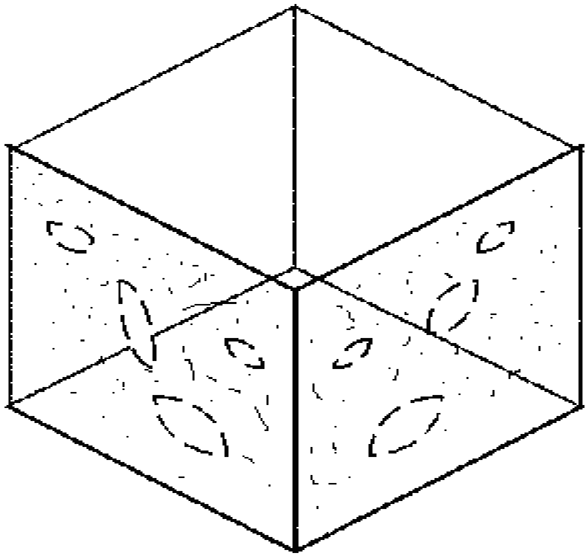


Figure x – Massive sulphides

Massive sulphide deposits are characterized by coarse mineralization with mineral values confined within gangue material. High grade zones are common. Mining methods are typically mechanized methods such as post-pillar cut and fill or blasthole/longhole methods. Pre-concentration of the ore with a moderate degree of waste rejection at good metallurgical recoveries is possible for these ores (Bamber et al, 2005), and substantial cost savings can be realized in deeper deposits. Additional planned and unplanned dilution in these ores can be 20-30%, which would also be removed through pre-concentration. An additional benefit can be the recovery of the low-grade metal values in the dilution though pre-concentration.

An idealized grade tonnage curve for a massive sulphide deposit with several high grade zones is represented below:

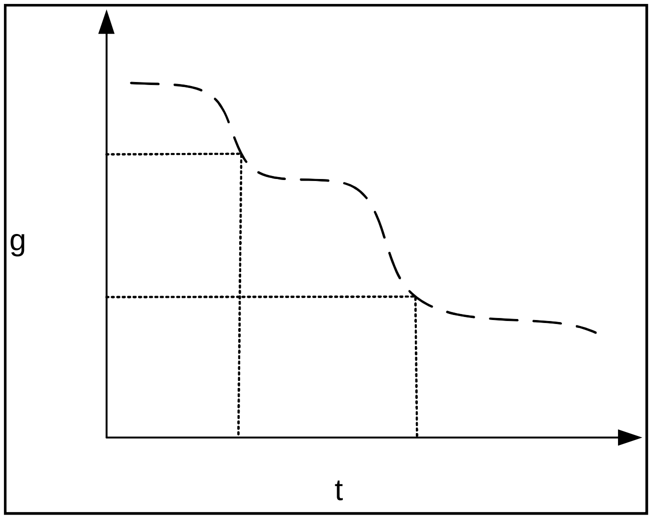


Figure x – Massive Sulphide Grade Tonnage Curve

In low cost operations or alternately a situation of high metal prices, the ore may be unselectively mined. However, a decrease in metal price, or increasing cost with depth or extent of mining can massively increase the selectivity required of the mining method. This isolates large portions of the ore necessitating a change in mining method which can be costly in capital and operating costs. Pre-concentration of the ore can bring tonnage into the mineral inventory, but more importantly regularize the grade of ore mined and thus stabilize the profit margin of the mine, thus stabilizing the conditions under which a particular cutoff grade and mining method are profitable.

d) Porphyry deposits

Porphyry deposits are characterized by dispersed and discontinuous mineralization interspersed by gangue material. Mining methods are typically high productivity mechanized methods or bulk mining methods such as VCR stoping or even block caving. Pre-concentration of the ore with a moderate degree of waste rejection at good metallurgical recoveries is still possible (Munro et al, 1999), and substantial cost savings can be realized in deeper deposits. High grade zones, in the case of porphyrys areas of supergene enrichment, are also common. Dilution can be 15-20% which could possibly be rejected. A typical porphyry orebody model with dispersed and discontinuous values is shown:

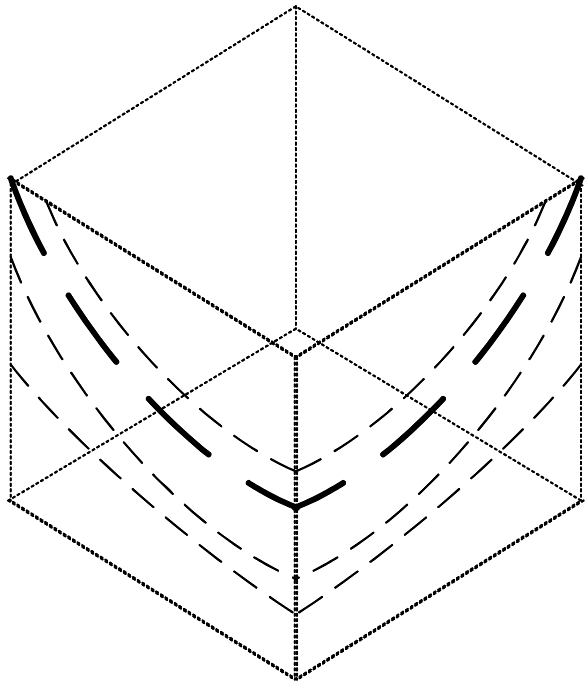


Figure x – Copper Porphyry Model

The cutoff grade curve for such a deposit is represented in the figure below:

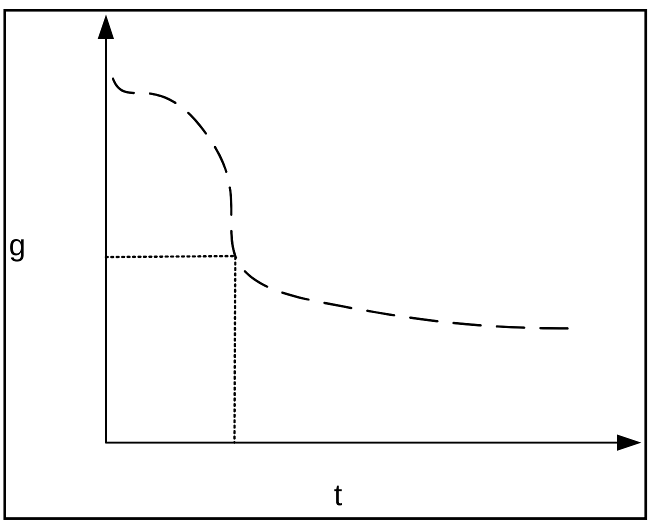


Figure x – Grade Tonnage for a Copper Porphyry Deposit

As improved technology enables the cutoff grade to be lowered, a massive increase in economic tonnage is realized for these types of ore.

e) Disseminated sulphide ores

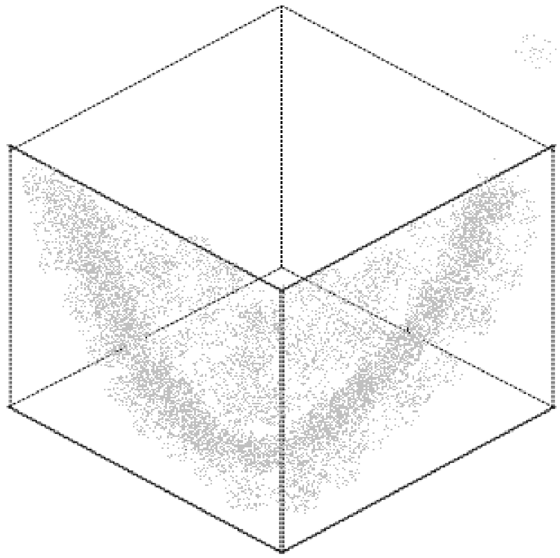


Figure x – Disseminated sulphides

Such deposits, where ore values vary continuously and uniformly throughout the orebody, are common. A range of mining methods are possible depending on the thickness and grade distribution of the ore zone, and costs can vary but are generally low due to the typically low grade of such disseminated deposits. Pre-concentration of this ore is difficult as can be seen by examination of a typical grade-recovery curve for this ore: either the proportion of waste rejected is low, or there is a high degree of metal loss due to the need to raise the level of rejection criteria to maximize the percentage rejection. Pre-concentration is practiced more commonly on these ores in order to simply reject low grade ore streams from high grade, or to reject deleterious elements, such as high phosphorous in iron ore, or high MgO in base metal sulphide ores. These deposits represent most closely the idealized grade-tonnage curve. Changes in method, technology or cost structure and thus cutoff grade result in a massive increase in reserve tonnage.

Appendix D – Integrated Mining and Processing at Placer Dome Group

1. Introduction

A scoping study into the potential for underground pre-concentration at Placer Dome's Musselwhite and South Deep Mines has been conducted by BC Mining Research for Placer Dome Technical Services. The study comprised desktop data gathering, sampling and metallurgical evaluation. A literature review has been conducted and an evaluation model was developed for the study and both have been submitted under separate cover. This report summarises the results and interpretation of the results of the investigation, preliminary outputs from the evaluation model and makes recommendations for future work.

The primary aim of this project was to complete a scoping evaluation to determine the amenability for and the potential for the application of integrated underground mining and milling for two Placer Dome sites in order to motivate further development of the concept for PDG. Two operations, Musselwhite Mine in Northern Ontario, and South Deep Mine in Carletonville, South Africa were investigated.

Basic mining parameters such as tonnage, depth, grade mining method etc, and physical features of the existing operation that are presented have been developed from information supplied by PDTS and from public domain literature on the operations. The results of the basic amenability evaluation for pre-concentration of the ore and selected results from testwork by PDSA on South Deep ore are presented for discussion. Details of the opportunity, ideas for implementation, as well as projected impacts and benefits of underground pre-concentration at the two mines are presented.

A financial evaluation using the model which has been configured for the purposes of this study has been undertaken using data from literature, material and reports supplied by PDTS, and the results from the amenability evaluation by BC Mining Research.

2. Background

Substantial research has already been done into the underground pre-concentration of both gold ores (particularly Witwatersrand ores) and massive-sulphide base-metal ores. P.J.D. Lloyd, of COMRO in South Africa, investigated the potential of an integrated mining and processing system for deep gold mines in some depth between 1978 and 1989. Testwork was conducted at the pilot scale for an underground pre-concentration facility producing coarse waste rejects for backfill utilizing coarse particle flotation followed by scavenging of the flotation tails using gravity concentration. The research concluded that underground pre-concentration was possible on such ores producing a concentrate of 6.53g/t from a feed of 4g/t through the rejection of 60% by mass at a metallurgical recovery of 98%. The coarse particle size was targeted in order to maximize the strength of backfill made from the waste, resulting in an expected UCS of 10 – 20 MPa for the fill, with the addition of 30% cement by mass. This improvement in the quality of cemented backfill was predicted to lead to an 85% improvement in ground conditions, resulting in a reduction in engineered support requirements, a decrease in uncontrolled rockbursts as well as decreased ventilation requirements post-fill. Such n improvement in ground control can potentially increase the safe depth at which mining can occur. Furthermore, there would be an increase in hoisting capacity through the rejection of waste, and a further opportunity to hoist the concentrated ore using hydraulic hoisting. A brief comparison of the economics of producing

50% backfill from underground waste produced through underground pre-concentration, versus hoisting the waste to surface for a 240 000 tpm gold mine, mining an average grade of 6,6 g/t indicates potential operating and capital cost savings to the mine of 40%. A further benefit arises from the basic assumption that with a constant tailings grade, recovery at the surface metallurgical plant will improve with increasing feed grade, offsetting the recovery penalty of the underground pre-concentration step. On this basis it is considered that underground pre-concentration has great potential to improve the economics of mining deep or otherwise marginal orebodies.

From previous scoping studies and evaluations, cost savings through underground pre-concentration are derived in a number of ways. Savings are generally proportional to the percentage of mass rejected underground through pre-concentration. A maximum waste rejection of 60% by mass has been determined through previous work as optimum due to space constraints consequent from the bulking effect on broken density over the in-situ density of ore. An estimate of the cost saving potential for South Deep, and the basis of the saving is explained below:

- Mining – 10%: potential savings in mining costs are derived through an increase in productivity at the mining face through a reduction in selectivity required and thus a potential increase in the minimum mining width allowable.
- Backfill – 50%: savings in fill costs are possible through the generation of a large quantity of cheap backfill material close to the stopes as a by-product of the pre-concentration process. The balance of backfilling cost constitutes the batching of waste rejects with water, sand and cement from surface, and the cost to place the cemented backfill. Cost impacts arising from consequences such as a reduction in rockbursts and greater stope- and drift reliability through improved ground control are not accounted for here.
- Hoisting – 30%: savings in hoisting are directly proportional to the proportion of waste rejected. However, assuming a ratio of 50% fixed to 50% variable costs, a 30% saving only is expected from a 60% reduction in the tonnage of ore.
- Power – 30%: hoisting and material handling constitutes more than 65% of all energy usage in underground mines. A reduction of 60% in the quantity of material to be hauled, hoisted and handled on surface results in an estimated 30% savings in the overall power cost for the operation. Additional power savings are derived from the elimination of a large quantity of hard siliceous waste in the feed to the mill, with a consequent reduction in grinding effort, as well as in overall tons to be processed in the mill.
- Milling and tailings disposal – 30%: 60% of the ore has been rejected underground prior to surface processing. This results in lower unit cost/ROM t for processing and tailings disposal. Savings due to a reduction in Bond Work index and thus grinding effort are noted above.

Negative cost impacts – underground pre-concentration constitutes an additional processing cost which must be accounted for. The metallurgical processes under consideration are typically cheap, high capacity coarse-particle processes such as sorting, dense media separation, and coarse-particle grinding and flotation. Furthermore, crushing is commonly undertaken underground, the unit cost of operating and maintaining this is included in the typical cost breakdown. Pre-concentration would incorporate the cost of underground crushing as part of the feed preparation process; in previous studies, pre-concentration costs have been estimated at an additional 2.5% of the base cash cost. This must be included in the evaluation. The cost of waste disposal is accounted for in the residual backfill cost as described above.

3. Pre-concentration of Ore at Musselwhite Mine, Northern Ontario, Canada

3.1. Background – mining geology and processing

Musselwhite uses both open-pit and underground mining, open-pit production having been designed to ensure mill feed at a rate of 3300t/d for about five years following mill commissioning. In 1997, phase one underground production from the T Antiform (T-A) deposit built up to a design rate of 2700t/d, supplemented by ore, at a rate of 1000t/d, from the PQ zone. The T-A rate was increased to 3300t/d by expanding the fleet of 40t haul trucks.

3.1.1. Geology and Mineralogy

The Musselwhite Property is within the Weagamow/North Caribou Lake Greenstone Belt of the Sachigo Subprovince of the Archean Superior Province. Supracustal rocks have been regionally metamorphosed to amphibolite grade. At least two major deformational events have occurred. There is no consensus as to whether Musselwhite mineralization is pre- or syn/post major deformation. However, it is clear that the present distribution of mineralized zones is structurally controlled. Much of the greenstone belt, including the mine area, is covered by water and glacial overburden with only rare outcrop exposures. For this reason, interpretation of the geology from the surface is difficult and heavily reliant upon geophysical techniques. The regional surface geology around Musselwhite is presented in Figure 1.

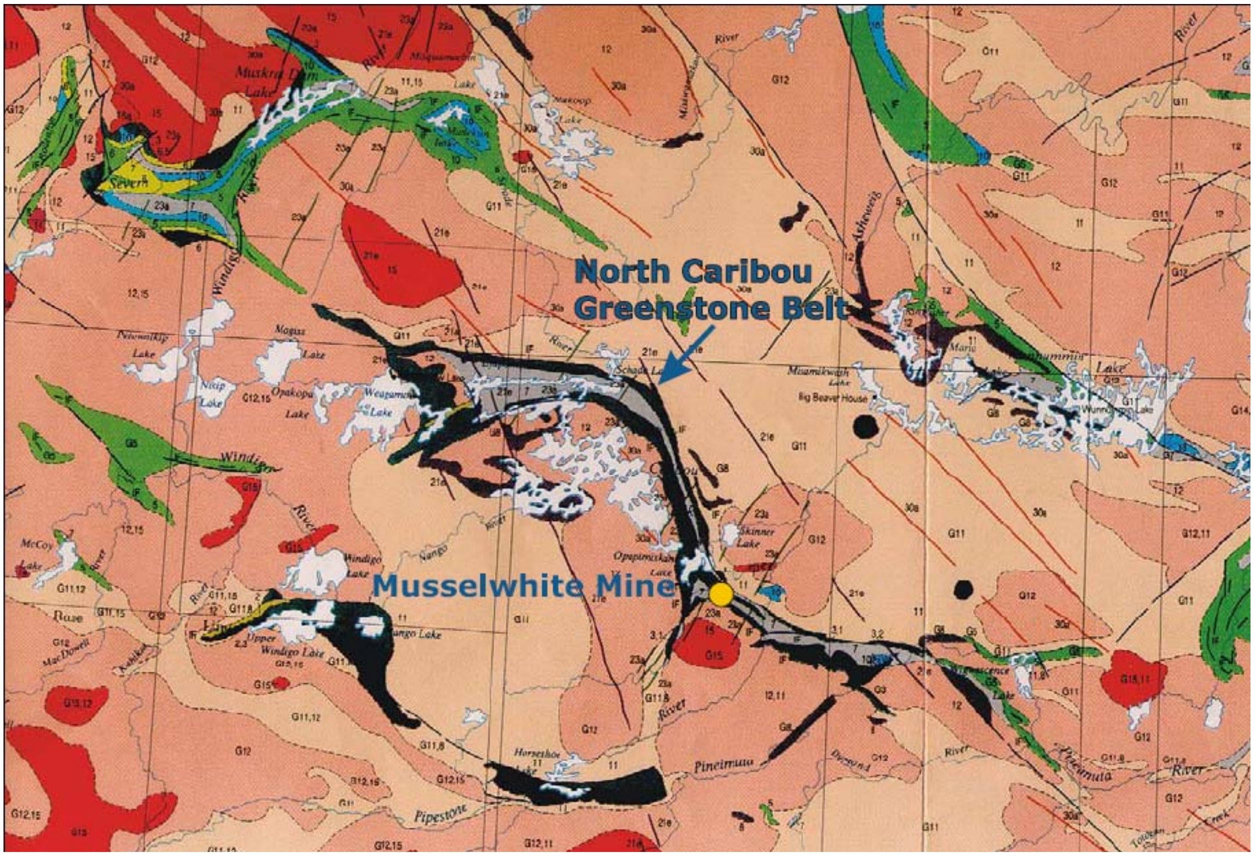


Figure 1 – Musselwhite Surface Geology (after Galley et al, 2007)

Strata-bound gold mineralization at Musselwhite occurs primarily within tightly folded silicate/oxide iron formation in a dominantly volcanic sequence. Three major deposits occur on the east Bay Synform and were identified as the T Antiform, PQ and OP deposits. The T Antiform is the largest and most significant deposit. The T Antiform is an antiformal wrinkle that occurs within the hinge area of the East Bay Synform. The T Antiform system plunges northwest at 12 to 15 degrees and the Deposit has been traced by drilling over a strike length in excess of 3km. The tightly folded structures extend over a vertical amplitude on 150m and a horizontal wavelength of 100m. The east limb of the East Bay Synform is referred to as the PQ Limb, and hosts the PQ Deposit and several other mineralized zones. The OP Deposit occurs on the southern extension of the east (S) limb of the T Antiform.

Grunerite-garnet-amphibole-chert iron formation is the most common host rock to mineralized zones that characteristically contain abundant pyrrhotite, quartz flooding and native gold. Mineralized zones in the T Antiform, in general, are subvertical tabular bodies concentrated within third-order antiformal fold hinges and adjacent limbs, and are subdivided into the S, C, T and WA zones. Individual zones extend over minimum strike lengths of about 1km. Mineralization is visually distinct and easily recognized in underground headings. Geological data analysis indicated that pyrrhotite content is the principal criteria for defining and modeling gold mineralization domains. Due to the massive and well foliated nature of the mineralization, the presence of native (gravity recoverable) gold and the strong association of the gold with the pyrites, initial indications for pre-concentration were deemed to be positive, which was borne out in the preliminary pre-concentration testwork. Mineral reserves as at December 2003 are shown in Table 1.

Table 1- Mineral Reserve by class at a cutoff grade of 3.00 g/t Au

Class	**Tonnes000)**	**Au Grade (g/t)**
Measured	3,278	6.24
Indicated	2,410	6.41
Subtotal	**5,688**	**6.31**
Inferred	3,517	7.35

*Note: excluding the Mineral Reserve.

3.1.2. Mining

Mining of the TA deposit at Musselwhite has now developed in several phases. The first phase involved developing a twin decline from surface at 12.5% grade to access approximately 5Mt of ore reserves above the 275m level, a conveyor ramp from the crushing station on 275 level to surface and a ventilation raise. The decline also serves for personnel and material access. During phase two, reserves were accessed via a production shaft, achieving a mining rate of 3500t/d. A new ventilation shaft was created and the existing ventilation shaft converted into a production shaft. A new sub-decline with a loading level, ore passes and a new underground crushing and conveying station was commissioned in 2003 which has now increased production to over 4000t/d.

The mining method predominantly in use at Musselwhite is sublevel blasthole stoping with backfill. This could be described as a modified Avoca method, where the stope mining face retreats away from rock fill being dumped and advancing from the other side of the stope. The ore horizons are accessed from the main decline (in the footwall to the east of the ore) via a 5 m wide x 5.5 m high cross-cut, and silled out to their full extents to the north and south. There are typically 4 sublevels per level, giving a drift spacing of approximately 25m. Mining is by longitudinal retreat along strike. Mucking is done by a fleet of diesel LHDs (8, 9, and 11 yd3) and 40 ton capacity haulage trucks. Development ore is hauled to the underground crusher, crushed and conveyed to surface, while development waste is typically dumped into open stopes. Completed stopes are backfilled with a combination of uncemented rockfill (URF) and/or cemented rockfill (CRF). URF is produced in waste development headings and from a seasonal crushing/screening plant on surface. Surface waste is delivered to underground through a backfill raise that extends from surface to the 200 level. A cement batch plant on surface produces a cement/fly ash slurry, which is fed to the underground workings through boreholes. The cement slurry is sprayed onto loads of URF before subsequent placement as CRF in the required stopes.

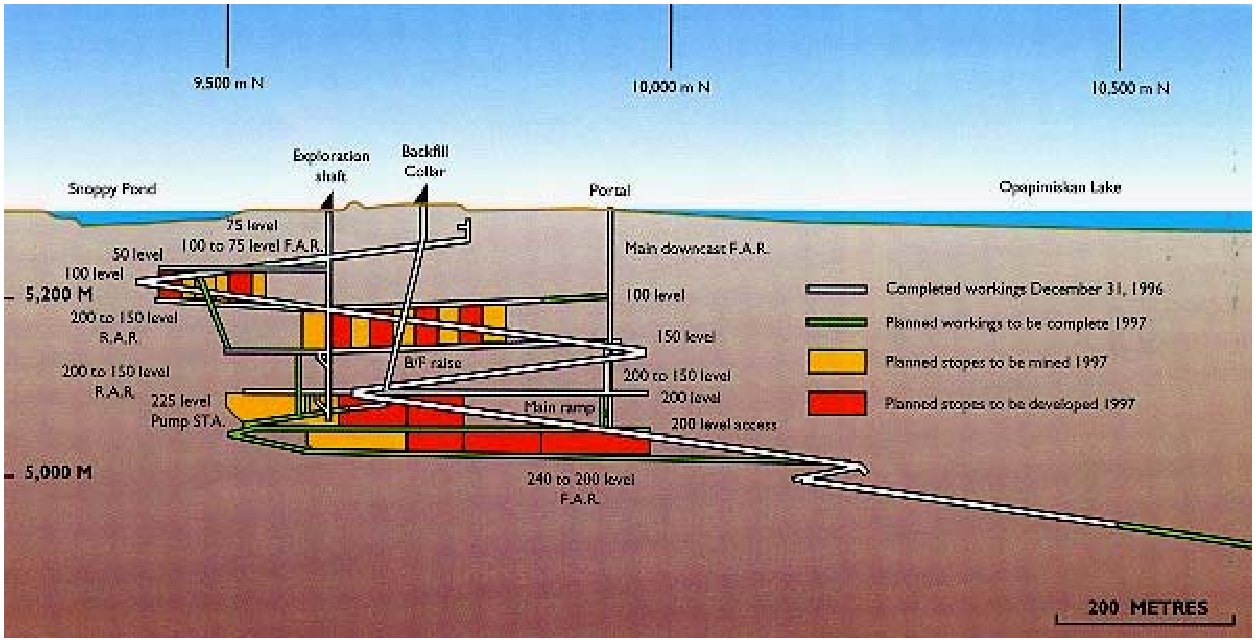


Figure 2 – Musselwhite Simplified Cross Section (www.placerdome.com/musselwhite.htm)

3.1.3. Milling

The present mill is designed to treat 3300t/d at a typical head grade of 5.4 g/t and employs conventional gravity separation, cyanide leach and carbon-in-pulp (CIP) processes to recover the gold from the ore. Primary crushing is carried out on surface using a jaw crusher, the output being fed directly to a secondary crushing plant. Grinding is via a rod mill - ball mill combination, with gravity concentration of the coarse free gold in the cyclone underflow via a Knelson concentrator. Shaking tables are used to upgrade the gravity concentrate prior to smelting; gold recovery in the gravity circuit is of the order of 23%. The presence of gravity recoverable gold circuit installed in the mill is a good indication of positive potential for pre-concentration of this ore. Mill discharge is thickened and leached for 32 hours in agitated leach tanks. Gold in solution is recovered by absorption onto activated carbon in the CIP circuit. Loaded carbon is eluted using the pressurized Zadra technique. Gold in the pregnant solution is recovered onto stainless steel cathodes by electrowinning. The gold sludge is pressure-washed from these cathodes, filtered, dried, and melted into bullion bars. Stripped carbon is reactivated and returned to the CIP circuit. Leach recovery is of the order of 72% for an overall plant recovery of 95%.

CIP tailings flows by gravity to a two-stage CCD washing circuit to recover cyanide. The washed tailing is pumped to a reaction vessel for cyanide destruction. Tailings grades are typically of the order of 0.27 g/t Au. Cyanide destruction is achieved using the Inco/$S0_2$ air process with two stages of washing, discharging an effluent containing less than 5ppm cyanide into the tailings pond. A representation of the Musselwhite Mill flowsheet is shown in Appendix 1.

Of great interest for the pre-concentration study is the fact that Musselwhite expects the leach/CIP recovery to be linked to the feed grade by the following equation:

Leach Recovery (%) = 95.56 - $\underline{(3.2 + 100\text{*Leach tails (g/t Au)})}$ {1}

Head grade (g/t Au)

While it is generally expected that the metallurgical plant recovery would improve with an improvement in feed grade based on the 2-product formula, this model for Musselwhite is confirmation of the assumption and has been built into the evaluation model. The impact of this will be explored in more detail in the analysis of results section.

3.1.4. Operating Costs

Operating cost information was supplied by Musselwhite for the study. Summary operating costs are as follows:

Table 2: Average Unit Operating Costs – 1998 to 2003 ($/t)

Cost	**1998**	**1999**	**2000**	**2001**	**2002**	**2003**
	Actual	Actual	Actual	Actual	Actual	Budget
Mine	18.74	20.78	23.54	24.02	31.15	23.06
Mill	9.76	9.54	9.45	10.58	10.74	9.97
Plant	0.36	3.59	3.80	4.41	4.23	2.10
Admin	11.89	11.07	11.20	13.67	19.04	13.95
Total	**40.75**	**44.98**	**47.99**	**52.68**	**65.16**	**49.08**

Cash costs in 2003 were $57/ton vs. $49.08/t planned. A detailed breakdown of the actual 2003 cash costs is shown in Table 3:

Table 3 - Activity Based Costing for Musselwhite Mine (2003 basis)

Activity	**Cost**
Drilling & blasting	$ 4.15
Mucking	$ 5.50
Support	$ 0.15
Backfill	$ 1.81
Haul costs	$ 9.05
Crushing/conveying	$ 1.64
Other u/g	$4
Development	$7
Mill	$7
Other surface	$3
Admin, Engineering, Geology	$13
Total	**$57**

These costs have been used as input costs into the evaluation model in order to estimate the magnitude of savings to be expected from the implementation of pre-concentration.

3.2. Testwork Results

Bench scale testing was conducted on drill core samples from the Musselwhite PQU and PQD deposits. The objective of the testing was to assess the potential for pre-concentration of these ores. Testwork comprised visual inspection of the core for mineralogical data and RQD evaluation, size-assay analysis, ore and waste characterization and sink/float analysis by heavy liquid separation. The main results of the testwork are summarized in the following table.

Table 4 – Musselwhite core evaluation results

Parameter	**PQU**	**PQD**
Au Grade (Float Sink)	7.36 g/t	3.32 g/t
RQD	85	85
80% passing size	12 mm	12 mm
S.G. - metal bearing rock	3.5	3.5
S.G. – barren rock	2.8	2.8
Concentration Criteria	1.4	1.4
Float-Sink @ SG 2.9		
Sinks Au Grade	8.32 g/t	4.14 g/t
Sinks Au Distribution	93.2 %	94.6 %
Floats Au Grade	2.84 g/t	0.75 g/t
Floats Mass Distribution	17.6%	24.0%

RQD results are good indicating competent ore hosted in competent rock, an observation which is borne out by the longhole mining method selected by Musselwhite Mine. The RQD results also indicate good potential for large, stable underground excavations, again borne out by the recent installation of an underground crushing station at Musselwhite. Also of interest was the size-assay analysis results for the ore (Figure 3). These results indicate that there is a possibility of rejecting a coarse, barren fraction (+19mm core size). There is a slight increase in grade towards the finer fractions, but not sufficient to merit concentration by size classification alone. However, for the PQU ore, there was a distinct concentration of gold in the -1.7mm fraction indicating some preferential distribution and liberation of the pyrite in this fraction, and potential for separation based solely on size.

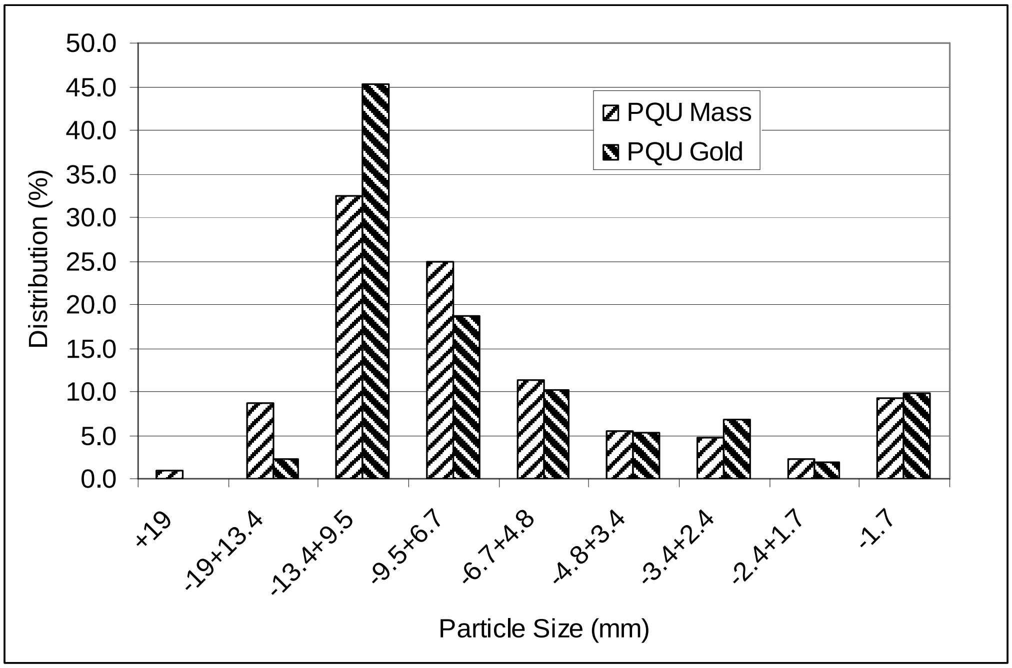


Figure 3 – PQU Size Assay Results

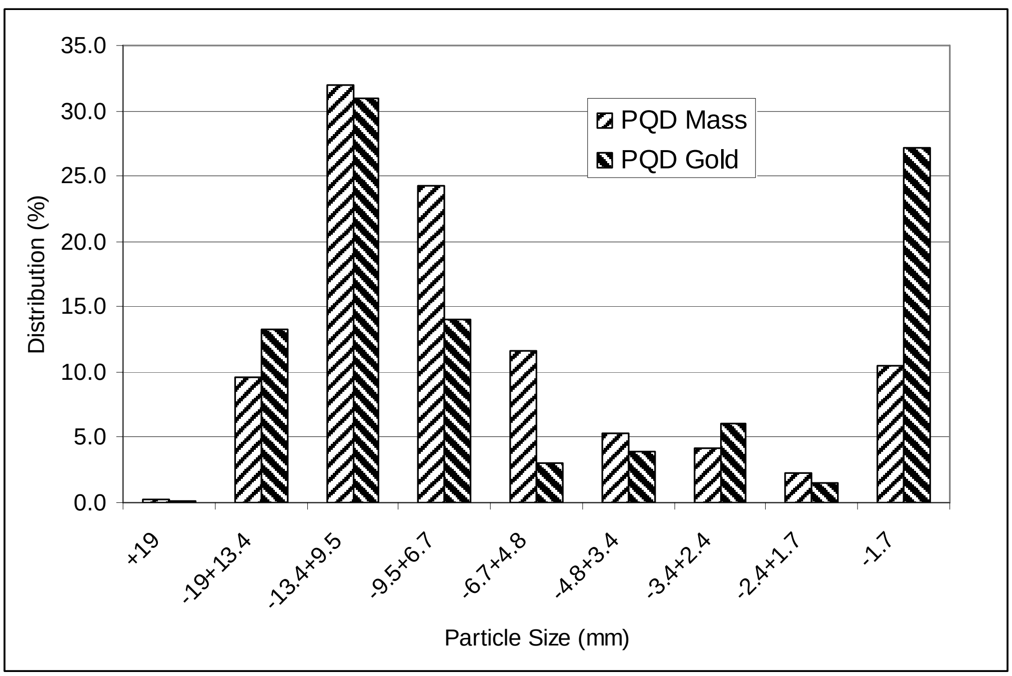


Figure 4 – PQD Size Assay Results

As shown in the table, there is a clear S.G. distinction between the metal bearing rock and the barren waste. This SG differential allows the application of gravity-based separation processes such as dense media separation. Desne media separation was performed on the crushed core at an SG f 2.9. Results are presented in the table below.

Table 5– Musselwhite Sink/float testing results on -13mm core at SG 2.9

Sample	Total Wt%	Assay g/t	Sinks Wt%	Assay g/t	Floats Wt%	Assay g/t	Recovery %
PQD 9	21.7%	3.54	72.6%	4.76	27.4%	0.31	97.6%
PQD 6	28.6%	3.22	71.6%	4.18	28.4%	0.81	92.9%
PQD 3	18.0%	5.74	64.1%	7.40	35.9%	2.78	82.6%
PQD -3	31.8%	2.29	100.0%	2.29	0.0%	0.00	100.0%
PQD Total	**100.0%**	**3.45**	**79.5%**	**4.01**	**20.5%**	**1.29**	**92.3%**
PQU 13	12.8%	3.55	80.4%	4.36	19.6%	0.22	98.8%
PQU 9	17.3%	3.13	79.8%	3.54	20.2%	1.53	90.1%
PQU 6	14.5%	11.41	79.6%	12.90	20.4%	5.58	90.0%
PQU 4	16.5%	7.70	74.5%	9.18	25.5%	3.38	88.8%
PQU 3	9.6%	5.11	72.6%	5.82	27.4%	3.24	82.6%
PQU 2	7.6%	15.25	68.1%	21.50	31.9%	1.87	96.1%
PQU -2	21.8%	10.80	100.0%	10.80	0.0%	0.00	100.0%
PQU TOTAL	**100.0%**	**7.92**	**81.8%**	**9.07**	**18.2%**	**2.73**	**93.7%**

Preliminary testwork results indicate that 20.5% of the PQD ore could be rejected at a nominal particle size of -19mm using dense media separation with a recovery of 92.3%. Similarly, 18.2% of the PQU core was rejected at a recovery of 93.7%. The results indicate good potential for pre-concentration using this method, although processs optimization is now required in order to maximize the degree of waste rejection and improve metal recoveries.

A preliminary process flowsheet is suggested comprising crushing, coarse screening to remove the barren fraction, fines screening to remove the -3.4mm high grade fraction and dense media separation of the +3.4mm -19mm fraction. The fines would then be re-combined with the concentrated sinks fraction resulted in a projected gold recovery of 93.2% while rejecting 17.6% of the mass for sample PQU. For the PQD the recovery was 94.6% and the waste rejection was 24%. A conceptual process flowsheet is shown in Figure 5. These preliminary results are sufficiently encouraging to warrant further metallurgical work to assess the maximum degree of waste rejection possible from the ore.

3.3. System Design

Testwork at UBC as described above, as well as previous testwork by Placer Dome on Musselwhite ore indicates good potential for pre-concentration by means of dense media separation due to the coarse nature of the mineralization, the presence of gravity recoverable gold, a strong association of the gold with dense sulphides and good liberation arising from preferential liberation of the sulphides along the foliated boundaries in the rock. Thus a design comprising appropriate feed preparation, followed by dense media separation, hydraulic hoisting of the concentrate and automated preparation and disposal of the waste rejects as backfill is presented.

Basic design criteria for the underground processing system are suggested from previous research:

- Plant should be as compact as possible
- Processes should be robust in the face of variations in feed grade and tonnage
- Open-circuit processes are preferred to closed
- Recovery should be maximized in favour of grade
- Waste products should be suitable for backfill

Previous research further suggests that a maximum waste rejection of 60% only is possible due to space constraints arising from the bulking factor of broken ore compared to in-situ ore after fragmentation. While reject ratios of 17% and 24% respectively were achieved in the testwork, the approach was very preliminary, and further comminution and liberation evaluation should lead to the maximization of the degree of waste rejection possible for this ore. The reliability and operability of the plant is critical, and it is essential that the unavailability of the plant should not impact on the mining process, thus sufficient surge capacity ahead and after the process facility as well as a full bypass alternative should be provided in the design. A conceptual flowsheet of the underground dense media plant, embodying the design criteria above is presented in Figure 5:

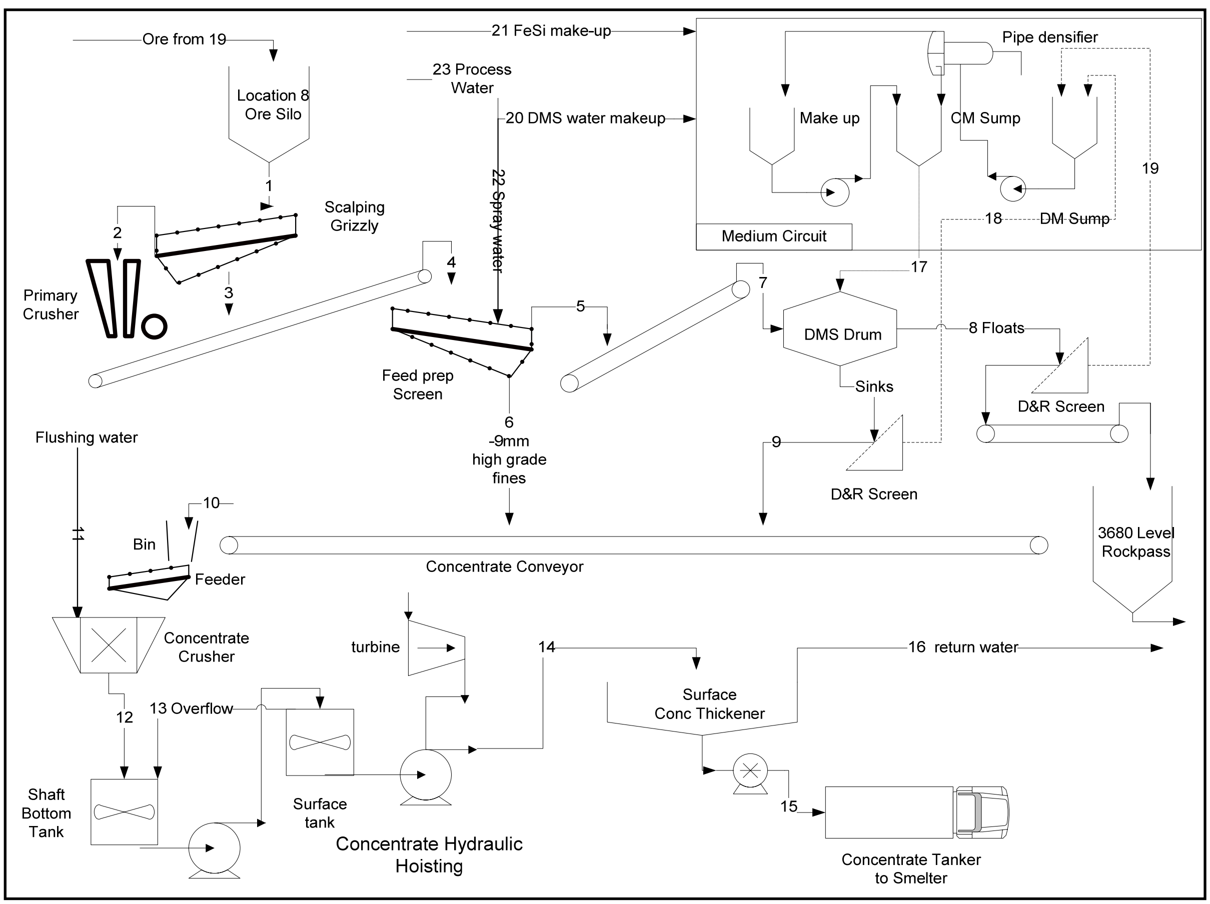


Figure 5 – Underground Pre-concentration through Dense Media Separation

Appropriate crushing and feed preparation design, sufficient surge capacity in the form of a 1000t feed bin ahead of the plant, 1000t waste and product storage provisions, as well as potential for the full bypass of the DMS plant are considered essential features of the design. The plant would be designed for a nominal 4000 tpd, translating to approximately 200 tph which is moderate for a DMS facility and the plant is thus compact and from previous designs, maximum excavation sizes, including allowance for operating and maintenance access, are of the order of 12m x 15m x 15m, linked by standard 3m x 3m drifts for conveyors and personnel access. Based on the high RQD values measured on the core during testwork, indicating high wall rock strength, the development of large, stable excavations will not be a problem at Musselwhite.

Location of the plant would be in a series of new and potential existing excavations on the main haul ramp above the T-antiform orezone. Locations around the existing underground crushing station should be investigated in order to capitalize on this existing infrastructure.

3.4. System Evaluation

Results for the metallurgical testwork have been used as input into the evaluation model that was previously developed for the project. The evaluation methodology includes considering impacts on the tonnage reporting to each process, and thus unit costs, capital cost estimates for the facility as well as impacts on revenue of the additional processing step and impact on surface plant metallurgical performance due to the increase in feed grade to the plant.

The evaluation model with the key study data inputs is attached. Based on an ore reserve of 25 Mt, and a mining rate of 4274 tpd, and with a waste rejection ratio of 20% at a recovery of 94.6%, results are as follows. Operating costs are reduced through the savings described above from $61.15 to $55.03/t. Overall gold recovery goes from 95.75% to 91.2% for a revenue penalty of $3.84/t at a gold price of $450/oz. Overall impact on the operation is a reduction in cutoff grade from $52,60 to $47.34 contained value, and a consequential increase in mineable reserve and life of mine of 3.12 Mt and 2.15 years respectively. Based on an estimated capital cost of $31.2m for the installation, NPV is increased on the present mineral reserve by $36 000 000.

The evaluation indicates that the NPV of the Musselwhite operation can be increased by $36m through timeous implementation of the concept via a lowering of the operating costs, decrease in cutoff grade and thus an extension of the life of mine through a related increase in the mining reserves. The NPV generated could be potentially increased by increasing the mining rate to accommodate the additional reserves, however no capital has been estimated for this.

4. South Deep

4.1. Background – Mining, Geology, Mineralogy and Processing

The South Deep Twin Shaft Complex has under development between 1995 and November 2004, consisting of a Ventilation Shaft to 2 760 metre depth and a Main Shaft to 2 993 metre depth. The South Deep Twin Shafts provides direct access for men and materials to levels previously serviced through subvertical shafts from the old South shaft, thereby obviating costly double handling. arrangements. In November 2004, as a result of the commissioning of the new main shaft, mining operations were transferred from the older Western Areas Gold Mine South Shaft and the sub vertical Shaft Complex to the new South Deep Main Shaft Complex. Two principal reefs are mined at South Deep, the Ventersdorp Contact Reef (VCR) and Elsburg reefs.

4.2. Geology and Mineralogy

VCR typically consists of a 1-2m thick conglomeratic reef structurally hosted in palaeochannels and erosional ridges on the underlying basalts (Henning et al., 1994). Gold is hosted by steep- and shallow dipping fractures associated with thrusting, hydrothermal fluid flow and alteration. Gold distribution is controlled by the interaction of these thrusts with the lithological architecture, as well as the fracture density. Gold grades are typically vertically distributed in the VCR, with a concentration in grade towards the footwall contact, which is characterized by the presence of carbonates, often referred to as the carbon-leader. This characteristic suggests that pre-concentration of the VCR would successfully reject a substantial portion of non-gold bearing rock above this contact but falling within the stope cut (Figure x).

Figure 5 – VCR / Wits Conglomerate Mesotexture

Elsburg reefs typically consist of massive, steeply dipping fan-shaped sequences of reefs hosted in ancient paleochannels, varying in thickness between 2 – 30m; again the gold is strongly associated with pyrite and less strongly with uraninite in this ore. Each reef package is separated by barren rock, and the alternating distribution of gold in the fan between the basals and the interleaving country rock suggests that pre-concentration could also reject a substantial quantity of non gold-bearing rock.

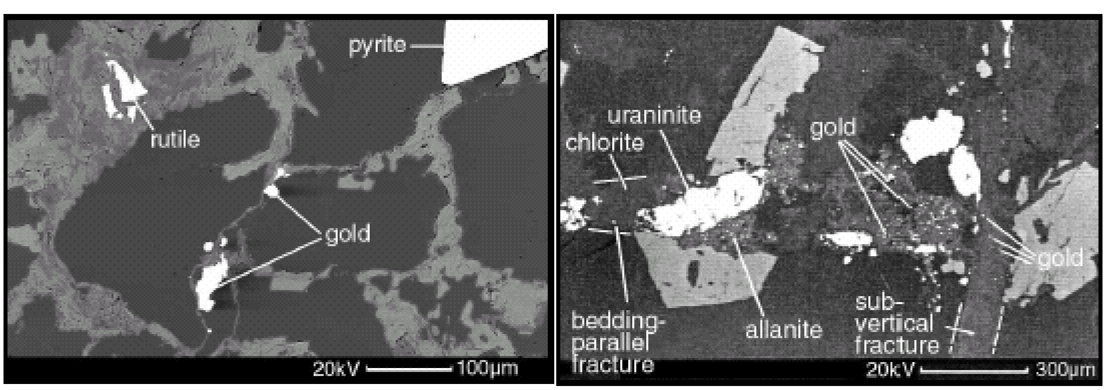


Figure 6 - Gold occurrence in the VCR and Elsburg Reefs (after Barnicoat & Jolley, 2000)

Quartz is the predominant mineral in the assemblage, with chlorite and pyrite as secondary minerals within the VCR and Elsburg reefs. Gold is primarily associated with pyrite, which occurs as nodular buckshot or as disseminated pyrite. Up to 40% of the gold is recoverable as free gold in the metallurgical plant. The strong association of gold with pyrite and uraninite in Witwatersrand ores, as well as the high degree of free gold in the ore suggests that pre-concentration by radiometric sorting or gravity methods may be a possibility. Radiometric sorters have been tested and put into active use on surface at several Witwatersrand gold mines, including East Driefontein, Hartebeestforntein Gold Mine in Stilfontein, Saaiplaas and Western Deep Levels between 1982 to 1986 prior to the collapse of the gold price and suspension of activities at many of many of these mines. South Deep themselves have tested radiometric sorting more recently, the results of which are reviewed here.

4.3. Mining

VCR comprises 50% of the planned production at the mine and is mined by conventional underground long wall mining methods. Elsburg reefs are and mechanized drift and fill trackless methods (50%). Typically the conventional stopes are drilled by jackleg,, blasted and scraped by scraper winch or washed into a series of gullies that feed an ore pass system. Ore from mechanized trackless sections is drilled, blasted and loaded with scoop trams to internal ore passes delivering ore to the main operating levels of the mine on 85, 90 and 95 level. ROM size distribution at South Deep as supplied by the mine is indicated in the Figure x. There is no size assay data available as yet for the South Deep ROM ore, but evaluation of this will enable further conclusions as to the pre-concentration potential of the ore.

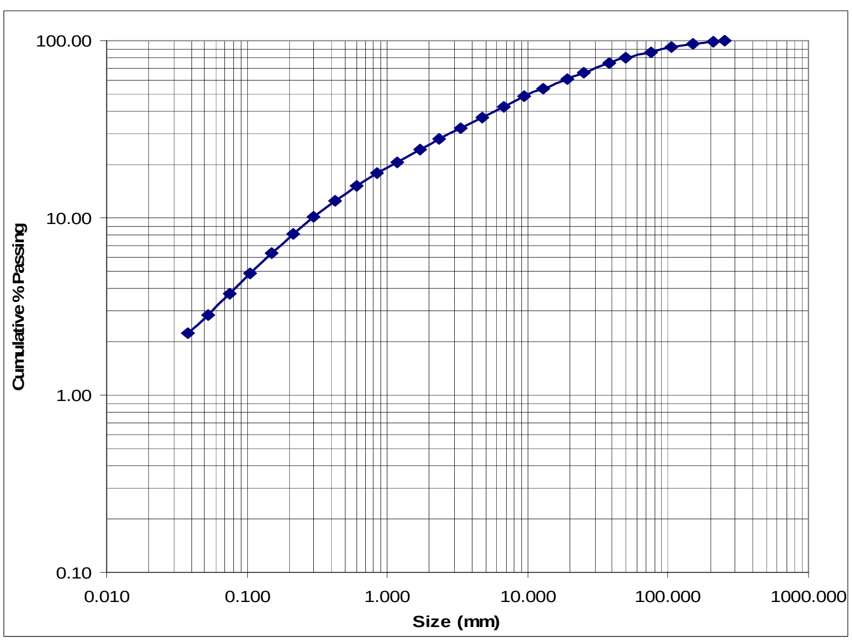


Figure 7 – South Deep ROM Size Distribution

Massive mechanized mining is made possible by previous de-stressing of the stopes via the mining of a 1.5m de-stress cut in ore by conventional methods prior to bulk mining. De-stress stopes contribute a small amount to ore production at South Deep. Tramming on all the main levels is done by means of electric rail locomotives with 14 ton hoppers. All ore from VCR and Elsberg sections is delivered to 90 level or 95 level for horizontal tramming and to 95A level for conveyance skip loading. Since the commission of the Main Shaft, people, material as well as waste rock are hoisted from 2565m through the old South shaft via No 3 subshaft. The localization of all ore handling around 90 level makes this a potential location for concentration of the ore underground, and rejection of the waste to backfill prior to hoisting. Of particular interest is the discrepancy between the grade of ore delivered from the various mining sections. Conventional mining of the VCR delivers the highest grade of ore, while mechanized sections on the Elsburg reefs deliver a lower grade, due to the high level of dilution incurred by this method. The high dilution and thus low grade, however, is offset by the low unit cost of the mechanized sections. Grades delivered by the various mining methods, and the unit cost of the methods (2003 data) are as follows:

Table 6 – South Deep Mining Methods: Grade and Cost

Method	**Grade (Value /t @ 450/oz))**	**Unit cost**
Conventional	10.72 ($151)	16.30
De-stress	4.10 ($57.65)	16.30
Trackless	5.94 ($83.50)	9.00

The deepest activities are presently at a depth of 2,692 metres below surface. Future mining sections will be extended down dip and ultimately in a Southerly direction along the sub-outcrop, further extending hauling and hoisting distance underground. The average wet bulb temperature in the stopes is 30.0 degrees Celsius, with ventilation velocity at the stope face of 0.74 m/s, vs. a planned air velocity of 1.0 m/s. Currently the mine is refrigerated by 35MW of cooling on surface located at South Shaft, 11 MW of cooling on surface at the South Deep Shaft Complex and 10MW of cooling underground on 80 level. Cooling is typically by mobile closed loop coil cars as well as fixed open-spray bulk air coolers. Ventilation and refrigeration is thus under pressure at the mine, particularly in the conventional sections due to the high virgin rock temperature of 49.5˚C and extensive surface area of the tabular longwall stopes.

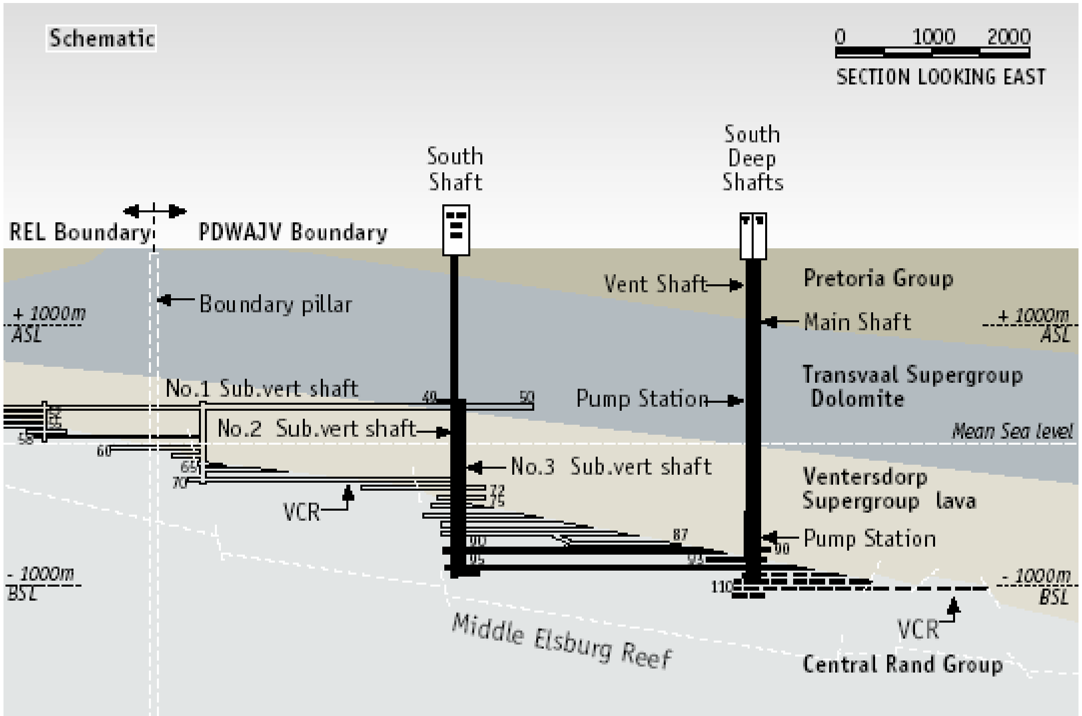


Figure 8 – Cross Section Through South Deep showing Arrangement of Shafts and Reefs (www.placerdome.com/south deep.htm)

Mineral Resources and Reserves

Proven and Probable Mineral Reserve categories at South Deep presently amounts to 55.6 million ounces of gold. In addition, the mine's attributable resources are estimated at some 74.8 million ounces (Table 4). At current levels of mining, the reserve alone is sufficient to support a mine life of over 60 years. Due to the practically unbounded nature of the reserves at this stage, a practical life-of-mine of 20 years has been selected for the evaluation in order to demonstrate the interaction between cost savings, cutoff grade and the increase in mineral reserves.

Table 7: South Deep Resources and Reserves (as at December 2003)

Class	**Tons (000's)**	**Grade g/t**	**Contained oz (000's)**
Reserve			
Proven	5345	8.3	1424
Probable	100 165	8.4	27 018
Total	105 510	8.4	28 442
Resource			
Measured	895	10.9	314.6
Indicated	66819	9.0	19 279
Total	67,714	9.0	19 593
Resource + Reserve			

*Cutoff grade varies between 4-6 g/t for these figures

As can be seen the grade of the reserve and the resource is fairly good at approximately 9 g/t, while two of the mining methods de-stress and trackless deliver grades of between 4 & 6g/t, conventional sections are achieving a head grade above the reserve grade. Thus efforts must be made to realise this grade potential in the head grade delivered to surface through novel means such as increased selectivity, innovative blasting methods such as SBM, resue mining or alternately underground pre-concentration by sorting or other means. For the purposes of the study however, a reserve of 105 Mt is unbounded, and a finite reserve of 25 Mt has been selected in order to demonstrate the principles of the concept.

4.4. Surface Processing

The new gold metallurgical plant was commissioned in June 2002. It includes a two stage milling circuit (SAG Mill - Ball Mill) with a gravity gold recovery circuit inclusive of leach reactors and a pump cell CIP plant with elution, electrowinning and direct-induction smelting. The plant has a planned capacity of 7,200 tons per day or 220 000 tons per month. Plant recovery is of the order of 91%, with an average feed grade of 7.4 g/t. While present plant performance is poor due to highly variable feed grade and tonnage, this is expected to improve after steady state feed conditions are achieved. Pre-concentration underground would be expected to both raise and regularize the grade of ore delivered to the surface leach plant, leading to further improvements in plant performance.

Potable water consumption averages 340,000 kilolitres per month. It is purchased from Rand Water, the sole potable water supplier in the area. Mine service water from underground is used to supplement the water requirements of the mine.

4.5. Operating Costs

Detail on the operating cost breakdown at South Deep is not presently available at the time of writing. Operating costs for South Deep have been estimated from cash cost and grade data in the 2003 Annual report, as well as typical cost data from the 2003 Canadian Mining Journal Sourcebook. Operating costs at South Deep are estimated as follows. All figures are in US$:

- Cash cost of production: $287 / oz
- ROM head grade: 0.27 oz/tonne
- Cash cost/ROM tonne: $78.51/t

An activity-based breakdown of this cost has been developed using a typical breakdown of mining and processing costs for Canadian underground mining operations responding to the 2003 Mining Sourcebook survey. The unit cost breakdown (in %) has been applied to the South Deep cash costs to obtain activity based unit costs for mining, hauling, hoisting, surface processing and tailings disposal. These costs may not be representative of the actual cost breakdown for a deep-level South African gold mine, but they can be used as a starting point for evaluation.

Table 8 – Activity Based Cost Breakdown for South Deep

Activity	**Breakdown-Typical U/GCanadian**	**% of Total Operating Cost**	**South Deep Breakdown**
Mining	10.48	13.4	10.53
Mucking/hauling	3.23	4.1	3.25
Crushing	1.48	1.9	1.49
Hoisting	3.85	4.9	3.87
Backfill	4.42	5.7	4.44
Materials	5.75	7.4	5.78
Power	5.23	6.7	5.26
Supervision & labour	21.13	27.0	21.24
Services	7.10	9.1	7.14
Surface haul	4.45	5.7	4.47
Milling	9.00	11.5	9.05
Tailings	2.00	2.6	2.01
Total	**78.12**	**100**	**78.52**

4.6. Metallurgical results

To date, no samples have been received from South Deep for metallurgical testing. A suite of mineralogical and metallurgical testwork is planned for the samples once received, including mineralogy, size-assay, coarse particle flotation and gravity separation testwork. However, metallurgical results from radiometric sorting tests undertaken by South Deep have been made available. A summary of the pertinent results, which have been used in the economic analysis and evaluation for indication of the potential of this concept at South Deep, have been presented below. A separate testwork report, analysis and evaluation will be submitted once testwork is complete. Several interesting results from the testwork at South Deep, Rossing Uranium and Optosort Sydney are presented below.

Radiometric Sorting at South Deep

A sample of ROM material at 8g/t was screened of undersize at +40mm and subjected to radiometric sorting using an Ore Sorter's Ultrasort 10 radiometric sorter (10 radiometric sensing heads) at a belt speed of 2m/s and using a range of sensitivities (measured in counts/cm^2). The range of weight rejection achieved and the correlating sorter setpoint is shown in Table 9.

Table 9 – Sorter setpoints and results

Sorter set point (cts)	mass passed	gold passed	heads g/t	tails g/t
15.00	43.3%	82.4%	8.13	1.33
11.00	49.9%	85.5%	7.33	1.24
8.50	55.9%	87.9%	6.72	1.17
3.75	70.5%	92.6%	5.62	1.07
0.00	100.0%	100.0%	4.28	0

The +40mm fraction constituted 25% of the total Rom sample taken. The grade/recovery curve obtained from the testwork is shown in the figure below:

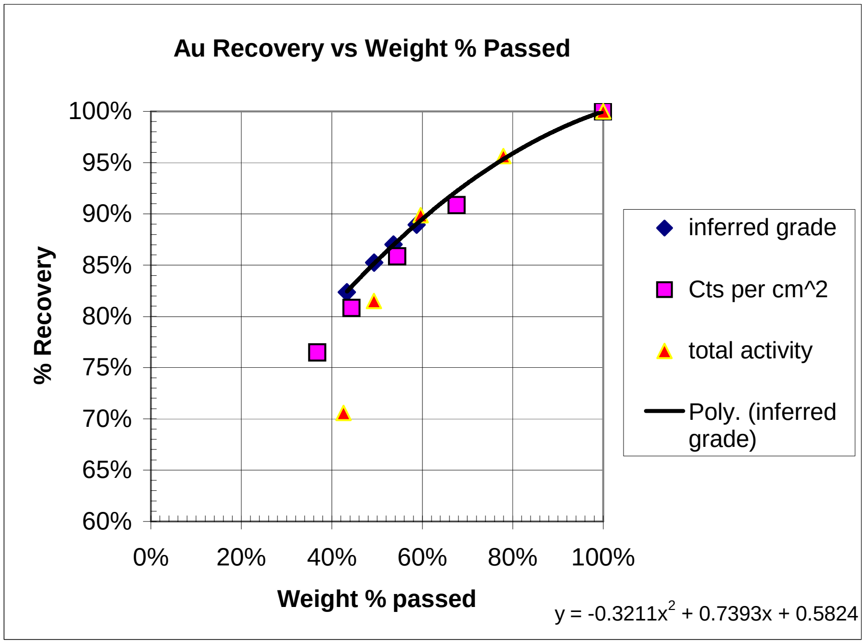


Figure 9 – Radiometric sorter grade-recovery curve

An optimum result of 50% rejection by mass of the +40mm fraction at a metallurgical recovery of 87% was achieved at a setpoint of 8 counts/cm2. Tailings grade (i.e. sensor cutoff) was 1.09 g/t Au. This translates into an overall metallurgical performance of 15% waste rejection at a recovery of 98.3% which indicates great potential for the concept. It is felt that results could be improved by crushing the ROM ore slightly to improve liberation and therefore exposure of the uraninite to the radiometric sensor, extending the size range of ore sorted down to +19mm by the introduction of a fine sorter for the +19 -40mm, and sorting of the +40mm fraction using the existing setup.

Optical sorting tests were carried out at Optosort in Sydney on a similar sample. Sorting results indicate that 5% of the ore can be rejected at a tailings grade of 0.6 g/t, translating into an overall metallurgical recovery of 99.6%. While the degree of waste rejection is not high enough to merit installation of such a sorter, the high recovery is indicative of an opportunity to increase the degree of waste rejection.

4.7. System Design

Previous testwork at UBC, radiometric and optical sorting testwork conducted by South Deep, together with the combined flotation and gravity concentration flowsheet piloted historically at COMRO by Lloyd et al suggest a number of process alternatives for underground pre-concentration at South Deep. In the light of these criteria, two process flowsheets are suggested.

Sorting Flowsheet

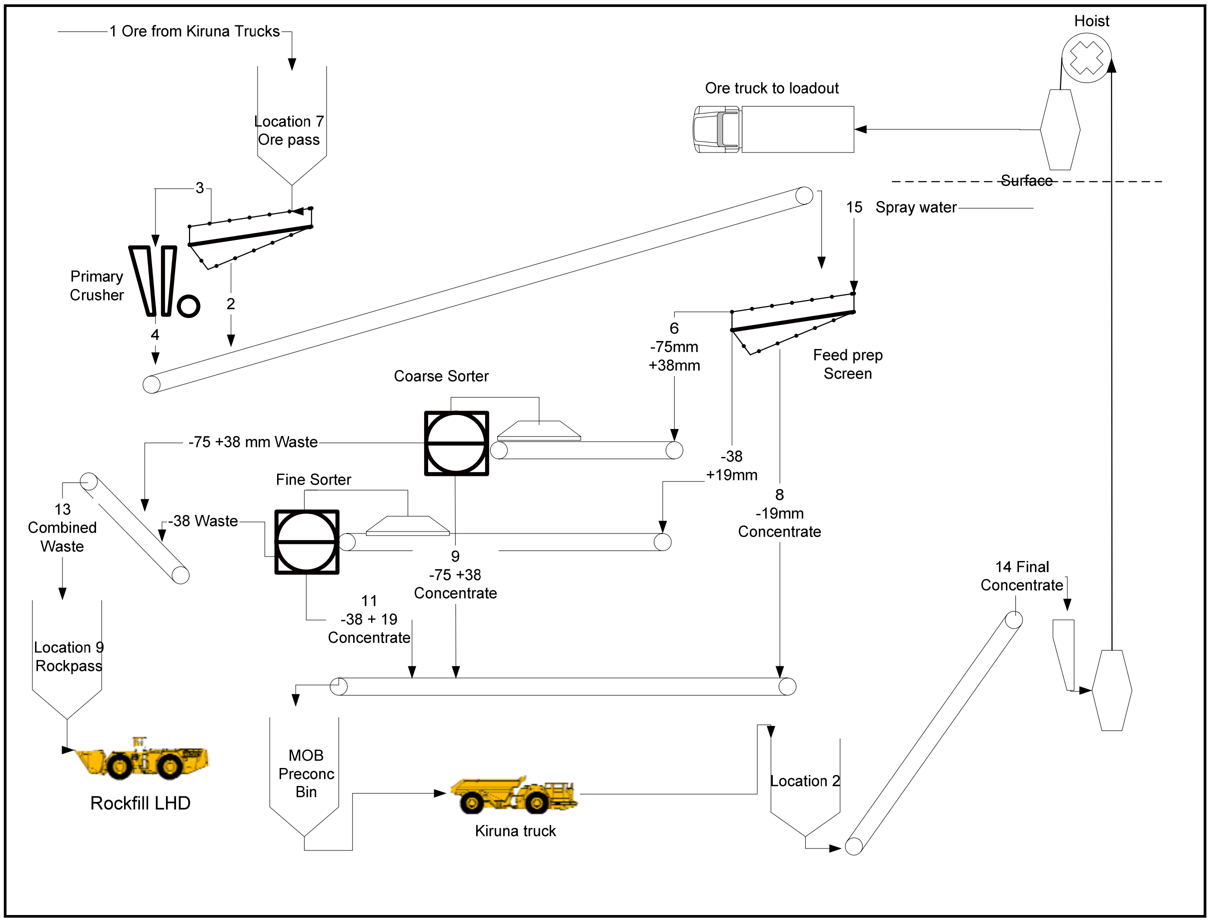


Figure 10 – Sorting-based flowsheet

ROM ore is dumped into the 1000t feed ore bin by LHD or haul truck. -80mm undersize is scalped out and -1000mm + 80mm oversize is crushed to -80mm in the jaw crusher. Combined streams are prepared by wet screening to produce 3 products: -80mm +38 is sorted in the coarse radiometric sorter, -38+19 is sorted in the fine sorter and -19mm undersize is delivered directly to concentrate. This arrangement maximizes the capacity and efficiency of the sorting process, and capitalizes on the naturally occurring tendency gold ores to exhibit a concentration of metal values in the finer fractions. +19mm waste rejects are delivered to a waste bin for preparation as aggregate for backfilling of the mechanized stopes; -80mm + 0mm concentrate is hoisted to surface using the existing skip system suitably modified to handle the finer size distribution of the ore.

An alternative for this ore, previously identified in the research is coarse particle flotation followed by scavenging of the flotation tailings by gravity methods. While testwork for this option still has to be completed on the forthcoming South Deep sample, a conceptual flowsheet for this process has been developed and is presented below:

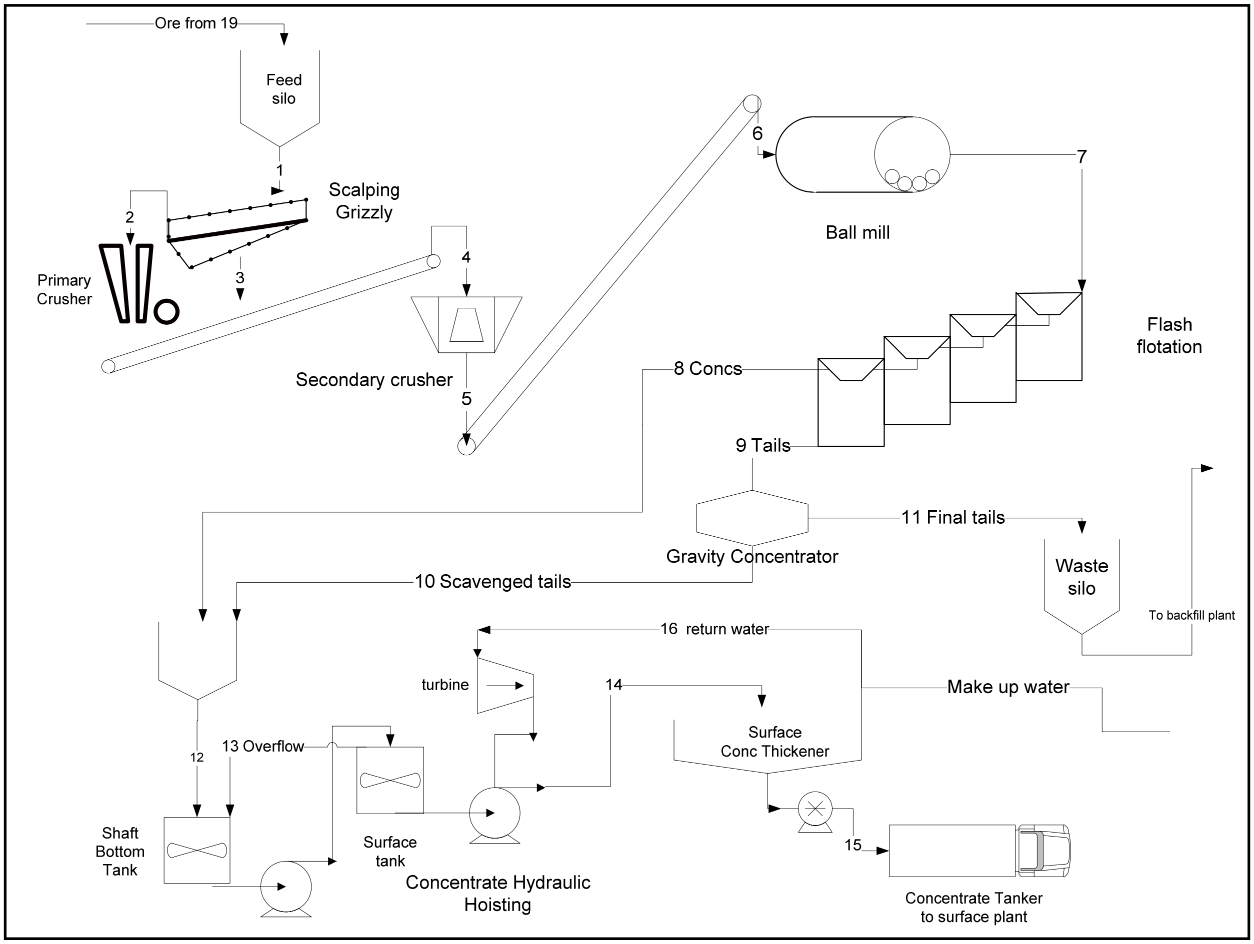


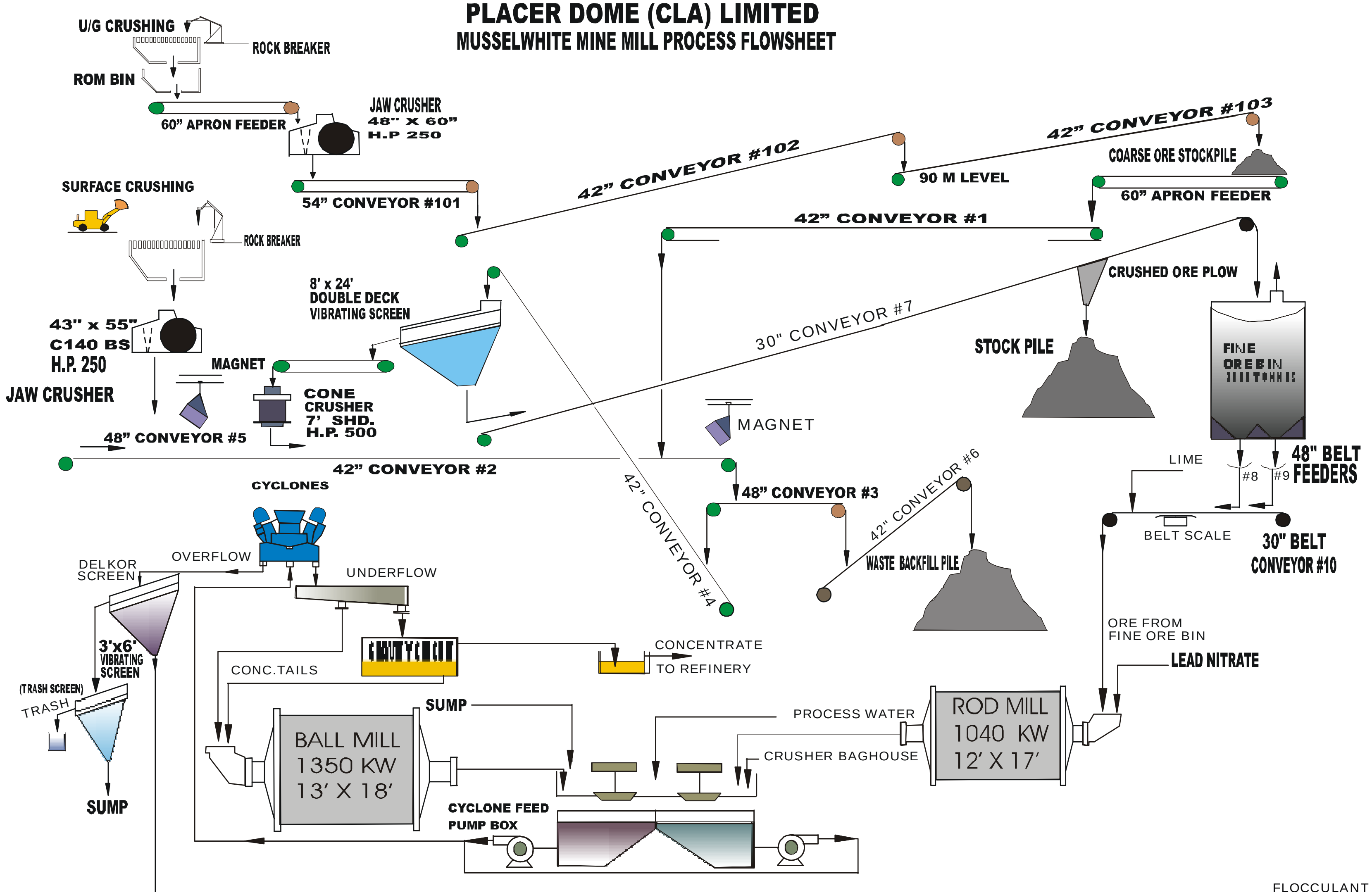
Figure 11 – Underground pre-concentration by coarse particle flotation

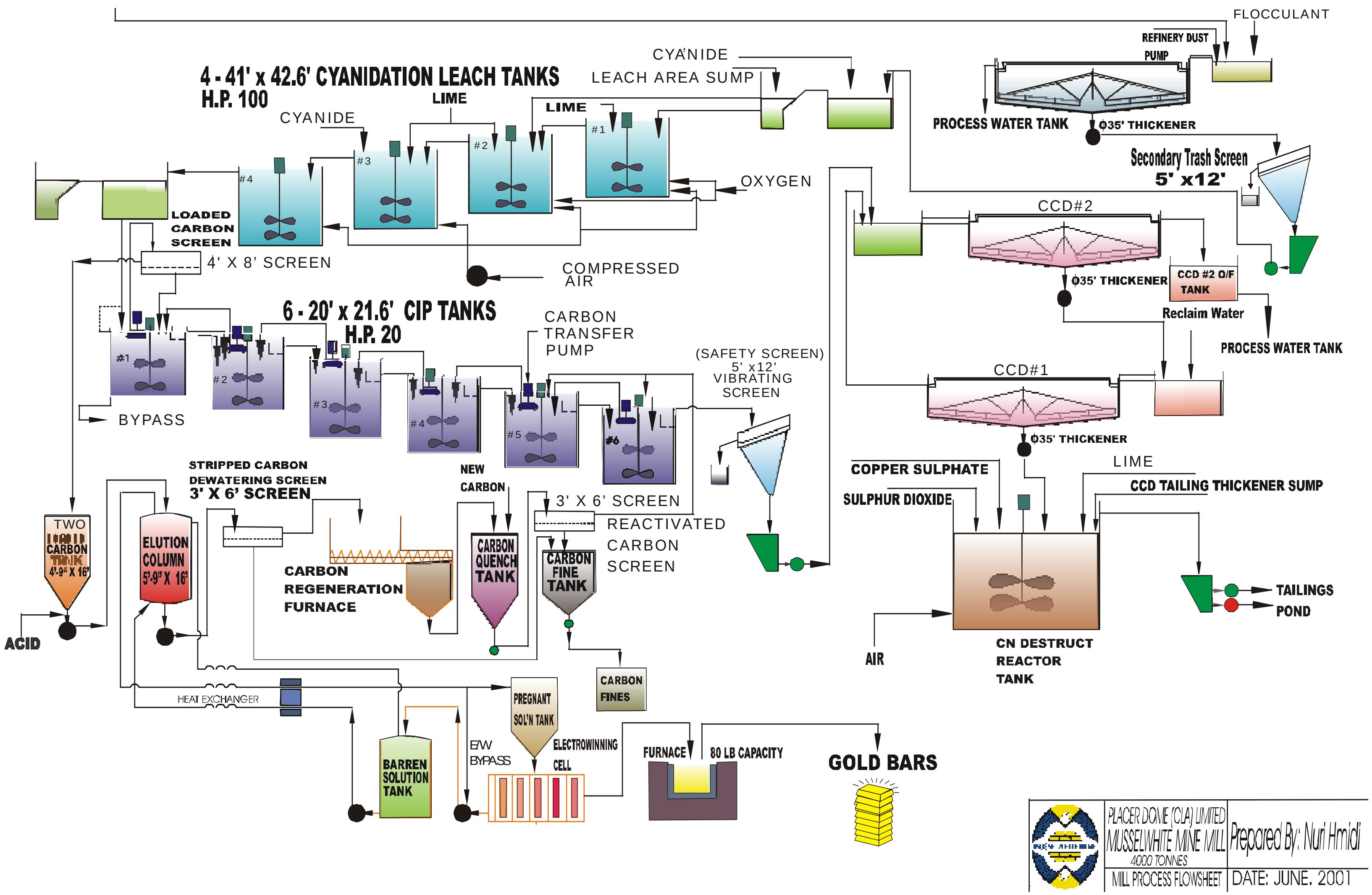
4.8. Evaluation

Attached is the evaluation model with the cost data as presented in the report and data input from the PDSA radiometric sorting. Based on the selected ore reserve of 25 Mt, and a mining rate of 7200 tpd, and with a waste rejection ratio of 15% at a recovery of 98.3%, results are as follows. Operating costs are reduced through the savings described above from $78.91 to $72.80/t, with a resultant decrease in calculated cutoff grade from $64.88 to $59.86/t contained value and an increase in ore reserve of 3.28 Mt. Overall gold recovery goes from 91% to 90.47% for a revenue penalty of $1.63/t at a gold price of $450/oz. With an estimated project capital cost of $44m, overall impact on the operation is a reduction in working costs and an increase in mineable reserve and thus life of mine of 1 year, translating into an increased NPV on the present mineral reserve of $83 145 000.

A second model run was done for comparison using the results of the Sydney optical sorting tests (attached). Overall results were 5% waste rejection at 99.6% recovery indicating possible operating cost savings of $3.56/t and negligible revenue losses at such a high recovery. For a similar cost of installation, mine life is extended by half a year and the NPV of the reserve is improved by $39.6m at a long term metal price of $450/oz.

PLACER DOME (CLA) LIMITED
MUSSELWHITE MINE MILL PROCESS FLOWSHEET





4 - 41' x 42.6' CYANIDATION LEACH TANKS
H.P. 100
FLOCCULANT
REFINERY DUST
PUMP
CYANIDE
LEACH AREA SUMP
LIME
CYANIDE
PROCESS WATER TANK
Ø35' THICKENER
Secondary Trash Screen
5' x12'
OXYGEN
LOADED
CARBON
SCREEN
4' X 8' SCREEN
COMPRESSED
AIR
CCD#2
Ø35' THICKENER
CCD #2 O/F
TANK
Reclaim Water
PROCESS WATER TANK
6 - 20' x 21.6' CIP TANKS
H.P. 20
CARBON
TRANSFER
PUMP
(SAFETY SCREEN)
5' x12'
VIBRATING
SCREEN
CCD#1
Ø35' THICKENER
BYPASS
STRIPPED CARBON
DEWATERING SCREEN
3' X 6' SCREEN
NEW
CARBON
3' X 6' SCREEN
REACTIVATED
CARBON
SCREEN
COPPER SULPHATE
LIME
CCD TAILING THICKENER SUMP
SULPHUR DIOXIDE
TWO
CARBON
4'-9" X 16'
ELUTION
COLUMN
5'-9" X 16'
CARBON
REGENERATION
FURNACE
CARBON
QUENCH
TANK
CARBON
FINE
TANK
TAILINGS
POND
ACID
AIR
CN DESTRUCT
REACTOR
TANK
CARBON
FINES
HEAT EXCHANGER
PREGNANT
SOL'N TANK
E/W
BYPASS
ELECTROWINNING
CELL
FURNACE
80 LB CAPACITY
GOLD BARS
BARREN
SOLUTION
TANK
PLACER DOME (CLA) LIMITED
MUSSELWHITE MINE MILL
4000 TONNES
MILL PROCESS FLOWSHEET
Prepared By: Nuri Hmidi
DATE: JUNE, 2001

Appendix E – Testing, Design and Evaluation of Pre-concentration Options for INCO's Pipe II Deposit in Northern Manitoba

8.1 Introduction

Following the successful conclusion of the work at INCO's Sudbury Operations, INCO was approached to initiate an expanded study for pre-concentration into the pre-concentration of low-grade ultramafic nickel deposits at INCO Thompson in Northern Manitoba as well as additional work on VMS nickel deposits at Voiseys Bay. Results of the INCO Thompson study are presented in this Chapter. Work on the Voisey's Bay study will start in January 2008 and is not considered a part of the scope of this thesis.

The Thompson Nickel Belt (TNB) is a 10-35 km wide geologically complex igneous intrusive belt located on the boundary between the North West Superior Province and the Churchill Province of the Canadian Shield. Since its discovery in 1961, two deposits in the region have been principally exploited, at the Thompson and Birchtree Mines. These operations are mature and have limited mineral reserves with which to keep the integrated Thompson milling and smelting Complex operating. However, several other so called 'ultramafic' Ni deposits have been identified throughout the TNB, including the Pipe II, Kipper and Opswagan Deposits, but these are generally low-grade, low Ni tenor and high MgO deposits which have been found to be difficult to process using the present mill configuration. The presence of chrysotile is a further challenge at the mill due to respiratory safety concerns and full respiratory protection is required when handling these ores. Ore has been mined from the Pipe deposit previously, but operations were ceased in 1986 due to the incumbent low Ni prices in conjunction with the problems mentioned above. Pipe is the southernmost of the Northern sequence of the ultramafic deposits in the Thompson Nickel Belt, which include the Moak, Manasan, and Opswagan deposits. Deposits to the South of Pipe include the South Mystery and Setting deposits (Figure 1).

Pipe is thus representative of a class of large, highly ultramafic Ni deposits in the TNB which have been previously mined and processed, but are typically low in Ni:S ratio and high in MgO which makes them unattractive from a technical and economic perspective. However, INCO presently has resources of over 200 Mt of this type of ore, and due to the diminishing reserves of the high tenor, low magnesium ores at Thompson and Birchtree, is interested in assessing the present economic potential of mining and processing these ores, starting with an assessment of the Pipe II deposit. INCO had previously developed a type conductivity sensor for low grade ores, and it was decided to investigate the impact of pre-concentration, either by conductivity sorting or dense media separation (DMS), on the mining, transport and downstream processing of the Pipe II ore.

UBC were approached to undertake pre-concentration research with INCO Thompson on the deposit for the purposes of assessing the potential for application of these technologies to Thompson ultramafic ores. Site visits for fieldwork and sampling were undertaken in April, June and August 2006. A prototype of the INCO sensor was made available to UBC and subsequently laboratory evaluation of the response of the core to conductivity sensing was undertaken in 2006, with a further planned phase of laboratory testwork on conductivity based sorting in 2007 should the ore prove amenable to the technology. Design criteria were developed from the laboratory testwork and a preliminary system design was undertaken at the scoping study level and evaluated with results for inclusion into the scoping study by INCO ITSL.

Options for the project included a ‘small mine’ scenario operating at 13 000 tpd and delivering feed to the existing Thompson mill, and a ‘large mine’ scenario delivering 45 000 tpd of feed to a custom designed mill located at Pipe. Both scenarios were to be evaluated with- and without pre-concentration in order to develop comparative economics for the options. An order of magnitude study including metallurgical testwork and process development for the pre-concentration option was conducted in order to assess this, the results of which are presented in this Chapter.

8.2 Background: Geology, Mining and Processing of Ores from the Thompson Nickel Belt

The Thompson Nickel Belt is a 10-35 km wide intrusive belt comprising highly mixed reworked Archean basement gneisses of the Superior Province, early Proterozoic cover rocks of the Opswagan Group, granitoid plutons and mafic to ultramafic dykes of the Molson dyke swarm, located on the boundary between the North West Superior Province and the Churchill Province (Bleeker, 1991).

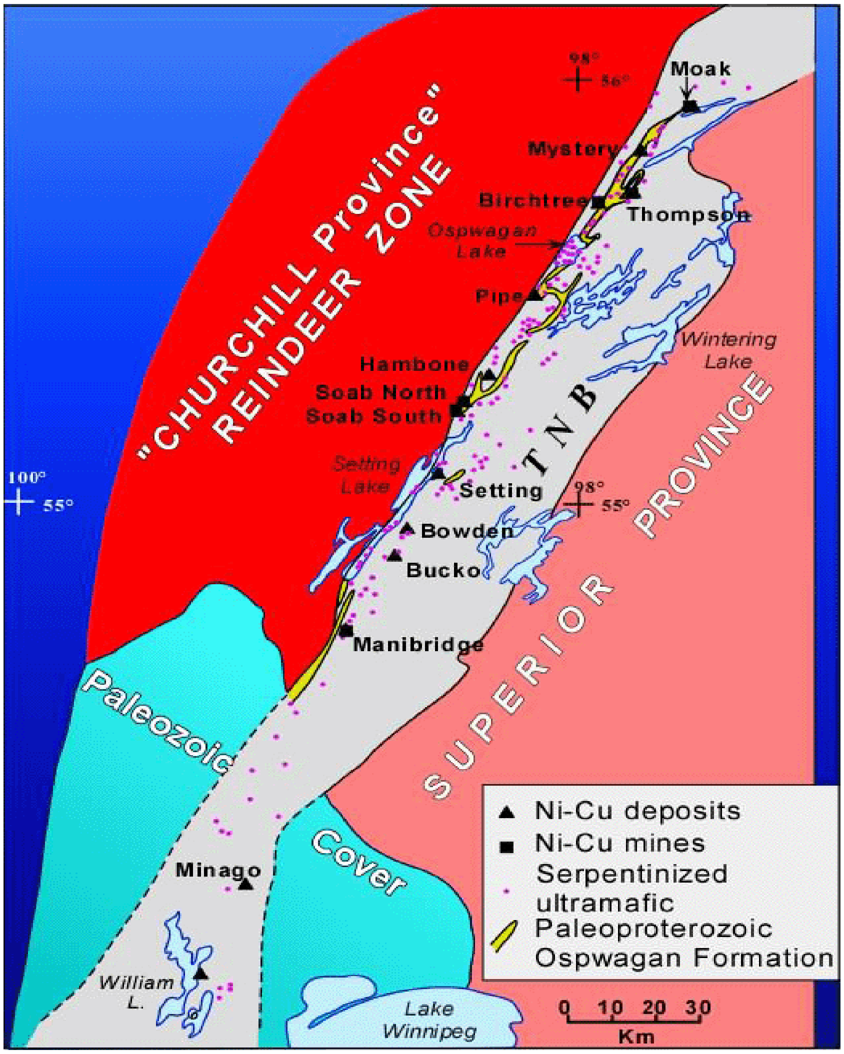


Figure 1 – Thompson Nickel Belt Regional Geology (after Hulbert et al, in press)

The belt is characterized by deeply dissected, multiaxial fold interference structures which, along with extreme deformation and subsequent erosional cover, makes the identification of conventional stratigraphy somewhat challenging. However, an empirical stratigraphy adopted by INCO geologists (Perederey et al, 1982) has been generally accepted as descriptive of the TNB.

Economic mineralization is associated with ultramafic sills which intruded co-evally with the dyke swarm into the Proterozoic cover sequence (Bleeker, 1991). Furthermore, all known deposits in the Northern Thompson Nickel Belt are lithologically and geochemically associated with major sedimentary sulphide concentrations in the graphitic schist, although the origin of the nickeliferous ores is clearly magmatic. Ore mineralization can be classified into 4 main categories (Rubingh, 2006):

- Ni enriched metasedimentary sulphides (minor occurrences, not considered at Pipe)
- Extraparental massive sulphides external to the ultramafic sills
- Intraparental sulphides occurring as massive, vein, net-textured and disseminated sulphides within the boundaries of the ultramafic sill (see Figure x+1)
- Breccia sulphides – massive matrix sulphides incorporating small zenoliths of schist wall rock (see Figure x+2)

The massive sulphides are of moderate tenor and have been determined to be the most economical ore to mine at Pipe. However, the mining of the intraparental and brecciated sulphides are considered challenging due to variable grade, high levels of internal dilution, and the presence of magnetite and chrysotile magnesium which negatively affects metallurgical performance in the Thompson mill.

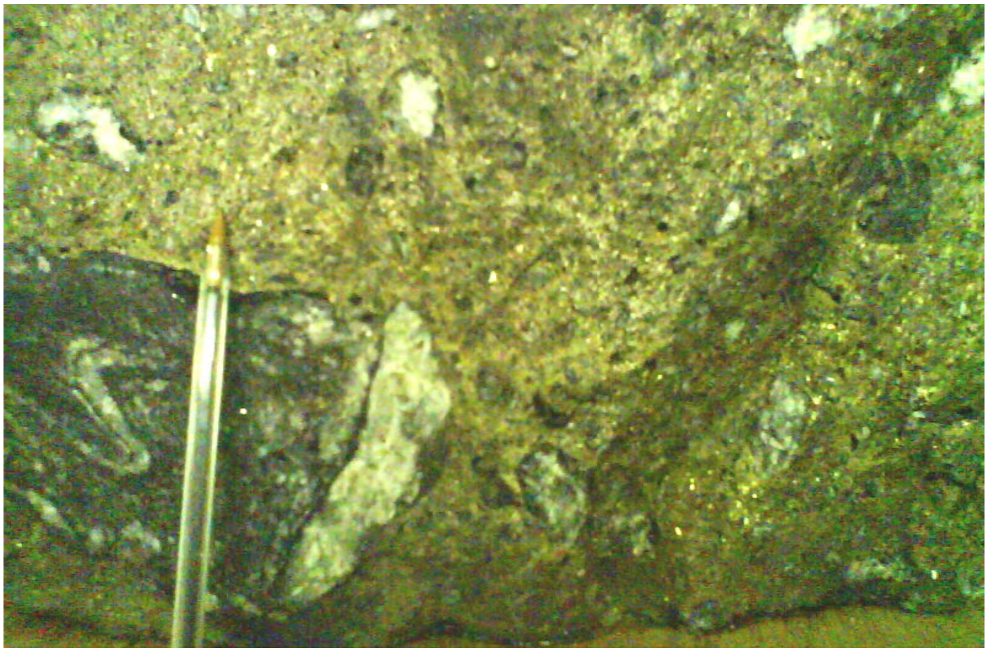


Figure 2 – Photograph of extraparental massive sulphide sample at Pipe. Note highly foliated nature of breccia inclusions that mimic the overall structural geology of the TNP

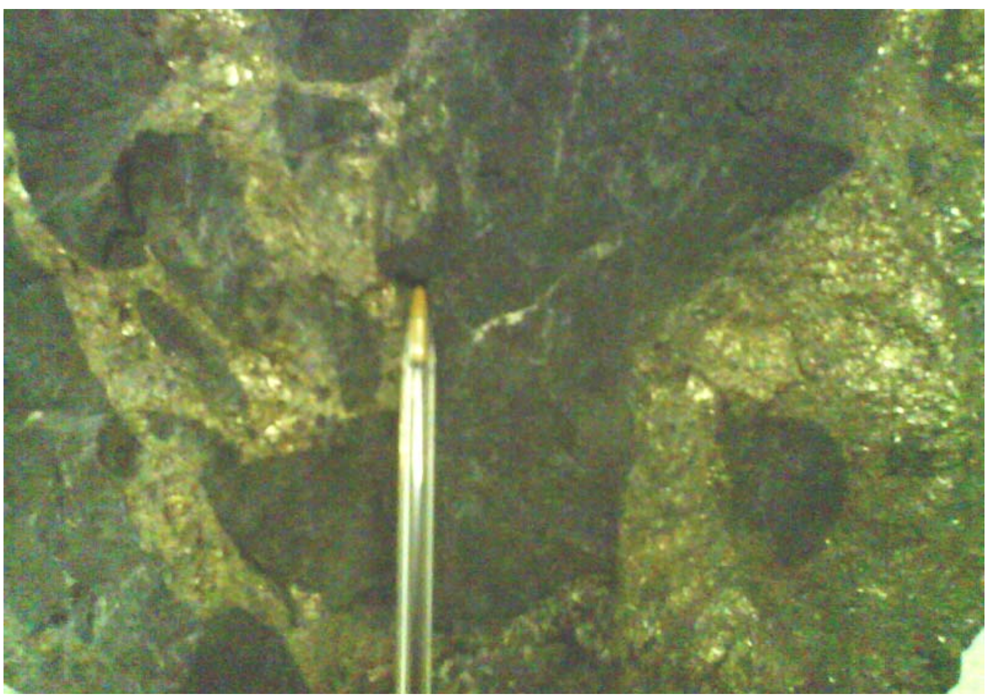


Figure 3 – Photograph of intraparental brecciated sulphide sample from Pipe. Note massive / net textured sulphides with incorporated schist inclusions

The Pipe deposit was discovered in 1957, and has been variously mined via shaft underground at Pipe I and II as well as by open pit at Pipe II from 1969 – 1985 (See Figures x and y). Pipe II is the southernmost of the Northern sequence of ultramafic deposits and is cut off to the South by an E-W conjugate fault, and incorporates all of the major styles of mineralization found in the Thompson Nickel Belt. Localised geology around the Pipe II deposit is presented in Figure x on the facing page. Mineralogically, the ore consists of pyrhottite, pentlandite, chalcopyrite, cubanite, shear-hosted magnetite and a large proportion of chrysotile magnesium oxide spinel (Figures x and y).

Mining

Since their discovery in 1961, two major deposits in the region have been exploited, at the Thompson (North Pit, South Pit and underground Mine) and Birchtree Mines. Ore from the Pipe Mine has been historically mined and trucked approximately 40km to the Thompson mill for processing. Operations were ceased in 1986 due to a combination of the low grade in the pit, the refractory nature of the ore and low nickel prices at the time. Ore from the Thompson and Birchtree Mines by comparison to Pipe appears to be high tenor with moderate levels of MgO. As the deposits are generally massive and polydimensional, mining is either by open pit near surface, or by longhole open stoping methods in the underground setting. Characteristics of the ROM ore from Thomspon, Birchtree and the ore expected from Pipe are presented in Table 1 for comparison.

Table 1 – Comparative ROM ore characteristics at INCO Thompson

Ore	**Ni %**	**MgO %**	**Throughput (tpd)**
Thompson	2.07		5300
Birchtree	1.81	17	4500
Pipe II	0.76	34	n/a

Processing

The integrated metallurgical complex at INCO Thompson includes a conventional grinding and flotation mill, and a dedicated Ni smelter and refinery. Copper concentrate is shipped from Thompson mill to the Copper Cliff smelting and refining complex in Sudbury. The Thompson Mill was designed with a capacity of 15 000 tpd, but is presently operating at nominally 10 000 tpd, and in the absence of new feed from Pipe or other local deposits will be reduced in future to 6000 tpd. Treatment at the Thompson mill is via a simple grinding and flotation (MF1) circuit, with secondary scrubbing of the rougher tails prior to scavenger flotation. Primary grind is 149 μm. Flotation is generally by Denver-type flotation cells; however the mill was recently upgraded with a new MgO rejection circuit comprising Outokumpu OK80 tank cells with enhanced CMC depression (Munionen, 2006). Current mill performance is presented below in Table 2. Combined recovery of the Thompson and Birchtree ores is approximately 85% (Drapak, 2006)

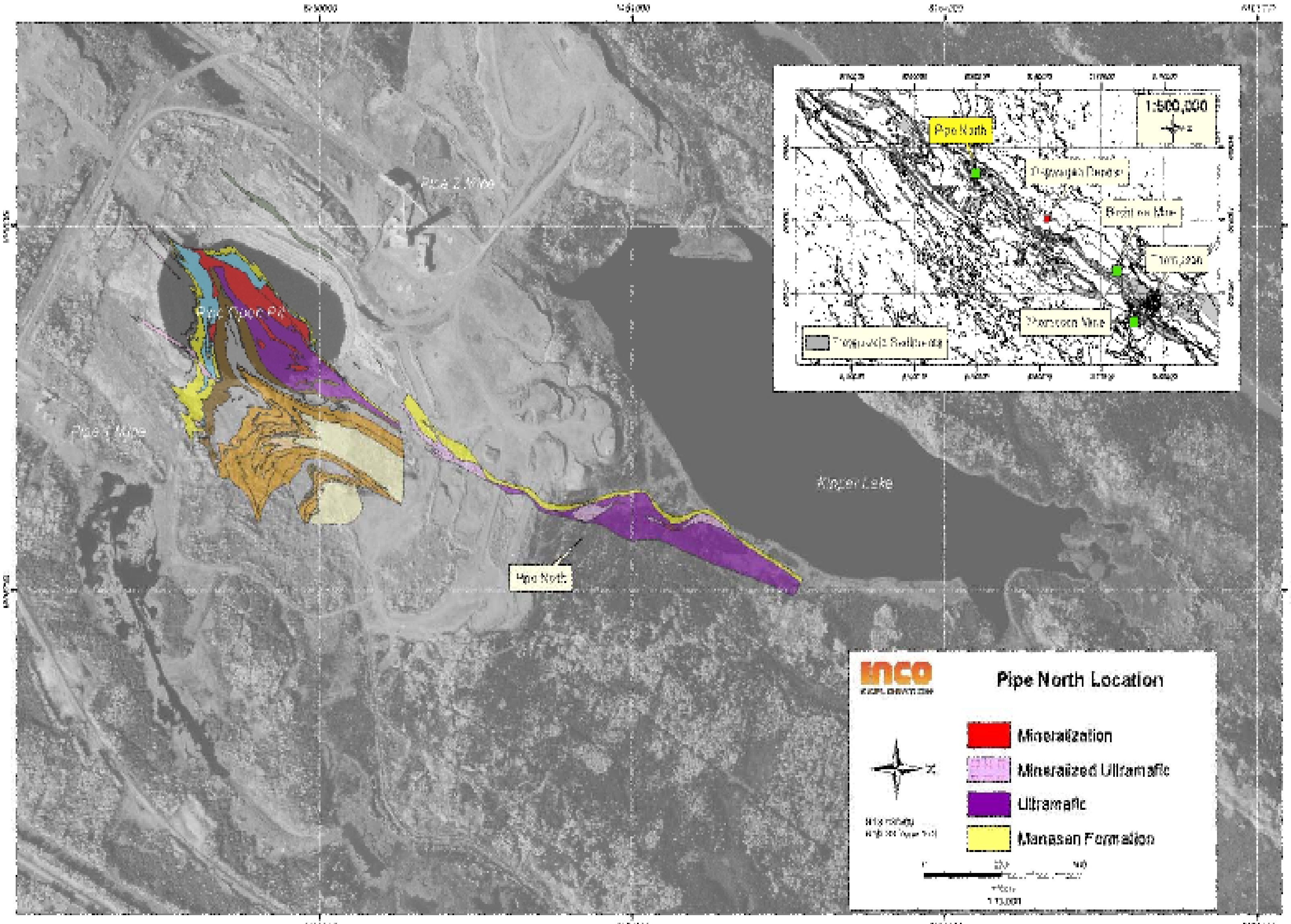


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Figure 4 – Pipe II Localised Geology (courtesy INCO ITSL)

Recovery in the mill is highly sensitive to feed grade and magnesium content. Mill recovery varies with feed grade according to the relation:

$$R = 93 - \frac{(16.1*96.3\ \%)}{g} \qquad - (1)$$

Where g is the feed grade to the mill in %Ni.

At an average between 0.46 – 0.76% Ni, it is expected by INCO that the recovery of Ni from Pipe ores in the Thompson mill would range from 69 – 76% (Penswick, 2006). The high levels of magnesium assayed in the ore also present a number of problems at the mill. The presence of MgO minerals in the feed decreases the pulp density and viscosity in the grinding mills, thus decreasing grinding efficiency. MgO also grinds preferentially and reports to the flotation circuit as fine, hydrophobic slimes which tend to report to the flotation concentrate, lowering concentrate grade, and increasing filtration times as well as final moisture content in the cake. MgO in the concentrate is a problem as it requires a higher temperature in the furnace to melt it to slag, which reduces the life of furnace linings. This can be countered by increasing the depression of MgO, however this also negatively affects Ni recovery. Pipe II ore at 0.76% Ni and 34% MgO contains over 100% more MgO than the Birchtree ore, thus the planned milling of this ore involves severely exacerbates these problems.

Table 2 – Present Thompson Mill Feed Parameters

Feed	Thompson	Birchtree
Throughput	5300	4500
Ni	2.07	1.81
MgO	~9	17
BWI	14	13
Ni Concentrate		
Ni	14.4%	
Cu	0.29%	
MgO	3%	
Cu Concentrate		
Ni	4.1%	
Cu	16.4%	
MgO	3%	
Tailings		
Ni	0.28%	
Recovery	90%	80%

8.3 <u>Methodology, Sampling and Metallurgical Testwork</u>

The main purpose of the study was to assess the response of selected Pipe ore samples to conductivity-based sorting methods. The results of preliminary testing would be generalized to form the basis of a metallurgical flowsheet for pre-concentration at Pipe, as well as the basis for the impact evaluation.

Two systems were selected for the testwork, a commercial sensor supplied by AST out of Sydney, Australia, and a conductivity sensor, the 'B2' sensor developed in-house by INCO through a previous project (Boucher, 2005). Both sensors were installed on a commercially available conveyor belt fitted with a pneumatic ejector system below the head pulley in order to undertake the sorting. The intention of the testwork on Pipe ore was to deliver a technical result for the pre-concentration of the Pipe ore as well as a preliminary business case analysis for pre-concentration at Pipe using the parametric evaluation model previously developed at UBC. Major impacts to be quantified included any impacts to NPV as well as mine life for Thompson as the project could potentially add significant life to the Thompson mining camp.

Grab samples were obtained in May 2006 for preliminary evaluation. Core logging and mineralogical evaluation were undertaken by INCO ITSL in Thompson.

Preliminary metallurgical testwork was conducted to establish the amenability of the ultramafic Pipe type ores to conductivity sorting. Dense media separation testwork was not performed, although testwork on similar ultramafic ores indicates that comparable metallurgical results could be achieved using either process. During core logging and sensor calibration tests (see Chapter 2) a good correlation between static EM reading and % Ni was established for all four holes using both sensors. The performance varied between holes with variation in mineralization, and the presence of magnetite in BH115140 was identified as a potential problem. The MgO content measured showed a reverse correlation with the EM reading, therefore the potential for preferential rejection of MgO was indicated (Stephenson, 2006). Data from the calibration testwork was utilized in a synthetic sort in order to provide an indication of the performance of the AST and 'B2' sensors on these samples. Results are shown in Table 3.

Table 3 – Preliminary sorting results.

	B-2 Sensor	AST Sensor
Feed Grade	1.87 % Ni	1.87 % Ni
Pre-conc Grade	2.43 % Ni	2.08 % Ni
Ni Recovery	94.9 %	94.1 %
Mass Rejection	27 %	15.5 %

Both sensors indicated a moderate degree of waste rejection with high Ni recoveries.

Based on the results of the amenability evaluation, a further test was performed on the mineralized zone of BH 115141 in order to provide a sample of sorter concentrate for downstream flotation testwork at Sheridan Park. An 80 kg sample of the core was crushed and screened into -50 + 26.9mm, -26.9 +10mm and -10mm fractions, then split in preparation for sorting using the sorting conveyor equipped with the AST and 'B2' sensors. The -10mm fine fraction was not sorted. Products from the sort were then sent to Sheridan Park for standard flotation tests.

Table 4 – Sorting Products for Flotation Testing

	INCO B-2 Sensor	**AST Sensor**
Feed Grade	1.03 % Ni	0.68 % Ni
Pre-conc Grade	3.21 % Ni	0.98 % Ni
Ni Recovery	68.4 %	79.9 %
Mass Rejection	78.2 %	45.2 %

Mass rejection using the 'B2' sensor was 78.2% at a recovery of 68.4% Ni. Mass rejection with the AST sensor was lower at 45.2% but higher in metal recovery. The results indicate potential f this technology with refinement to sort low grade ore with moderate mass rejection and high recovery. Flotation of sorter concentrate after grinding to -149um produced a concentrate grading 23% Ni and 2.5% MgO at a recovery of 70%. The product is considered acceptable smelter feed, with high Ni grade and low MgO content. However, overall Ni losses on the sample were considered too high.

A 1910kg sample of mineralized rock was selected from the old low-grade stockpile at Pipe and sorted (Figure 5).

Figure 5 – Low grade stockpile at Pipe. Note the relatively consistent particle size distribution of the rocks in the sample.

The sort was performed in order to provide a basis for conceptual design criteria, based on which an economic evaluation of the pre-concentration process for the ore in the 'small' and 'large' mine scenarios could be made. The sample was crushed and prepared into -300 +80mm, -80 +9.5mm and -9.5mm fractions, which were then sorted, sampled and assayed. The -9.5mm fine fraction was not sorted.

Table 4 –Sorting results on Pipe ore using INCO 'B2' and AST sensors

Mineral Type	Feed (% Ni)	Conc. (% Ni)	Recovery (%)
Brecciated Massive Sulphide	2.97	2.98	99.1
Extraparental Massive Sulphide	3.51	3.51	99.3
Ultramafic / intraparental	0.40	0.49	10.6
Ultramafic / extraparental	0.32	0.95	29.4
Disseminated matrix sulphide	1.86	2.07	87.2
Disseminated matrix sulphide	2.05	3.11	85.9

Individual results of the sort vary widely with the type of mineralization. Sorter performance on the Massive and Brecciated sulphides is almost 100%, with lower results for the disseminated sulphides. Improving the sorting result for the most ultramafic of the disseminated-type ore proved difficult, however there is still potential to scalp residual Ni sulphides from this material if mined using this method. Ni grades are greatly increased in the sorting, which will massively benefit the Thompson mill in processing this ore. The preferential rejection of MgO is also indicated in the results with possible major positive impacts at the mill and the smelter.

Based on these results, the pre-concentration of the Pipe ore by conductivity sorting on the -300mm and -80 + 10mm fractions, either using the INCO 'B2' or AST sensors is strongly indicated as possible. Due to the coarse liberation size of the massive sulphides, fine crushing appears not to be required.

8.4 Design Criteria, Process Design and Expected Performance of the Surface Pre-concentration Facility

The initial resource at Pipe was indicated by INCO to be 27Mt at a nickel grade of 0.76%. A preliminary mining and processing scenario was evaluated for a proposed 29Mt, 0.76% Ni deposit at a mining rate of 3250 tpd. Results were marginally positive at a relatively conservative long-term estimate for the long term Ni price of $4.50/lb, due to low Ni grades and low expected metal recoveries for the deposit. In consideration of the pre-concentration results, it was decided to lower the resource cutoff grade and evaluate the exploitation of a potential 150Mt, 0.46% Ni deposit, to be mined at planned rates of 13 000 tpd and 45 000 tpd respectively (Penswick, 2007). Stripping ratio was to be 9:1, with a planned bench height of 15m.

The PSD of the ROM feed was not known at this stage of the study, so this was modeled using the 'Kuz-Ram' blast fragmentation model (Laing, 2003). Basic open pit mining parameters were agreed with INCO in order to provide a blast design for input into the Kuzram model (Table 5).

Table 5 – Input data to the 'Kuz-Ram' Blast Fragmentation Model

Hole Diameter	241mm
Charge Length	6m
Burden	5m
Spacing	5m
Drill Accuracy SD	0.2m
Bench Height	15m
Powder Factor	0.23 kg/tonne
Charge Density	0.66 kg/m3
Charge Weight per hole	250 kg

The blast Fragmentation profile generated from the pit design criteria is presented in Figure 6. A topsize of ~3m was indicated with a nominal particle size of 800mm.

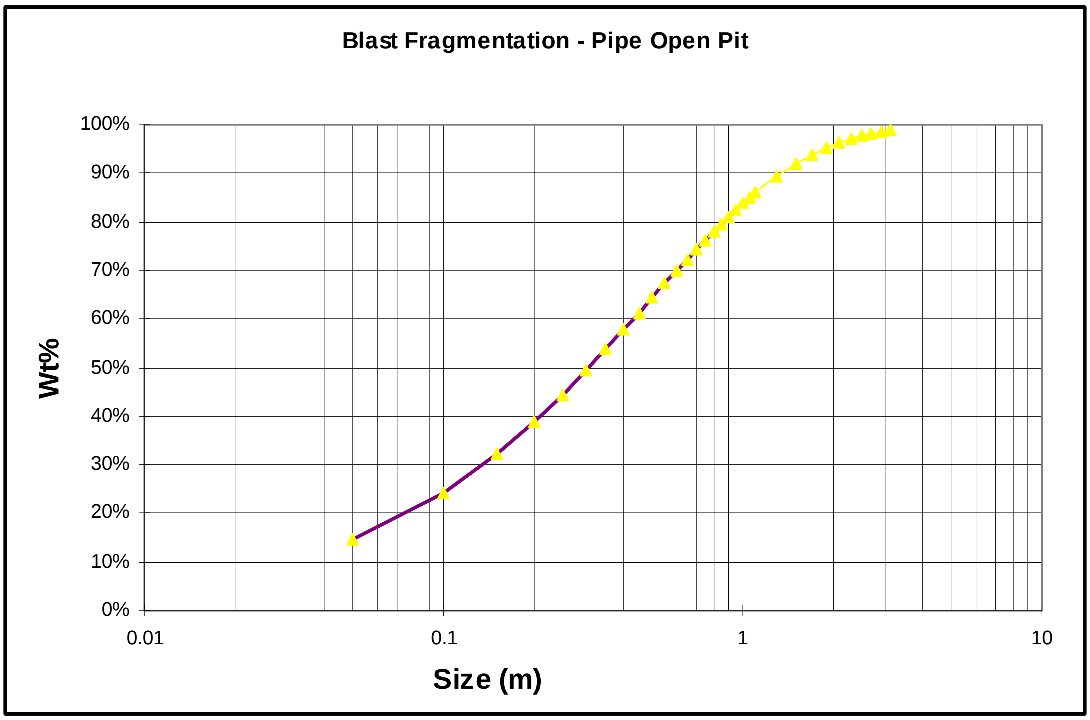


Figure 6 – Blast fragmentation profile generated by the Kuz-Ram model

While the bench layout and blast design were considered by INCO to be optimal for a pit of this size, this feed size was considered unsuitable for economical crushing. Scalping of the +1.5m material in the pit for secondary blasting was indicated in order to prepare adequate feed for a sorting plant. The topsize of the feed to the pre-concentrator after secondary blasting was thus modeled at 1.5m, which is considered reasonable for a medium-tonnage gyratory crusher such as a Metso 30-60 Gyratory.

Based on the results of the testwork, process plant design was to be for a conductivity-based sorting plant processing either 13000 tpd (small mine scenario) preparing feed for introduction into the present Thompson mill, or 45 000 tpd (large mine scenario) preparing feed for a dedicated mill to be built at the Pipe II site. Plant operation was to be 355 days/annum, 7 days/week, 24 h/day with 85% design availability, resulting in a nominal feed rate to the crusher of 637tph (small mine) and 2200 tph (large mine).

The fundamental plant design for the two scenarios is essentially the same, with primary differences being in the number of units selected for each duty (Figure 8). Sorter capacity is limited according to the topsize of material fed to the sorter, thus a relatively coarse topsize of 300mm was targeted for the design. Furthermore, the size range of ore reporting to the sorter is also to be limited to a maximum of 4:1, thus both fine sorters operating on -100 + 20mm material and coarse sorters operating on -300 +100mm material for the large mine, and -200 + 100 mm for the small mine were required. In sorting, as in Dense Media Separation, good feed preparation, with the minimum of adhering fines is essential, thus feed to the sorters was to be wet screened at 10mm prior to sorting (Figure 8 & 9). The fine- and coarse sorter product is then to be combined with the bypassed fines and delivered to the concentrate stockpile for transport to the Thompson Mill.

For the small mine scenario, 2 Mikrosort® Primary sorters operating at 200 tph were required for the coarse sort, and 3 Mikrosort® Secondary sorters operating at 100 tph were selected for the fine sort. The metallurgical performance of the sort was determined using the suite of results obtained from the scoping testwork as a basis. Significant waste rejection yet acceptable metal recovery was desired, thus from the data 30% waste rejection at 93% Ni recovery was chosen as the sorter operating point. The number of sorters required increases dramatically for the large mine scenario, and the plant design is presently considered unfeasible at this tonnage.

The projected metallurgical performance of the plant is indicated in the Table.

Table x – Process Parameters for Preconcentration of Pipe Ore

Metallurgical performance:	**Pre-concentrate Products:**	**Waste products:**
Waste rejection: 30%		-200 + 10mm
Mass pull to concs: 70%	0.728% Ni	0.013% Ni
Ni Recovery: 92%	0.058% Cu	0.001% Cu
MgO recovery: 30%	16% MgO	

Significant waste rejection is projected for the plant, with a 7% loss in metal value. Ordinarily this would constitute an unacceptable financial penalty as metal prices are high. However, as indicated by the testwork, there is preferential rejection of MgO during sorting with significant benefits in the performance of the mill. Recoveries on unsorted samples of Pipe ore indicate standard mill recoveries of 66.5% However, with sorting, the feed grade increases by 27%, MgO levels are reduced from 34% to 16%, and the mill recovery model indicates that mill recovery would improve to 71.9% which is in close agreement with the value from the flotation testwork. Thus the overall Ni recovery to concentrate on Pipe ore is unchanged with sorting, however the full operating cost saving is realized, thus the expected NPV of the savings in mining, transport and processing costs is maximized through the implementation of this technology. The impact of this result for the small- and large mine scenario has been evaluated using the parametric model that has been developed.

8.5 Parametric Evaluation of the Exploitation of Pipe II with and without pre-concentration

Capital costs for the open pit mining fleet, surface plant and infrastructure are estimated automatically in the parametric model using modified O'Hara indices. The base case estimate for a 13000 tpd mine and surface mill located at Pipe is presented in Table 10.The estimate is generally to ±30% accuracy, and includes for the design and construction of a winterized plant, including a 50% provision for supporting infrastructure such as offices, laboratories, workshops, stores, HT power transformation & power distribution as well as in-plant roads.

Access roads, overhead power lines, tailings facilities and closure costs are not included in the cost provision. The model indicates a capital requirement of $508.5m for the development of the small mine. Capital costs were developed in conjunction with CVRD-INCO and are considered accurate to ±30%.

Table 10 – Pipe II Base Case Capital Estimate

	Capacity	Base Estimate	Present Cost
Mining			
Scoops/trams	1	$7,164,603.40	$20,658,192.32
Haul trucks	1	$8,993,164.00	$25,930,606.49
Pre-strip & boxcut		$2,499,107.50	$7,205,848.02
Preproduction development		$52,000,000.00	$149,935,165.96
Infrastructure		$8,820,379.40	$25,432,404.78
Subtotal		**$79,477,254.30**	**$229,162,217.57**
Plant			
Crushing & Screening		$13,230,569.09	$38,148,607.17
Process Plant		$21,168,910.55	$61,037,771.47
Concentrator building		$8,820,379.40	$25,432,404.78
Subtotal		**$43,219,859.04**	**$124,618,783.43**
Feasibility Engineering & Design		$30,674,278.34	$88,445,250.25
Contingency	15%	$23,005,708.75	$66,333,937.69
Total		**$176,377,100.43**	**$508,560,188.93**

Capital costs are automatically adjusted for the pre-concentration scenario. Mining capital remains unchanged, and an additional capital amount of $54m is allowed for the construction of the sorting plant. Surface plant and infrastructure requirements are reduced to $98.3m for a plant treating 9000 tpd compared to the base case of $124.6m.

Operating costs were also examined. Costs for the sorting facility include for power, labour, consumables and maintenance and are estimated from operating cost estimates for similar plants using scaling factors for the tonnage and are considered accurate to ±30%. Projected plant operating costs were $1.80/t for the small mine.

The impact of pre-concentration on the economics of the Pipe operation are substantial. Capital costs are reduced overall by a small percentage. Operating costs are reduced by 7% due to the 30% reduction in tonnage after pre-concentration. Both extraction of the resource and metal recovery are improved over the base case, with extremely favourable economic impacts. NPV of the base case without pre-concentration is estimated at $610.9m at a discount rate of 10%. The NPV of the small mine with pre-concentration is increased by $371.6m to $982.5m.

After preliminary design of the 13 000 tpd sorting plant, it was considered unfeasible to extend the design of the plant to 45 000 tpd. The number of coarse- and fine sorters increases from 2 and 3 respectively to 6 and 6 respectively, which was considered presently unfeasible due to the increased cost of feed preparation and delivery as well as the raw cost of the sorters themselves. A pre-concentration option for the 45 000 tpd scenario was thus not examined. However, the NPV of the small mine scenario with pre-concentration was compared to the base case NPV of the large mine and found to be comparable.

8.6 Conclusion & Recommendations

For the Pipe II study it was shown that sorting of the low grade ultramafic nickel ore using induction-balance type conductivity methods dramatically improves the economics of exploitation of the deposit. Further development of the technology in terms of increasing, the power, sensitivity and accuracy of the sensors will also dramatically improve this situation. Environmental and permitting impacts were also found to be significant as it was determined that the NPV of the small mine scenario with pre-concentration was competitive with the NPV of the large mine scenario, indicating that the same value can be extracted from the resource with a significantly reduced total surface impact.

Further research and development on the sorting technology is required to improve the power, sensitivity and accuracy of the sensors. Further development is required also in the general field of sorting technology in order to develop sorters appropriate for the large mine scenario. Preliminary design work indicates that a sorter with a capacity of 400 – 600 tph would be appropriate for such a duty. For the project in question, a bulk sample is recommended to confirm and improve confidence in the metallurgical parameters. Sampling and testwork is recommended to be followed by a full pre-feasibility study to compare in detail the outcome of the pre-concentration scenario to the base case.

Appendix F: A Model System - Integrated Mining and Processing at Cameco's High Grade Uranium Operations in Saskatchewan

1. Introduction

Cameco Corporation is the world's largest uranium producer. Cameco operates a number of mines in Northern Saskatchewan, including Key Lake, Rabbit Lake, and the high grade McArthur River and Cigar Lake mines. At both McArthur River and Cigar Lake an integrated mining and processing approach is used where the ore is hydraulically mined, ground to -75um in a SAG mill and hydraulically hoisted some 300m to surface (Edwards, 2003). A development of this approach is proposed where the high grade U3O8 ore is mined, ground in a SAG mill as before, then concentrated prior to hydraulic hosting. As has been previously discussed, the pre-concentration of ore is not a new approach. It is the proposed pre-concentration of amenable ores underground, where that approach can be demonstrated to be economically and environmentally superior to the conventional approach, which is novel and is the focus of this thesis. The installation of non-mining infrastructure underground is also not novel – the installation of substantial non-mining infrastructure such as refrigeration, pumping, ventilation, crushing and hoisting plant underground is an established practice in large, deep-level mines. While backfill preparation is more common, the installation of backfill plants underground has been identified as superior in some case and examples of such installations are becoming more common (refs). An integrated mining and processing approach approach which comprises mechanised mining, pre-concentration of the ore underground, preparation and disposal of the waste rejects as fill, followed by hydraulic hoisting of the concentrated ore has previously been proposed in conceptual designs for base metal sulphide ores. The conceptual system model is shown in Figure 1 (Bamber, 2004; 2005).

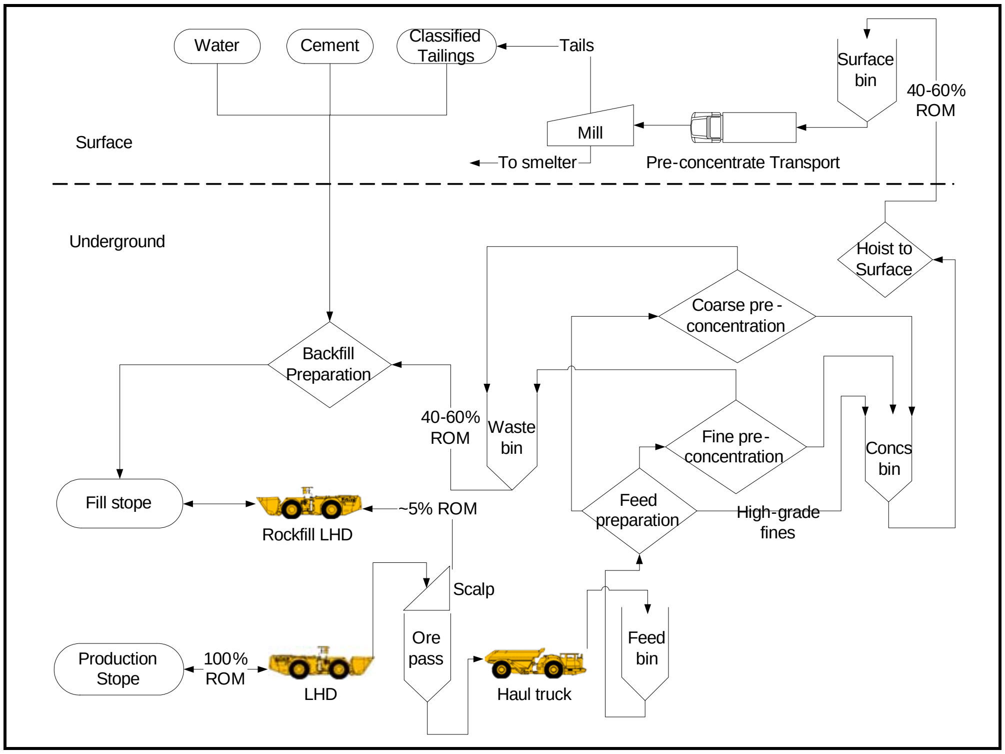


Figure 1 – Proposed Integrated Underground Mining, Processing and Waste Disposal System for Base Metal Ores

It is the integration of these technologies simultaneously and in combination and applied as a Lean Manufacturing approach in the mining of hard rock base metal ores which is considered both novel and highly original and is the focus of this thesis. Several case studies have been undertaken evaluating the potential of the conceptual approach on a range of ores including base metal sulphide ores, pyrite- and paleo-placer hosted gold, lead-zinc as well as some copper porphyry ores. All ores in the case studies have shown amenability to pre-concentration by various means, including optical, conductivity and radiometric sorting, dense media separation, and coarse-particle flotation, with metallurgical results varying between 20 – 54% waste rejection at recoveries between 91% and 99% (Scoble, 2006). Based on these results, significant benefits in terms of reducing the total environmental footprint of the mine are achievable, and further economic evaluation of the opportunities indicates positive economic benefits for mines with high operating cost to ore value ratios (Scoble, 2006). Adapted for use in a high-grade uranium application, this approach is suggested as an economically and environmentally superior approach for the mining and processing of these ores as well. Some advances have been already made at these mines in terms of mine mill integration, and these are themselves used as a precedent for some of the concepts developed elsewhere in the research. A case study focusing on Cameco's high grade uranium operations is presented as an example of the potential application of this approach outside of the hard rock mining industry.

2. Underground Processing Approaches at Cameco

2.1. Present Approach

Cameco presently operates two high-grade uranium mines at McArthur River and at Cigar Lake located in the in Athabasca Basin, Northern Saskatchewan (Figure 2).

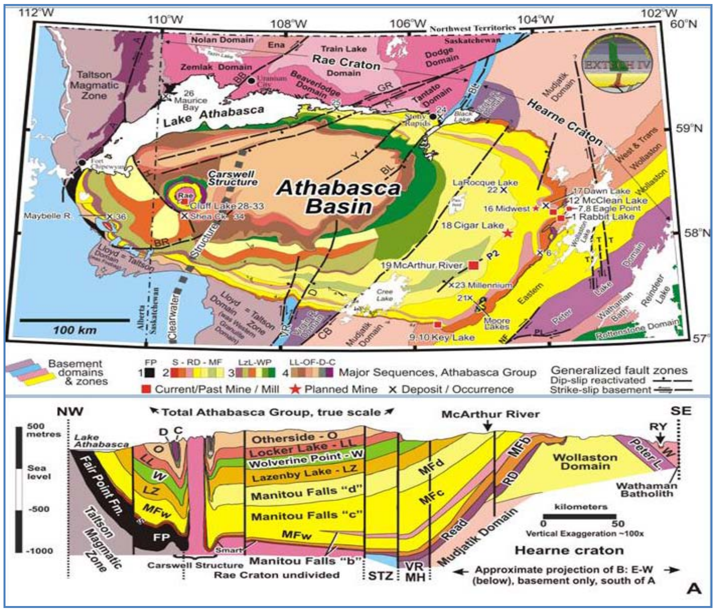


Figure 2 – Geology of the Athabasca Basin showing Uranium Deposits (after Card et al. (2003, 2005a, b), Portella and Annesley (2000a), Ramaekers et al. (2005b) and Thomas et al. (2002))[1].

At these mines, the high-grade uranium evaporites are found sandwiched between archean granite basement and overlying Athabasca sediments at a depth of between 450 and 640m (Cameco, 2006). Ore is mined using a combination of novel and highly innovative techniques (Edwards, 2003). The overlying sediments are saturated and unstable and are thus firstly stabilized by freezing of the outer core of rock into a ‘freeze curtain’ using refrigerated brine. The ore is typically a combination of uraninite, pitchblende and Athabasca sediments averaging 21% U3O8 (ref), and is hydraulically mined within the stabilized freeze curtain at nominally 4 tph by unmanned, automated 3mØ jet-boring machines operating at 20 MPa (Figure 3).

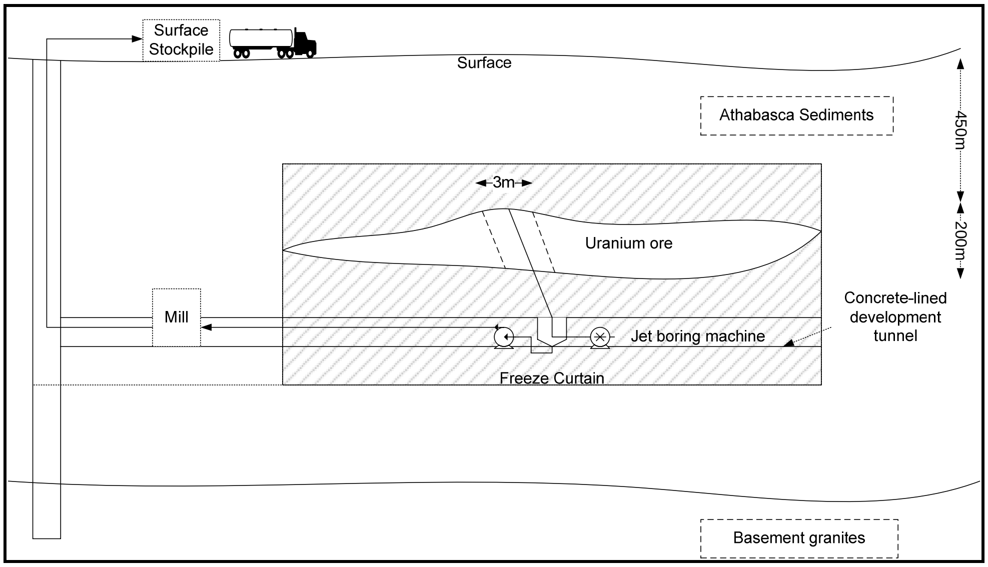


Figure 3 – Freeze curtain / Jet boring mining method at Cameco's McArthur River

The resulting -75mm (nominal) rock sludge is gravitated from the jet boring head via a surrounding downpipe to the borer sump where it is pumped centrifugally at 5% solids to an underground silo containing up to 3 days production, where the excess water is decanted off. Ore is recovered from the silo by clamshell grab and fed via hopper to the primary Metso waterflush cone crusher. Crusher underflow is passed to a 300kW SAG mill where the ore is ground to -75um and pumped to surface using one of 2 300kw Wirth positive displacement pumps (Figure 4).

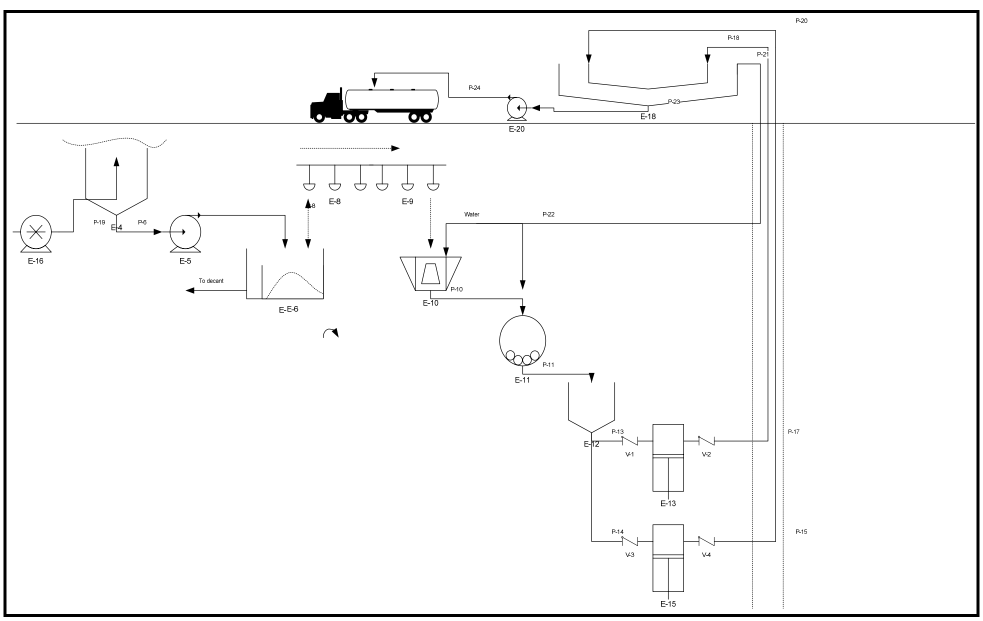


Figure 4 – Underground Mining and Processing of McArthur River Ore

All process facilities are unmanned and operated from surface. Maintenance is undertaken on a breakdown basis by specially shielded crews. The diluted ore is thickened on surface and shipped to the mill where it is combined with other U3O8 ores for processing. Thickener overflow is returned underground for use in the mining and comminution process.

2.2. Discussion of Present Approach

Experiences at the mines to date have been mixed. The freeze-curtain/jet-boring approach was justified on the requirement to have unmanned operations in high grade areas. Ongoing problems are being experienced in maintaining the freeze-curtain, and flooding of the mines has occurred a number of times. Cigar Lake is currently experiencing major delays due to a failure of the freeze wall and subsequent flooding of the mine in October 2006, and is presently planned to be in operation again in 2010 (Cameco, 2006). For these small scale operations a major challenge was encountered in the installation of the SAG mill underground, where the size of the mill was severely restricted by the small size of the shaft. At the time it was also felt that the hydraulic hoisting system was a high risk operation and a fully installed standby pumping system was installed. Since operations began, it has been found that the hydraulic hoist is highly reliable and one pump suffices generally for the duty. The value of this novel approach is also somewhat diminished as the high grade ores from McArthur River and Cigar Lake are presently delivered to existing process plants at Key Lake and Rabbit Lake respectively, where the ore must be combined with other low grade ores to meet existing feed specifications. The ore is thus diluted from 27% to 4% in the process, and most advantages presented by the high grade of the ore are lost in the process.

2.3. Developments in Integrated Underground Mining, Processing and Waste Disposal at UBC and Cameco

While embodying many good precedents in terms of mine-mill integration, the present installations at McArthur River and Cigar Lake cannot be fully considered to be an integrated mining and processing facility in terms of this thesis as no concentration of the ore occurs. No waste is rejected from the ore thus the full tonnage of ore is delivered to surface and thus no underground waste disposal facility is required. The integrated underground mining and processing approach seeks to reject and dispose of a significant component of unvaluable material from the ore prior to hoisting, and major benefits arising from this have been identified in previous research, including the environmental benefit of reducing the quantity of waste material disposed of on surface, together with significant projected cost savings and a resultant decrease in cutoff grade and increase in the mineable reserve (Bamber, 2005). Capital and operating cost savings of 30% and 20% respectively have been projected for a waste rejection of 50% using the parametric evaluation model that has been developed (Bamber, 2006). Several discussions have been held with technical personnel at Cameco (Edwards, 2005; 2006) and a significant opportunity was identified to improve the economics of this approach by integrating an underground concentration and waste disposal step into the design. While UBC has not been directly involved in the study, Cameco have proceeded independently with a pre-feasibility, and now feasibility study for Cigar Lake based on this model.

3. Proposed Approach

3.1. Integrated underground mining, processing and waste disposal at Cameco's High Grade Uranium Mines

A pre-feasibility for the underground mining, processing, and hydraulic hoisting of U3O8 ore has been undertaken for the proposed Cigar Lake mine in Saskatchewan by Cameco. The design also necessitated development of an appropriate underground waste disposal strategy for the pitchblende tailings. Design of the proposed mining and processing system was undertaken by Cameco for the purposes of costing and evaluation. Conceptual system design, followed by modeling and evaluation of the system using parametric methods was undertaken independently at UBC for comparison to the outcomes of the Cameco study.

As at the existing mines, ore is to be mined by the jet boring method, again using the refrigerated 'freeze curtain' approach to enhance the stability of the Athabasca sediments. ROM ore is delivered as a slurry at 4tph to the underground crushing and grinding circuit as in the previous design. However it is the intention to integrate the previous ore grinding and pumping arrangement with the front end of a uranium processing mill (Edwards, 2005). -75um SAG mill underflow would now be delivered to underground storage pachucas for thickening prior to leaching of the ore (Figure 5). Thickened ore is pumped from here to a series of atmospheric acid leach vessels for extraction of the U3O8 to solution. Pulp from the leach tanks is pumped to a Larox filter where the pregnant U3O8 solution is removed from the solids. Filtered leach solution is then pumped to the surface for precipitation and extraction of the yellowcake directly using hydrogen peroxide. A 53% U3O8 precipitate would be produced at nominally 1.5 tph from an ore grade of 21%, for a total uranium production of 20 mlb/annum, making Cigar Lake a significant producer of finished yellowcake in its own right. Barren solids at 2.5 tph and nominally 12% moisture are delivered to an underground tailings handling facility where they are disposed of in remnant underground excavations.

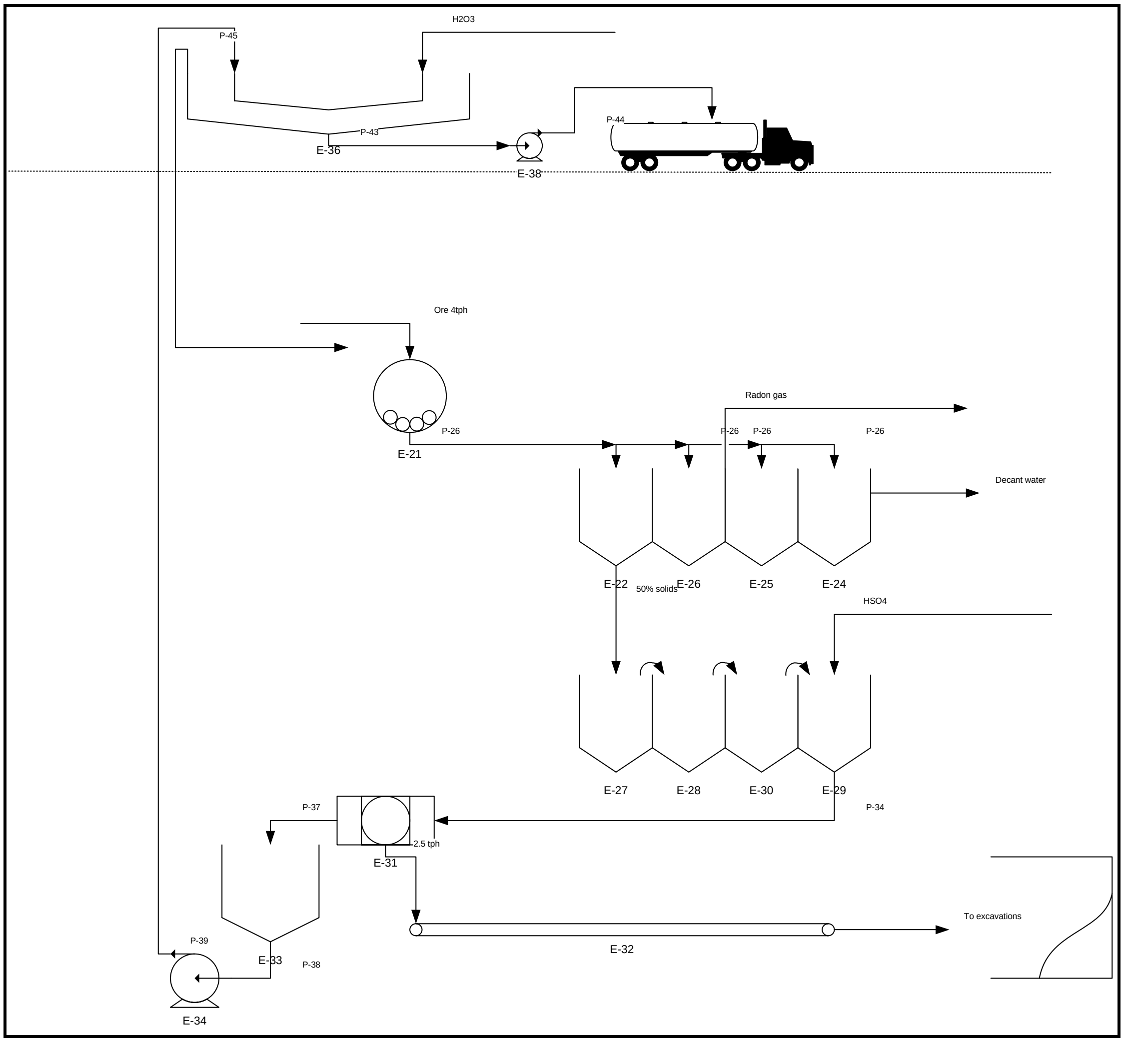


Figure 5 – Proposed Integrated Uranium Mining, Processing and Waste Disposal System for the Cigar Lake Mine

3.2. <u>Benefits of the Proposed</u>

Based on the outcomes of the Cameco study as well as modelling and evaluation by UBC, the benefits of this approach over the previous approach are projected to be significant. Uranium mill tailings is typically high in heavy metals and high in acid generating potential (ref), and disposal of this material underground is a significant benefit. Again, the integrated mine and mill design is planned to be operationally unmanned with resultant reduced exposure of workers to radon gas.

The grade of ore delivered to surface from these mines is extremely high, and strict radiation management procedures must be followed during surface transport and milling of these ores.

Milling facilities have to contain the radon gas arising from the ground U3O8 ore, and all structures must be fully α- and β-radiation shielded to meet Canadian safety regulations. This is typically achieved for surface mills with the provision of sophisticated and costly radon gas extraction and containment systems and costly lead/concrete shields for plant and buildings. However, it was identified in the study that these levels of protection would not be required for the underground mill as it is already a highly irradiated environment, and sufficient shielding is provided by the surrounding rock. Radon gas management is also simplified as the radon gas arising from the ground product is extracted from the underground mill product pachucas and sequestrated directly in the ground. Significant challenges were encountered in permitting McArthur River operation due to the high grade of ore arriving on surface for processing, however, based on discussions with the Saskatchewan Ministry of Mines, the permitting of the mine as it is now proposed would be much simplified, as the only product arriving on surface is a yellowcake precipitate and all other solid wastes are retained underground.

Substantial capital and operating cost savings are also anticipated. The decision to process the ore underground faciltates consideration of atmospheric leaching of the U3O8 over pressure leaching using autoclaves despite their poorer process performance (ref). Further savings are anticipated in terms of radiation containment and shielding as discussed above. Operating cost savings arise principally from savings in material handling and transport – up to 60% of the uranium ore would be removed and disposed of underground prior to hoisting. Hydraulic hoisting as previously utilized has been demonstrated to be economically superior to conventional hoisting (Edwards, 2004), and in this case the U3O8 to be pumped to surface is now in a solution, with further anticipated savings in terms of pumps and pipelines. Capital savings of 25-30% and overall operating cost savings of 32% overall were expected compared to the base case feasibility (Mainville et al 2007), which compares well to the predictions of the parametric model, and thus further validates the model (Figure 6). Together with the anticipated permitting advantages this substantially improves the feasibility of the proposed approach over the conventional approach.

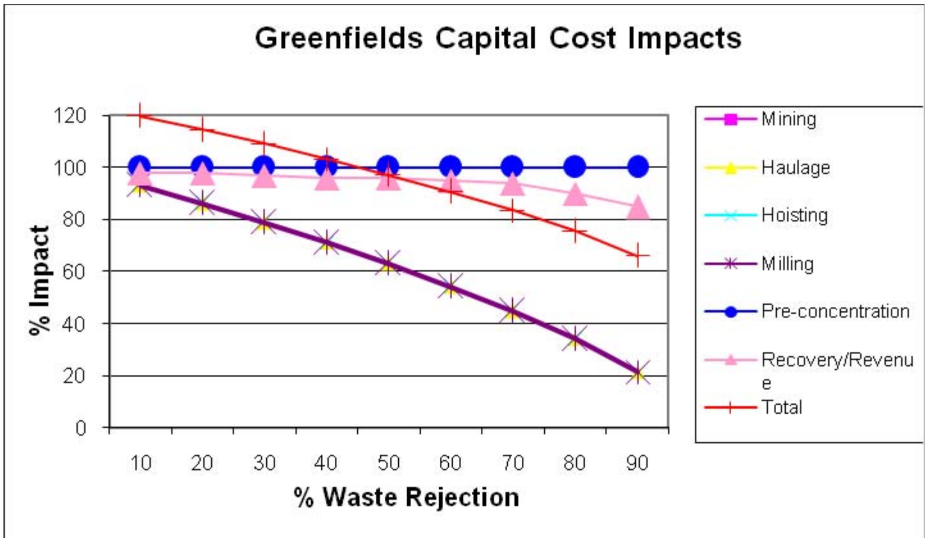


Figure 6 – Economic Impacts of Ore Pre-concentration (after Bamber, 2004)

4. Conclusion

A mining and processing approach without pre-concentration has already been adopted for the McArthur River and Cigar Lake mines by Cameco, and itself forms the basis for a number of features of the approach proposed in this thesis. The integrated mining, processing and waste disposal approach further developed by UBC for base metal mines, and now applied to high grade uranium ores is highly recommended for application by Cameco on future isolated, high grade deposits in Northern Saskatchewan in order to significantly improve the economic and environmental potential of exploiting these ores.

Appendix G : Density, Magnetic Susceptibility and Conductivity Values for Selected Rock Types and Minerals (from Ford, 1986)

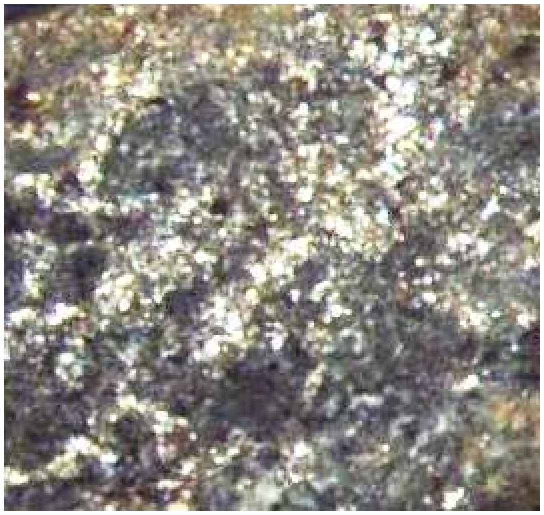
	Density g/cm^3			Magnetic SI x 10^{-3}			Conductivity mS/m		
Rock Type	**Min.**	**Max.**	**Av.**	**Min.**	**Max.**	**Ave.**	**Min.**	**Max.**	**Av.**
Sediments and Sedimentary Rocks									
Overburden			1.92						
Soil	1.2	2.4	1.92	0.01	1.26				
Clay	1.63	2.6	2.21				10	300	
Glaciolacustrine Clay						0.25	10	200	
Gravel	1.7	2.4	2				0.1	2	
Sand	1.7	2.3	2				0.1	2	
Glacial Till							0.5	20	
Saprolite (mafic volcanic rocks, schist)							50	500	
Saprolite (felsic volcanic rocks, granite, gneiss)							5	50	
Sandstone	1.61	2.76	2.35	0	20	0.4	1	20	
Shale	1.77	3.2	2.4	0.01	18	0.6	30	200	
Argillite							0.07	83.3	
Iron Formation							0.05	3300	
Limestone	1.93	2.9	2.55	0	3	0.3	0.01	1	
Dolomite	2.28	2.9	2.7	0	0.9	0.1	0.01	1	
Conglomerate							0.1	1	
Greywacke	2.6	2.7	2.65				0.09	0.24	
Coal			1.35			0.03	2	100	
Red Sediments			2.24	0.01	0.1				
Igneous Rocks									
Rhyolite	2.35	2.7	2.52	0.2	35				0.04
Andesite	2.4	2.8	2.61			160			
Granite	2.5	2.81	2.64	0	50	2.5			
Granodiorite	2.67	2.79	2.73						
Porphyry	2.6	2.89	2.74	0.3	200	60			
Quartz Porphyry				0	33	20	0.04	1.7	
Quartz Diorite	2.62	2.96	2.79						
Quartz Diorite, Dacite				38	191	83			
Diorite	2.72	2.99	2.85	0.6	120	85			
Diabase	2.5	3.2	2.91	1	160	55			0.03
Olivine Diabase						25			
Basalt	2.7	3.3	2.99	0.2	175	70			0.2
Gabbro	2.7	3.5	3.03	1	90	70			0.02
Hornblende Gabbro	2.98	3.18	3.08						
Peridotite	2.78	3.37	3.15	90	200	250			
Obsidian	2.2	2.4	2.3						
Pyroxenite	2.93	3.34	3.17			125			
Monzonite, Latite				33	135	85			
Acid Igneous Rocks	2.3	3.11	2.61	0	80	8			
Basic Igneous Rocks	2.09	3.17	2.79	0.5	97	25			
Mafic Volcanic Rocks							0.09	0.27	
Dacite	2.35	2.8	2.58						
Phonolite	2.45	2.71	2.59						
Trachyte	2.42	2.8	2.6	0	111	49			
Nepheline Syenite	2.53	2.7	2.61						
Syenite	2.6	2.95	2.77	0	111	49			
Anorthosite	2.64	2.94	2.78						
Norite	2.7	3.24	2.92						
Metamorphic Rocks									
Quartzite	2.5	2.7	2.6			4			
Schist	2.39	2.9	2.64	0.3	3	1.4			
Marble	2.6	2.9	2.75						
Serpentine	2.4	3.1	2.78	3	17				
Slate	2.7	2.9	2.79	0	35	6			
Gneiss	2.59	3	2.8	0.1	25				
Amphibolite	2.9	3.04	2.96			0.7			
Eclogite	3.2	3.54	3.37						
Granulite	2.52	2.73	2.65	3	30				
Phyllite	2.68	2.8	2.74			1.5			
Quartz Slate	2.63	2.91	2.77						
Chlorite Schist	2.75	2.98	2.87						
Skarn	2.95	3.15				2.5			1.25
Hornfels	2.9	3				0.31			0.05
Sulphide Minerals									
Chalcopyrite	4.1	4.3	4.2	0.02	0.4		1.11	6.67	
Galena	7.4	7.6	7.5			-0.03	1.11	1.47	
Pyrite	4.9	5.2	5	0.03	5.3		1.67	8.33	
Pyrrhotite	4.9	5.2	5		3200		6.25	5.00	
Sphalerite	3.5	4	3.75	-0.03	0.75		0.08	3.70	
Other									
Magnetite			5.04	1000	5700				1.92
Graphite	1.9	2.3	2.5	-0.08	0.2		1.01	3.57	

Compiled from Grant and West (1965), Keller and Frischknecht (1966), Carmichael (1982), Hunt et al. (1995), Palacky (1986), and Telford et al. (1990).

Appendix H - Optical Results from the Research

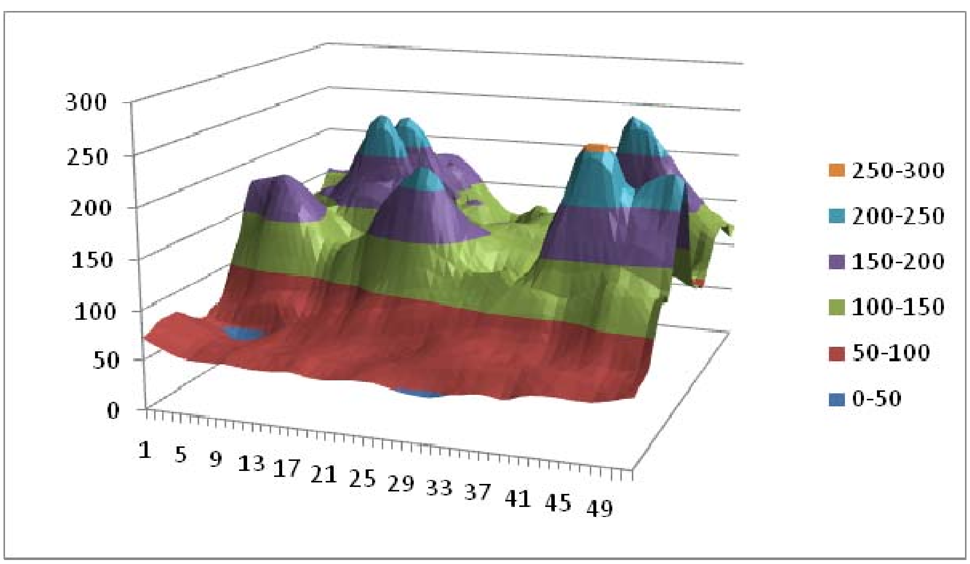
Measurements and correlations of colorimetric analysis with ore grade for selected ore and waste samples: RGB data; Intensity Scale 0-255
Craig 8112 Colorimetric Analysis

Craig 8112 Ore: 1.26% Ni, 0.57% Cu



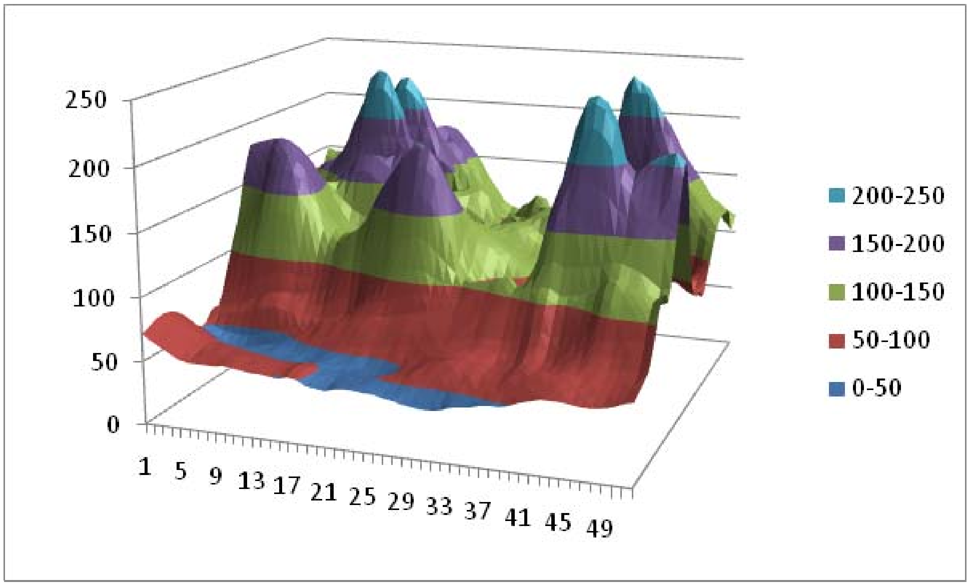
Red

Ave	66.93891
Median	61
Max	255
Min	0
Sdev	38.83589



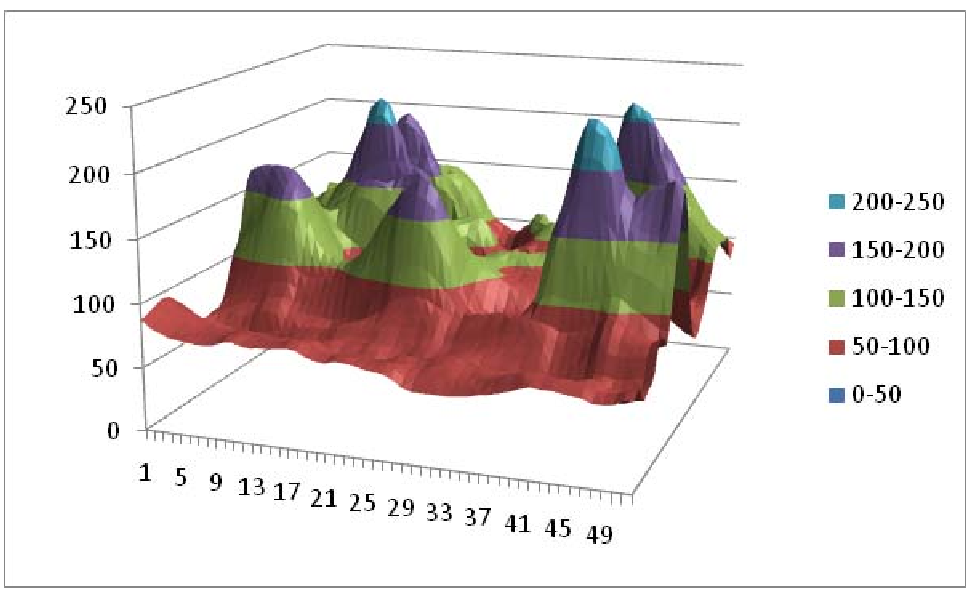
Green

Ave	66.93891
Median	61
Max	255
Min	0
Sdev	38.83589



Blue

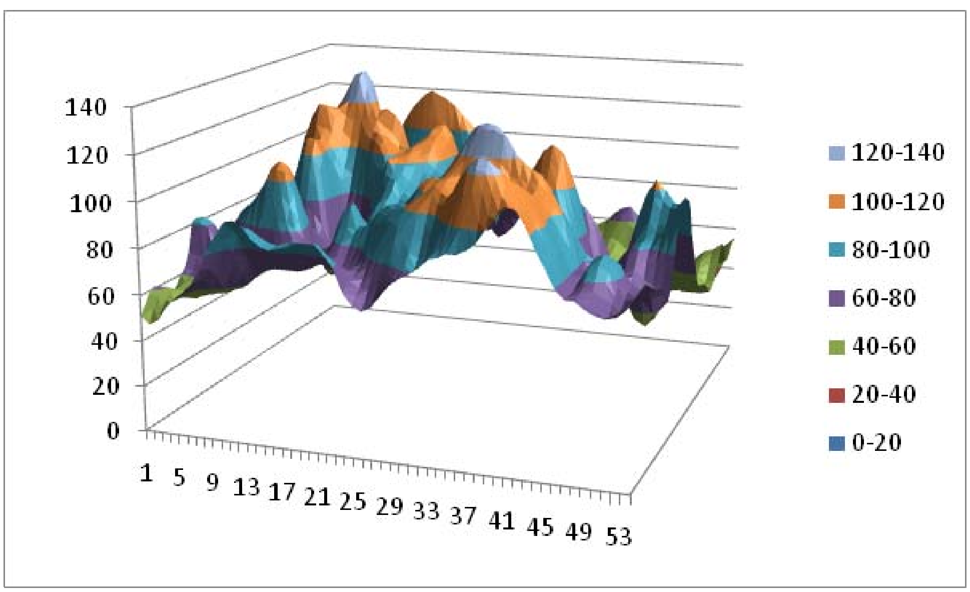
Ave	108.282
Median	99
Max	255
Min	10
Sdev	50.55641



Craig 8812 Waste: 0.19% Ni, 0.12% Cu

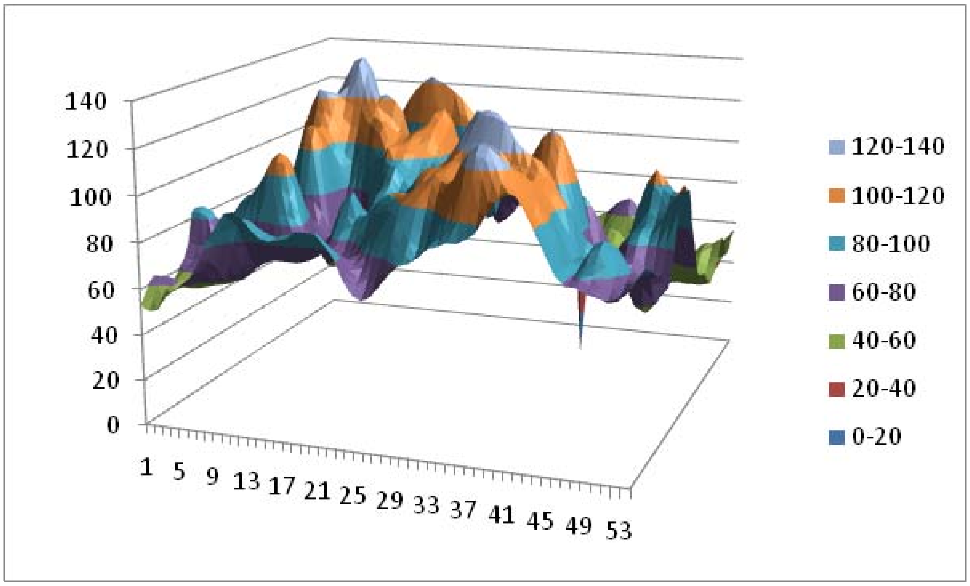
Red (Scale 0-255)

Ave	77.03692
Median	71
Max	255
Min	4
Sdev	36.90777



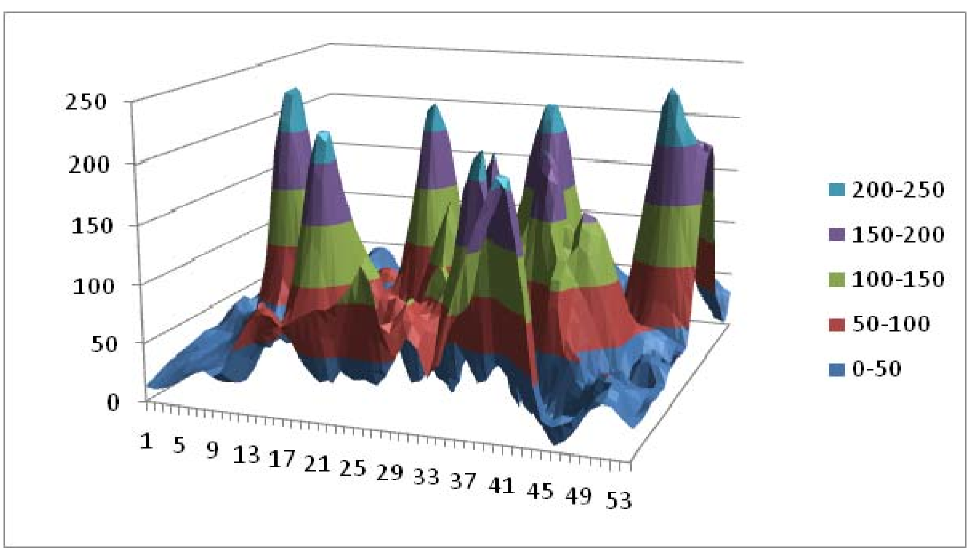
Green

Ave	66.93891
Median	61
Max	255
Min	0
Sdev	38.83589

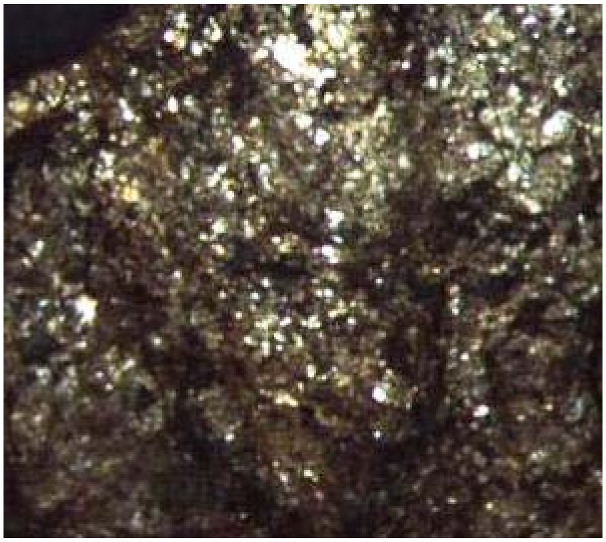


Blue

Ave	66.93891
Median	61
Max	255
Min	0
Sdev	38.83589

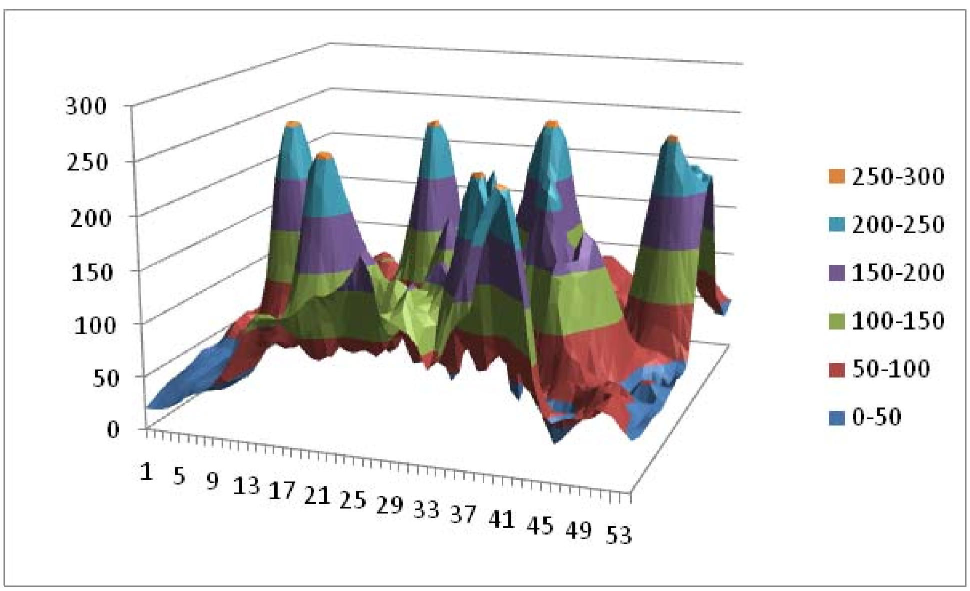


Craig LGBX Ore: 3.5% Ni, 0.38% Cu



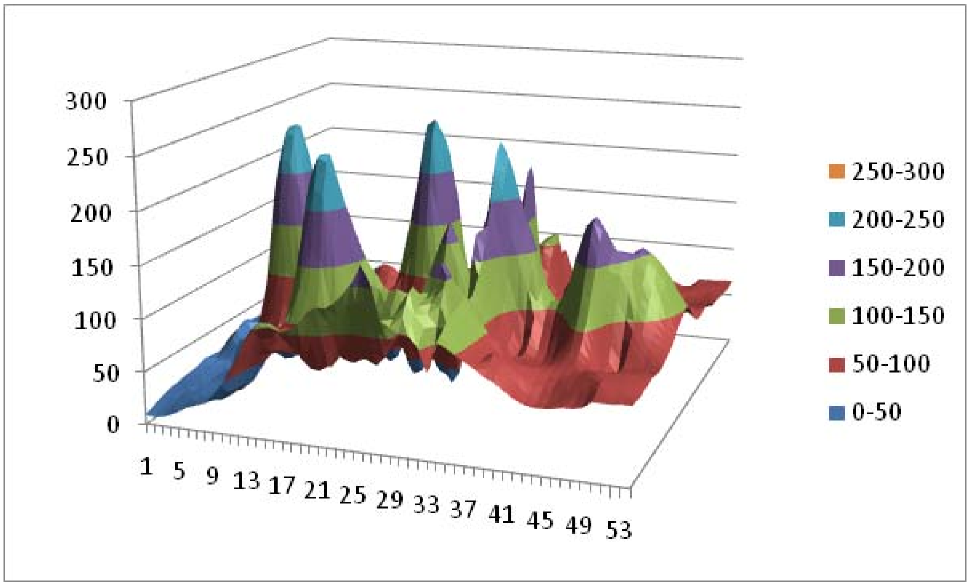
Red

Ave	109.4111
Median	99
Max	255
Min	0
Sdev	51.33175



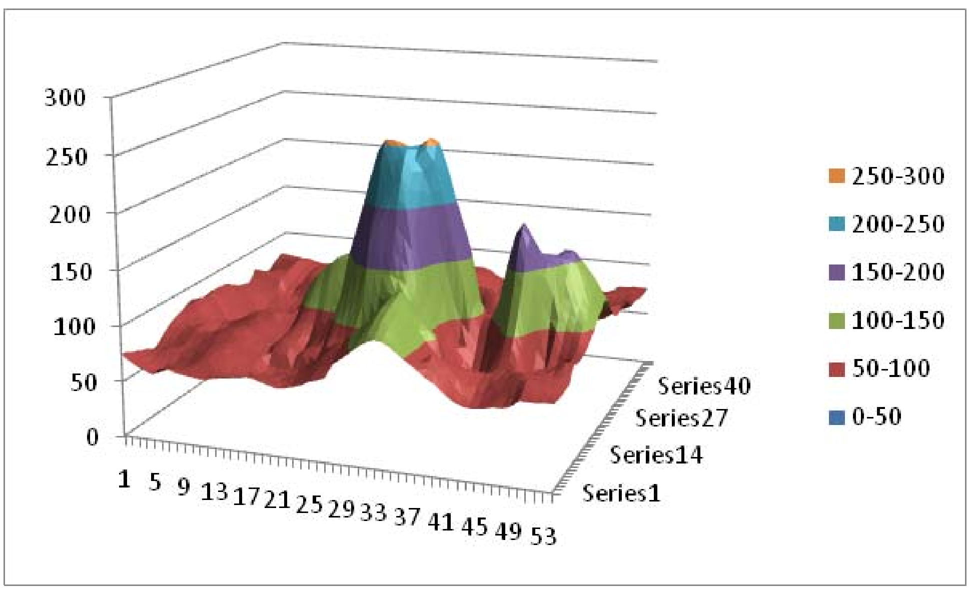
Green

Ave	104.7625
Median	93
Max	255
Min	0
Sdev	58.43261

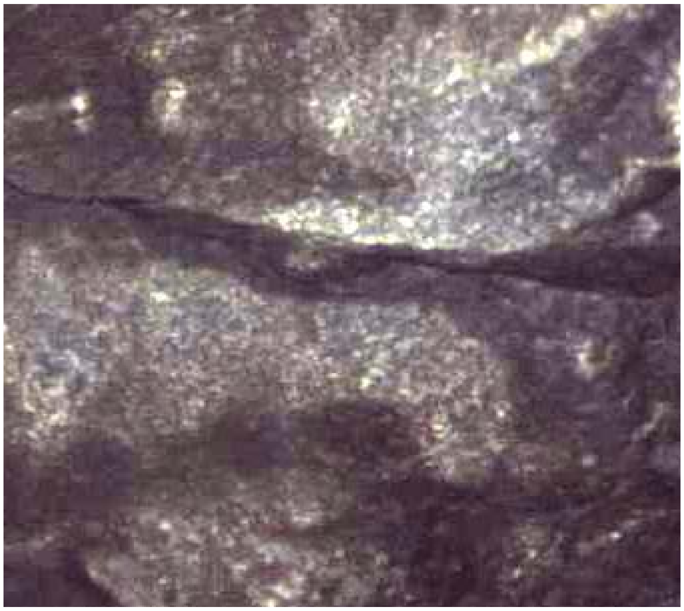


Blue

Ave	98.21985
Median	90
Max	255
Min	0
Sdev	56.69748

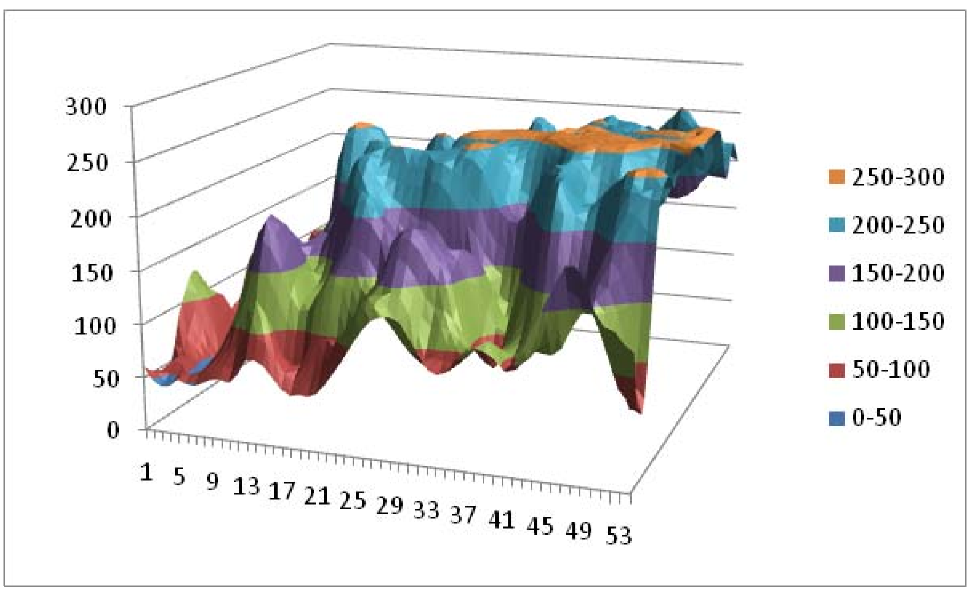


Craig LGBX Waste: 0.21 % Ni, 0.18% Cu



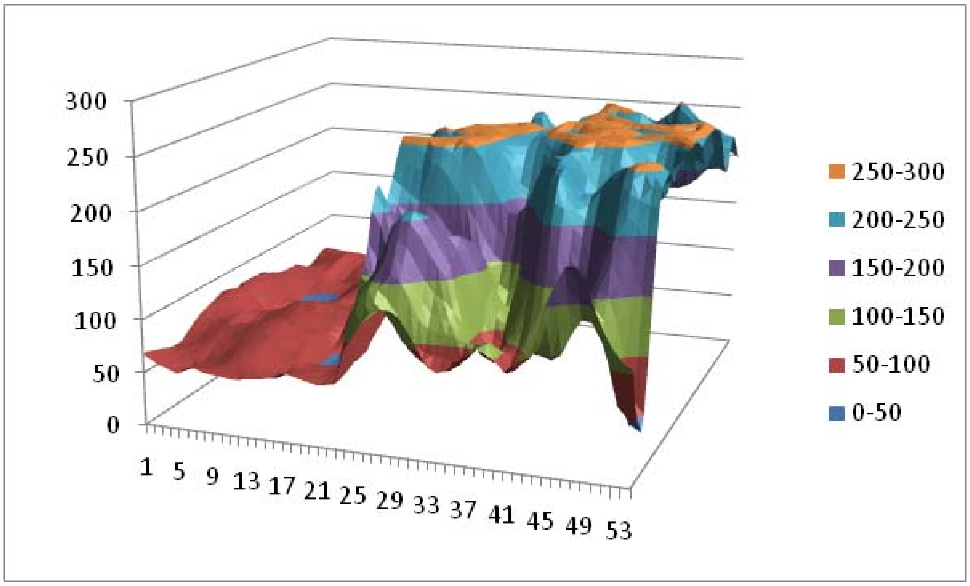
Red

Ave	104.2697
Median	90
Max	255
Min	8
Sdev	56.84553



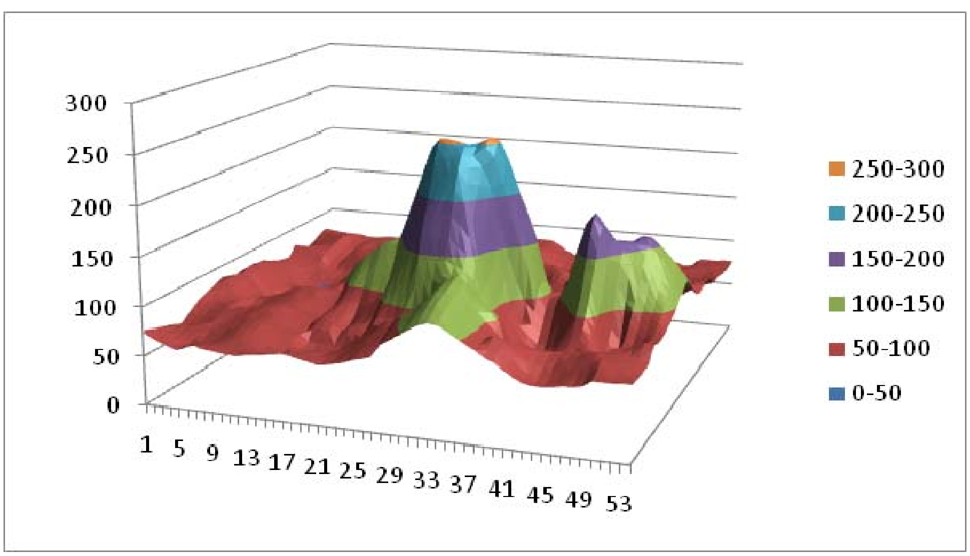
Green

Ave	101.1501
Median	86
Max	255
Min	0
Sdev	58.52855

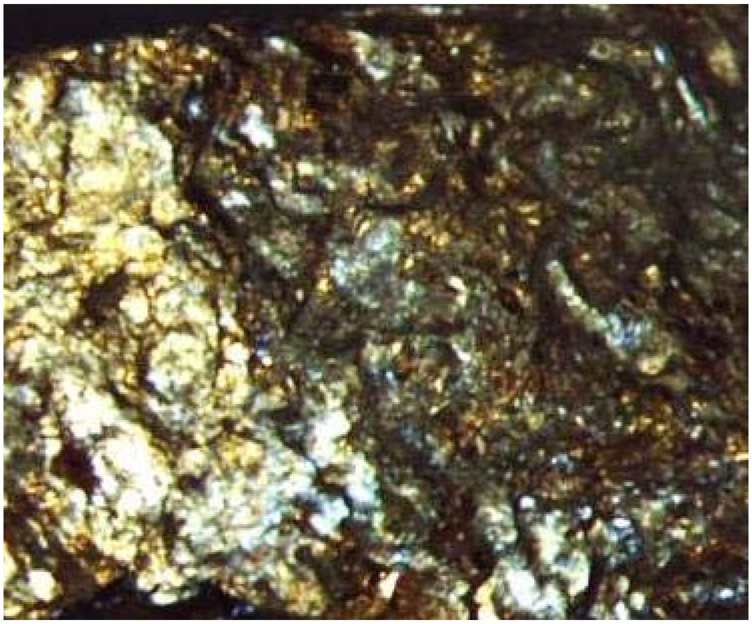


Blue

Ave	87.43752
Median	78
Max	255
Min	0
Sdev	53.38275

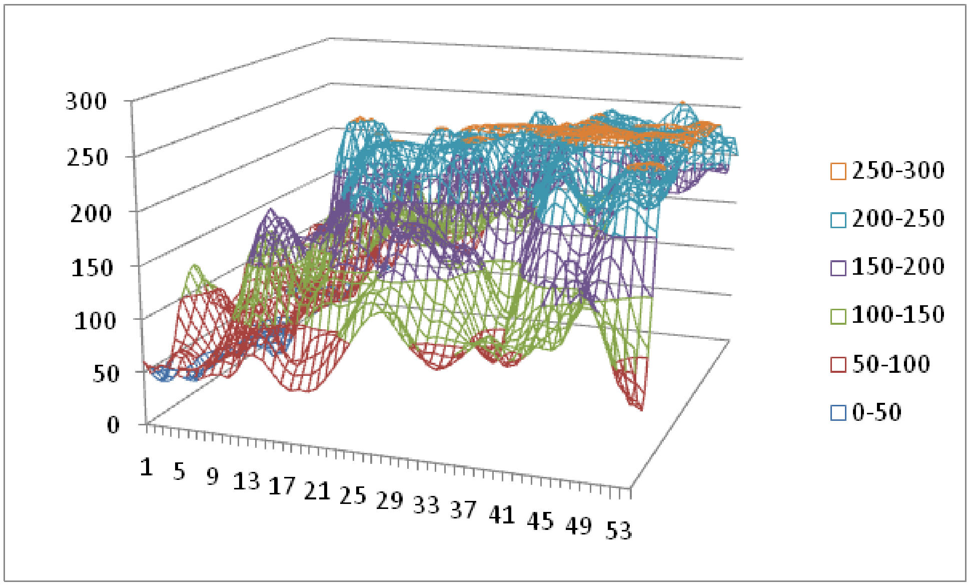


Fraser Cu Ore: 0.84% Ni, 22.01% Cu



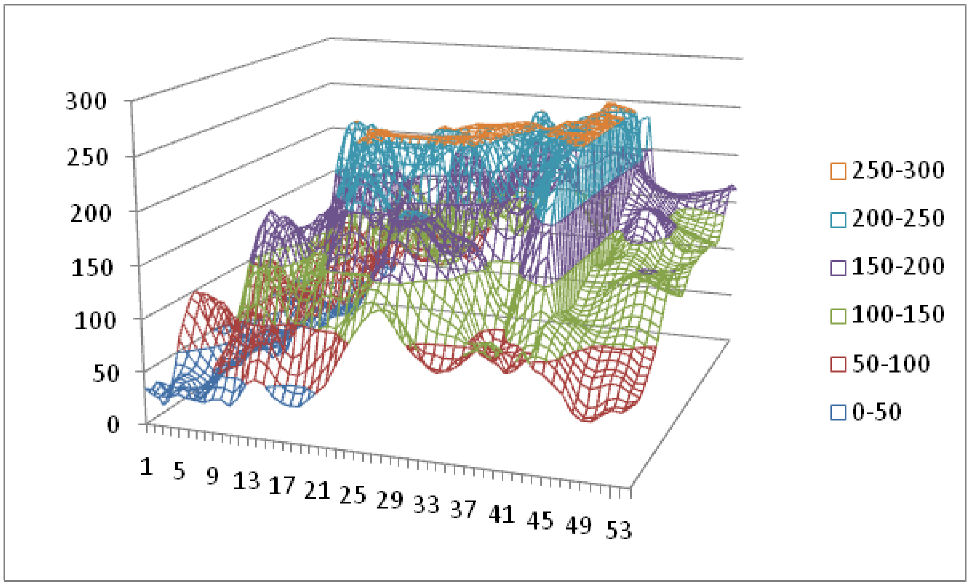
Red

Ave	107.5085
Median	89
Max	255
Min	8
Sdev	64.09539



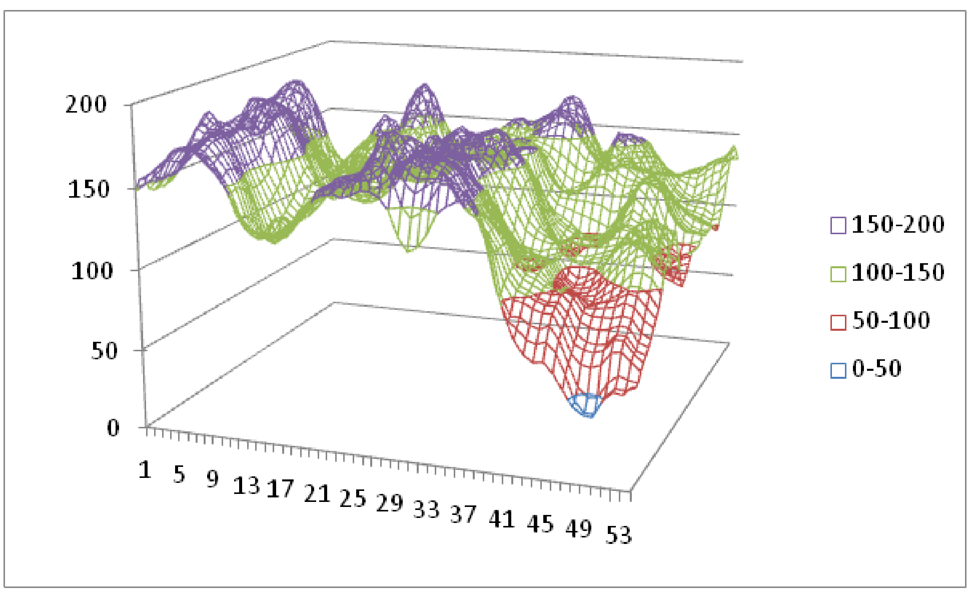
Green

Ave	92.23201
Median	72
Max	255
Min	3
Sdev	61.09994

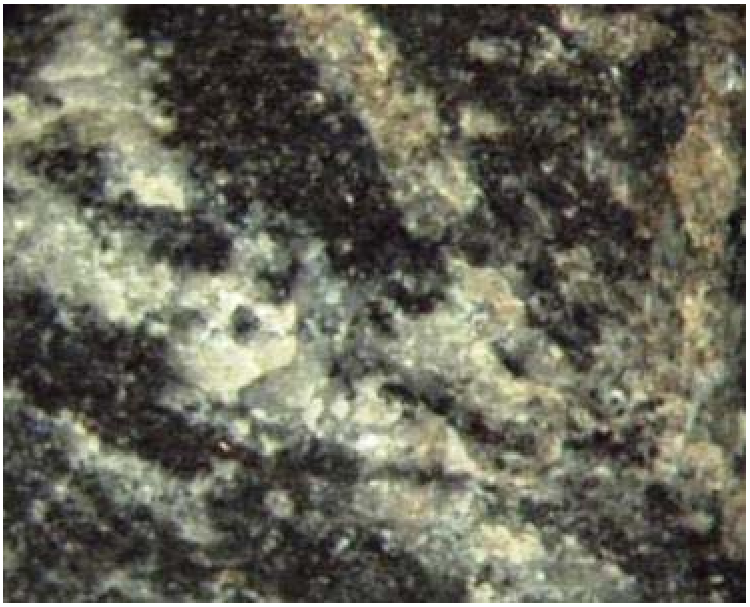


Blue

Ave	75.94285
Median	60
Max	255
Min	0
Sdev	47.78652

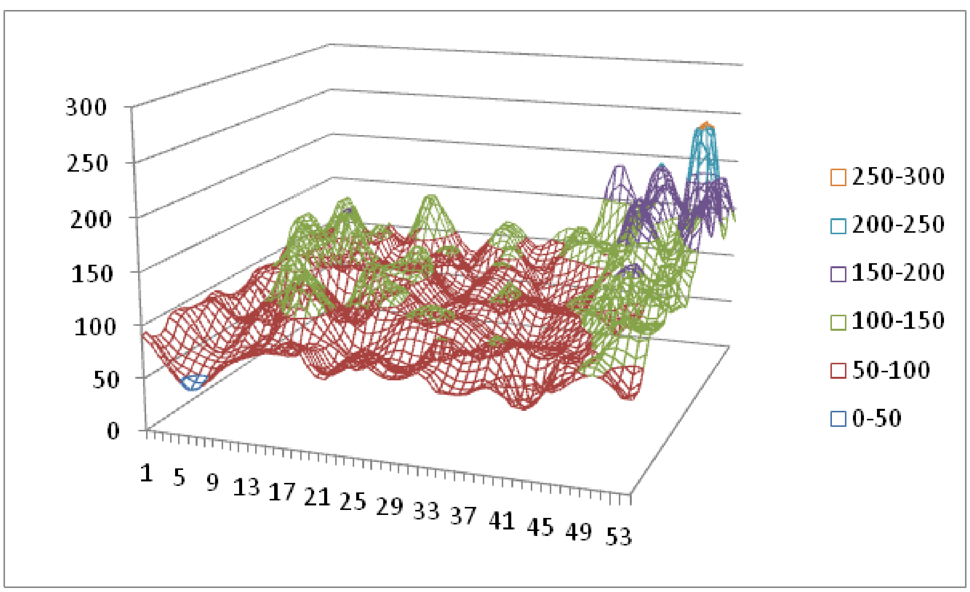


Fraser Cu Waste: 0.03% Ni, 0.4% Cu



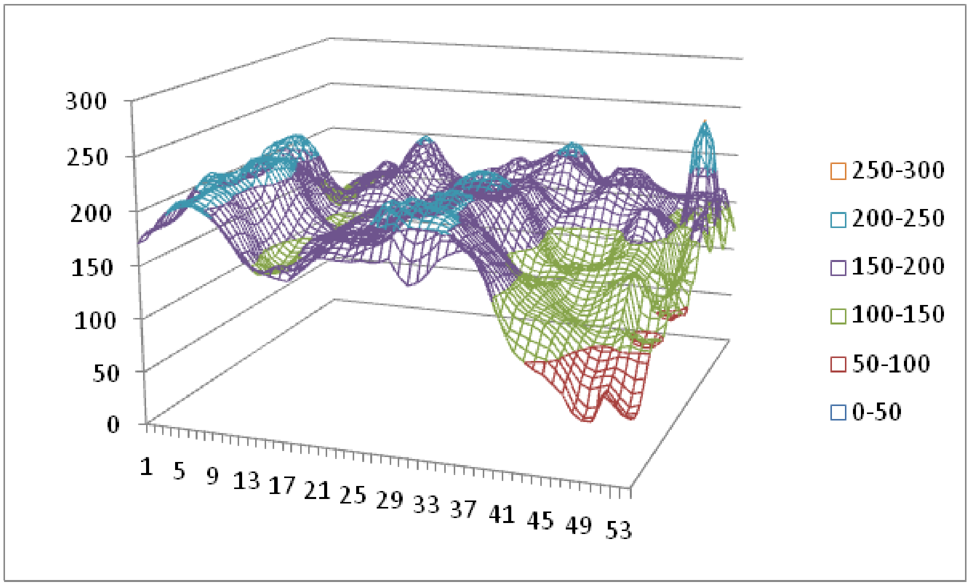
Red

Ave	99.01471
Median	93
Max	255
Min	14
Sdev	37.48492



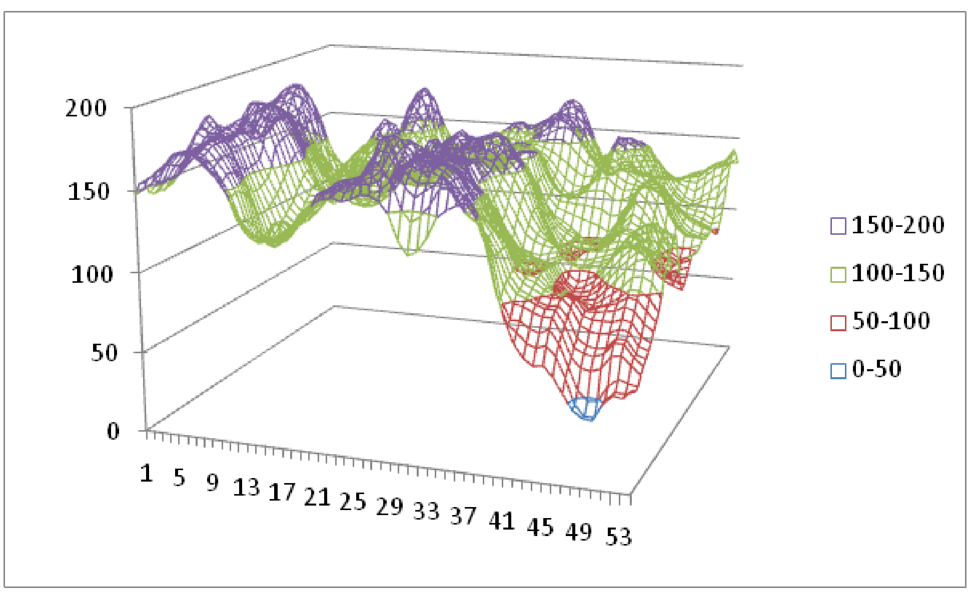
Green

Ave	89.37101
Median	82
Max	255
Min	5
Sdev	42.15757



Blue

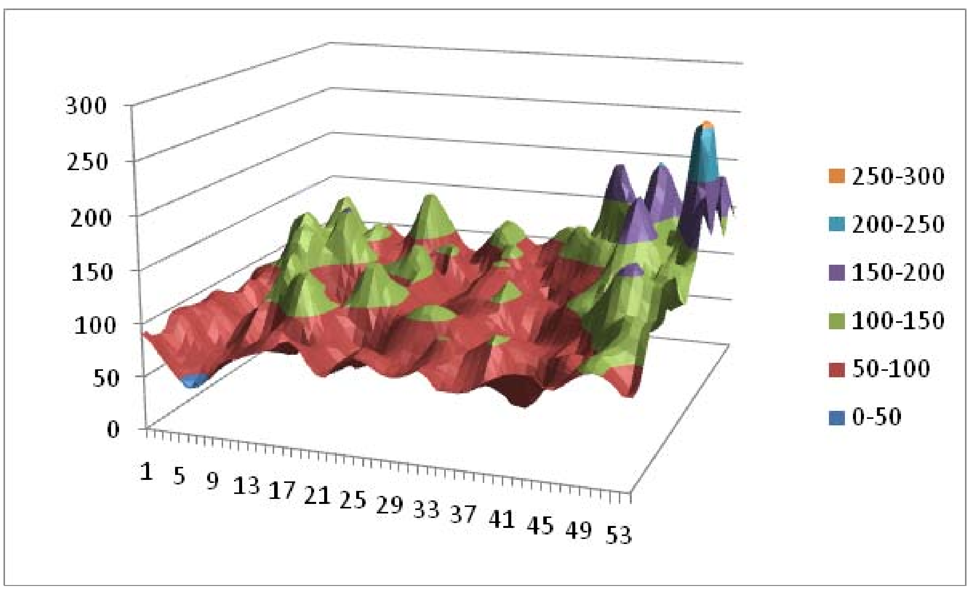
Ave	68.76598
Median	59
Max	255
Min	0
Sdev	37.55012



Fraser Ni Ore: 1.97% Ni, 2.8% Cu

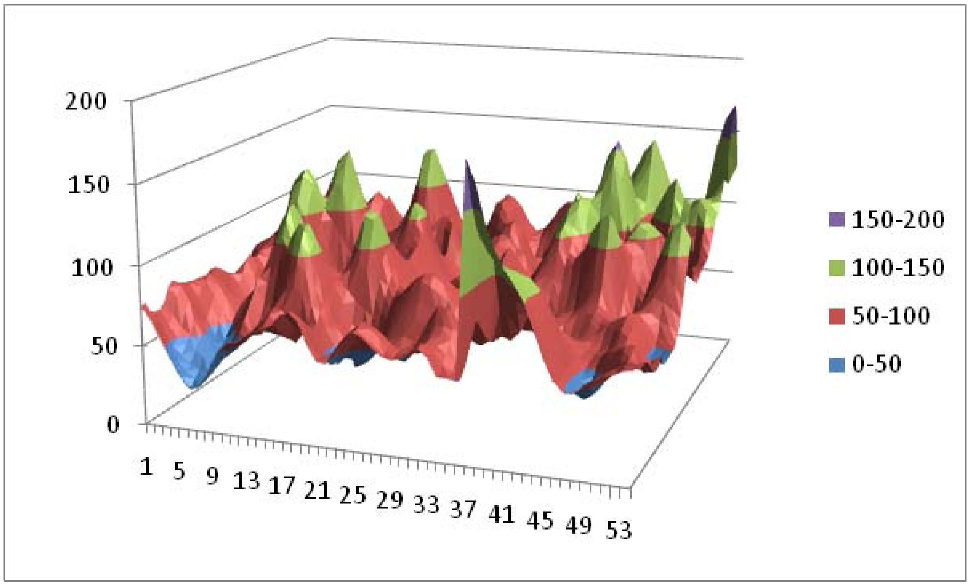
Red

Ave	95.73302
Median	89
Max	255
Min	25
Sdev	36.02689



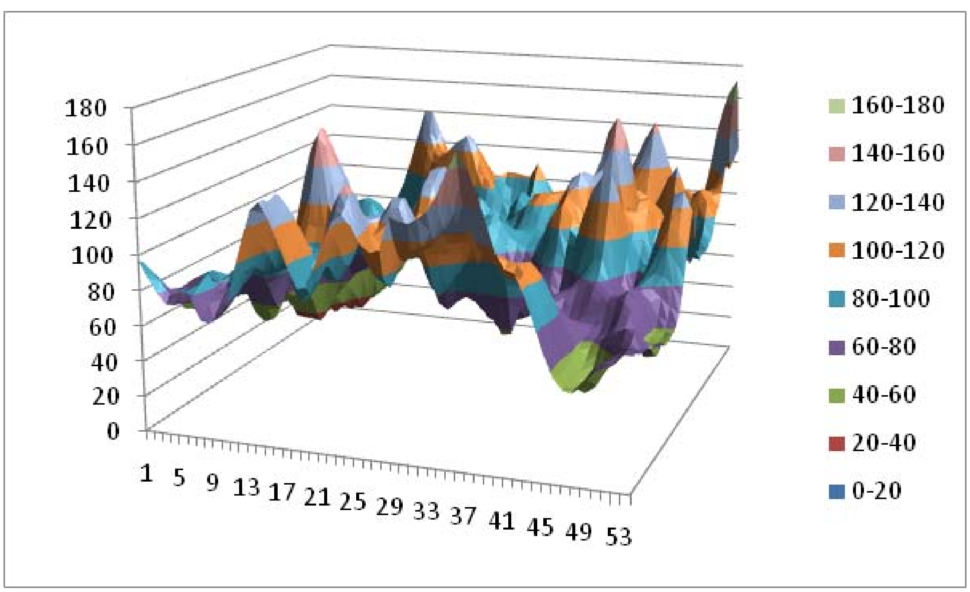
Green

Ave	97.45357
Median	90
Max	255
Min	10
Sdev	40.95302

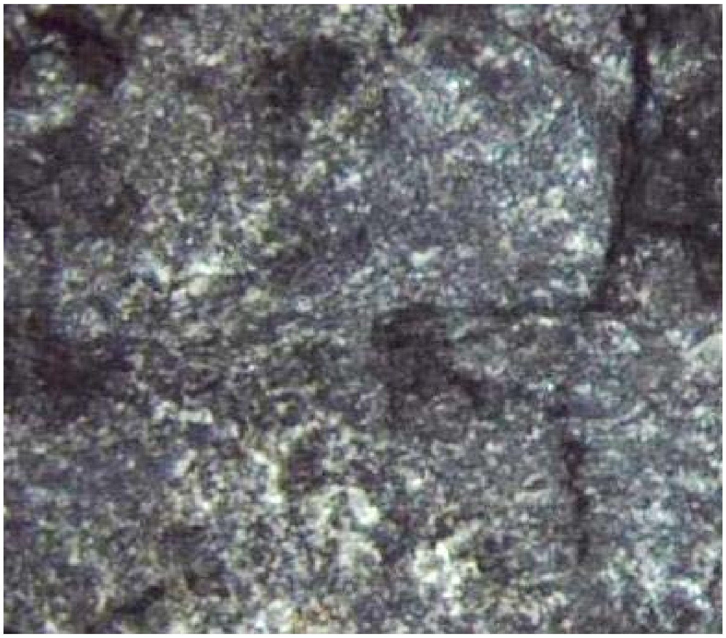


Blue

Ave	105.3811
Median	98
Max	255
Min	19
Sdev	41.55017

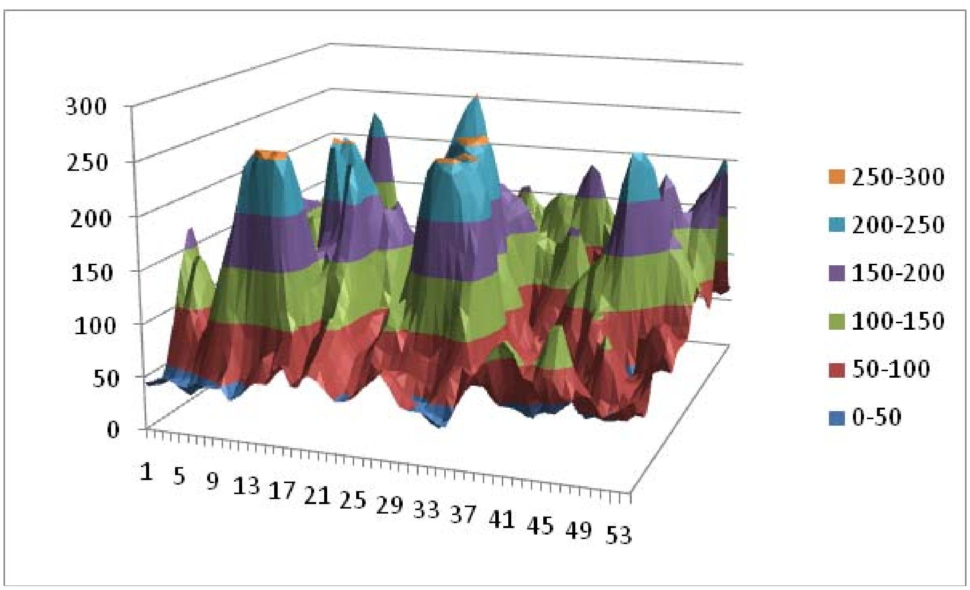


Fraser Ni Waste: 0.20% Ni, 0.12% Cu



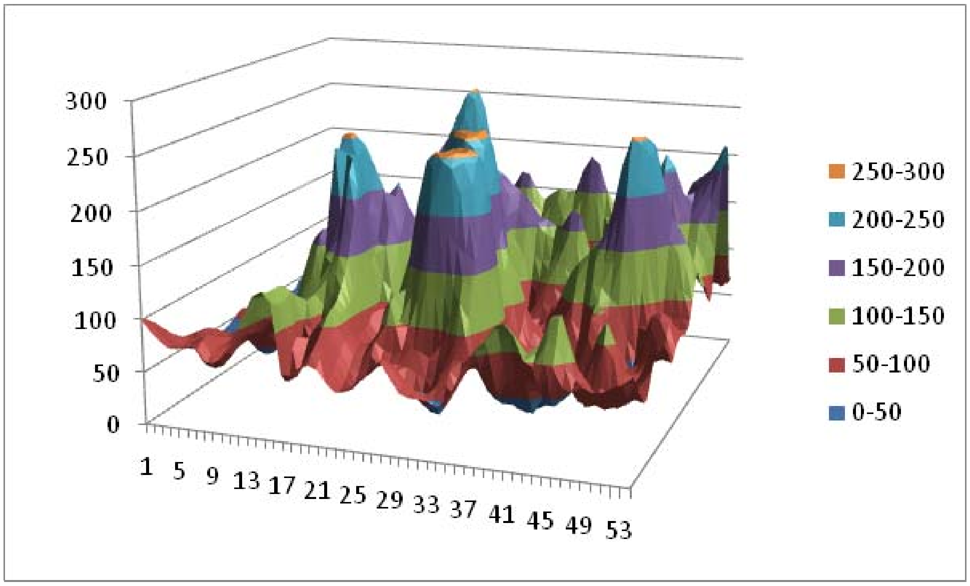
Red

Ave	107.679
Median	96
Max	255
Min	15
Sdev	52.8305



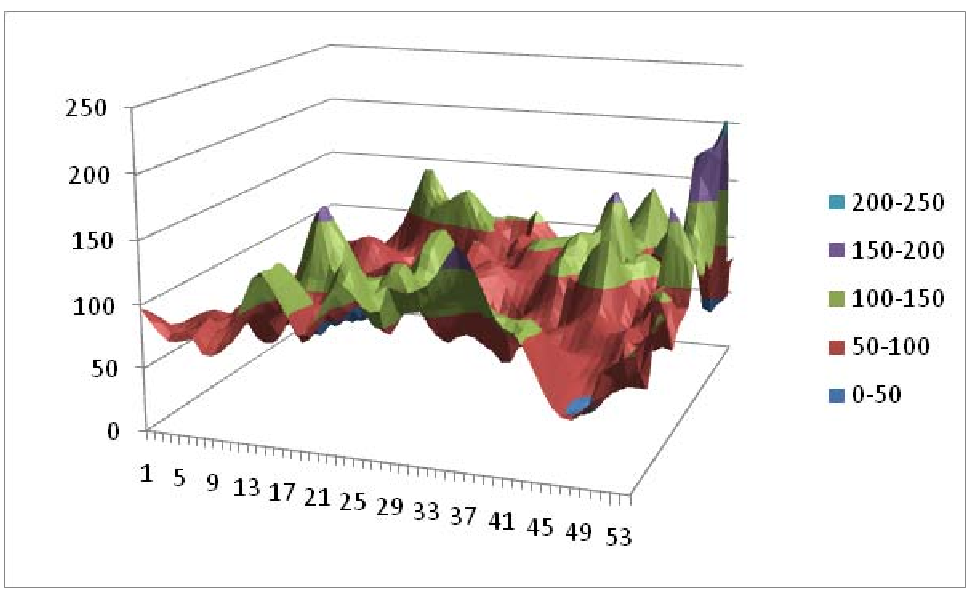
Green

Ave	100.9488
Median	93
Max	255
Min	10
Sdev	47.76216

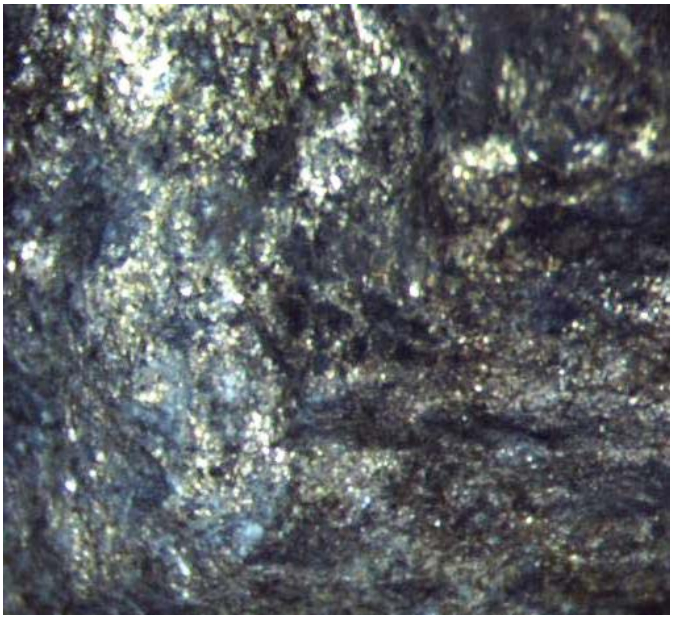


Blue

Ave	105.3192
Median	99
Max	255
Min	18
Sdev	41.1028

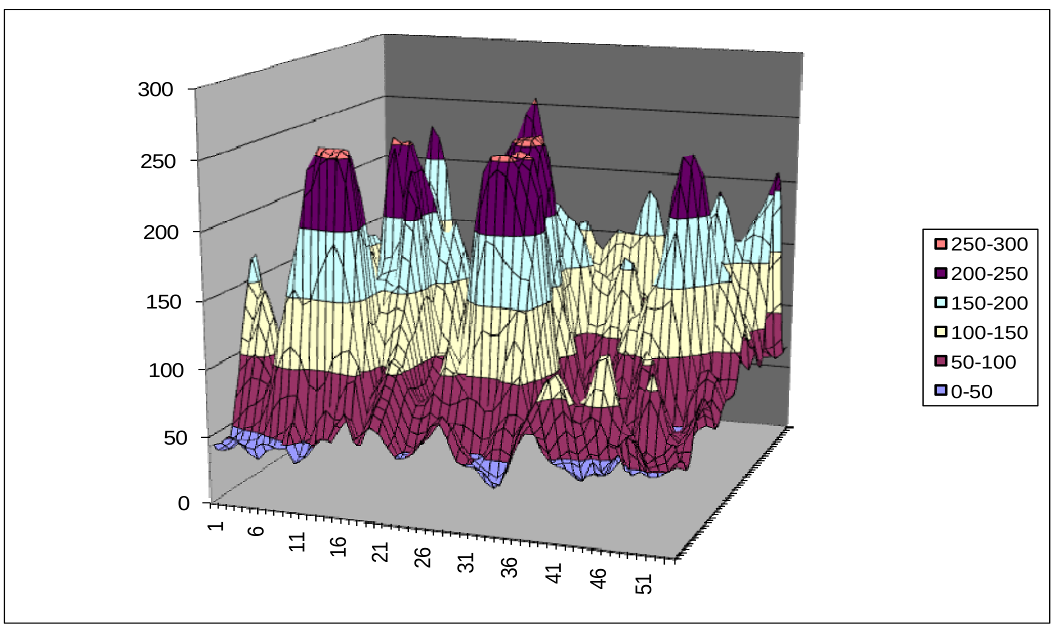


Montcalm H Ore: 2.12% Ni, 0.82% Cu



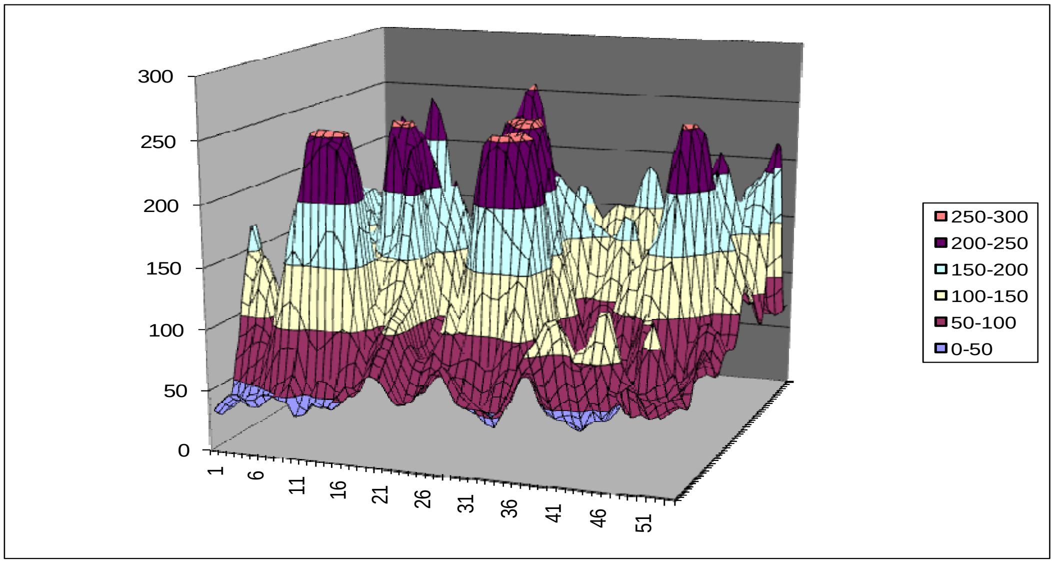
Red

Ave	116.2747
Median	104
Max	255
Min	15
Sdev	56.21671



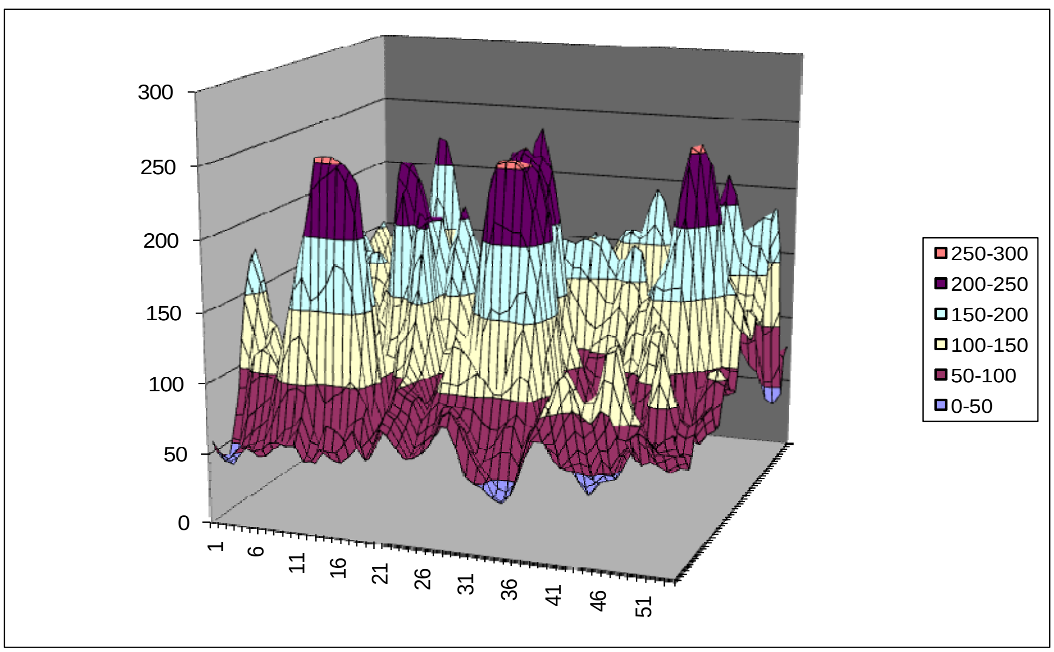
Green

Ave	120.2834
Median	110
Max	255
Min	11
Sdev	54.04509

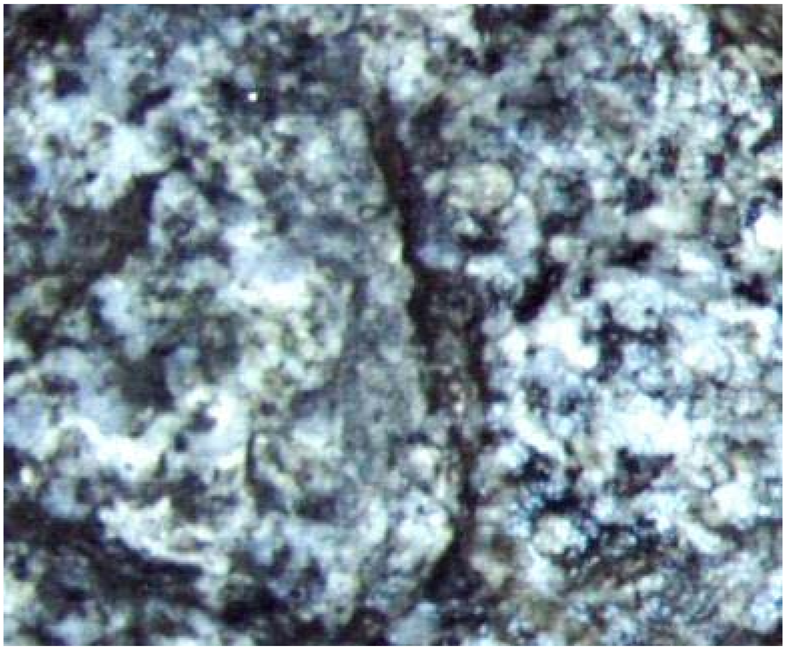


Blue

Ave	117.5521
Median	109
Max	255
Min	14
Sdev	45.42211

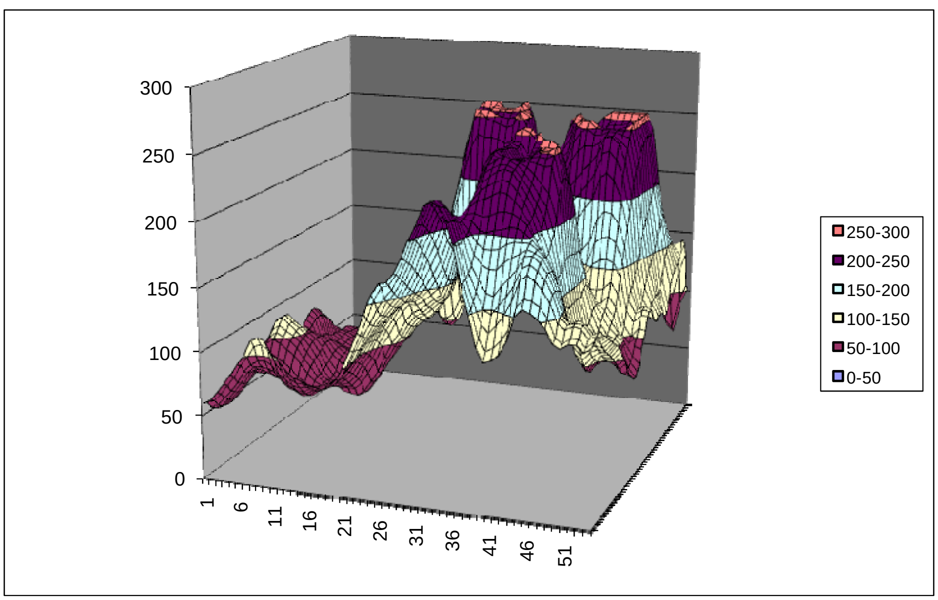


Montcalm H Waste: 0.15% Ni, 0.18% Cu



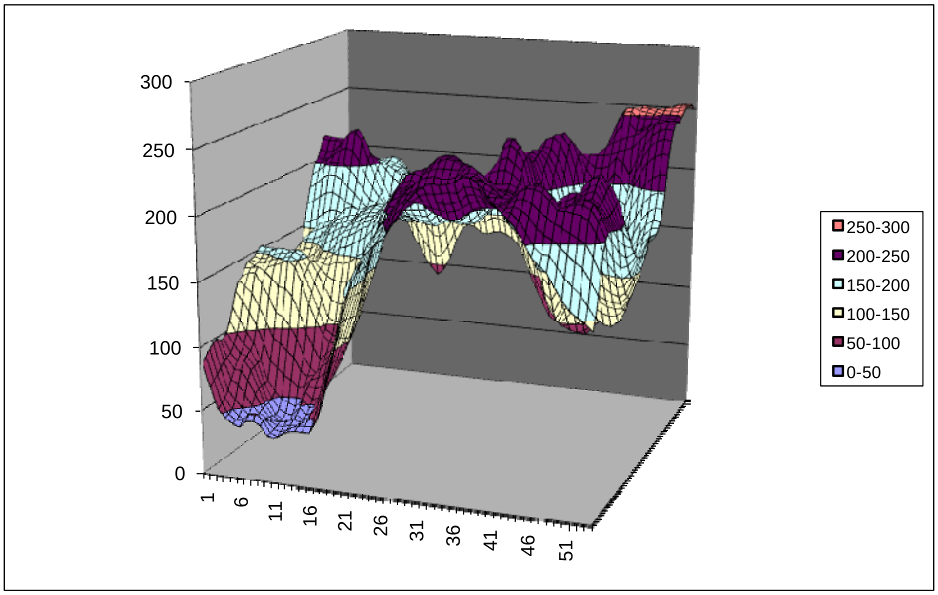
Red

Ave	95.06152
Median	85
Max	255
Min	0
Sdev	48.84085



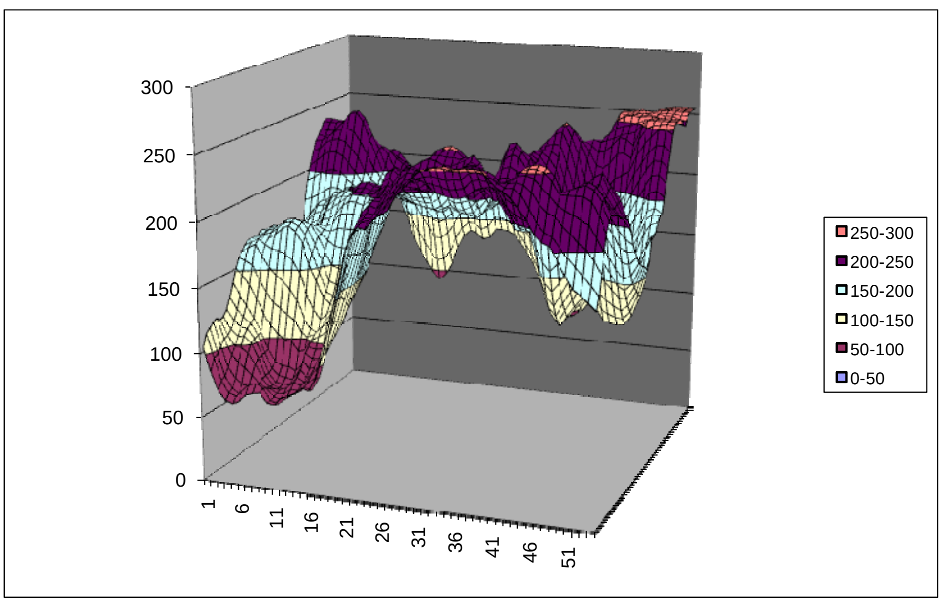
Green

Ave	111.0511
Median	101
Max	255
Min	2
Sdev	52.22056



Blue

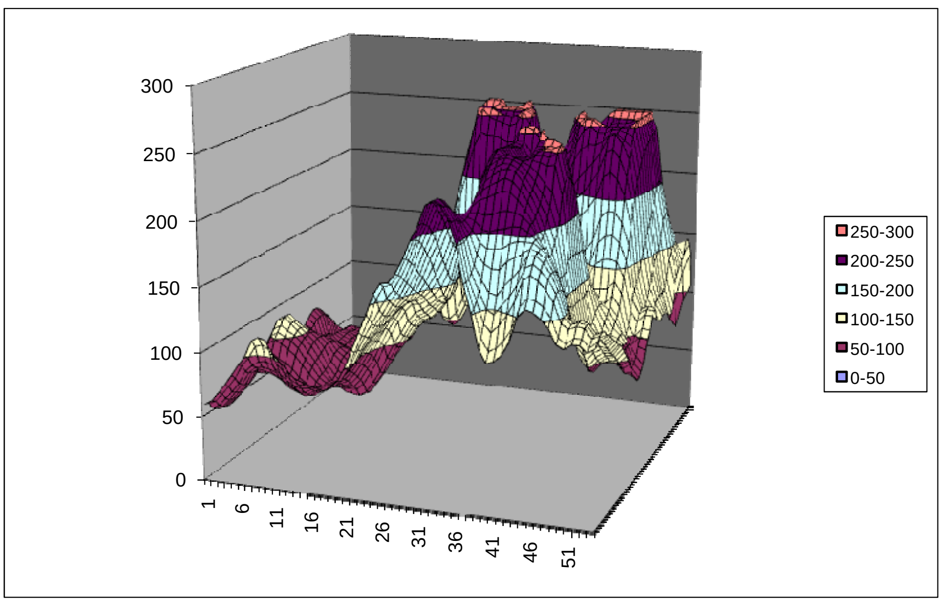
Ave	125.7835
Median	117
Max	255
Min	9
Sdev	49.99223



Montcalm L Ore: 1.4% Ni, 0.56% Cu

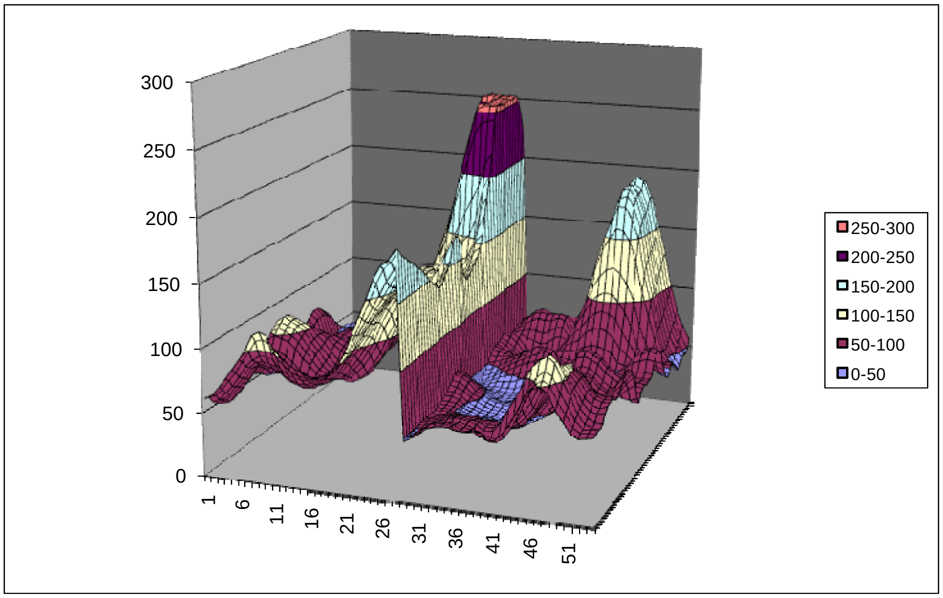
Red

Ave	88.62756
Median	75
Max	255
Min	0
Sdev	51.93579



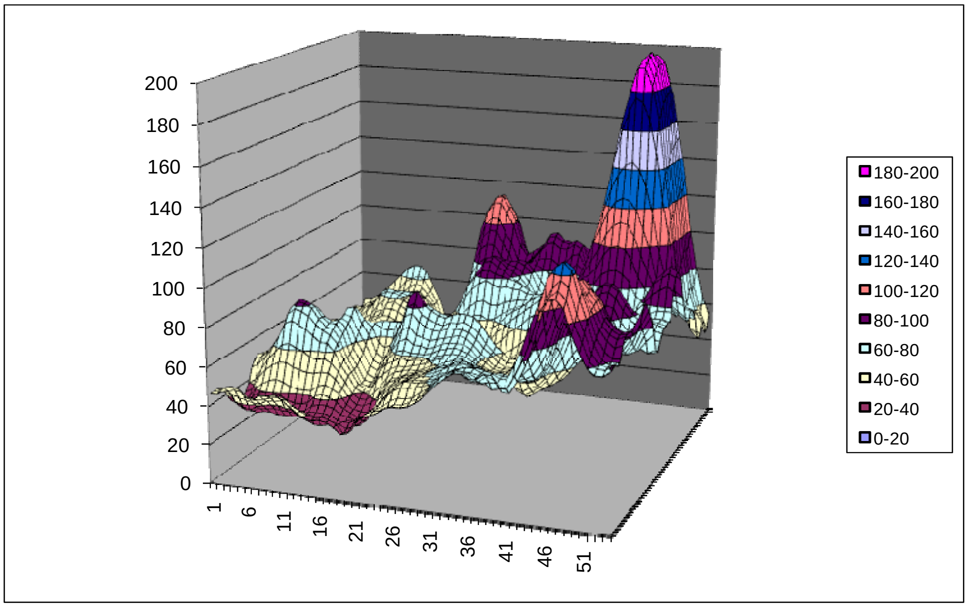
Green

Ave	83.22521
Median	72
Max	255
Min	0
Sdev	47.46631

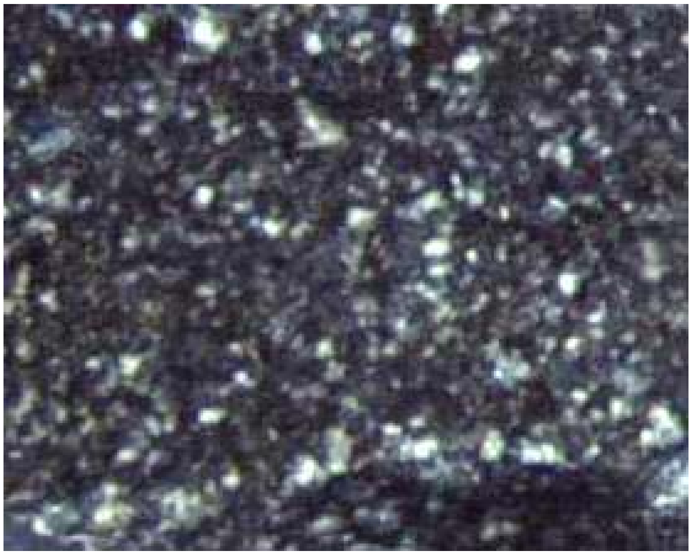


Blue

Ave	76.33774
Median	69
Max	255
Min	0
Sdev	39.3499

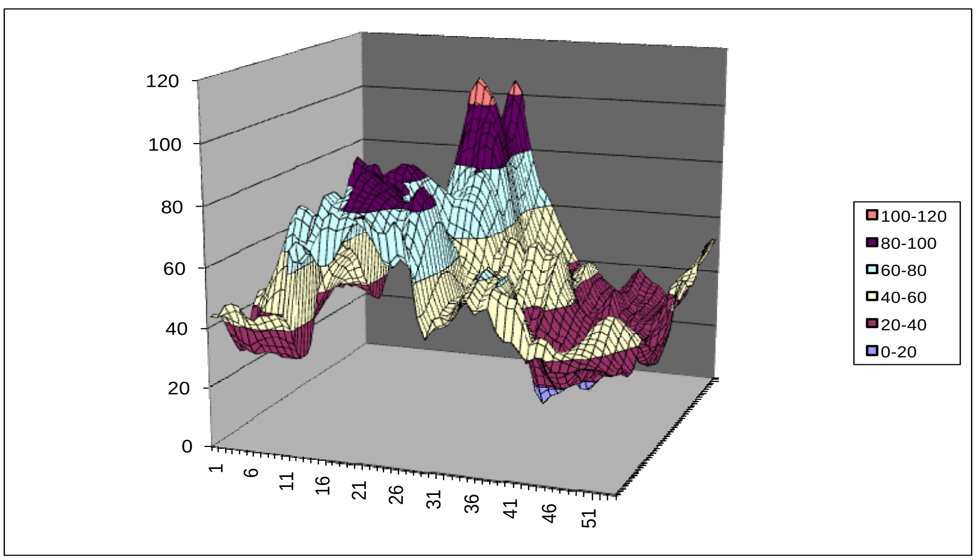


Montcalm L Waste: 0.15% Ni, 0.1% Cu



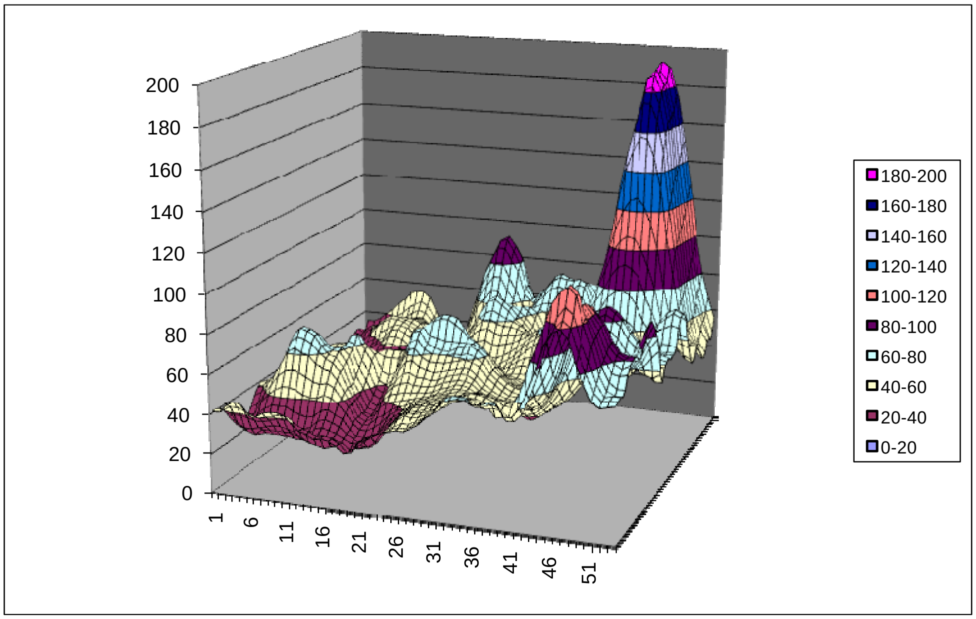
Red

Ave	86.8213
Median	76
Max	255
Min	10
Sdev	46.23356



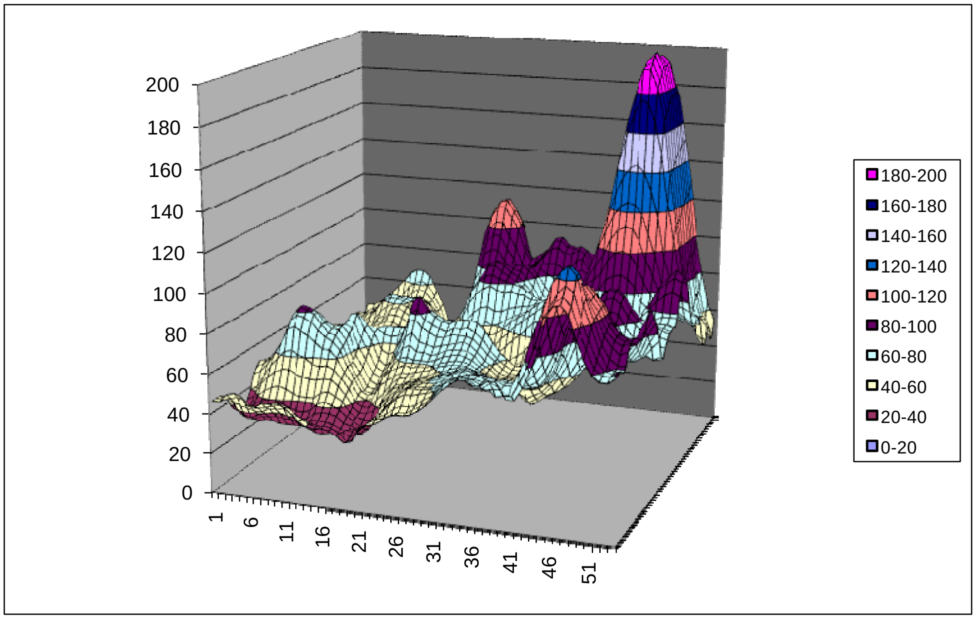
Green

Ave	81.76139
Median	71
Max	255
Min	0
Sdev	49.24515



Blue

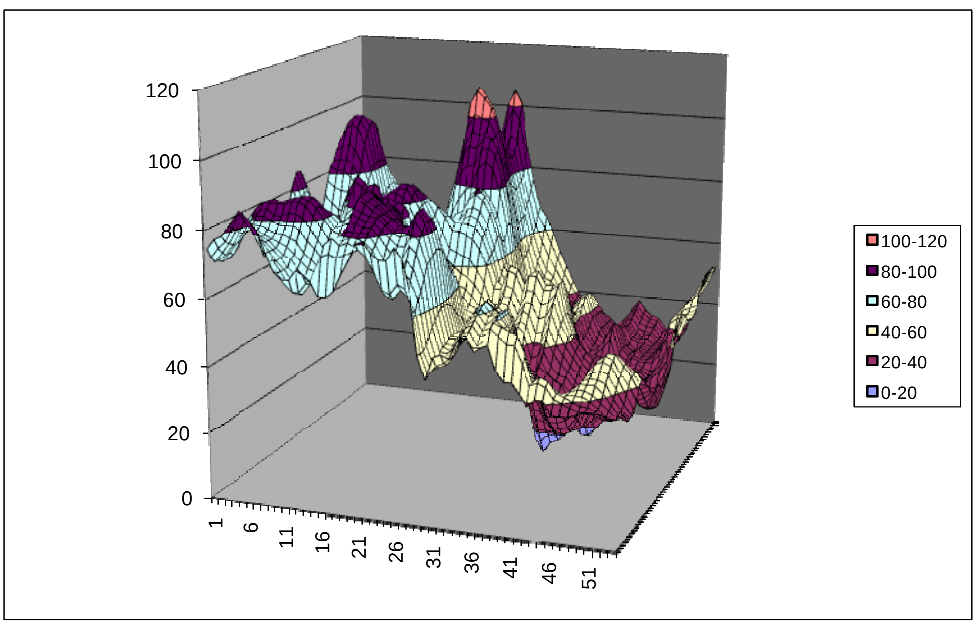
Ave	75.51526
Median	66
Max	255
Min	0
Sdev	43.62071



T-L 15 Ore: 1.83% Ni, 10.79% Cu

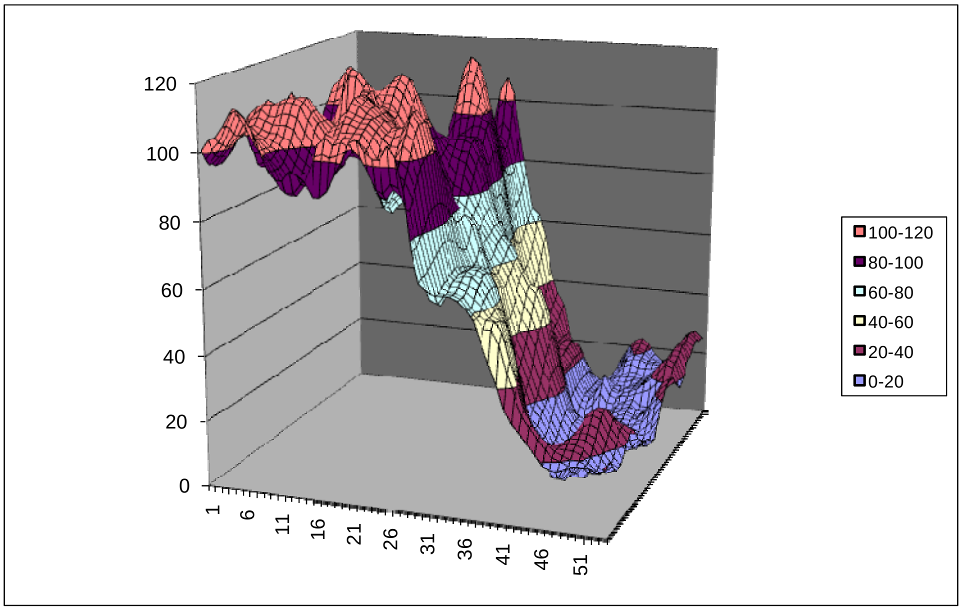
Red

Ave	92.33209
Median	81
Max	255
Min	10
Sdev	48.53768



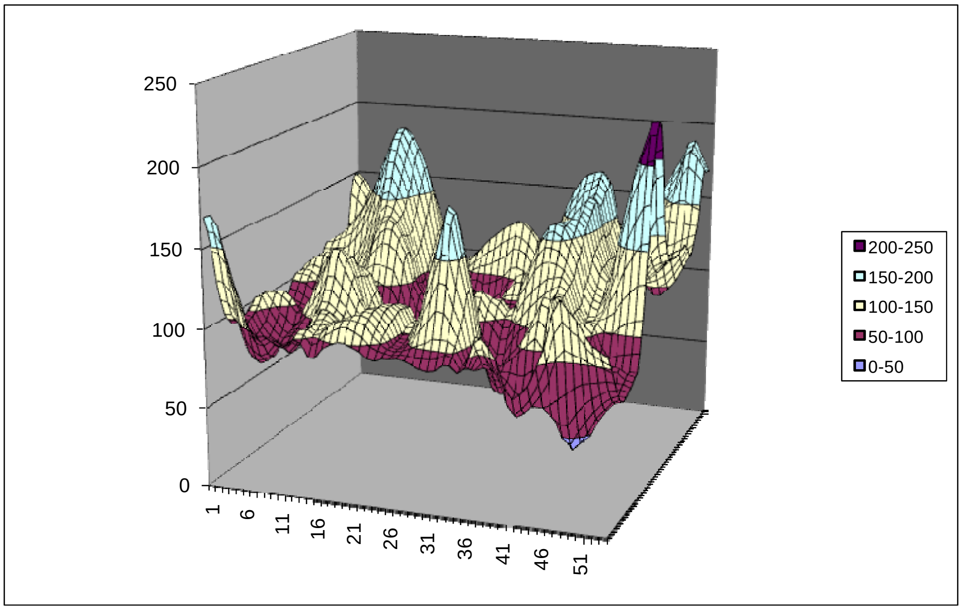
Green

Ave	104.6583
Median	96
Max	255
Min	0
Sdev	51.71257

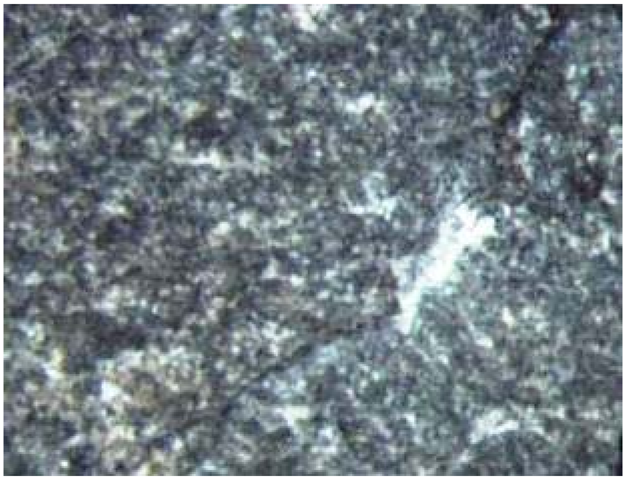


Blue

Ave	108.9964
Median	104
Max	255
Min	2
Sdev	46.91959

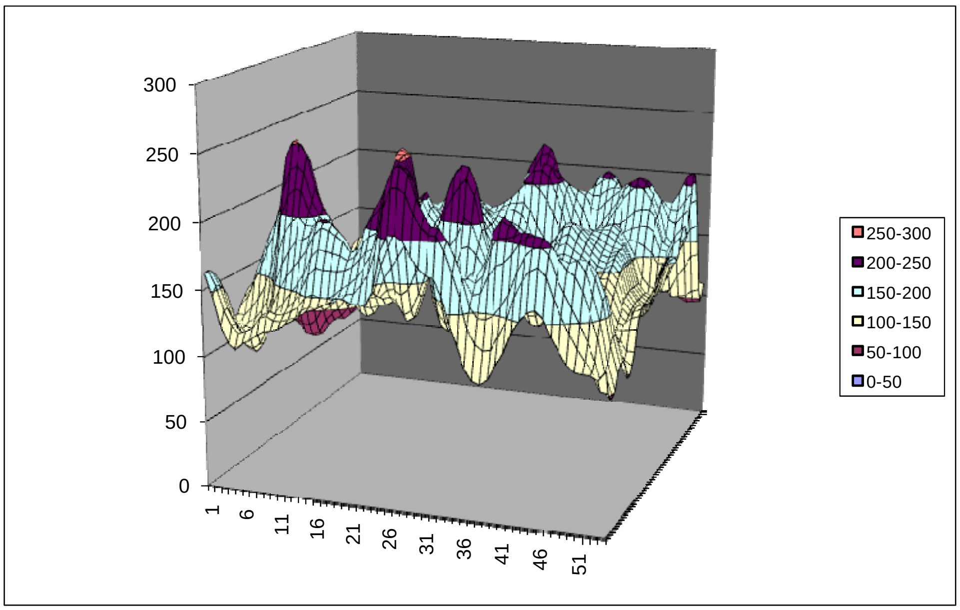


T-L 15 Waste: 0.076% Ni, 0.4% Cu



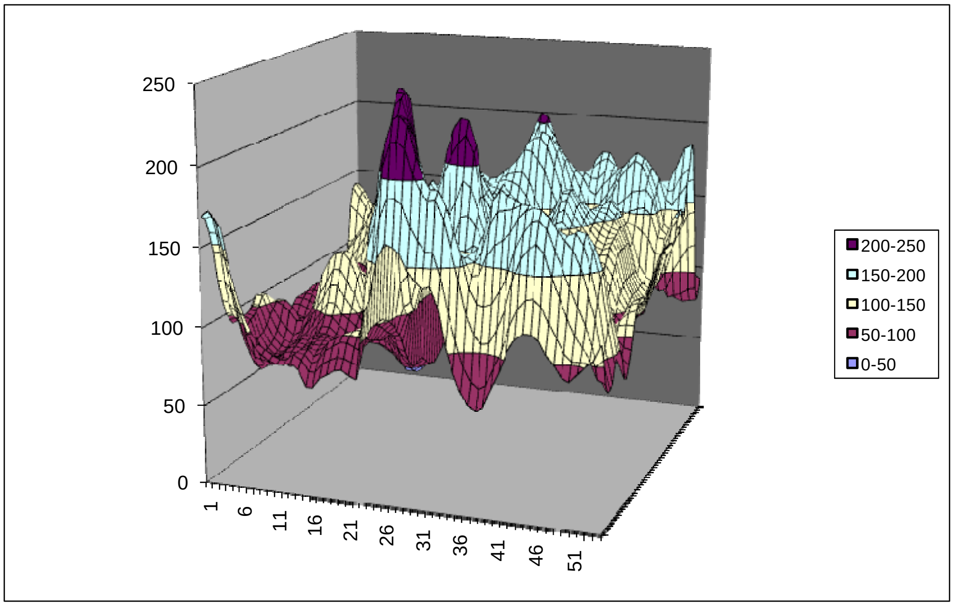
Red

Ave	129.7359
Median	122
Max	255
Min	12
Sdev	48.12655



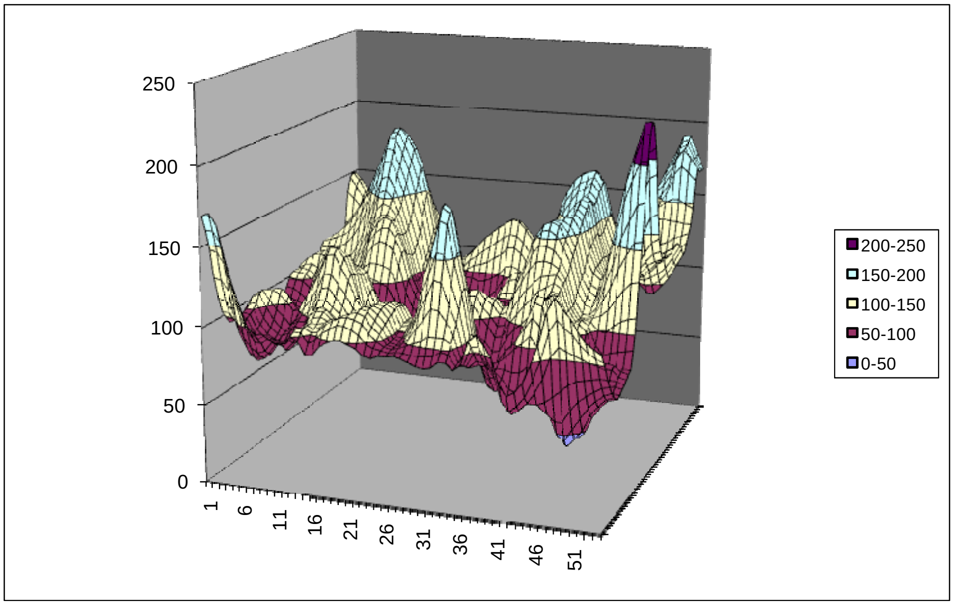
Green

Ave	121.5984
Median	112
Max	255
Min	7
Sdev	50.7195

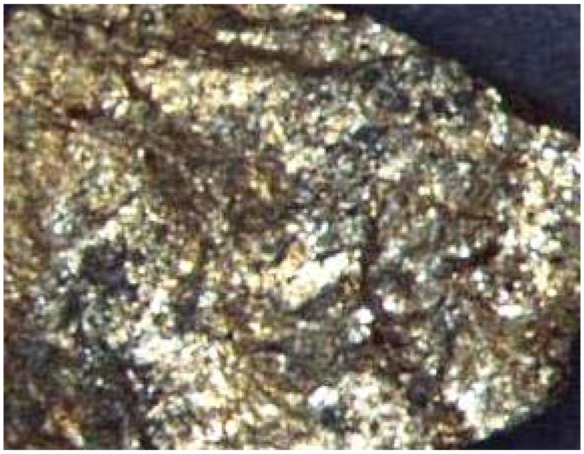


Blue

Ave	104.0819
Median	97
Max	255
Min	0
Sdev	47.11492

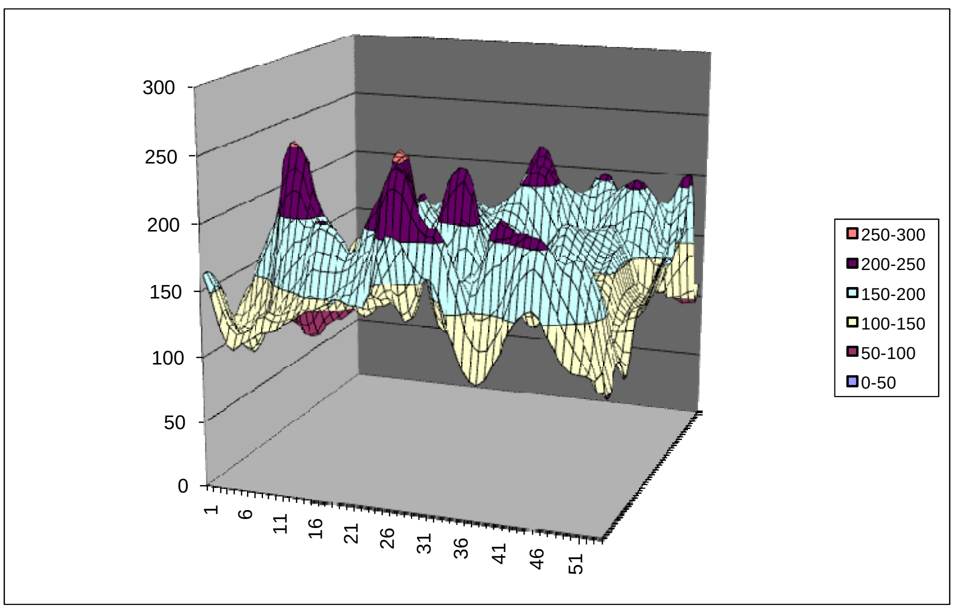


TL-80 Ore: 1.70% Ni, 1.11% Cu



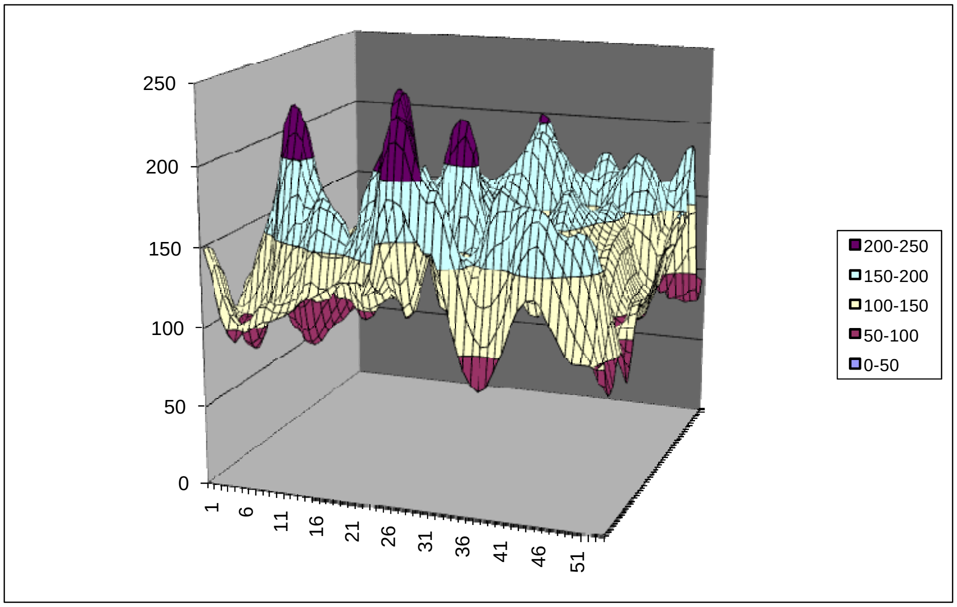
Red

Ave	131.2821
Median	125
Max	255
Min	11
Sdev	47.1173



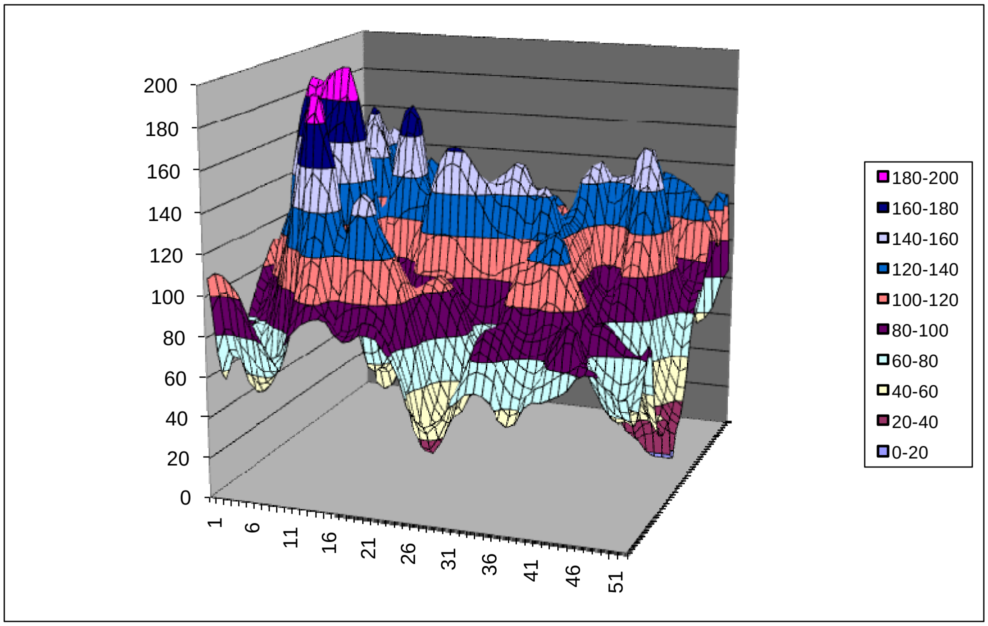
Green

Ave	117.0428
Median	110
Max	255
Min	1
Sdev	48.35397

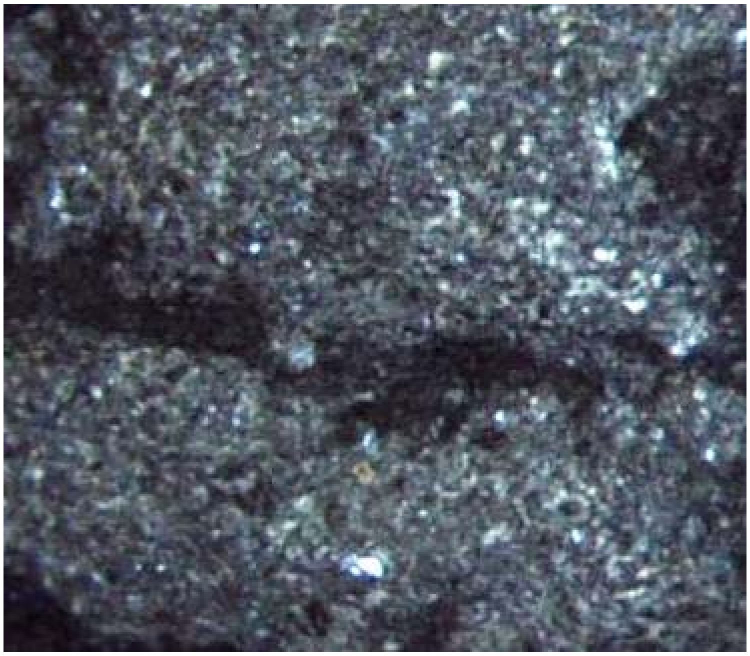


Blue

Ave	90.40864
Median	83
Max	250
Min	0
Sdev	43.32921

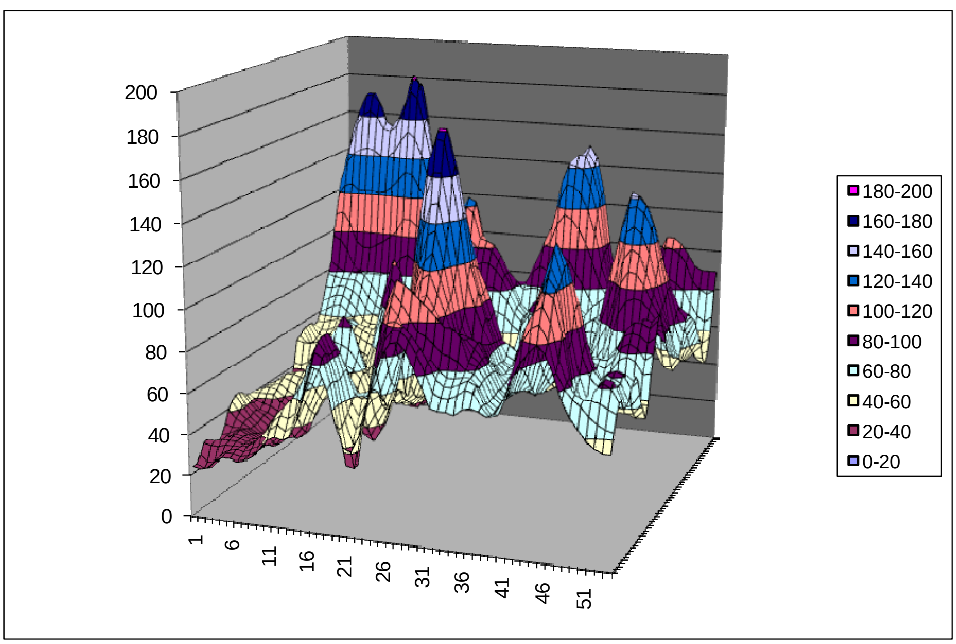


T-L 80 Waste: 0.114% Ni, 0.15% Cu



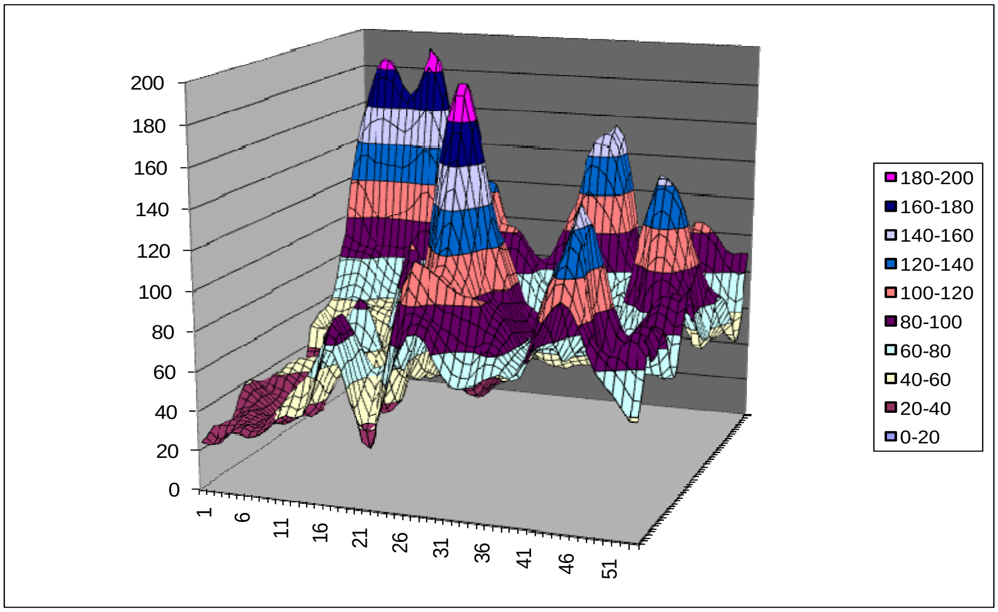
Red

Ave	74.97849
Median	70
Max	255
Min	0
Sdev	34.40829



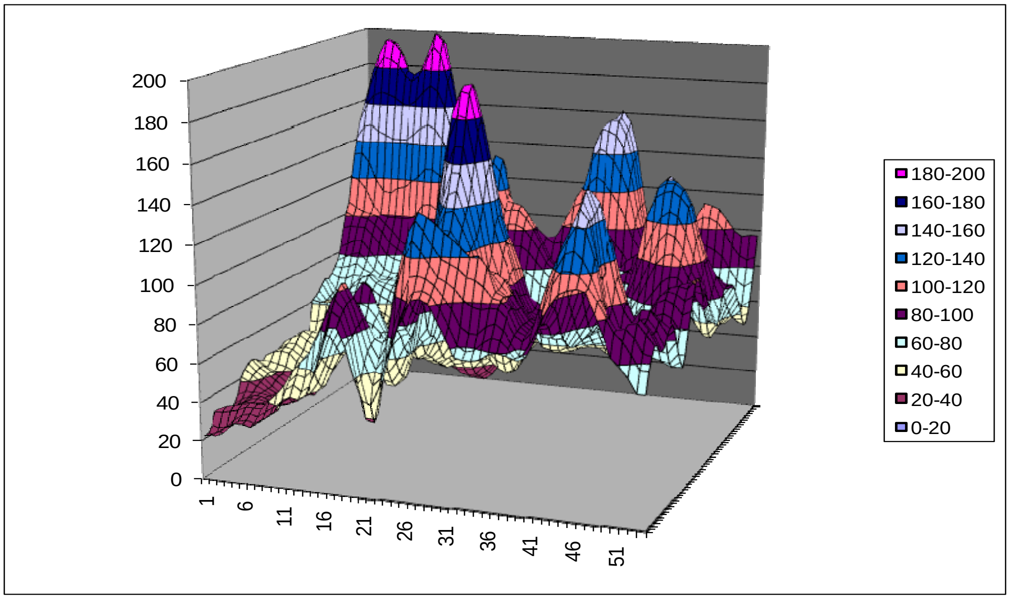
Green

Ave	82.5371
Median	75
Max	255
Min	6
Sdev	38.09737

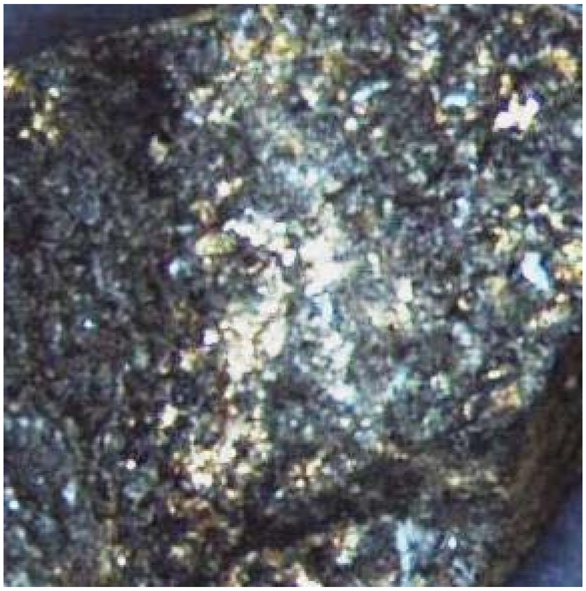


Blue

Ave	82.39788
Median	81
Max	255
Min	12
Sdev	32.61907

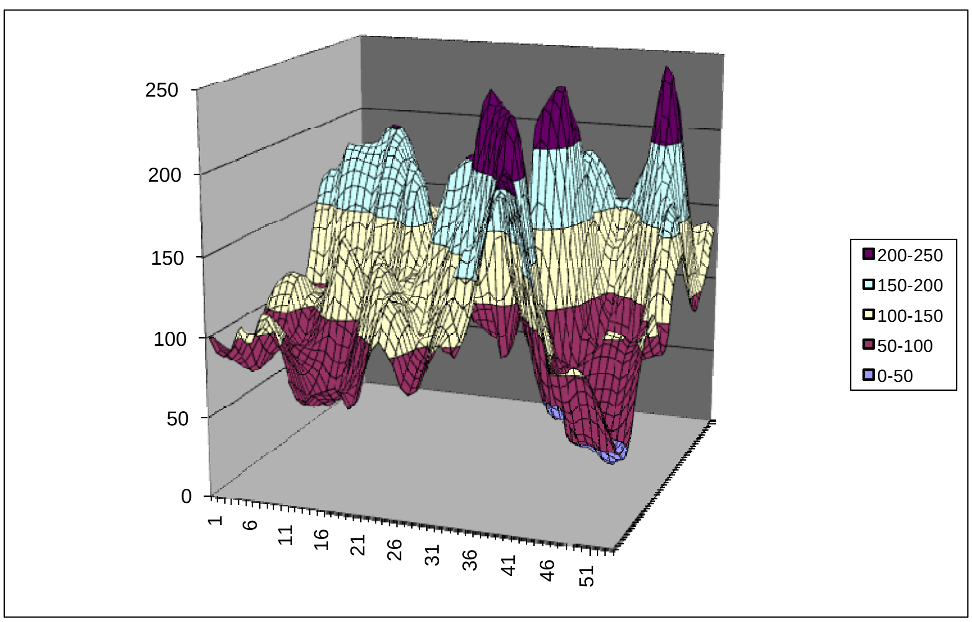


T-L 670 Ore: 0.82% Ni, 0.45% Cu



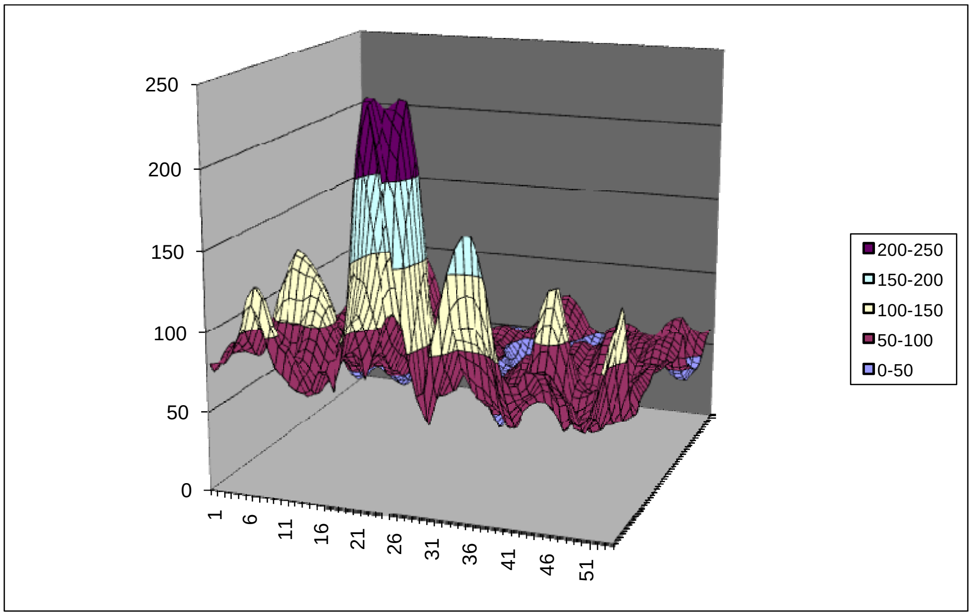
Red

Ave	119.0898
Median	115
Max	255
Min	8
Sdev	46.10237



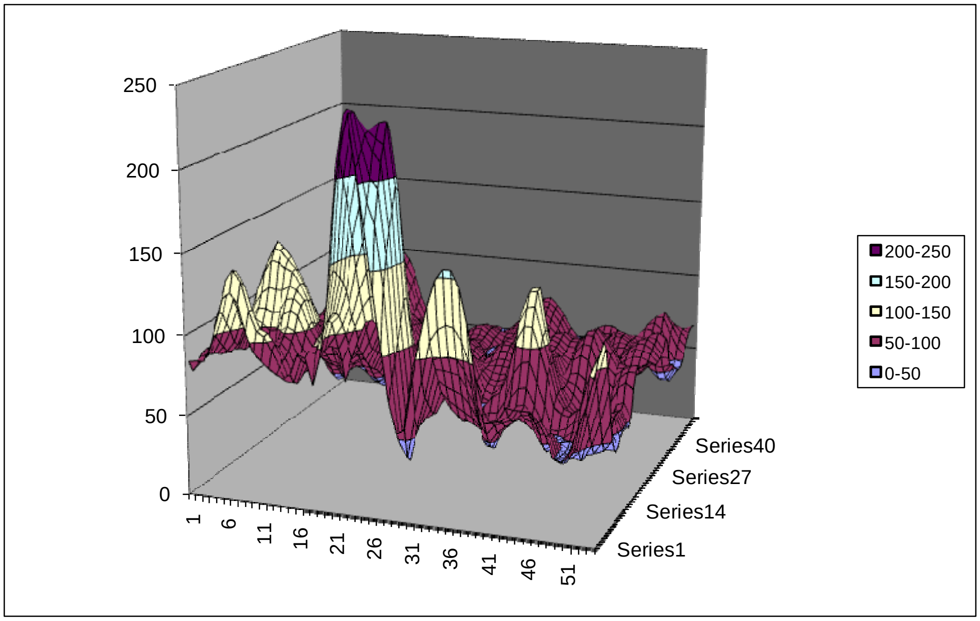
Green

Ave	112.0511
Median	107
Max	255
Min	0
Sdev	49.65724

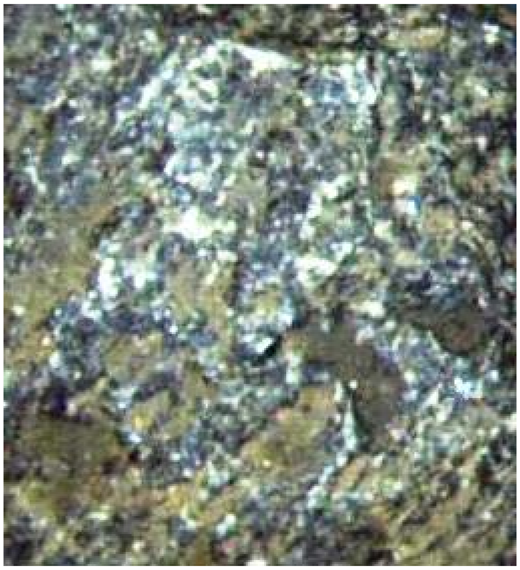


Blue

Ave	92.27257
Median	85
Max	255
Min	0
Sdev	37.86008

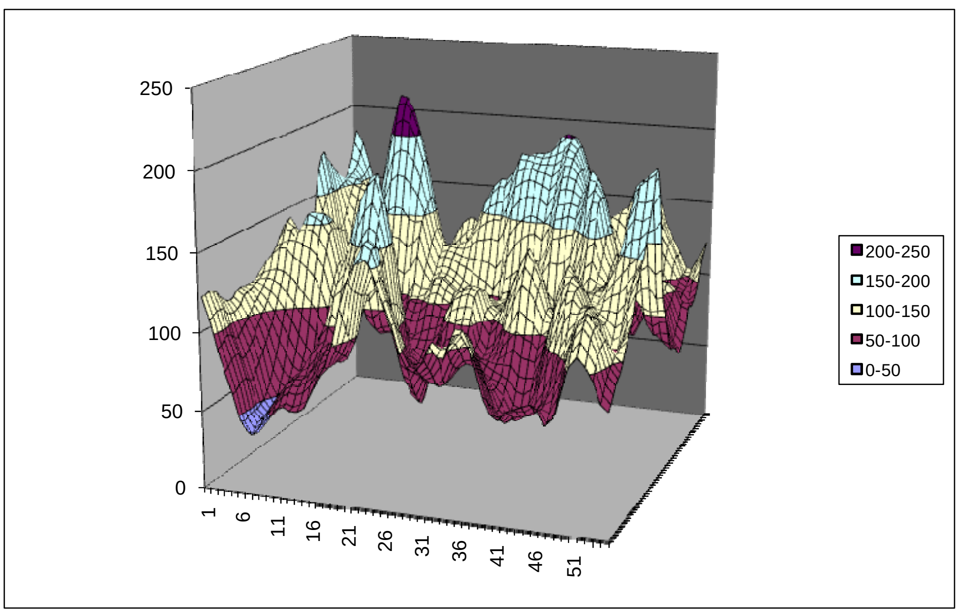


T-L670 Waste: 0.16% Ni, 0.15% Cu



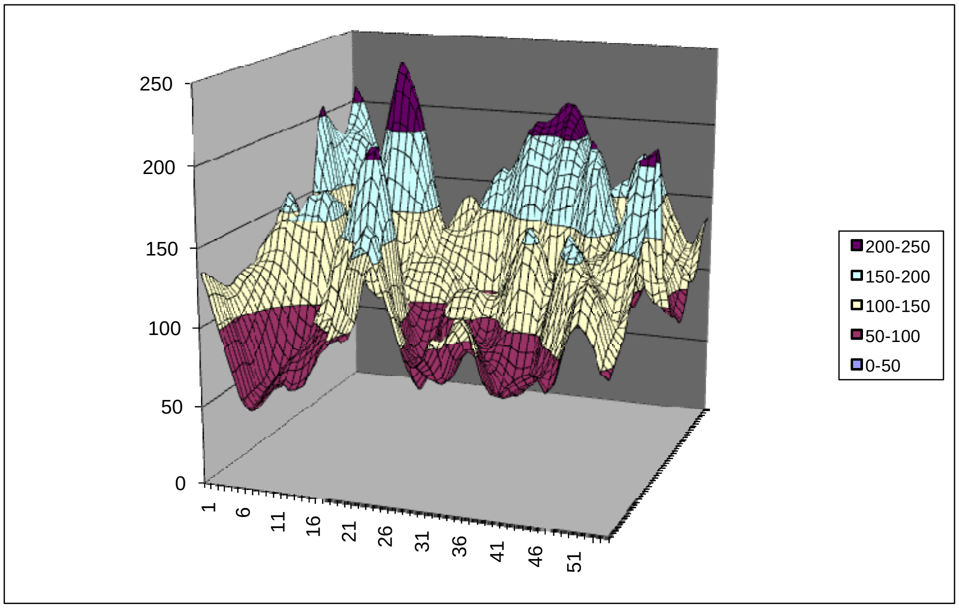
Red

Ave	130.3955
Median	126
Max	255
Min	17
Sdev	44.64572



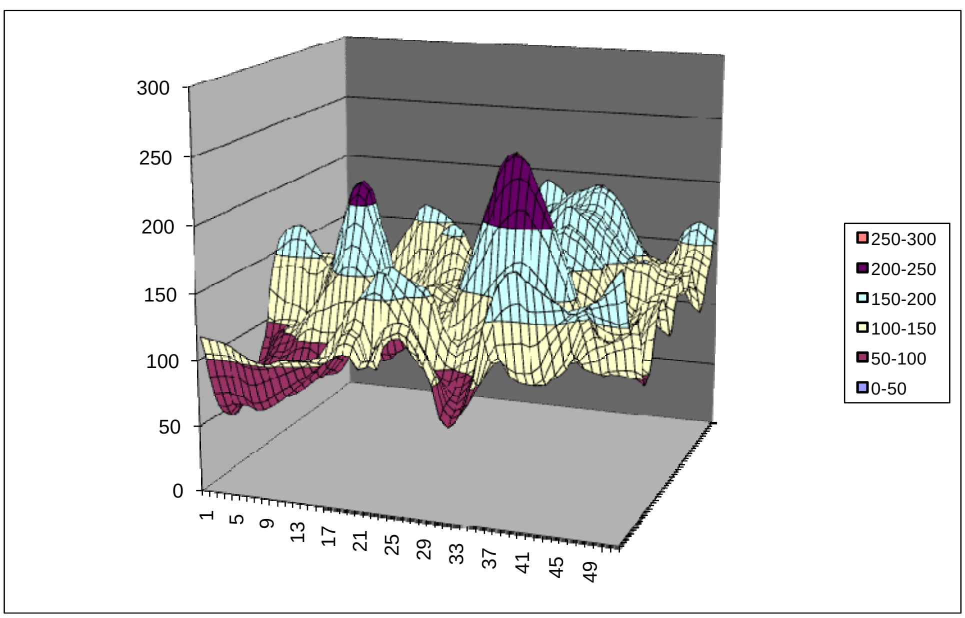
Green

Ave	138.0024
Median	132
Max	255
Min	18
Sdev	47.54835



Blue

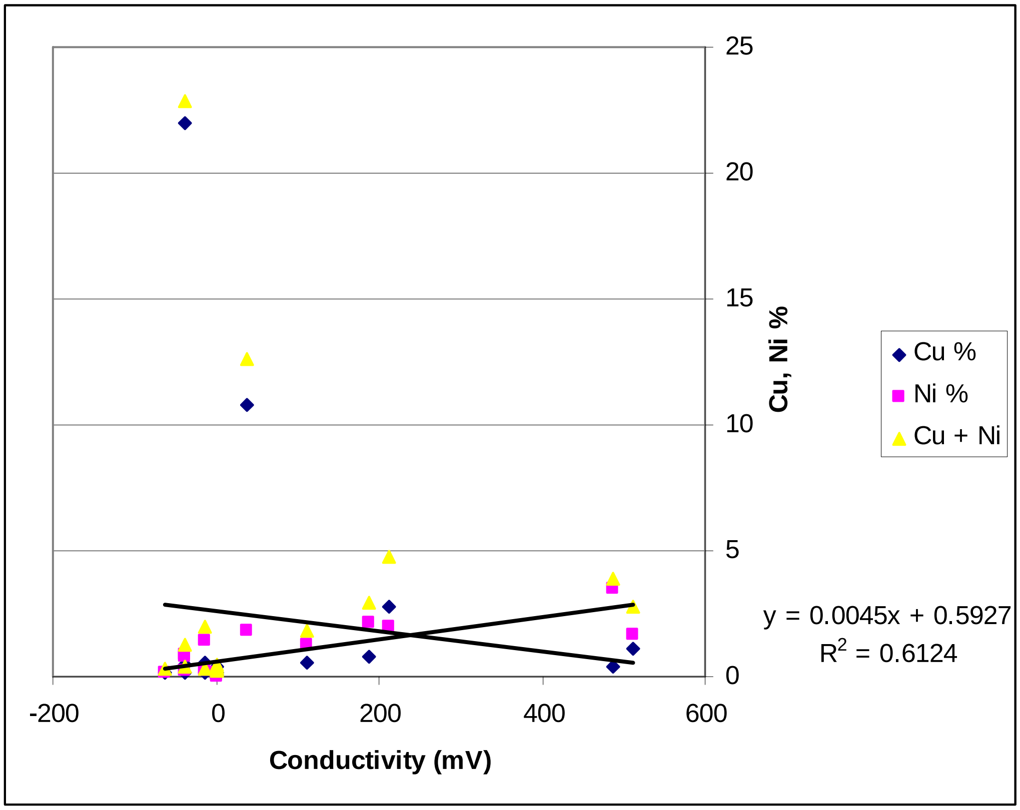
Ave	124.3988
Median	117
Max	255
Min	15
Sdev	44.22189

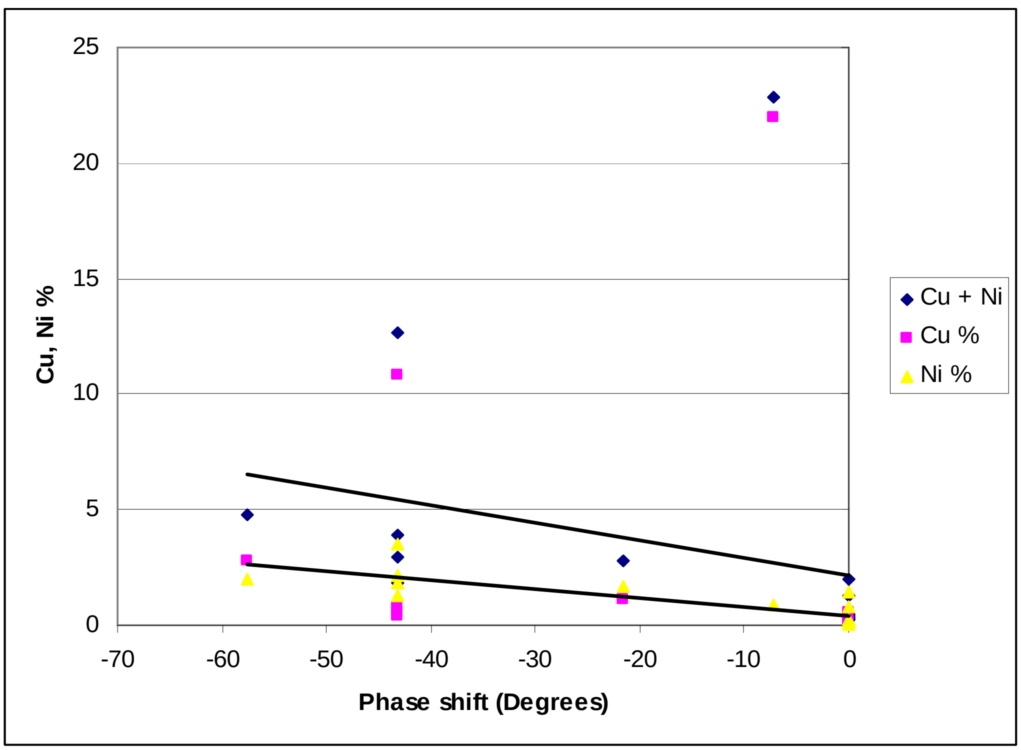


Appendix I – Conductivity and Susceptibility Data from the Research

Measurements of conductivity for selected ore and waste samples were taken with the MineSense B2 'MkII' induction-balance sensor that has been developed. Sensing signal was 100kHz, 3V peak to peak amplitude driving 2 coils, 1 sensing and 1 balancing coil. Input and response signal readings were taken on a Tektronix 255M Analogue Oscilloscope. Readings were correlated with previous data for Cu and Ni content of the samples.

Sample	Grade			Amplitude		Susceptibility	
	Ni %	Cu %	Cu + Ni	Reading	mV	Reading	Ø degrees
Craig 8112 Ore	1.26	0.57	1.83	7	112.5	-3	-43.2
Craig 8812 Waste	0.19	0.12	0.31	0	0	0	0
Craig LGBX Ore	3.5	0.38	3.88	22	487.5	-3	-43.2
Craig LGBX Waste	0.21	0.18	0.39	1	-37.5	0	0
Fraser Cu Ore	0.84	22.01	22.85	1	-37.5	-0.5	-7.2
Fraser Cu Waste	0.03	0.4	0.43	0	0	0	0
Fraser Ni Ore	1.97	2.8	4.77	11	212.5	-4	-57.6
Fraser Ni Waste	0.2	0.12	0.32	2	-12.5	0	0
Montcalm H Ore	2.12	0.82	2.94	10	187.5	-3	-43.2
Montcalm H Waste	0.15	0.18	0.33	0	0	0	0
Montcalm L Ore	1.4	0.56	1.96	2	-12.5	0	0
Montcalm L Waste	0.15	0.1	0.25	0	0	0	0
T-L 15 Ore	1.83	10.79	12.62	4	37.5	-3	-43.2
T-L 15 Waste	0.076	0.4	0.476	0	0	0	0
TL-80 Ore	1.7	1.11	2.81	23	512.5	-1.5	-21.6
T-L 80 Waste	0.114	0.15	0.264	0	0	0	0
T-L 670 Ore	0.82	0.45	1.27	1	-37.5	0	0
T-L670 Waste	0.16	0.15	0.31	0	-62.5	0	0


25
20
15
10
5
0
-200
0
200
400
600
Conductivity (mV)
Cu, Ni %
Cu %
Ni %
Cu + Ni
y = 0.0045x + 0.5927
R2 = 0.6124


25
20
15
10
5
0
-70
-60
-50
-40
-30
-20
-10
0
Phase shift (Degrees)
Cu, Ni %
Cu + Ni
Cu %
Ni %