A CRITICAL REVIEW OF THE INPUTS TO LONG RANGE MINE PLANNING OF OPEN PIT PORPHYRY TYPE COPPER DEPOSITS

by

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A thesis submitted to the Department of Mining Engineering

In conformity with the requirements for

the degree of Master of Applied Science

Queen’s University

Kingston, Ontario, Canada

(April, 2013)

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Abstract

Long term planning is the process used by a mining organization to develop a strategic business plan. The plan describes how the ore is going to be extracted over the mine life. As such, it is routinely updated in order to declare annual reserves, evaluate options and react to changes in the initial assumptions. Inputs into this planning process are the parameters that drive profitability.

The purpose of this research is to understand and document the open pit long range planning process in current use by mining operations, isolate the input parameters that feed into this process, and conduct a critical review of these parameters in an effort to develop a more robust plan.

The thesis also searches for answers to the following questions: Can the copper metal price be correlated to a factor (or a set of factors)? Can the price be predicted? How useful is the work of O’Hara and Taylor in predicting the mine life and milling rate at the scoping study stage? How can the pit by pit graph be used to better guide the selection of the ultimate pit? Is there a realized benefit from operating at an elevated cutoff grade strategy with low grade copper porphyry deposits?

The research concludes with a proposal (not common in the industry) for the selection of the metal price as an input into the mine planning process. This approach, if implemented, can give a corporation a dominant position in the future. The research also presents a modified approach for the selection of the ultimate pit. Furthermore, the use of Taylor’s rule in predicting the mine life was tested and verified on an open pit copper porphyry deposit and the benefits of operating at an elevated cutoff grade strategy was demonstrated for the deposit.
Acknowledgements

To my dear wife who shed tears watching me juggle work and school the past couple of years; to my son and daughter who stayed home and didn’t enjoy a fun weekend with Dad so I can complete this research; to my father, mother, brother and sister for their continuous support and motivation up to the last second, I say thank you.

I would also like to thank my supervisor and my mentor Professor Garston Blackwell for being patient with me all these years and for his continuous guidance throughout my mining career.

Finally how can I forget to thank the three beautiful women in the mining department: Wanda, Tina and Kate? For the warm welcome every time I entered the mining office.
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Chapter 1

Introduction

Planning is a fundamental function of any business venture. It is a path that an organization construct to achieve its goals. These plans are routinely updated as the organization is required to continuously shift its direction to adapt to changes in the business environment.

Similarly, the mine planning process is the development of a business plan that describes how a mining organization is going to extract ore. Mine planning is divided into three main stages based on the time frame imposed on the plan: long term planning (years to life of mine), medium term planning (weeks to months), and short term planning (days to weeks).

Long term planning is essentially the work performed by mining and other engineers in conforming to the strategic plan of a mining organization, and world class mining corporations maintain a strategic long term planning group in their corporate head office. Planning techniques are used to generate mine plans as part of the evaluation of undeveloped ore bodies (commonly called “Greenfield” projects), extensions of ore-bodies and working mines (“Brownfield” projects), and the evaluations of potential mergers and acquisitions of mineral deposits owned by other mining entities. A function of the long term planning group is the continued re-evaluation and updating of plans for operating mines in order to declare reserves, evaluate options and react to changes in the input parameters.

The inputs into the mine planning process are the parameters that drive profitability; metal price, mill recovery, and capital costs. Other parameters include processing costs, wall slope angles, mining cost, and general and administration (G&A) costs. The effect of such variables on mining cash flows are
examined in order to demonstrate that such as spider diagrams are a suitable means of identifying those items having the greatest effect on revenue and expenditures.

Such spider diagrams are a means of focusing planning on the most important aspects of mining cash flows, but they are not the whole story. Cash flows must be optimal, and there are many parameters available and suitable for optimization including payback time, the minimum metal price producing positive cash flow, total profit, total cash flow, the net present value or the internal rate of return.

Discounted methods such as NPV, DCFROR and IRR are the preferred methods. No corporation can satisfy its stakeholders with a strategy which does not pay dividends or increase equity value. To accomplish this, investments have to return value commensurate with corporate goals.

1.1 Thesis Objectives

The purpose of this research is to understand and document the open pit long range planning process in current use by world class mining operations, isolate the input parameters that feed into this process, and conduct a critical review of these parameters in an effort to improve on the process.

The thesis is directed toward the open pit planning of suitable copper porphyry deposits. The various inputs required for such work are discussed along with a brief introduction to short range planning. Most porphyry copper open pit mines require detailed knowledge of some inputs that is beyond the expertise of the long range mine planning and feasibility engineer, and is best left to specialists.

Consequently some aspects of mine planning are not described in detail (e.g. detailed mill design and costing; property ownership, claims and leases; detail of vegetation required for reclamation and etc.). This does not imply that planning engineers with training and experience should not use that knowledge. However, the items listed are beyond the scope of this thesis.
The objectives of this research are to:

- Document the long term open pit mine planning processes currently in use by world class mining operations.
- Develop a step by step process flow diagram of the long range mine planning process.
- Identify the input variables driving profitability.
- Understand the effect of change in each variable on the project value.
- Recognize the predictability of each variable.
- Determine how the planning engineer can utilize this knowledge to improve value throughout the mine planning process.
- Document potential future research topics, especially where the work of this thesis has been withdrawn because the topic would obviously lead an abandonment of the goals set out at inception.

The thesis also searches for answers to the following questions:

- What are the inputs into long range mine plans.
- Can the copper metal price be correlated to a factor (or a set of factors). Can the price be predicted? How can the mine planning engineer utilize this information and develop robust long term strategic plans.
- Is the engineer captive to the vagaries of metal prices in the time period of a decade.
- How useful is the work of O’Hara and Taylor in predicting the mine life and milling rate at the scoping study stage.
- How can the pit by pit graph be used to better guide the selection of the ultimate pit.
- Is there a realized benefit from operating at an elevated cutoff grade strategy with low grade copper porphyry deposits.
- Can deposits be identified which will become profitable future mines ensuring corporate longevity.
1.2 Thesis Contributions

The thesis presents a summary of the mine planning process in terms which can be understood by personnel who have some mining experience. This is accomplished in Chapter 2 by including a photograph of an open pit at the stage of completing the original pit A at the highest grades available in the deposit, mining the first pushback (pit B) at rates approaching full mine/mill capacity, and starting the next pushback pit C in waste. The photograph clearly shows the process of mining the highest grades as soon as possible to avoid discounting, followed by the mining of lower grade ore prior to reaching further high grade. With debt repayment completed the next pushback can mine waste and low grade. Marginal material mined in the earlier pits and stockpiled close to the mill now becomes ore. In such a scenario the original mine design is only run for 12 years as discounting significantly reduces the value of any ore mined after that year.

In order to start the long range feasibility and planning process, published spider diagrams are used. Their applicability to the task will be investigated during the thesis.

The factors defining the price of copper are explored within the framework of the commodities markets. The influence of supply and demand, the availability and need for primary metal versus scrap and substitution, the uses and consumption patterns, and the prediction of copper prices are researched.

In order to start a mining operation not only must the metal price be known, but so has the value of cash flows (revenues minus costs). The revenue comes from metal in the ground and methods of estimating the amount of metal and the confidence in such estimates are also researched at length. Mill recovery is a consideration in that it determines how much of the metal in the ground can be recovered and sold usually as concentrates. Sales are dependent on contracts for the sale of quantities of concentrates from trainloads
to shiploads to years of production. The balance between what smelters need and what mines can supply is also examined.

The life of the mine is also considered as a balance between short life and a costly mill, or long life and a small mill whose capacity can be increased with cash flows from the mine. This avenue of research leads to empirical methods of providing the first scoping estimates on mine life and concentrator throughput. This in turn enables capital costs to be estimated with operating costs being determined from other properties or the data bases commercially available. Equipment is required for mining in the pit and such equipment must be fit for purpose. Equipment sizing leads back to grade estimation to determine what sizes of loaders and trucks are most beneficial for mining the orebody in terms of the spatial distribution of grades and the recovery and dilution inherent in the size of the loader. The grade estimation procedures developed from the research allow the loader size, capital and operating cost to be optimized for the grade model. The loaders must be sized to take advantage of the estimated grade values and spatial distribution of grade and be productive. Again, empirical methods are assessed in achieving the goal of improved cash flow. One of the key areas researched is the provision of methods of assessing equipment utilization and maintenance, and simpler statistics are preferred.

There are important parameters not usually included in conventional spider diagrams such as wall slopes, pushback and ramp design and location, all of which are inter-dependent. This interdependence is examined as a potential research topic.

Finally, the methodology used to determine the important parameters affecting long term feasibility studies of open pit porphyry copper mines are used to define a case study long term mine plan which is optimal and uses methods not commonly used in industry. The research in this area confirms statements made by other well-known contributors to the field of study, and concludes that some areas of research are not warranted based on the study.
1.3 Thesis Organization

The following is a summary of the work as it will be presented in the upcoming chapters:

**Chapter One** – This chapter provides an overview of the research objectives and organization of the thesis.

**Chapter Two** – This chapter reviews the three stages of mine planning, identifies the variables driving profitability, and considers the sensitivity of mining projects to these variables.

**Chapter Three** – This chapter explores the possibility of predicting the price of Copper and Gold. To accomplish this the following questions are asked: Can the price be correlated to a factor (or a set of factors)? and Can the price be predicted?

**Chapter Four** – This chapter presents an overview of grade estimation and discusses some of the errors associated with the measurement of grade. Reserves and resource classification, net smelter contracts, recovery calculation, and economic modeling are also introduced.

**Chapter Five** – This chapter provides an overview of procedures used at the scoping study stage to: select mill throughput, select mining rate, estimate capital and operating costs, investigate equipment selection and productivity, and describe the relationships between them.

**Chapter Six** – In this chapter, a number of parameters usually not considered as profitability parameters are reviewed. Wall slopes are discussed as an input to mine planning. The general method for pushback design is introduced along with ramp placement.
**Chapter Seven** – The open pit long range mine planning process is demonstrated on a copper porphyry mine. The application of Taylor’s rule in selecting the plant size and mining rate is investigated, and the project sensitivity to the profitability parameters is tested and presented in a spider diagram. The techniques for selecting the ultimate pit design were evaluated. The pushback selection and design process is shown, and cutoff grade strategies evaluated.

**Chapter Eight** – This chapter presents the conclusions reached during the course of this research and identifies the recommendations for future work related to the research.
Chapter 2

Mine Planning Process

This chapter reviews the open pit mine planning process. The objective is to map the process in use by mining corporations to develop long range plans, evaluate new undeveloped ore bodies (Greenfield or Brownfield Projects) and isolate the input variables (profitability parameters) that drive value.

Four sources of information are available; the published reports from the Ontario Securities Commission SEDAR (The System for Electronic Document Analysis and Retrieval, 1997), journal papers by software vendors, personal experience of the author and the experience of peers and colleagues in the mining and associated industries, and text books such as “Open Pit Mine Planning and Design” (Hustrulid, W., and M. Kuchta, 2006).

Although corporations do not publish their processes, corporations listed on the Toronto Stock Exchange (TSX) and other jurisdictions must publish reports on their mining operations which enable reviewers to assess the material in the mine plan, usually in the form of a report. The TSX and SEC/CSNX (2010), CNSX is a Designated Offshore Securities Market of the US Securities and Exchange Commission per Rule 902(b) of Regulation ‘S’ under the US Securities Act of 1933) demand a report written by a “qualified person” as defined by the Canadian National Instrument 43-101, (2010) “Standards of Disclosure for Mineral Projects” reporting guidelines (2010 is the most relevant of revisions to the original 2003 document, and at the time of writing, 2010 is subject to major revisions). Section 18 of NI 43-101 details the mine plan as well as the grade estimation, extraction and processing methods, and economics of the ore-body, and material from these sources is used in this research. Manuals published by software providers are a limited source of information on how their propriety tools are tailored to the planning process.
The mining equipment must be suitable and sufficient to mine the deposit given the volume and material to be mined. Availability and utilization of equipment must meet or exceed that planned such that extraction rates of ore and waste can be met at all times. The geological evaluation and ore estimation procedures must produce an acceptable “resource”, as defined by NI 43-101, in terms of grade and tonnage. The ore must be “treatable” or “blend-able” to produce a saleable product of sufficient value to pay for its extraction plus corporate profitability goals. Slope stability analyses must indicate that wall slopes, both temporary and final, will allow mining without serious interruption to the ultimate pit. At this point the mine will be able to produce a “bankable” “ore reserve”, and be able to raise funding from loans, equities etc. to satisfy securities regulators. The terms “resource” and “reserve” are defined in Canada by National Instrument 43-101, and similar securities regulations exist worldwide, e.g. the JORC in Australia (2004).

The evaluation of undeveloped ore bodies (commonly called “Greenfield” projects) are planned using long range planning techniques. Such deposits may be newly found, or their existence (as uneconomic deposits or old mines) known for decades or millennia. Feasibility studies and planning include extensions of ore-bodies and working mines (“Brownfield” projects) and include the potential for mergers and acquisitions of mineral deposits owned by other mining entities.

2.1 Open Pit Mining Overview

An open pit mine is an excavation in the ground for the purpose of extracting ore and which is open to the surface for the duration of the mine’s life. Figure 2.1.a is a picture of an open pit mine with text describing important items and terminology. This particular operation has an “initial pit (A)” which is expanded on the right and centre rear of the picture by a “pushback (B)”, an extension of the pit which is usually restricted to one or two sides starting a few years after the initial pit. As pushback (B) deepens it is followed by “pushback (C)” starting at the top back of the pit and in this case the pushback (C) will
mine everything except the right hand side. Provided access ramps and mine scheduling can be coordinated, a pushback can extend all-round the pit by moving the ramp in stages to the outer edge.

In Figure 2.1.a, it is normal to be mining at least 2 adjacent pits/pushbacks at once to ensure waste stripping and ore production are synchronized. A sinking cut (D) has been completed and expanded. The drill (E) is drilling blast holes to expand the bottom bench further, and the drill cuttings piles from the completed holes for the next blast are shown to the right of the drill and behind the broken material from the latest blast. These cuttings will be sampled and assayed for grade control. This blast will be dug and loaded when the water in the starting cut to the next bench (front right) has been pumped out. The shovel (F) is cleaning up the last of the bench above the bottom. Another shovel (G) is under maintenance while mining the first pushback (B). A third shovel (H) is digging on the top left of the second pushback (C). Another drill (I) is drilling material for the second pushback (C). Truck (J) is returning down to shovel (F) on the haul ramp from the crusher to initial pit (A). The temporary ramp (K) allows the drill to access the top of the upper bench in the first pushback (B). Of note is the spill material from the upper blasts for shovel (G) sitting on berms (L), and similar spill has filled or is filling berms all around the pit. The long range planner has to first co-ordinate pushbacks to safely mine lower material without having spill rock fall on the ramps from higher pushbacks, and second ensure ore (from initial pit (A)), and waste from second pushback (C) is available. Material from pushback (B) may be ore, low grade or waste depending on location, and the shovel (G) can be quickly moved to the bench above and back to mine lower grade ore for the crusher or stockpile/waste for the dumps depending on the drill cuttings assay metal content or ‘grades’.

In this scenario, developed by the short range planner for this shift (8-12hrs), the crusher is operating at less than full capacity allowing maintenance of one shovel, and maintenance of mill equipment further along the material handling flow from crusher to the live ore stockpile of the mill.
The common organizational structure in the mining industry divides an operation into two components: the mine division and the process (mill) division. The mine division usually includes: drilling, blasting, loading, hauling, de-watering and mine engineering. The process side of the operation usually includes process engineering, primary crushing (in some companies a mine responsibility), secondary crushing, conveying, live ore stockpile(s), concentrating, assay laboratory, leaching, solvent extraction and electro winning, and load out of products.

There are three common processes to recover metal from mined ore;

- Crushing and grinding followed by floatation and concentrating.
- Crushing and leaching in heaps followed by chemical extraction.
- Run of Mine (ROM) leaching where the ore is broken only by blasting followed by heap leach and chemical extraction.

Based on the time frame imposed on the plan, mine planning is divided into three stages (Camus 2002);

- Long term planning
- Medium term planning
- Short term planning
Figure 2.1.a: Initial pit A, pushbacks B & C  
(For caption see text section 2.2)  
(Photograph courtesy of G. Blackwell)
2.2 Long Term Planning

Long term planning is concerned with variables that determine the overall value of the mine as mining progresses. Although these variables are set at mine start-up (feasibility stage) they must be revised as the business environment continues to change with time i.e. the beginning of each individual mining business cycle, and annually with some detail. It should be noted that the annual plan (in fact the annual budget) is actually dated up to six months before it is issued prior to the start of the period (year). Time is required to analyze changes in costs, grade estimation and equipment and obtain approval of mine management and senior corporate personnel. The initial assumptions such as metal price, costs, overall wall slope angles, ramp access and mineral content are continuously changing and as such the long range mine plan is in essence an annual revision to the initial feasibility study. At this (say annual) stage the planner has the freedom to revise push-back design, mining sequence, cut-off grade strategy, extraction rate, and ongoing equipment requirements, e.g. effect of metal prices, new technology, equipment scrapping and new (or used) equipment purchases.

The purpose of long term planning is to ensure that;

- ore at a grade meeting or exceeding mine or corporate profitability requirements will be found
- continuous efficient mine and mill operations will process the ore without interruption
- suitable equipment availability and maintenance (both planned and breakdown) is assured
- slope stability (avoiding mine closure from wall failure) will be assured, and should failure occur, mine planning can be revised to ensure continued production.
- maintenance of planned stripping ratios (waste/ore tonnage) will ensure the future availability of ore
- material of lower “marginal” grade can be identified and stockpiled by tonnage and grade in order to continue operations in the event of an ore shortfall. Such material may presently be waste due to capital cost payback or present metal prices. It may become ore stockpile or even ore at an
unknown higher future price of the product mined (metal price), or as a result of technological change. The material may also be available from marginally economic stockpiles for emergency production or processing at ‘end of mine life’ when all debts have been repaid and corporate profit expectations have been realized.

- the mine is profitable. The forward metal price may be known over the first three months of the long range planning revisions, but only forward selling can assure the metal price years in advance. Forward selling can also be carried out over periods of years if deemed advisable, but there is a risk that the large cash flows from a metal price peak (a price ‘window’) will be lost.

The overall process can be summarized in the following steps:

1. Receive an updated resource model of grades by location and bench from geology/mine planning
2. Receive slope guidelines from geotechnical staff
3. Develop a long range economic model
4. Run a ”pit optimizer” such as those using the Lerchs Grossmann optimizer (L-G) algorithm (Lerchs, H and I.F.Grossman, 1965) to determine the ultimate pit limits.
5. Run incremental product price increases (alternatively by operating cost or cut-off grade decreases) to make narrow circular “shells” to guide the design of the pushbacks
6. Design pushbacks with ramp(s) access from the current topography to the ultimate pit
7. Run “Life of Mine” Schedule
8. Design dumps and stockpiles with adequate capacities based on topography
9. Determine equipment requirements
10. Issue the life of mine plan with cash flows and discounted cash flows at an interest rate provided by corporate staff, and determine cash flow, net present value (NPV), rate of return (ROR) or internal rate of return (IROR) or use similar project evaluation techniques both before and after applicable taxation (Fraser, N.M., E.M. Jewkes, I. Bernhardt and M. Tajima, 2009)

The process is shown in Figure 2.2.a.
Figure 2.2.a: Long Range Open Pit Mine Planning Process.
2.3 Short Term Planning

Although the main focus of this thesis is long range planning one needs to understand the complexity of moving to a detailed mine plan. This section provides a brief review of the short term planning process and demonstrates the importance of considering operability during the development of the long range plan. Short term planning deals with maintaining the goals of the annual budget. Daily or weekly ore grade forecasts and production scheduling form part of these goals. Activities such as drilling and blasting, temporary and permanent ramp access for trucks, and moving drills and shovels are addressed. An important function is to ensure that a sinking cut to the next bench is planned and expanded in time to provide sufficient digging room for productive mining as shown in Figure 2.1.a. The sinking cut usually becomes part of the main haul road, takes time to dig in the small space, and probably requires a submersible water pump which has to be advanced regularly. Where a temporary ramp is developed, the permanent ramp can be “raised” working up from the new open bench. The photograph Figure 2.1.a is a practical explanation of most of these details.

The short term planning process is similar in many respects to long term planning, but has a greater certainty of finding ore grades in the locations, quantities and values expected, and having waste at the required stripping ratio available, all based on ongoing blast-hole drill cuttings assays and ongoing geological interpretation. Short term planning also has up-to-date information regarding mine equipment and crusher availability and the need for repairs, and also the state of any pit wall instability and routine pit dewatering. The planning process also looks ahead a month or so to ensure that drill and shovel moves are discussed with mine operations and prepared for.

The metal price is also known but will not be realized immediately due to the commodities markets where prices are quoted a few months ahead (delivery date) or when transfer of ownership of concentrates to the
smelter occurs. Barring a sudden collapse or spike in metal prices, this latest forward price will be close to that realized.

Although blast holes are drilled for the purpose of loading and blasting with explosives, a small part of the “cuttings”, 1 kg plus or minus of material from the tonne or so from the hole, usually 15mm minus gravel, sand and dust, are assayed from each hole. These assay results allow the grade control personnel (or short term planner) to lay out ore, stockpile and waste “on the ground”. Short term planning can then be carried out with some certainty, allowing the drill and shovel operations to be planned for the week.

Starting with blast layouts prepared typically using AutoCAD or a mine planning software package (for example, MineSight 3D Blast Pattern Editor), all data relevant to each blast-hole can be stored, (i.e. such as co-ordinates, bench number, hole depth, water depth, whether de-watered and how difficult, amount and type of explosive, rock type as well as sample grade by blast number and hole number). Present technology allows the radio link transfer of the engineered pattern to the drill where the driller can position the drill over the hole planned with a graphic computer screen and return the co-ordinates to engineering. This has been made possible by the adoption of Geographic Positioning Systems (GPS) and the technology can also facilitate fully automated drill operations.

Unfortunately, electronic means of measuring the average grade of the hole “in-situ” have not been applied as discrete “down the hole” sampling methods do not provide an unbiased or accurate result. Accurately measuring the average metal grade in the walls of a blast-hole containing a minority of interspersed small veins and vein-lets millimeters or less in width is difficult. The barren host rock contains within it small individual veins and vein-lets with values of say 25% copper. This material cannot be measured accurately and repeatedly with some form of down-hole on-stream analyzer, and would be even more difficult to measure for low concentrations and numeric values of gold. This leaves some form of manual method of sampling blast hole cuttings conducted by personnel as the most popular,
and several procedures have been developed to produce representative small samples of the drill cuttings left on the surface after the hole has been completed.

Modern large shovels are also equipped with similar GPS systems enabling the shovel operator to use a computer display to indicate the bucket location, making it possible to dig to ore grade outlines, planned pit walls and un-blasted drill patterns accurately. Maintenance of the bench elevation and sinking cut gradient (elevation) is both simple and accurate using such technology. This in turn gives the mine planning engineer better control of grade and tonnage mined, and combined with computerized dispatching systems on trucks with GPS location, a better accounting of ore and waste. Maintenance of pit equipment and the primary crusher can be planned along with pit equipment moves. Usually this is determined by the scheduled crusher maintenance. Breakdown maintenance must also be allowed for if possible, and is usually accomplished in discussions with the pit, maintenance and crusher general foremen. The processing plant can be warned of the arrival and duration of impurities in the ore (e.g. lead or softer altered rock) that may affect the mill throughput, product for sale, recovery, reagent use and etc. Activities for the following week can be discussed, including the state of any wall instabilities and dewatering, completing the short term planning process.

The overall process can be summarized in the following steps:

1. Receive a plan of blast-hole grades by location and bench outlining ore, stockpile and waste.
2. Estimate outlines of ore, stockpile and waste where blast-hole drilling and sampling has not yet been completed.
3. Ensure ore grade mined will produce acceptable cash flow.
4. Produce digging plans matching the stripping ratio.
5. Develop digging plans which minimize shovel moves.
6. Produce plans of drill holes for individual blasts. Allow for long drill moves where required.
7. Note and allow for scheduled maintenance of shovels, drills and crusher.
8. Start sinking cuts early enough to enable full production from the new bench as required, and ensure dewatering pumps are available.

9. Ensure mine plan allows for any pit wall instability by ensuring availability of “safe” ore.

10. Prepare for potential breakdowns of any or all of the equipment (e.g. which shovel can be re-mobilized fastest regardless of future damage to parts).

Medium range planning merges aspects of both short and long term planning, and is not commonly used as some overlap of long and short term planning can be expected from the short term planning engineer on a continuous basis.

2.4 Isolating Important “Profitability” parameters

The second objective of this chapter is to isolate the input variable that drives value through the mine planning process (Profitability Parameters). These are the input parameters to the pit optimization step, commonly referred to in the industry as LG input parameters (Christensen, K., Connaughton, G., Ogryzlo, P., pp. 75, 2011).

These parameters in addition to capital expenditure cost (CAPEX) are the primary drivers of value for a mining project. Prior to considering a mining venture, a corporation determines the suitability of an investment by testing the sensitivity of the venture to changes in these variables.

There are several methods in common use, including two types of “spider diagrams” and “tornado diagrams”. These variables are described in their typical order of importance and assume that a mine plan has already been completed to define this order, a typical “chicken-and-egg” scenario.
2.4.1 Spider diagrams

An example of the more common mining application of a spider diagram (Figure 2.4.1.a) is that given by Inmet Mining Corporation (March, 2010) in a SEDAR submission to the Ontario Securities Corporation by AMEC for Inmet.

![Sensitivity of After Tax IRR](image)

Figure 2.4.1.a: Inmet Mining Corporation Spider Diagram showing Impact of Metal Price Change on the Cobre Panama Project, Inmet SEDAR FEED study, their Figure 9-2: Sensitivity Spider Graph for After-Tax IRR (after AMEC and Inmet, March 2010).

Typically such diagrams indicate the effect of a “cost” or “revenue” on the project, in this case on the After Tax Internal Rate of Return (after tax IRR). The most sensitive (important) items (lines with steepest slope, change is the X axis) are Metal Price, Steel, and Operating Expenditure. It should be noted that the relationship with “Impact on IRR” is often not linear.

The second application of a spider web diagram (Figure 2.4.1.b) involves scoring the fundamental inputs to mining projects such as “human” and “financial”. A score of “5” indicates that a resource is readily available and “0” not available at all. As part of the Climate Adaptation and Minerals Down Under Flagships of the Commonwealth Scientific and Industry Research Organization (CSIRO), Australia
(Loechel, B., J.H. Hodgkinson, K. Moffat, S. Crimp, A, Littleboy, and M. Howden, August, 2010), a Goldfields-Esperance Regional Mining Climate Vulnerability Workshop produced the spider web diagram shown in Figure 2.4.1.b.

![Spider web diagram](image)

Figure 2.4.1.b: Spider web diagram produced by the Industry Sector Group (blue line: industry [general expertise]; blue dotted line: industry [specific climate adaptation expertise]; red line: consultants to industry). The CSIRO Australia spider diagram (their Figure 13) shows the importance of various factors on climate in the Goldfields Esperance area as assessed by various groups. Note that human factors (construction, operation) are presented as very available but specific expertise in climate change etc., (social) is lacking. Consultants believe that there is social acceptance of climate change, but industry believes that there is little acceptance. (After CSIRO, Loechel, B., J.H. Hodgkinson, K. Moffat, S. Crimp, A, Littleboy, and M. Howden, August, 2010)

Diagrams such as the second type of “spider” (spider web, Figure 2.4.1.b) enable broad sectors such as the availability or cost of labour, which impacts almost everything, and steel, which mainly impacts construction and shovel-rock wearing parts, to be recognized at an early stage of project development.
2.4.2 Tornado diagrams

A further type of diagram which expresses importance and sensitivity is the “Tornado” and Figure 2.4.2.a shows a typical example from the Strategy Consultant, Graham Jeffery website, (Jeffery, G., 2010).

The Tornado diagram describes ranges of uncertainty for all variables making up a project rather than on some point estimate. This allows risk and reward to be quantified in order to make decisions based on “what if” scenarios. A range of three values (10th, 50th (or median) and 90th) is assigned to each variable. There is then a 10% chance the true value will turn out lower and a 10% chance higher, and the base case (median) is the “best estimate”. The chart represents 80% of all outcomes with the variables least understood at the top of the “tornado”. Even binary (yes or no) variables can be included as “yes” representing the NPV target and “no” the cost of abandonment. Inherent in the model is some indication of NPV linearity where the bar has a different length (implied affect) depending on which side of the NPV target it falls. For example, the Launch Year is estimated as 2015. Completing the launch in 2016 has a substantially lower NPV over a 2014 launch, and the NPV is not linear with respect to launch date.
2.5 Choice of a sensitivity indicator

In summary, all three methods (spider, spider web or tornado), of determining the importance of variables on some form of “profitability”, “cash flow” or “discounting” basis are useful. However, the first type of spider diagram is most likely to be used for economic analysis, figure 2.5.a (Doggett, M., and Mackenzie, B. W., 2007). This thesis begins by accepting the importance of such as spider diagrams, and proceeds to replicate them in order to critically judge their use in the mine feasibility process.
Figure 2.5.a: Typical spider diagram showing sensitivity of net present value to possible changes in input variables. A vertical line indicates that a change in the variable has no effect on NPV (the % change is the Y axis). As the various lines describing cost and revenue centers approach the horizontal, their effect on NPV is larger. (After Doggett and Mackenzie, 2007)

The approximate percentage change in NPV from a one percent change in any particular variable is sown in table 2.5.1;

Table 2.5.1: Ten key factors from commonly used spider diagrams

<table>
<thead>
<tr>
<th>% Change</th>
<th>Factor</th>
</tr>
</thead>
<tbody>
<tr>
<td>12%</td>
<td>Metal Price</td>
</tr>
<tr>
<td></td>
<td>Head Grade</td>
</tr>
<tr>
<td></td>
<td>Process Recovery</td>
</tr>
<tr>
<td>8%</td>
<td>Operating Cost</td>
</tr>
<tr>
<td>3%</td>
<td>Pre-production Capital</td>
</tr>
<tr>
<td>2.5%</td>
<td>Reserve Size</td>
</tr>
<tr>
<td></td>
<td>Mine Recovery</td>
</tr>
<tr>
<td>1%</td>
<td>Dilution</td>
</tr>
</tbody>
</table>
The ten key variables (Table 2.5.1) individually, over their respective percentage ranges, endanger the economic viability of a mining project used in Figure 2.5.a. Some of the variables have effects so similar that they are combined (e.g. metal price, grade and process recovery have almost identical effects on NPV).

Items such as wall slope angles do not appear to be included in such diagrams. This may be because most ore-bodies under review in this thesis are either porphyry copper or epithermal gold with extensive low grade mineralization around a central core. A shallow wall slope might be of benefit in allowing near surface low grade to be mined and stockpiled, perhaps extending mine life. This does not imply that wall slopes have no effect, and the effects of varying wall slopes in porphyry copper and epithermal gold open pits will be discussed. Given good management, which recognizes the contribution of geology and modern slope monitoring systems, wall slopes can be classified as “somewhat predictable”. This was not the case decades ago when expensive slope monitoring systems were in their infancy making wall slopes “somewhat unpredictable”.

Other factors such as the size/location of the initial pit, number, size and location of push-backs, and ramp placements (ramp design), are also not included in the list but are discussed in upcoming chapters.

Using an arbitrary predictability scale comprising 6 discrete units;

- very unpredictable
- unpredictable
- somewhat unpredictable
The six key variables are rated in order of importance and with notes as to their predictability:

1. Metal Price, Grade, Processing Recovery (in Figure 2.5.a, all three follow the same curve)
   - metal price - very unpredictable
   - head grade – somewhat unpredictable at the feasibility stage, predictable when blasted
   - processing recovery – predictable
     ➢ a 1% change in any of these variables produces an 12% change in NPV

2. Operating Cost
   - somewhat predictable at the feasibility change, predictable for a production year
     ➢ a 1% change in this variable produces an 8% change in NPV

3. Pre-Production Capital
   - somewhat predictable
     ➢ a 1% change in this variable produces an 3% change in NPV

4. Reserve Size, Mine Recovery
   - somewhat unpredictable
     ➢ a 1% change in any of these variables produces an 2.5% change in NPV

5. Dilution
   - somewhat unpredictable
     ➢ a 1% change in this variable produces a 1% change in NPV

6. Production Capital, Sustaining Capital
   - somewhat predictable
     ➢ a 1% change in any of these variables produces an 0.25% change in NPV
This general predictability rule does not account for differences depending on the metal(s) mined, ore-body shape, stripping ratio and pit wall slope. For example at an open pit gold mine with deep high grades and a large (say 20/1) strip ratio, mine operating costs would have a severe impact on NPV. The date of initiating the project may make costs (pre- and production, operating) “unpredictable” as opposed to “somewhat predictable” if monetary inflation or cost inflation due to an unsustainable demand for equipment (e.g. shovels) and wearing parts (e.g. tires) result from too many positive independent production decisions being made at the same time.

2.6 Summary

The long range mine planning process has been mapped as a step by step process. This process will be applied in chapter 7 on the Brenda block model. The profitability parameters where identified and ranked in the following order of importance (based on Doggett and Mackenzie, 2007): metal price, head grade, process recovery, operating cost (OPEX), capital expenditure (CAPEX), reserve size, mining recovery, dilution, production and sustaining capital. The effect of change in parameter on the NPV of the Brenda project will be tested in chapter 7. The efficacy of the spider diagram, at this stage of the thesis, has not been proven, but merely assumed.

As the metal price is the main driver of value in mine planning it will be the focus of the next chapter
Chapter 3
Metal (Copper) Price

The objective of this chapter is to explore the possibility of predicting the price of Copper. To accomplish this; the following questions are asked: Can the price be correlated to a factor (or a set of factors)? And can the price be predicted?

3.1 Copper Prices, Demand, Processing, Usage and Forecasts

A recent study by Ernst & Young (2008) has revealed that, despite the high level of expertise found among metals analysts, "the accuracy of outcomes for the recent metal price forecasts has been consistently disappointing." The study authors, Downham and Williams (2008), claim metals analysts’ prediction of metal prices “have consistently and significantly lagged behind the actual spot market,” and that mining and metals equities have been undervalued. They also noted that "the pace of consolidation in the mining industry shows no signs of slowing. The continuing, robust levels of metal and minerals prices are fuelling the drive for growth through acquisition." and US$70 billion in transactions took place in 2006. "All of these transactions have taken place in an environment where commentators speak of the sector being near the ‘top of the cycle' with analysts predicting huge declines to current metals prices in the long term. ” They asserted that the market "is undervaluing mining assets by not fully appreciating how long demand will outstrip supply”. "With a near impossible task of predicting future prices, it is no wonder that metals analysts are keen not to stray too far from the comfort zone of historic averages . . .”

McIsaac (2008) shows a decreasing overall trend in 2001 dollar copper prices from 1970 to 2000 (his figure 19, page 63). This trend is due to overproduction by mining operations attempting to remain in business and pushes down prices. This trend is exacerbated by positive technological change reducing
operating cost and further increasing supply. In a short time of a few years, reserves will be depleted leading to an undersupply and higher prices and this eventually brings more mines into production. Prices have actually increased to or held steady at about US$7700/mt (US$3.50/lb.). McIsaac’s work was written in 2008 based on material from up to about 2000. It is a matter of opinion as to whether McIsaac in 2008 could have predicted “todays” 2012 and 2013 prices. This thesis is concerned with the metal price in effect at the year a mine starting construction today (2013) would reach production (2017) and maturity, i.e. about 2024. By carefully watching worldwide supply and demand and applying McIsaac’s work, (and assuming these can be updated in “real time”) the extent and rate of price changes might be estimated, but this does not lead to an accurate price forecast acceptable for the biggest factor influencing mine feasibility, metal price. Figure 3.1.d shows that China, population 1.3 billion, can influence metal prices for reasons that can only be guessed at. The far-east comprises many developing countries with demographics that will influence demand in the future.

In order to predict the price of copper, factors such as the supply and demand of the metal must be studied and answers to the following found:

- What is the price history of the metal
- Where does “scrap” copper go
- Where does copper come from and how much
- Where is it smelted and refined
- Where and how is it alloyed
- Where is it formed into useable shapes
- Where is the final product used
- Can new uses and future consumption be predicted
- Can technological change and substitution with other metals be predicted
Most of the data used in this section (3.1) is from the International Copper Study Group (ICSG) in Lisbon, Portugal and their 2010 World Copper Factbook (http://www.icsg.org/index.php?option=com_docman&task=doc_download&gid=278&Itemid=61).

An excellent review of 100 years of resource growth for copper is given by Schrodde, R., (2010), (Figure 3.1.a) which explains the impact of costs, grade and technology, exploration success, trends in copper discovery rates and the key drivers. Key points of the review are:

- The world’s copper resource grew by a factor of 25 times over the last 100 years
- Much of this was through discovery
- Technical innovations enabled giant “disseminated” porphyries to be mined, and then Cu-oxide deposits (SXEW)
- Costs were reduced through economies of scale (30%) and new technologies (70%)
- As costs reduced, so did cut-off grades thereby further growing the resource. Halving the ore grade increased ore tonnes by a factor of 6 and tonnes of metal by a factor of 3.
- Prices are an output, not an input. They are driven by supply and demand
Figure 3.1.a: 1900 to 2010 copper prices and all mining operation costs stated in constant 2009 US$ per pound of copper. Prices can be classified into 4 groups, 1900-1920, 1920-1975, 1975-2005 and after 2005. The reduction in unit costs of mine operations after 1975 is due to economies of scale (mill tonnage throughput and increasingly larger mining equipment). A note of concern is that after 2005, costs are estimated to rise. The fall in copper price in 2008 has not continued, and is at the time of writing about US$7720/mt (US$3.50/lb). (After Schodde, R. 2010)

The copper price must also be viewed in more modern terms, 1960 to 2006, Downey, W., (2009). The period to 2003 is of a small increase in a stable price punctuated by sudden price increases as business cycles ebb and flow (Figure 3.1.b). In 2003 the price rises “exponentially” from US$1750/mt to US$8800/mt (US$0.79 to $4.00 /lb.).
Figure 3.1.b: Copper trading chart in US cents/lb in currency of the time. The period 1960 to 2003 is characterized by a slowly increasing flat price for copper broken at intervals by large swift increases and equally large and swift decreases linked to the ebb and flow of business cycles. Post 2003, copper prices rise exponentially from US$1750 to 8800/mt (US$0.80 to US$4.00 per pound). (After Downey, 2009)

The most up-to-date material at the time of writing is that from the International Monetary Fund (IMF), (2011), prepared by their Research Department Commodities Team, Figure 3.1.c. From 1980 to 2003, the slow increase in copper price can be seen. This is punctuated by sharp increases and returns to the “flat” price which have been the norm for a half century. In 2003 an exponential increase in price occurred indicating an imbalance between supply and demand. Demand decreased in 2008/9 from a recession which reduced the copper price to levels in keeping with the earlier period, and then rose exponentially again.

For the long term mine planner and feasibility stage planner, important conclusions can be drawn. If a more serious recession occurs then copper prices may reduce to “approximately” US$2000/mt (US$1.00/lb). This price level would demand operating mine re-design, reduction in mine life or “mothballing”, and the postponement of new mines. If the copper price remains near US$8000/mt (US$3.60/lb) for the foreseeable future, investments in new mine discoveries, acquisitions and mergers
would be prudent. In the short term, provided studies show that total supply will not greatly exceed consumption, mines should continue to be brought on-stream. For the long term, a company must have deposits on hand to maintain their business, and “greenfield” exploration, acquisitions and mergers are in order. A business cannot allow a competitor to “tie up” future supply. Either the business or the competitor will emerge dominant a decade or so in the future if the business allows this to happen.

Figure 3.1.c: Copper prices (log scale) 1980-2012. The price patterns shown and described in (Figure 3.1.a) can be seen until 2003. In 2003, an exponential increase to US$8000/mt occurs which is ended by the 2008 recession reducing demand and lowering the price to US$3000/mt. Another upswing in 2010 brings the price to US$7000-9000/mt. Any reduction in consumption of copper would apparently reduce the price to levels more in keeping with those of 1980 to 2002, which should be of concern to producing and potential copper mines. (After IMF research department commodities team, 2011)

The general rule that when copper stocks fall, prices rise and vice-versa would appear to be true until July 2009 when there is evidence of both stocks and prices rising. This is shown in Figure 3.1.d, LME copper stocks and spot prices from Kitco Metals (2011) for the period December 2006 to December 2011. A full study of what makes copper prices rise and fall is beyond the scope of this thesis, and also beyond the scope of 90% of mine planning engineers. This does not prevent the mine planning engineer from making
an estimate using a series of five values; worst conceivable, expected low, most likely, expected high and highest conceivable based on intuitive guesses and historical prices. This series of mine plans is then readily available when required by management.

Figure 3.1.d: LME copper stocks and spot prices December 2006 to December 2011 (www.kitco.com). When stocks fall, the copper price rises. When stocks rise, the copper price falls. However, these rules are not substantiated in the period covered by the right hand side of the charts. From Dec08 to Feb10, copper
prices rose despite a rise in stocks from Jul09 to Feb10. Quoting Ernst and Young; “metal price forecasting has been consistently disappointing”.

Work by Ioannou S. and E.Young, (2012) (Haywood Securities Base Metal Analysts) shows that the arrival of China as a major importer changed the copper market in 2003, significantly increasing copper prices. By 2005, China had developed a strategy of buying (and presumably stockpiling) when the copper price was low, thereby giving its copper fabricators an advantage over spot buyers. These conclusions can be seen in Figure 3.1.e below.
Figure 3.1.e: China’s effect on copper prices 2005-2011. Green (right scale) shows LME spot price and the entry of China into the copper market as a major buyer in 2009. Red (left scale) shows the strategy of China to buy more copper when prices are low. Purchases have slowly decreased since the 2009 peak and increased again recently. (Ioannou S. and E. Young, 2012) (Haywood Securities Base Metal Analysts).

In 2009 (the latest complete data available from ICSG 2010 World Copper Factbook) about 16 million tonnes of primary copper were produced annually from mined concentrates (12.7 million tonnes) and solvent extraction – electro winning (SX-EW) (3.3 million tonnes and growing). Chile continues to be the
biggest producer at 5.5 million tonnes. However, worldwide scrap copper continues to be a major source, (Figure 3.1.f) adding a further 8 million tonnes, representing about 35% of the total, with various regions having a higher or lower Recycling Input Rate (RIR). Comparable RIR values for 2002 and 2008 are as follows

<table>
<thead>
<tr>
<th>Recycling Input Rate (RIR)</th>
<th>2002</th>
<th>2008</th>
</tr>
</thead>
<tbody>
<tr>
<td>Asia</td>
<td>30.9%</td>
<td>34.0%</td>
</tr>
<tr>
<td>Europe</td>
<td>44.4%</td>
<td>42.7%</td>
</tr>
<tr>
<td>North America</td>
<td>32.4%</td>
<td>33.0%</td>
</tr>
<tr>
<td>Rest of the World</td>
<td>16.3%</td>
<td>16.3%</td>
</tr>
<tr>
<td>Total World</td>
<td>34.6%</td>
<td>35.1%</td>
</tr>
</tbody>
</table>

Figure 3.1.f: Copper sources. Primary refined (mined) production makes up 65% of the total, and there are two sources of recycled metal which make up 35% of the metal entering the market; new (scrap from manufacture of copper alloys and shapes) and old (copper in obsolescent articles or “end of life”). (After ICSG 2010 Fact-Book, page 39)

The Recycling Input Rate (RIR) measures the proportion of metal and metal products that are produced from scrap and other metal-bearing low-grade residues. The RIR is mainly a statistical measurement for
raw material availability and supply rather than an indicator of recycling efficiency of processes or products. Europe has the best RIR record, and Asia is improving. Overall, in 2008, recycling is probably not as high as it could be despite having an energy requirement of one fifth that of “primary” copper, (EuroCopper Annual Report, (2011), their page 9, (http://www.eurocopper.org/doc/uploaded/File/20110422_annualReport_eurocopper_v14.pdf)

The rise in consumption averages about 4% per year, and appears to follow the growth of world population (ICSG 2010, their page 37) except for the years 1975-1990. This period is shown in Figures 3.1.g and 3.1.b where the copper price can be seen to be constant despite some volatility. The employment of better technology in mines (economies of scale with larger equipment and throughput) allowed many mines to continue operations albeit at a lower copper price and flooding the market. In Figure 3.1.e and 3.1.b the copper price seemed to decline in the period 1995-2003, implying that low copper prices spurred consumption and lowered inventories, restart the correlation between copper consumption and world population.
Figure 3.1.g: World copper usage and population. The period 1975-1990 shows that per capita consumption of copper remained stagnant. Before 1975 and after 1990 there was a correlation between consumption and population. (After ICSG World copper fact book, 2010, their page 31)

In 1990, political change in China to a more market driven economy followed by the entry of China into the World Trade Organization in 2001 (http://www.wto.org/english/news_e/pres01_e/pr243_e.htm) resulted in a continuing increase in demand and price for copper after 2000 once inventories in the chain of copper production were depleted. China now uses 30% of world copper (Figure 3.1.h).

The largest twenty producing mines account for about 8 million tonnes or 50% of the total supply, so constraints on production at of any of these operations has a significant effect on supply (ICSG 2010, their page 12). Constraints include:

- Falling Ore Grades: a serious issue in developed copper areas such as the USA and Chile
- Project finance: cost of capital is a central factor. High interest rates may reduce supply significantly
- Capital cost overruns: in the past, underestimations of US dollar inflation were the source of many cost overruns. Today it is lack of skilled labour (human resources), steel, etc.
- Water supply: a critical issue in dry mining districts
- Energy: coal is the fuel chosen to power many copper mines, mills and processing plants. Climate change issues may increase costs.
- Shipping costs: not an issue for copper at present
- Sulphuric acid supply and price: typically 16% of cost for SX-EW projects
- Skilled labour: open labour markets would help address this constraint
- Labour strikes: tend to increase when refined prices are high and GDP is growing fast, but tend to be longer and less frequent in more stable economic times and also when copper prices are down
- Differential between imported inputs and domestic input costs affected by the currency strength of the producer
- Market power/concentration: risks have moved to the import demand side versus export supply side in recent years
- Peace and security is also a key factor

The worldwide movement of concentrates is also a concern where continental canals (Suez, Panama) and straits (Gibraltar, Malacca, Bering, Hormuz and etc.) are key possible restrictions to ship transportation.

![Major International Trade Flows of Copper Ores and Concentrates]

Figure 3.1.h: Trade flows (not actual routes) of copper concentrates and ores and list of major exporters and importers. Of note is the west coast of South America exporting to the far east via the Pacific ocean. This is shown travelling east not west in this idealized map. (After ICSG Fact-Book 2010, page 28).

The importers of the concentrates then employ various techniques in smelting to produce such as impure copper ingots. The worldwide smelting capacity in 2010 was about 18.5 million tonnes. The top twenty smelters can produce 8.7 million tonnes of copper, a similar concentration to that of mines (20 mines produce 50% of copper), but not in the same geographic areas. China has the largest smelting capacity at 3.5 million tonnes, 25% of the total, and is the largest importer of ores and concentrates.
From the smelter, the ingots (often referred to as “blister copper”) are refined. With SX-EW processes, refined copper is produced without the costly smelting stage, accounting for 1% of production in the late 1960’s and 18% in 2009. Worldwide refined copper capacity was 24 million tonnes in 2009, and again the top twenty refineries are capable of producing 8.7 million tonnes. Although China has the largest refining capacity (4.2 million tonnes), Chile has 3.3 and Japan 1.4 million tonnes each. Refining is the only phase of copper production where facilities are not heavily concentrated in one jurisdiction.

The penultimate phase for copper usage is the production of alloys such as brasses (copper + zinc), bronzes (copper plus tin and/or aluminum) and the production of semis (semis or sumis is a word derived from ancient coinage meaning half). The copper is then used for:

- Electrical
- Electronics and communications
- Construction
- Transportation
- Industrial machinery and equipment
- Consumer and general products

The “ICSG World Copper Factbook, 2010” page 25, provides statistics for the capacity to produce semis by region and by end product shown in Figure 3.1.i. Power requirements to re-melt semis and re-form the metal are major cost considerations in semis manufacture shown in the size of the copper semis industry in Norway and Switzerland where hydro power is available. The end use of semis shows wire rod (mainly electrical) is the major use (3.1.i right), after ICSG World Copper Factbook, page 25. Other important products include plates, sheets and strips, rods, bars and sections, and copper and copper alloy tubes.
Figure 3.1.i: Production of copper semis is influenced by the cost of power to melt copper and copper alloys (left). The end use of Semis shows wire rod (mainly electrical) is the major use (right), after ICSG World Copper Factbook, page 25.

Labour costs are a relatively large component of semis costs, hence emerging nations typically have large semis production. A typical operation chosen at random is the Saru Copper Alloy Semis PVT Ltd. of India. It produces simpler rods, bars and pipes and also finished products such as valves, hose connectors and sophisticated shapes for various engineering uses. These are shown in figure 3.1.j taken from Saru Semis website (http://www.sarucopper.com/gifs/e-catalouge.pdf).
The end uses of copper and copper alloys are shown in Figure 3.1.k from the ICA and ICSG Factbook, their page 50. Engineering accounts for 11.5 million tonnes per year, followed by building construction (7.3 mt) and infrastructure (3.3 mt) for a total of 22.1 million tonnes in 2009. Within these groupings, electrical power (5.3 mt), industrial (2.7 mt), power utility (2.5 mt) and diverse (2.4 mt) are major users. The conclusion is that electrical and associated uses account for 7.8 mt, over a third of the 22.1 mt total. The remainder can be said to be diverse usage and copper is a factor in all aspects of our civilization.

The building and construction industry is a significant user of copper, and home, condominium and “high rise” construction play an important part in defining the copper price. The other major user, engineering and manufacturing, is more dependent on the growth and state of the economy.
Figure 3.1.k: Users of copper and copper products (left), and ultimate end use by building, infrastructure and equipment. (After ICA & ICSG Fact-book, 2010, page 50)

The flow of copper from mine to end use is shown in Figure 3.1.1. The supply-demand is influenced somewhat by the amount of metal equivalent in transit and on inventory throughout the world. This flow gives rise to some volatility as inventories must be emptied at the price paid for, and filled at the prevailing price which is in turn dependent on supply and demand.

The flow starts with sources of copper, primary mined and secondary scrap which is then refined to produce metal. The metal ingots or sheets are then converted to semis of many types. The semis are converted to finished copper and copper alloy products and used as such by consumers. The last stage is returning “end of life” copper to the scrap metal input.
Figure 3.1.1: The flow of copper in detail showing mine to refined usage (top) and semis products to scrap (bottom) and eventual recycling. (After ICA & ICSG Fact-book, 2010, page 50)
The copper demand is also influenced by substitution which lowers the demand and price. When a new usage for the metal is developed, this increases demand and price. Such a scenario can be examined very approximately by the potential change from internal combustion motor vehicle engines to electrical.

Assuming the number of vehicles per person remains constant and the number of vehicles is 500 per 1000 persons (an average European figure from "Motor vehicles statistics - countries compared – NationMaster" (2009), a typical family of four would require 2 vehicles. These vehicles require an efficient AC motor (7 to 11 Kg Cu) with an inverter (4 to 5 Kg Cu) or approximately 13 Kg of copper total each (from "The Copper Development Association", (1998)) (http://www.copper.org/publications/newsletters/innovations/1998/02/ev_intro.html).

If 1.3 billion Chinese citizens purchase 650 million electric cars using 13 Kg of copper each, the total copper required is 8.5 million tonnes. This represents an increase from the present 22 million to 30 million tonnes of copper, or 36%, or about 3.6%/yr over a 10 year implementation. If supply and demand is close to balanced now, the long term (10 to 20 year) price for copper must rise significantly. Only if substitution and/or new extractive technologies plus new and improved electrical systems are employed will the balance be maintained and copper prices remain stable. The population of other nations in the far-east is of the order of that of China and their standards of living are also rising.

Given the chain of copper metal and alloy production from mine to consumer, “bottlenecks” and “excess inventory” will probably arise even in a very stable market. This leads to price volatility which is exacerbated when supply is readily available or when demand increases during periods of high copper...
consumption. When supply exceeds consumption, figures 3.1.a and 3.1.c would suggest that volatility decreases but volatility is still evident.

Given the discussion of copper presented under the heading 3.1 “Copper Prices, Demand, Processing, Usage and Forecasts” above, it should not be surprising that the choice of a longer term copper price is difficult to make. The “intuition” of the mine planning engineer at a remote copper mining operation may be as valid as the expert working in the “city”. The Ernst & Young study by Downham and Williams (2008) regarding metal price forecasting cannot be ignored. This is especially true where Asian economies, especially China, are driving the demand and challenging the older European and North American economies.

For immediate estimates (say up to 6 months) copper price forecasting would appear feasible. First, the copper markets are for up to 3 months delivery, so the price is known for this period. Study of demand (Figure 3.1.k) and supply (Figure 3.1.h) can also be used to determine an approximate copper price for up to say 18 months. On a five year basis and longer, estimating “life of mine” copper prices is far more difficult. The best estimates would include analysis of world population and the variable standards of living of that population.

On a long term basis, the price of copper will rise to satisfy supply but there will be unexpected volatility lasting months to years. It must be remembered that placing a significantly large copper mine into production can take about 5 years, longer for primary sulphides, shorter for SW-EX heap leach oxides. Only technological change in mining, substitution and recycling can moderate the future rise in “real” copper prices. Also on a long term basis, inflation will increase costs from mining through to end use. To
maintain supply balance, copper prices must also increase at the rate of inflation. It is within this framework that the copper mine long range planner and planner of the feasibility of new mines must work.

3.2 Summary

The discussion in this chapter has led to the following conclusions:

- Ernst and Young appear to be honest in their conclusion that metal price forecasting is disappointing in providing metal price valuations.

- There are sudden peaks followed by returns to normal prices lasting from 1 to 5 years (e.g. 1974, 80, 89, 95, Figure 3.1.b). Prior to the year 2000, mine feasibility studies would find benefits from the inclusion of such “profit windows” and with 5 to 10 years between peaks, longer life operations would find these peaks beneficial, but the rationale for the peaks is hardly “bankable”. There would also be a perceived advantage in starting a copper operation close to the approach of a peak price. This is very difficult to achieve as it would require securing financing and board approval when prices are low! Nevertheless, a large mining company with many long life (+10 years) operations can benefit by placing low grade operations (requiring a high copper price to be profitable) on care and maintenance when copper prices drop. The mining engineer must utilize (say) 5 mine planning scenarios (see page 3-7 above and figure 3.2.a below) to run multiple mine plans at various copper prices and then restart these low grade operations quickly when the peaks occur.

- Supply and demand appear to be the governing factor for copper price. The London Metal Exchange (LME) stockpiles are inversely proportional to the copper price over periods of relatively stable copper prices with some up and down spikes. Major changes in price are caused by recessions/depressions, new purchasers, producers of the metal from concentrates (e.g. China), and technological change (e.g. the use of electricity, or a transition from internal combustion engines to
electric motors), or an increase in supply (e.g. technological change in mining/processing/smelting/refining or a price level where more scrap metal (e.g. from copper phone cables replaced by glass fiber, etc.) becomes available.

- The mine planning engineer should develop a set of plans for a range of copper price estimates. This strategy could give a corporation a dominant position in the future, figure 3.2.a.

![Figure 3.2.a: Author recommended copper price inputs into the mine planning process.](image-url)
Chapter 4

Grade Estimation, Recovery and Smelter Contracts

The objective of this chapter is to review the estimation of grade as an input into the long range mine planning process and demonstrate how grade estimates in the resource block model are converted into net revenue.

4.1 Grade Estimation

Grade estimation is a continuous process starting with regional and local geology, exploratory surface mapping and various electronic or ground vibration techniques which locate a potential target. Typically a junior mining company or exploration group from a larger mining company drills the area by the least expensive method available, often with reverse circulation drill rigs employing a down-hole hammer drill. Using this method, drill cuttings and dust are sucked up inside the hollow drill rods (drill stem).

The cuttings are removed by pushing cuttings and air up outside the drill stem, or (preferred) sucking the cuttings up the centre of the drill stem. Minimal water is used as a dust suppressant. Contamination of the samples from material washed in from higher up the hole can be problematic and is usually observed when a sudden increase in payable grades slowly decreases as the hole deepens. Drilling below the water table is also problematic due to the washing in of such material which is forced up inside the drill stem as drill sludge.

Once the deposit can be identified and grades are indicated to be sufficiently good, diamond drilling is the preferred method of proving up the deposit. Drilling for cores (short solid cylinders of unbroken rock as
found in-situ) from 25mm to 70mm (1 in to 2.5in) is directed across the dip to ensure that cores are representative of the deposit and surrounding waste and low grade. Assay lengths can be determined by geological inspection such that similar types of material are assayed independently, or a fixed length of core can be used.

Based on the observation of the author at open pit copper porphyry mines; The amount of material actually drilled and sampled in an individual 50mm diameter hole drilled down at 45 degrees and 425m long (300m vertical depth) might be 2.2 tonnes using a density of 2.6 tonnes/cubic meter. A deposit might cover an area of 1000m x 1000m and 300m deep with 100 holes at 100m intervals on a square pattern in plan, making 220 tonnes of sample. Sample reduction, if representative, would reduce this tonnage to (say) 22 tonnes for actual assay. The deposit measuring 1000x1000x300 cubic meters might contain (say) 800 million tonnes, and is represented by 220 tonnes of sample of which 22 tonnes might be assayed. Extra in-fill drilling in “5 on a dice” pattern might double the assay samples to 44 tonnes. Trusting that mines can estimate the deposit grades so well given the huge volume represented by such a small sample base is extraordinary!

The mitigating factor is that the sampling of each hole might be conducted on consecutive 1.5m segments making 300 individual assays per hole. Each sample of 8 kg might be represented by assaying 0.8 kg of material which in turn represents 2.7 million tonnes of material. It is the consecutive crushing and splitting (cone and quartering) (Gy, “Safety Line”, 1979) that determines the “accuracy” of the eventual grade estimate. The samples may, however, be subject to core losses during the diamond drilling process. High pressure water must be used to cool and lubricate the drill bit, and “muds” are added to the recycled
water to achieve this. Assays of this recycled water and mud clearly demonstrated higher grades than were being found in the core assays, and underestimation of the orebody resource was anticipated.

In order to confirm this, a raise was driven along two vertical drill holes, an original hole and a “twin”. The raise rounds were processed individually in a portable pilot mill, and the 3 grade estimates examined. The first conclusion was that the original diamond drill hole and its twin had surprising deviations in grade. The work of Soregaroli (1968, 1974) suggests that the ore is in very narrow short stringers with little alteration, and in large clay gouge zones depending on the period of mineralization of the quartz diorite/monzonite host rock. That the twin holes assayed so differently should not have been surprising. The raise rounds (2x2x2m or 20 tonnes) indicated a higher grade than the diamond drill holes, and it was concluded that core losses were undervaluing the deposit. Noranda Mines were convinced of this by the Brenda Consortium and invested in the property at the feasibility stage, taking a 48% ownership when the operation was deemed profitable.

The material originally contained in Chapter 4 is quite detailed, involving such as sample variograms, grade distributions, plots of exploration vs production drilling and grade tonnage curves. A section on selective mining units (SMU’s) and the development of SMU factors to determine the “child” SMU grades based on the “parent” block grade is described and blocks sized based on David’s (1977) recommendations. Gy’s (1979) sampling line and its impact on blasthole sampling is discussed. The software supplied as part of the Mine 326 and Mine 441 courses in the Buchan Department of Mining, Queens’ University at Kingston was used extensively.
The reasons for a systemic undervaluation of the blasthole data (and consequent undervaluation of the deposit) are investigated, and washing out of softer high grade material in the exploratory drilling appears to be the reason. Unfortunately, adding grade because it might have been washed away is hardly “bankable” and certainly not acceptable as part of a SEDAR submission to securities agencies. The information detailed in the original Chapter 4 is simply attached as Appendix A, and a less detailed description of the work conducted using the exploration and blasthole data follows (Sections 4.1.1 to 4.1.4 inclusive).

4.1.1 Grade Continuity or Variography

The hole spacing (exploration grid) at the grade estimation stage is, to some extent, dependent on the deposit type and size. The preliminary data is then used to test deposit continuity using the (semi) variogram (shown in figure 4.1.1.a), relative variogram (relative to mean grade squared), or some other related investigative tool (e.g. covariogram) describing increasing variability with distance between known sample data. The best, but not commonly used measure of continuity is the correlogram. It is conventional to refer to the more often used semi-variogram simply as “variogram”.

Figure 4.1.1.a: Idealized (semi) variogram and terminology. The axes are X distance and Y variance. At zero distance the Y intercept or nugget describes the variability between many collected and assayed samples of the “same” material. The maximum variance is the sill or population variance for the deposit (or a “local” volume within the deposit). The distance at which the variogram reaches the sill is the range beyond which samples are not correlated. For distances below the range, samples grades are more or less similar depending on distance. It should be noted that the sample data making up the variogram must be of the same size (support) and often of the same orientation and grade grouping.

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Initially, with minimal diamond drill hole assays completed, the drill spacing is so large that the variogram at short distances merely describes continuity along the hole. The data is usually composited by weighted averaging of adjacent core assays to such as a possible bench height. If the bench height is changed later, the data can be re-composited and variograms revised. When the drill spacing (usually in a large test area representative of both ore and waste) is small enough, the range of the relative (to grade squared) variogram can be determined. The drill hole spacing can then be set as some fraction of the variogram range. Figure 4.1.1.a shows the nugget (the variance between the collection and assay of material from the same core), population (deposit) variance between grades from similar sized samples (support), and the range beyond which sample to sample variance is the same as that of the parent variance. The prediction of grade from samples more distant than the range is not appropriate, as the variance between samples is the population (or “local”) variance. In this case the best estimate of grade is the average grade of the deposit (or locality).

**4.1.2 The Resource Block Model - Estimating Grades of Selective (or Small) Mining Units (SMU’s)**

David, M., (1977), in “Geostatistical ore reserve estimation”, Elsevier, Amsterdam, Netherlands, p 283, recommends that the block size be no smaller than a quarter of the drill spacing. If the rule is ignored and blocks are too small, adjacent estimates will be of similar value, in which case why make so many small block estimates whose grades are not usually found in practice? Modern geostatistical techniques such as probability kriging or simulation (Journel and Isaaks 1984), which are beyond the scope of this thesis, can also be used to estimate the grades (and tonnages) of all the blocks making up the orebody, and may be capable of estimating smaller parts of individual blocks referred to as selective (small) mining units or SMU’s. If the variogram has a long range and a very small nugget, grades of small SMU’s may be
reasonably well estimated. A pictorial presentation of SMU’s, blocks and equipment size is shown in figure 4.1.2.a.

A better definition of the SMU is given by Leuangthong, O., C. Neufeld and C.V. Deutsch (2003) (http://www.uofaweb.ualberta.ca/ccg/pdfs/2003%2027-smuselect.pdf ) “the block model size that would correctly predict the tonnes of ore, tonnes of waste, and diluted head grade that the mill will receive with anticipated grade control practice”.

Figure 4.1.2.a: Blocks and SMU’s and their relation to selective mining. The SMU’s are shown as smaller cubes 15x15x15m (9000 tonnes) which can be dug in a shift by a small cable shovel using “double back-up” of trucks. The black square on right represents a block containing 4x4 or16 SMU’s, and the block size is 60x60x15m (140,000 tonnes). Assuming exploratory drilling on 90m spacing, the larger block grade can be well estimated, but not the grade or location of SMU’s within the block unless more sophisticated kriging estimation procedures are applied. This is because the SMU’s are exactly 1/6 of the drill spacing, breaking David’s quarter spacing rule for estimation. The production drill is shown on the left, and the shovel is operating in “double back-up” mode with room for a truck left and right of the shovel. The figure is to scale. The figure was generated by the author with the assistance of Professor Blackwell using AutoCAD.

A block has a large internal variance because of its size and because it spans both high and low grades. An SMU has a small internal variance because it is small and is located in material of more similar grade (ideally either ore or waste). The block or SMU variance is described by the average variogram value within the volume. In a block there are several SMU’s and the variance “between” individual SMU’s would be large when a large block contains high and low grades i.e. when the variogram has a short range. As variance is proportional to the square of the grade (the Proportional Effect, Isaaks & Srivastava, (1989), pp 49-50), variance also increases when the block is of higher grade. Therefore the block grades (from David, (1977)), are well estimated as long as they are dimensioned greater than ¼ of the drill spacing. Because the drill spacing is based on some fraction of the variogram range (say ½), this “¼ rule of thumb” would appear to be appropriate. Many smaller mining companies use large drill spacing as a cost cutting measure. As will be seen, the money spent using obscure computer data processing methods to determine “bankable” resources would be better spent on extra drilling and assaying.

From these relationships, given that the block size is 60x60x15 m and the 16 contained SMU’s are 15x15x15 m, and the variogram is that for 15m composites in the ore-body under study, 16 factors can be estimated which, when multiplied by the block grade, provide the grade distribution and grades of the 16 contained SMU’s. This theory is described as “change of support” where the block and SMU are of very different sizes (support), and can be proved in practice using exploration and production blast hole data (Blackwell, G.H. (1998) “Geostatistics Course Notes”, Dept. of Mining Engineering, Queen’s University
at Kingston). Typical example results for copper with a block grade of 0.5 % Cu would have the lowest grade SMU as (0.6x0.5) or 0.3 % Cu, the highest grade SMU as (1.4x0.5) or 0.7 % Cu, and the mean grade of the 16 SMU’s as 0.5 % Cu. If the cut-off grade were 0.6 % Cu, as a block the material would be waste. By changing the support from 60x60x15m to 15x15x15m, three SMU’s in the waste block are ore which can be selectively mined. The SMU grades are individually estimated for the block grade and a list of 16 factors such that (block grade x factor) gives the grades of SMU’s. The average grade of the SMU’s must always equal the block grade, or a bias will be introduced.

For long range mine planning purposes the SMU’s and their grades are randomly placed as a grid inside the block. This is not accurate as the SMU’s also have a variogram based on (usually) 6 individual blasthole cuttings assays per SMU. However, for long term planning the knowledge of the existence of the SMU’s given they are only a maximum of (say) 60m from their true location is sufficient. The blast-hole drill cuttings will be used as the ultimate ore/waste decision making criteria.

Based on the authors observation at two world class mining companies (supported by the observation of Blackwell) one can state that the mining industry has been reluctant to accept the maxim of block size being related to drill spacing, preferring to use small blocks whose grade estimates are almost meaningless swathes of ore or waste seldom found in practice. This leads to inappropriate selection and small numbers of large equipment which cannot be mined selectively. The selective mining unit (SMU) is required for tonnage and grade purposes in long and short term planning. A typical definition of an SMU can be the tonnage which equipment can dig in a shift, usually 15m x 15m in plan for most large loading equipment. This definition implies correctly that blocks of “ore” will contain some waste and blocks of “waste” some ore, and the loader must be sized to differentiate the ore/waste boundaries. Costs are also an
issue as small equipment can dig more selectively but have higher operating costs and vice-versa. The
solution is to select equipment such that higher operating costs are more than balanced by better grade
control of small SMU’s (reduced dilution, increased recovery), resulting in increased head grades and
cash flow.

4.1.3 Confirming the Accuracy of the Resource Block Model

The long rang mine plan is based on the estimates of grade (and tonnes) from the resource model
(60x60x15m blocks) based on exploration data. The performance of this model must be evaluated (after
production starts) against the blasthole model (15x15x15m SMU’s). The blasthole model is also known in
the industry as the short range mode or the grade control model. In the Brenda case, the number of loaders
was based on loader moves, availability and spatially inaccurate inverse distance power grade estimates.
The shovels and trucks were located on the docks in Montreal awaiting payment by an overseas customer.
The customer could not raise the funding to buy the equipment, so it was purchased by Brenda. Luckily,
the equipment was quite suited to the work of mining the deposit.

A number of graphical methods can be used to complete the comparison of sample grades from the two
separate sources (exploratory diamond drilling and grades as mined and milled based on production
blastholes). Grade probability distributions, X/Y plot of grades, and comparative grade/tonnage curves are
the three primary means of comparison. The grade tonnage curve method is described in this research.

The Grade-Tonnage Curves

This curve provides the user with the fundamental worth of a mine (or bench in the 4860 bench case
described). The X axis is cut-off grade which can be transformed to revenue or cash flow quite easily. At
cut-off zero, the grade is the “deposit” or bench average and the ore tonnage is the total tonnes in the
bench. By mining material below cut-off as waste, the average grade for “ore” increases, and the tonnage of ore decreases. If this curve is incorrect, then the mine will be under or over-valued, possibly significantly.

The author used the Brenda Mines exploratory diamond drilling grade files and variograms to “krige” the grades of 60x60x15m blocks and then find the SMU grade distribution using the volume/variance relationship. The author also used the blasthole cuttings grade data to find the “realistic” grades of blocks and SMU’s by simple averaging of the contained blastholes. The two databases are independent of each other, even the assay laboratories were independent as the mine assay laboratory was built during mine construction.

The monthly average of blastholes sent to the mill was compared with the mill head grade and tonnes. The agreement between of the two was excellent over the life of the mine, due in part to the mine operations acceptance that production drill hole sampling was important to the long term viability of the project, especially in periods of severely reduced metal prices.

Two grade tonnage curves (Figure 4.1.3.a, Figure 4.1.3.b) were generated; first from the 60x60x15 block grades in the resource model (generated from kriging the blocks using diamond drill hole information); and second using the “parent” kriged block grade to estimate the SMU grades using only exploratory sampled data and comparing these SMU’s with the average grade of blasthole samples found in the SMU when drilled and sampled prior to blasting and production.
Figure 4.1.3.a: Grade tonnage curve for Brenda, 60x60x15m blocks. The grade curve shows the kriging grade for this block size is less than the blast-hole grade for the same 60x60x15m block. This loss of grade is probably due to core losses, but reduces to almost no difference at 0.5 % Cu cut-off. The tonnage curve shows a small loss of ore tonnes which is insignificant at a 0.35 % Cu cut-off. Generally it can be stated that bigger blocks, less than ¼ drill spacing, are well predicted. This graph was generated using software from courses in the Buchan Department of Mining, Queen’s University at Kingston.
Figure 4.1.3.b: Grade Tonnage curve for 15x15x15m SMU’s. Blasthole grades (BH) are the average of the ~6 holes in the SMU. The SMU equivalent grades (KG) are found using kriged parent blocks of 60x60x15m to estimate 15x15x15m SMU grades. Only exploration data and the exploration variogram is used for the KG plots, and as such is precisely the estimate that would be produced prior to mine feasibility and construction. The grades show that the KG SMU’s clearly underestimate the BH blastholes, undervaluing the deposit. The tonnage curve shows a closer match, but has a systemic over estimate of lower grade tonnes and underestimate of high grade tons. This graph was generated using software from courses in the Buchan Department of Mining, Queen’s University at Kingston.

The 60x60x15m kriging block model prediction of grades and tonnes (figure 4.1.3.a) is adequate given the shortcomings of the input diamond drill assays. This estimation is referred to as the “parent” block model, and the underestimate of grade is assumed to be core losses in softer high grade material.
The 15x15x15m model using kriging and the volume/variance relationship is inadequate. The original block model (figure 4.1.3.a) indicates that any estimate of SMU grade based on the kriged block grade will produce a systemic undervaluation. However, the undervaluation in figure 4.1.3.b is magnified by another factor. The blasthole production data cannot be conducted to Gy’s (1979) specifications because sample reduction in the field cannot economically produce the sample required by the assay laboratory. The sampling procedure consists of taking pipe cuts (a 100mm pipe is driven into the cuttings pile 12 times all round and dumped into a box), The box is then riffled into a second box such that 50% of the gravel, sand and dust is kept. The riffling procedure continues until the remaining sample of 1 to 2 kg is bagged and tagged by bench, blast and hole number in order to match the sample with routine survey of the blastholes. The blasthole data therefore has a higher variance due to the addition of the nugget. On a grade distribution plot this materializes as a steeper curve, high grades being overestimated and low grades underestimated. Consequently the grade curve of BH versus KG in figure 4.1.3.b should show a loss of KG grade becoming larger as the cut-off grade is increased. The tonnage curve can be expected to be similar to that of figure 4.1.3.a but the differences should be larger.

The process of:

- obeying David’s (1977) drill spacing criteria
- having an awareness of the core losses of high grades in diamond drilling
- accepting that the exploration data variogram can model the volume/variance relationship
- having an inability to follow Gy’s (1979) sampling “safety line” for production blasthole cuttings assays

has been demonstrated, and the resulting effects have all influenced the data to an extent. Shortcomings of the exploratory model without blastholes have been noted and explained. The process of model
development has, despite the shortcomings of both exploratory and blasthole data, provided a resource model satisfactory to the mine planning engineer at the feasibility and production stages.

4.1.4 Classification of Grades and Tonnes of Resources

In the mining industry the definition of ore is contentious. Good thought provoking definitions of ore include 'Ore’ is rock that may be, is hoped to be, is or has been mined and from which something of value may be extracted” (Taylor, H.K. 1991) and “the valuable solid material that is sought and later extracted from the workings of a mine for the hoped or expected (though not always achieved) advantage of the mine operator or for the greater good of the community” (Taylor, H.K., 1986). Perhaps “dug up for money” is equally fitting.

The classification of mineral resources generally has two components, increasing level of geologic knowledge and increasing confidence in economic considerations. These concepts are combined in the “McKelvey Box”, McKelvey, V.E. (1972) on which most is classification systems are based.
Figure 4.1.4.a: The McKelvey Box with geologic knowledge as (X) and confidence in economics as (Y). The “economic - identified" Measured, Indicated and Inferred” categories are common to nearly all natural resource classifications including coal, industrial minerals and oil as well as metalliferous mines and are fundamental to the NI 43-101 regulations.

The common categories into which resources are placed are “measured, indicated and inferred”. Given the need to protect investors from enthusiastic over-reporting to outright fraud, the Ontario Securities Commission (OSC) and Canadian Securities Administrators, with the input of mining interest groups, developed National Instrument 43-101 covering most aspects of disclosure for mineral properties. (Wikipedia: National Instrument 43-101 (2007 and updates to 2013)).

The National Instrument also defines who may prepare reports and draw conclusions, and the format of such reports. The reports are made available to the general and investing public in “SEDAR” which is the electronic system for the official filing of documents by public companies and investment funds across
Canada (and often internationally). Its purpose is “to provide protection to investors from unfair, improper or fraudulent practices and to foster fair and efficient capital markets and confidence in capital markets”. SEDAR (1997) “http://sedar.com/sedar/background_on_sedar_en.htm”

The report must be compiled by or under the direction of a “qualified person”. The definition of "qualified person" requires a university degree or equivalent accreditation in an area of geoscience or engineering rather than a membership designation in a professional association. A qualified person (QP) must also be subject to continuing professional development within the professional associations.

The placing of material into measured, indicated or inferred is still the opinion of the QP which is subjective. Generally for porphyry or epithermal deposits, the variogram (or similar) range is used to determine drill spacing. The resource category is then determined by such as

- the number of drill assay intersections used in the kriging estimate
- the number of intersections closer than (say) half the variogram range
- the requirement of close intersections in a majority of octants (upper and lower quadrants)

Only the last of the above requires data around the unknown volume. The number of intersections required is subjective, and using a number which provides the required result is common. This information is made public via SEDAR and the reader (investor or investor’s adviser) can draw their own conclusions.

A solution to the subjectivity problem is given by Blackwell (1988) where the kriging error (a natural product of the kriging process) is used in a similar manner to the Dilution of Precision in GPS survey measurement. It was found that misclassification due to sample geometry, distance away and number of
intersections was reduced. By using the kriging standard deviation (the square root of kriging variance) and making variogram sill values unity regardless of metal (copper porphyry or epithermal gold) it was found that these RKSD values could also be used to define measured (RKSD<=0.3), indicated (RKSD>0.3 and <=0.5) and inferred (RKSD>0.5).

4.2 Smelter Contracts, Concentrate and Metal Sales

Grade has to be converted to net revenue after in-pit dilution, process recovery and all smelter and transportation costs and losses. This is referred to as “net smelter return of rock in the ground” (or the Mine Economic Model) and is most useful to mine feasibility and planning engineers in that grade can be converted to net revenue directly with no more deductions other than “operating costs”. Subtracting all net costs (mine, mill, operating, capital, return on investment and etc.) provides the net cash flow before taxes. When the net costs equal the net revenue, the net cash flow is zero and the grade producing such zero net cash flow may be considered as the cut-off grade below which material is classified as waste or stockpiled marginal mineralization.

The economic model of a mine is defined by

- the required corporate rate of return of discounted cash flows or some other measure
- the revenue generated by the sale of products (copper concentrate, dore gold bars, copper anodes, and contained by-products)
- the cost of operating the mine including product transportation, smelting and refining.

The objective of the economic model is to easily convert grade mined to dollar cash flow at the smelter, i.e. the revenue obtained from the smelter or metal sales organization including losses for impurities, costs from concentrate transportation and gains from associated ‘payable’ products, typically precious metal
credits, minus capital payback, equipment replacement, return on investment and mine site (and corporate in most cases) costs. It does not generally include such items as replacement of resources and reserves.

Revenues in copper mines correspond to the net value of the ore after accounting for physical losses in the mill, concentrate transportation, physical losses in transit and smelting/refining charges and credits.

For gold mines, it is common to strip gold containing cyanide solution with carbon and precipitate the gold. The gold is then poured as dore bars from a small furnace. These bars typically contain silver and precious metals which pay the costs of transportation, refining and insurance. Physical losses in the mill are accounted for by assigning the expected mill recovery and are in the vicinity of 80-95% for copper sulfide ore and less for gold depending on the complexity of gold mineralization.

Smelting recovery accounts for losses that occur in the roasting of the “green” copper concentrate. In a leach pad operation, chemical losses are accounted for by assigning a leach recovery which is usually in the vicinity of 60-70% for oxide ore. Consequently just using the metal price does not provide the revenue, and in the case of copper the revenue to the mine might be as low as 60% of the published metal price. In the case of gold, silver and precious metals in the dore bars produced at the mine often pay for any associated costs.

Transportation losses (up to 5%) are usually included in the smelter recovery. Prior to the recent increase in demand for copper, the relationship between smelter and mine was governed by a legal document, the smelter contract.
Every mine once required this document in order to be sure of selling concentrates based on metal prices prevailing on a certain date during transportation (e.g. crossing the International Date Line). “Price Participation”, whereby the smelter receives more money when metal prices are high and less when low, are an integral part of such contracts and ensure that the smelter shares in good times and bad. Costs chargeable include basic smelting and refining, allowance for extra environmental and fuel costs and various other deductions including impurities (e.g. arsenic and lead in copper concentrate). Revenue comes from the sale of refined copper and precious metals. In the case of gold, gold plus copper, silver and precious metals.

Recently, the high demand for copper concentrates has enabled large mining corporations to sell on the open market at advantageous terms, negating the need for a smelter contract. Some mines and nearby smelters are owned by the same corporation and profit sharing is determined by corporate objectives including tax and royalty minimization.

One method of finding the value of mine site concentrate, anodes, etc. is to calculate the Net Smelter Return (NSR). NSR is the value of metals recovered from each tonne of ore minus the cost of smelting. Mine site costs (mining, milling, site, etc) are not considered at this stage as these apply irrespective of the grade of various metals, even in a poly-metallic deposit (Annels, 1991, pp115-7). For a copper mine with a Cu recovery of 84% producing concentrate containing 24% Cu, a smelting cost of US$370.00 per tonne might be applicable. The metal price (at the time of writing) is in the US$3.00 to $4.00/lb range. Using US$7700/tonne ($3.50/lb) as a copper price and a mine head grade of 0.5%, (0.84 x 0.5) or 0.0042 tonnes of copper metal will be produced per tonne of ore at a value of (7700 x 0.0042) or $32.34. This will be contained in (0.84/24) or 0.035 tonnes of concentrate costing (0.035x370) or $12.95 to smelt. The
NSR for copper at this particular project and grade is then (32.34-12.95) or $19.39 per tonne of ore. The NSR assumes the grade of concentrate and recovery to be constant which is not true when the head grade varies or when constant mill tailings grades are the norm.

A better system for the mine planning engineer is to charge all smelting and associated costs and mill tailings grade to the cost per tonne of rock mined as ore given the rock grade. The challenge is that costs are not reported in the same units. Milling (and mining) costs are usually reported on a per tonne milled basis and mining costs added on a cost per tonne milled basis using the stripping ratio. Smelter costs and precious metals credits are reported on a per tonne (Cu) or per gram (Au) (or equivalent unit, pound Cu or troy ounce Au) basis. Shipping and freight is quoted on a per wet tonne of concentrate basis and water contents of 7% (drum filter followed by natural gas) to 9% (drum filter) are typical. The task of accounting for costs and revenues after the product leaves the mine site requires a good understanding of the overall transportation and smelting business, and the collection of information in numerous measurement units, sometimes including currency conversions if US dollars are not appropriate.

An example follows for a conventional floatation mill where non-oxide copper is floated in cells using recirculated air (oxygen depleted, mainly nitrogen) and chemicals. For low grade high tonnage open pits, this requires a “Rock in the Ground” formulae of the type;

\[
\text{Mine Site Revenue} = \text{Head Grade } \% \text{Copper} \times \text{constant (1)} - \text{constant (2)}
\]

A typical example using a head grade \%Cu of 0.3% and constants of 50 and 6 respectively;

\[
\text{Mine Site Revenue} = 0.3 \times 50 - 6 = 9.00
\]
The constants vary depending on copper metal price, mine and smelter locations, smelter terms and mill and smelter recovery, and the result is the amount of cash the mine receives after all off site costs (smelter, refinery and all transportation) have been paid. The mine operation in the above example has $9.00 to pay for mining, milling, and G&A costs. Return on equity, capital replacement, and etc. may also be included.

The following is from “Applied mineral inventory estimation”, Sinclair A.J. and G.H Blackwell, (2002), pp24-26: and describes the fundamentals of the smelter contract, and what must be quantified to enable cash flows to be calculated.

Table 4.2.a: From Sinclair et al (2002) Table 1-8, Smelter Topics.

(TABLE 1-8: PRINCIPAL TOPICS DEALT WITH IN A SMELTER CONTRACT FOR THE PURCHASE OF METAL CONCENTRATES)

| (i) commodity market or publication that determines prices to be used |
| (ii) date and time for period for fixing the price |
| (iii) currency used for price determination and payment |
| (iv) method and time of payment |
| (v) weights and measures used and any abbreviations |
| (vi) approximate contents of the concentrates |
| (vii) estimated production and amount of notice for increase/decrease |
| (viii) regularity of shipments and identification by lot/car |
| (ix) types of rail car/vessel acceptable for shipment |
Table 4.2.b: From Sinclair et al (2002), Table 1-9 Items affecting cost estimates in shipment of concentrates.

(TABLE 1-9: FACTORS TO BE QUANTIFIED IN ESTIMATING COSTS RELATED TO SHIPMENT OF CONCENTRATES)

- (i) road freight
- (ii) rail freight
- (iii) ocean freight
- (iv) loading
There are always difficulties with currency exchange rates. The currency at the local mine may change in value against the US dollar, as may the smelter costs when the smelter is located in a foreign country. The mine planning engineer will probably have to work with several currencies, and at present, the United States dollar is the base “reserve currency” in which mines in less developed countries report.

4.3 Process (Mill) Recovery

Mill recovery defines how much of the product being mined and sold is lost at the mine-site because not all the valuable component minerals (or metals) can be separated from the gangue or waste. The definition of mill recovery is given by (Pryor (1965), Mineral Processing, Elsevier, Amsterdam, Netherlands, pp 259-261) for a two product (concentrate and tailings) process;

\[
\text{Recovery (\%)} = \frac{100 \ c \ (f-t)}{f \ (c-t)}
\]

where

\[
c = \text{concentrate assay value (\%)}
\]

\[
f = \text{mill feed assay value (\%)}
\]
A typical example for a copper sulphide mill has a feed assay of 0.4% Cu, concentrate of 24% Cu and tailings of 0.025% Cu.

Recovery is \(100 \times \frac{24 \times (0.4 - 0.025)}{0.4 \times (24 - 0.025)} = \frac{100 \times 9}{9.59} = 93.8\%\)

Developing a recovery model for each process and ore type combination is critical for planning and forecasting, and recovery calculations where two or more concentrates are produced are more complex. It is essential that assay grades of feed, concentrate and tailing, and recoveries are provided to the planning engineer both daily and at month end. Problems exist and adjustments must be made. For example the mill conducts a “metal balance” at each month end which includes the tonnages of dry material as feed, concentrate and tailing. At some operations, principally gold, a “mine call factor” may be used to balance the mine and mill feed tonnages and grades. Such a factor, usually about 0.95, is almost always the result of over reporting by the mine rather than under reporting by the mill. Similar problems exist between mill and smelter, but the smelter contract defines the error between them and an umpire assay can be called for.

Based on recovery and/or tailings grade models, an equation is developed for each process and ore/stockpile host rock type. This equation depends on the interaction of mineralogy and host rock in the extraction processes. Recoveries must be applied to the entire ore/stockpile block model, and can also be applied to the waste blocks if there are problems such as acid leaching from waste piles. In the block model data-base, recovery is added as another field using a simple program described as a “script” by
most software vendors. (A script is a program or sequence of instructions that is interpreted or carried out by another program rather than by the computer processor as a compiled program.)

Mill recovery is usually expressed as a function of sulphide mineral and oxide mineral content. The recovery is reduced by a factor based on the acid soluble percent in the block because acid soluble material does not float well in the mill and typically:

\[
\text{Mill recovery} = 95\% \text{TotalCu} - 65\% \frac{\text{AcidCu}}{\text{TotalCu}}
\]

It should be noted that the total copper and acid soluble items are stored in the model database to which recovery is added by the “script”.

The major changes in copper sulphide flotation involve significant increases in the size and design of float cells, and the treatment of oxide copper. Crushing and grinding have been improved by the introduction of computer sensors to increase throughput and downstream recovery.

The processing methods used to extract gold have changed significantly (Marsden, J.O. and C.I. House, 2006), Chemistry of Gold Extraction (Second Edition). Littleton, CO: Society for Mining, Metallurgy and Exploration, Inc., Littleton, Colorado. 651 pp. ISBN: 9780873352406, pp12-16). In an update to “Thirty years of turbulent change in the gold industry”, Figure 3, page 59 (Flemming, C.A., 1998, CIM Bulletin Nov/Dec., vol. 91, No. 1025, pp55-67) the changes are indicated for the years 1990 and 2006 as shown in Figure 4.3.a., and discussed in the following paragraphs.
The Merrill-Crowe process is applied to high grade gold ores which are crushed and milled (with gravity separation if applicable) prior to cyanide leaching. The solution is then treated with zinc powder and the zinc dissolved and gold liberated and precipitated in the absence of oxygen. The process has often been replaced by competing processes due to the depletion of high grade ores, and Flemming, 1998, notes that if less were spent on research into deep mining and more into processing, a better outcome for the South Africa industry might have resulted.

Refractory ores are sulphide bearing and also may contain graphite (carbonaceous) material resulting in low recoveries due to trapping of gold in carbonaceous material in the sulphides, or from consumption of cyanide by other metals (e.g. copper). This is referred to as “preg-robbing” which increases reagent costs. Efficient and cost-effective treatment of these ores is by prior roasting, leaching, pressure oxidation and bacterial leaching replacing the conventional approach.

Heap leaching with cyanide solution is suitable for less complex low grade deposits which have replaced the now less common high grade ores. It is a relatively simple process requiring drip feed or sprays onto heaps of ore (pads) which may be run-of-mine or crushed. Once the gold recovery no longer pays for the chemical reagents, the pad is discarded as waste, or lightly blasted or moved and rebuilt to allow the reagents (sometimes simply water at this stage as enough cyanide remains in the material) to pass through different faces of the rock or gravel to access the gold.
Figure 4.3.a: Changing trends in gold processing from 1970 to 2006. In 1970, South Africa was paramount as a gold producer via high grade Rand type deposits employing the Merrill-Crowe extraction process. By 2006, China has taken over as the paramount producer, tending to use CIP/CIL/CIC extraction processes. (An update of “Thirty years of turbulent change in the gold industry”, Figure 3, page 59 (Flemming, C.A., 1998, CIM Bulletin Nov/Dec., vol. 91, No. 1025, pp55-67))
The other processes CIP (Carbon-in-Pulp), CIL (Carbon-in-Leach), CIC (Carbon-in-Column) use granulated activated carbon to adsorb gold dissolved from ores as a concentration step (loading) which is separated from the ore slurry by screens. Jacobi Carbons AB, Sweden, a producer of activated charcoal from coconut shells, describe the CIP, CIL and CIC processes in the web page (http://www.scribd.com/doc/43829076/46580-Gold , 2011). Later in the process, the gold-loaded carbon particles are stripped of their gold (unloading) into an aqueous solution (electrolyte) by a reverse reaction and sent to electro-winning for recovery of metallic gold. Success of these approaches is related to general cost efficiency due to the technical ability of activated carbon to concentrate gold species that may be leaching out at very low (weak) strength in a variety of process environments (low-cost heap leaching, tank cyanidation etc.).

Carbon in pulp (CIP) involves crushing of the ore to enable efficient leaching of the gold. Classified ore/water slurry from the milling circuit is treated by thickeners where the underflow gold bearing pulp is then pumped to a series of leach vessels where the gold is leached out of the ore using sodium cyanide (NaCN) solution. Sodium hydroxide (caustic soda) or calcium carbonate (lime) is used as a cyanide guard to maintain the pH at >10.5. Air is sparged into mechanically agitated leach vessels to provide oxygen which is essential for the dissolution of gold. The leached pulp is transferred to a series of mechanically agitated CIP contactors, each containing activated carbon. The loaded carbon is acid washed before being transferred to the elution circuit. Using the Zadra method, the loaded activated carbon is stripped and fed directly to a series of electrowinning cells, and the gold depleted solution is then returned to the stripping column. This continuous recirculating technique is commonly used to elute gold from the activated carbon.
The initial processing step of the carbon in leach (CIL) process is similar to CIP. The gold bearing pulp is then pumped to a series of CIL contactors where the gold is leached out of the ore using sodium cyanide (NaCN) solution. The leached gold is simultaneously adsorbed onto activated carbon.

A series of mechanically agitated contactors are employed along with screens to allow the pulp to flow from one contactor to another while at the same time retaining the activated carbon. The activated carbon is transferred, countercurrent to the flow of pulp, using recessed impeller pumps or air lifts. Loaded activated carbon is then taken from the CIL contactors, washed in acid to remove calcium and base metals, and washed again. The activated carbon is then transferred to an AARL elution circuit where the gold is then stripped from the activated carbon with pure water and the resulting concentrated strip solution is treated by either zinc precipitation or electrowinning to recover the gold.

Gold heap leach operations use carbon-in-column (CIC) circuits. The pregnant solution from the pond containing the heap leach run-off is pumped through vertical columns containing activated charcoal. The permeability of the ore and the percolation of the leach solution through the heap are considerations when using CIC. In the CIP, CIL and CIC methods, the carbon is then washed, stripped of gold and then thermally regenerated at 600-700 degrees C. Care must be taken when organics or oils from pit equipment contaminate the carbon. This batch treatment technique is commonly used to elute gold from activated carbons in CIL, CIP and CIC circuits.

Operating costs for gold operations using a number of techniques are given by Camm (1991) in Figure 4.3.b. The use of energy in autoclave and roasting give these processes the highest operating costs followed by Merrill-Crowe CCD methods. The heap leach and more especially SX-EW type deposits have significantly lower unit costs, and hence their popularity.
Marsden, J.O. and C. I. House (2006) give recoveries of gold between 50 and 90% with 80% as an acceptable average. Recoveries are highly dependent on fineness and associated minerals especially where gold is entrained. Camm (1991) claims 89% for CIP, 93% CCD (Merrill-Crowe), 88% for float leach model, 93% for gravity and 70% for heap leach. Throughput for cyanide processes is of an order larger than for gravity separation and of similar unit costs. Processes involving energy such as roasting and autoclaves have higher costs.

4.4 Economic Models

For a multi process operation each mined SMU has five possible destinations from the pit:

1. Crusher to Mill
2. Crusher to Leach Stockpile
3. Leach Stockpile(s) (ROM)
4. Low Grade Stockpile(s)
5. Waste Stockpile(s)

Mined blocks, however, can be sent to one or more of the above destinations. Unfortunately, in the mining industry, the term “block” is often used to describe “SMU”. Where the two are of the same size and based on the same origin, and/or the use is for tonnage rather than grade calculations, this is acceptable. The mining industry is still using the term block to erroneously describe grade, and this thesis from this point on follows this convention.

In order to determine which blocks (SMU’s) to mine and where to send the mined blocks (SMU’s), each block (SMU) in the model must be assigned values. The value of each block in the model is a function of
a number of variables. These variables will directly influence the size of the pit, life of mine schedule and ultimately the NPV or IRR of the operation. A block value calculator must be executed in order to assign a net dollar value to each block in the model. For gold, the silver content, and some precious metals, are usually measured from the dore bars. For copper, the gold and silver in the copper concentrate might be measurable, but may be too small to measure in the mined ore. The mine is therefore often dependent on the honesty of the smelter to report precious metal contents.

The block value is defined by the following equation:

\[
\text{Block Value} = \text{Block Revenue} - \text{Costs} \quad \text{or} \quad = \text{Block Cash Flow} - \text{Costs}
\]

If Block Value < 0 : Block Value = cost of waste

Generally if the Block Value is negative it is given the cost of waste mining (drill, blast, load, haul, dump and where necessary, environmental).

Block revenue is the gross revenue obtained from selling the final product and is a function of the following:

1. Metal(s) price(s)
2. Metal recovery (for all processes)
3. Smelter recovery
4. Refining (if required)
5. Credits (other payable metals)
6. Discount rate (if used)
7. Reserve Replacement (if used)
The cost associated with mining a block is a function of process assignment. A mill block will have mining and processing cost, a leach block will have mining and leaching cost and a waste block will have mining costs (and environmental and etc., if applicable). At some mines, with very erratic grade values, it is common to route blocks based on “least loss” or “best value” at the short term planning stage. Each block is valued for all possible destinations and the block is assigned to the destination with the highest positive value or the least negative one. (Anderson, M.J. (1999)), (Rossi and Parker, 1993).

Costing in mining is a very intensive topic with numerous items. For the purpose of long term planning the costs must be summarized to include the following, and an estimate of % of total cost is included:

- **Mining cost (10%)**
- Incremental haulage cost per bench (variable, 0 to 40%)
- **Processing cost (for mill, ROM leach and crush leach) (60%, 10% and 20% respectively)**
- Transportation, Smelting and Refining cost (5-15%)

A block value calculation could range from a simple equation with 2 variables to complex depending on the operation. As an example a block value calculator for a copper porphyry deposit containing copper and molybdenum products and gold by product will be discussed. Mill, ROM (run of mine) leach, crush leach and waste are possible destinations. For this example, stockpiles will be assumed to bear the same costs as waste).

Table 4.4.a shows sample inputs for a porphyry copper deposit containing copper and molybdenum metals with mill and leach processing. Figure 4.4.a maps an algorithm that calculates and routes blocks based on the best value process.
Using the following paragraphs and tables as a template, sampling, grade estimation and verification, smelter contracts, and an NSR or “NSR of rock in the ground” model completes the process of estimating grade and converting that grade to cash flow (revenue) before operating, capital payback, shareholder dividends and etc. costs.

Table 4.4.a: Block Value Calculator Input Items.

<table>
<thead>
<tr>
<th>Item</th>
<th>Units</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Metal Price</strong></td>
<td></td>
</tr>
<tr>
<td>Copper market price</td>
<td>$/lb</td>
</tr>
<tr>
<td>Molybdenum market price</td>
<td>$/lb</td>
</tr>
<tr>
<td>Precious metals credit</td>
<td>$/lb of Cu</td>
</tr>
<tr>
<td><strong>Mining Costs</strong></td>
<td></td>
</tr>
<tr>
<td>Mining cost</td>
<td>$/Ton</td>
</tr>
<tr>
<td>Incremental haulage cost</td>
<td>$/bench</td>
</tr>
<tr>
<td>Sustaining Capital Cost (Mining)</td>
<td>$/Ton mined</td>
</tr>
<tr>
<td><strong>Milling Costs</strong></td>
<td></td>
</tr>
<tr>
<td>Milling cost</td>
<td>$/ Ton</td>
</tr>
<tr>
<td>G&amp;A cost</td>
<td>$/Ton</td>
</tr>
<tr>
<td>Sustaining Capital Cost (Milling)</td>
<td>$/Ton milled</td>
</tr>
<tr>
<td>Downstream Costs and Credits</td>
<td></td>
</tr>
<tr>
<td>-----------------------------------</td>
<td>-------</td>
</tr>
<tr>
<td>Smelter conversion cost</td>
<td>$/lb of Cu</td>
</tr>
<tr>
<td>Smelter recovery</td>
<td>%</td>
</tr>
<tr>
<td>By Products Credits</td>
<td>$/Lb</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Recovery</th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Copper Mill Recovery</td>
<td>%</td>
</tr>
<tr>
<td>Moly Mill Recovery</td>
<td>%</td>
</tr>
<tr>
<td>Copper Leach Recovery</td>
<td>%</td>
</tr>
<tr>
<td>Smelter Recovery</td>
<td>%</td>
</tr>
</tbody>
</table>
Figure 4.4.a: Flow diagram of block routing. The diagram includes mill, leach pad and waste (which would include stockpile). The values are stored in the block model which must be updated as inputs such as metal price and time (interest rates) change.
4.5 Summary

The key findings from this chapter are:

- The financial analysis of the estimated grades is the foundation of the long range mine plan value.

- The basis for grade estimation in long range planning is the resource block model. This model is based on the exploration data and the planner should consider that the grade estimates are based on a small number of samples that may be biased.

- The selective mining unit (SMU) is required for tonnage and grade purposes in long term planning. A better definition of an SMU can be the tonnage which equipment can dig in a shift, usually 15m x 15m in plan for most large loading equipment. This definition implies correctly that in blocks, “ore” will contain some waste and “waste” some ore, and the loader must be sized to differentiate the ore/waste boundaries. Costs are also an issue as small equipment can dig more selectively but have higher operating costs and vice-versa. The solution is to select equipment such that selective mining and better grade control of small SMU’s produces more cash flow than higher operating costs loose.

- The “net smelter return of rock in the ground” (or the Mine Economic Model) are used to convert the grade in the block model to net block value.

- The process of routing a block (to ore, stockpile or waste) is documented and demonstrated in the process flow map.
Chapter 5

Mine Life, Mill Through-Put, Capital Cost, Operating Cost, Equipment Selection and Productivity

This chapter completes the review of the profitability parameters in the spider diagram. The chapter starts with a review of mine life and mill throughput and their relationships to cutoff grade and reserve size. The second part of the chapter reviews capital and operating costs as they relate to mine planning. Finally equipment selection and productivity, as an input to mine planning, are reviewed.

The starting points for mine life and throughput are published papers and a set of rules written and developed by H.K. Taylor (1972, 74, 77, 85, 86, 88 and 91). It should be understood that Taylor studied operating mines and his rules are a result of reviewing North American open pit copper porphyry’s at the time. Taylor’s does not claim the rules to be optimal, but are a “rule of thumb” and a starting point which will produce a working model for further analysis and discussion.

The plant size (and mine size) and throughput are determined by financial analysis of the project, usually involving several initial scenarios at the scoping feasibility stage, followed by detailed selection of a mine design(s). In order to start the process, a relationship between “resources” and annual production determines the mine life and production rate or daily throughput. (Note that the word “resources” is used because the word “reserves” implies that a profitable and practical mine plan has been developed as per NI 43-101). The “annual production - mine life” relationship was investigated for a number of mines by Taylor’s (1977) who produced a “rule of thumb” or guide known as Taylor’s rule;
Tons per year = 5.0 * Tons of Reserves $^{0.25}$

Or  Tons per year = $(1 +/- 0.2) \times 6.5 \times$ Million Tons Reserves $^{0.25}$

From which

$$\text{Life (years)} = 0.2 \times \sqrt[4]{(\text{expected ore tonnage}/1.102)}$$

or  $$6.5 \times \sqrt[4]{(\text{expected ore tonnage (millions)}/1.102)}$$

where the tonnage may be short or metric as the rule is within $(1 +/- 0.2), (+/- 20\%)$.

Capital cost is determined by the size of the processing plant, the processing method and mine location.

For heap leach, the capital cost of the plant may be approximately equal to the mine equipment capital cost. As the size and complexity of the plant increases (e.g. high pressure leaching of gold; copper sulphide comminution; flotation; etc.) the plant cost might easily be ten times that of the mine equipment.

Capital costs for gold operations are given by Camm (1991), and leach or process plant costs for copper are similar.

Although the costs for gravity separation are low, throughput may be inadequate. Merrill Crowe (CCD) and Autoclave/CIL have the highest capital cost of the large throughput processes.
All costs are subject to inflation and cost over-runs when, for example, shortages of labour or increases in prices of steel and concrete occur. Given the impact of capital costs on project viability, (figure 2.5.a) it is important to estimate these costs as closely as possible as the project progresses. Delays in construction might double capital costs every 10 years. The commissioning of a mine in say central Africa or the Andes is logistically more challenging and therefore more costly than a mine in the southern interior of British Columbia. An example is Barrick’s Pascua-Lama which was estimated at $3 billion in 1991 is now $8 billion and rising, (Jordan, P., Globe and Mail, 11 April 2013, pp B1 and B11.) However using doubling of costs every 12 years (6% inflation) the figure might actually be $14 billion.

In order to determine operating costs, the type of equipment and suitability for the task must be known. To accomplish mining, other than artisanal, mechanical equipment is required which must be maintained by scheduled and breakdown maintenance. The availability and utilization of such equipment has a major bearing on the mine planning process and on its operating costs. Most mining operations follow a process of drilling, blasting, loading and hauling. A few operations include primary crushing of the hauled rock as a mine function to ensure mine scheduling of ore, and primary crusher costs are part of mining. The processing and upgrading of the rock to produce a concentrate, or using impregnating solvents, runs under the general description of milling (or processing). The detail of milling (processing) is beyond the scope of this thesis other than where the processing plant demands that mining must be controlled by mine planning of individual ore types; for example, the Kemess mine in northern B.C. where the mine produces oxide or sulphide rock where both exist in the ore-body. Here the mill can be adjusted every say 3 months to handle oxide instead of sulphide and vice-versa. Other examples involve mining ore with or without byproducts or deleterious substances such as lead. In such examples the role of mine planning is
paramount in providing mills with the most suitable feed and stockpiling the material which has to be processed separately ready for the next processing cycle.

5.1 Mine Life and Through-put

Mine life can be approximated using Taylor’s rule (Taylor’s, H.K., 1986) which has been revised based on newer data by McSpadden and Schaap (1984). The newer rule used 0.18 instead of 0.2 as follows:

\[
\text{Life in years} = 0.18 \times (\text{expected tons} / 1.102)^{0.25}
\]

Where expected ore is in millions of short or metric.

<table>
<thead>
<tr>
<th>Expected Ore (10^6 Tons)</th>
<th>Median Life (Years)</th>
<th>Range Of Lives (Years)</th>
<th>Median Output (TPD)</th>
<th>Range Of Outputs (TPD)</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.1</td>
<td>3.5</td>
<td>4.5-3</td>
<td>80</td>
<td>65-100</td>
</tr>
<tr>
<td>1.0</td>
<td>6.5</td>
<td>7.5-5.5</td>
<td>450</td>
<td>400-500</td>
</tr>
<tr>
<td>5</td>
<td>9.5</td>
<td>11.5-8</td>
<td>1,500</td>
<td>1,250-1,800</td>
</tr>
<tr>
<td>10</td>
<td>11.5</td>
<td>14.9-5</td>
<td>2,500</td>
<td>2,100-3,000</td>
</tr>
<tr>
<td>25</td>
<td>14</td>
<td>17-12</td>
<td>5,000</td>
<td>4,200-6,000</td>
</tr>
<tr>
<td>50</td>
<td>17</td>
<td>21-14</td>
<td>8,400</td>
<td>7,000-10,000</td>
</tr>
<tr>
<td>100</td>
<td>21</td>
<td>25-17</td>
<td>14,000</td>
<td>11,500-17,000</td>
</tr>
<tr>
<td>250</td>
<td>26</td>
<td>31-22</td>
<td>27,500</td>
<td>23,000-32,500</td>
</tr>
<tr>
<td>350</td>
<td>28</td>
<td>33-24</td>
<td>35,000</td>
<td>30,000-42,000</td>
</tr>
<tr>
<td>500</td>
<td>31</td>
<td>37-26</td>
<td>46,000</td>
<td>39,000-55,000</td>
</tr>
<tr>
<td>700</td>
<td>33</td>
<td>40-28</td>
<td>60,000</td>
<td>50,000-72,000</td>
</tr>
<tr>
<td>1,000</td>
<td>36</td>
<td>44-30</td>
<td>80,000</td>
<td>65,000-95,000</td>
</tr>
</tbody>
</table>

Table 5.1.a: Expected ore (short or metric tons in 000,000’s) versus mine life (years) and throughput (tons/day). The data collected are not precise enough to distinguish between imperial and metric values and ranges of mine life and throughput are given. (after Taylor, H.K., 1977, Mine valuation and feasibility studies, in Mineral industry costs, 2nd. revised edition, Hoskins, J.R. and W.R. Green eds., Northwest mining association, Spokane, WA 99201, table p7.)
Unfortunately, the term “expected tonnes” or “reserves” implies an “ore” cut-off grade which also has to be optimized to produce the best rate of return by whatever measure, and a stripping ratio which may change over the life of a mine. A degree of certainty that ore exists at a known location; what is ore at an assumed metal price, exchange rate and cost ranges, and what cut-off grades define any stockpiling of low grade ore will all be optimized in some manner.

The impact of newer technologies in processing and mining, and better definition of grades from in-fill exploratory drilling generally contribute to extended mine life and grade certainty. The increases in daily production at a reduced unit cost are also ignored by the rule, and well managed mines can double their life despite a 50% increase in throughput. A good example is Brenda Mines near Kelowna, B.C. where the original 90 million short tons (82 million tonnes) of ore with a 10-12 year life at 24,000 short tons per day (21,800 mtpd) of ore eventually became a 24 year life operation moving 34,000 metric tons per day. (Brenda Mines Factsheet, 1992).

Using short tons as in the original reference;
Examining Taylor’s (1977) original formulae for Brenda Mines using \( L \) for years of mine life and \( T \) the total short tons of ore to be mined over the life of the project, 90 million tons, (82 million tonnes) is;

\[
L = 0.2 \times T^{0.25}
\]

\[
and \text{ using the Brenda Mines example, } L = 0.2 \times (90,000,000)^{0.25} = 0.2 \times 97.4 = 19.5 \text{ years}
\]

or modified using 0.18 to replace 0.2

\[
L = 0.18 \times T^{0.25}
\]

\[
and \text{ using the Brenda Mines example, } L = 0.18 \times (90,000,000)^{0.25} = 0.18 \times 97.4 = 17.5 \text{ years}
\]

The project life at Brenda was 12 years.
Taylor’s original formulae using \( X_0 \) as the tons per day throughput and \( T \) as the total tonnes of ore to be mined over the mine life is;

\[
X_0 = 0.0147 \times T^{0.75}
\]

and using the Brenda Mines example, \( X_0 = 0.0147 \times 90,000,000^{0.75} \)

\[= 13,583 \text{ short tons per day}\]

The original tons per day processed at Brenda was 24,000 short tons.

The values found are of the correct “order”, confirming Taylor’s “rule of thumb”

Modifications to Taylor’s rule by Singer, D.A., W.D. Menzie and K.R. Long, (1998) at the US Department of Mines (USBM, now the US Geological Survey, USGS) are typical and where \( C \) is daily ore capacity in short tons per day and \( T \) is total ore tonnage in short tons;

\[
C = \frac{(T^{0.5874})}{2.404}
\]

again using Brenda Mines, \( C = \frac{(82,000,000^{0.5874})}{2.404} \)

18,500 short tons per day

Later modifications by Long K.R. and D.A. Singer, (2001) are a continuation of the USBM work on predicting tonnages and costs of open pit mines, and using the above definition for \( X_0 \);

\[
X_0 = 0.0236 \times T^{0.74}
\]

and again using the Brenda Mines example, \( X_0 = 0.0236 \times 82,000,000^{0.74} \)

17,000 metric tons per day

Further modifications by Long K.R., (2009) again continued the USBM work on Taylor’s original (1986) models,

\[
0.0147 \times (T, \text{expected tons})^{0.75}
\]

which can be re-written as;

\[C = b \times T^a\]
where $C$ is capacity in short tons per day, $T$ is reserve tonnage in short tons, and $a$ and $b$ are coefficients to be estimated.

Taking natural logarithms of each side of the equation;

$$\ln C = \ln b + a \ln T$$

Using ordinary least-squares methods for estimating the coefficients $a$ and $b$, the values found are:

$$a = 0.649 \quad b = -2.093 \quad \text{or} \quad C = 0.1233 \, T^{0.649}$$

$$C = 0.1233 \,(90,000,000)^{0.649}$$

$$C = 18,000 \, \text{short tons/day}$$

Long’s re-estimation of Taylor’s Rule relating capacity to estimated reserves used a large sample of mines representing many countries with free-market economies and several mineral commodities. Discussion in the paper explain that using various discounting methods, the tonnage mined per day exceeds the 2 dimensional area available, and that practicing planning engineers using optimization by discounting may produce results for daily tonnages in excess of those practically possible.

Nevertheless, the Taylor “rule of thumb” has become a starting point when estimating mine life and mill/mine capacity (through-put), and guides the feasibility and mine expansion planner towards a practical solution of mill throughput as demonstrated in the Brenda case. O’Hara, 1980 (as will be shown later), found that +/- 20% is acceptable at the “scoping” pre-feasibility stage.

Starting with Taylor’s rule, the USBM produced some detailed analyses using regression techniques to include capital and operating costs with the rule. Singer, D.A., W.D. Menzie and K.R. Long, (1998)
and also Long K.R. and D.A. Singer, (2001), revised work by Camm, T. W., (1991) to produce such capital and operating costs.

5.2 Capital Costs

Having decided the throughput at the scoping scale, capital costs must be estimated. The problem here is that capital costs are subject to all forms of inflation, and such inflation factors as the Consumer Price Index (CPI) may not be applicable. Taylor’s (1977), defines inflation as “a continuous depreciation of the value of money in terms of what it will buy” and that it is essential that all cost estimates for a project be estimated the particular date that a construction function is to take place, e.g. concrete is poured early but steel cladding takes place later. This allows cost conversion to a date in the future when the project starts and moves toward completion and is a practical application of such as Critical Path Methods (CPM) with Pert, Riggs, J.L., (1968).

It is possible to obtain the full detail of capital and operating costs from similar projects conducted by the corporations mine planning engineers, from corporate consultants, from equipment manufacturers, or from industry peers (classmates), and these sources, when available, are probably the best. Other solutions include buying subscriptions to companies selling such information, and “World mine cost data exchange” (http://www.minecost.com/ and http://www.minecost.com/faq.htm) and “Global info mine” (http://technology.infomine.com/reviews/MineCosts/welcome.asp?view=full) are typical. Other good sources include “SEDAR” documents from the OSC.

In the late 1970’s the mining industry became aware that capital cost estimation depended on a few practitioners, and that capital costs had a large effect on the profitability of mines. Several engineers
attempted to produce suitable mathematical models to rectify this situation, but all models suffer from inflation, and the material is outdated before it is published.

Mular, A. and R. Poulin (1998) produced “Capcosts, A handbook for estimating mining and mineral processing equipment costs and capital expenditures and aiding mineral project evaluations” and the following cost indexes are included:

- Engineering News-Record (ENR) for labour and building materials (steel, cement, lumber & labour)
- Marshall and Swift (M&S Mine/Mill) “all-industry equipment” for 47 industries
- Chemical Engineering plant construction (CE) for equipment, erection & installation, material, labour, engineering & supervision
- Nelson Refinery construction (NR petroleum industry) iron & steel, building material, miscellaneous, common & skilled labour

Engineering News-Record (ENR) builds its index on the following basis:

- 200 hours of common labor at the 20-city average of common labor rates
- 25 cwt. of standard structural-steel shapes at the mill price prior to 1996 and the fabricated 20-city price from 1996
- 1.128 tons of portland cement at the 20-city price
- 1,088 board-ft of 2x4 lumber at the 20-city price.
The graph of the ENR index in Figure 5.2.a shows a significant increase in inflation starting in 1970. Also shown are periods of similar rates of inflation, 1970-82, 1982-03 and 2003 to 2011, the last year with readily available data.

Figure 5.2.a: Engineering News-Record (ENR) inflation index, a typical index to convert costs estimated in one year to those of any other year. Note the increase in inflation after 1970, and the three distinct inflation rates, 1970, 1982 and 2003. The 1970-82 and 2003 to present rates are the highest. After McGraw Hill Companies Inc., Construction, ENR.com.

The “CapCost” recommended bases for calculating present costs are;

1. A phone call
2. \[ \text{Cost}_{\text{Now}} = \text{Cost}_{\text{Then}} \times \left( \frac{\text{Index}_{\text{Now}}}{\text{Index}_{\text{Then}}} \right) \]
3. \[ \text{Cost}_2 = \text{Cost}_1 \times \left( \frac{\text{Parameter}_2}{\text{Parameter}_1} \right) \]
4. \[ \text{Cost} = a \times \text{Parameter}^b \]

Two examples are given in Figures 5.2.b (shovels) and 5.2.c (trucks) in 1997 US$. 

97
Figure 5.2.b: Cable shovel costs (1997 US$) for various bucket sizes in cubic yards. An example of a 20 cubic yard (15 cubic meter) shovel (e.g. P&H 2800) gives a cost in 1997 US$ as $5,000,000 each.

For 20 yd$^3$ shovels the constants a and b are 535300 and 0.7395 respectively, making the price US$4,906,000 each. For 10 yd$^3$ shovels the cost is $2,938,000 each. The ENR indices for 1997 and 2011 are 5826 and 9100 respectively.

Cost now 20yd$^3$ shovel = cost then (index now/index then) = 4,906,000 * 9100 / 5826 = $7,663,000

Cost now 10yd$^3$ shovel = US$ 4,589,000, both of which values are reasonable.
Figure 5.2.c: Haulage truck costs (1997 US$) for various nominal load sizes in short tons. An example of a 200 ton truck (e.g. CAT 789D) gives a cost in 1997 US$ of 2,200,000 each.

For 200 ton mechanical drive trucks the constants $a$ and $b$ are 19570 and 0.8862 respectively, making the price US$2,292,000 each. For 200 ton electrical drive trucks the constants $a$ and $b$ are 72710 and 0.6513 respectively, making the price US$2,142,000 each. Large numbers of 100 ton electric drive trucks were built at costs approximately equal to mechanical drive, costing $1,459,000 (mechanical) and $1,159,000 (electrical).

Cost now 200 ton trucks = $3,580,000 (mechanical) and $3,346,000 (electrical)
Cost now 100 ton trucks = $2,279,000 (mechanical) and $1,810,000 (electrical) which are reasonable.
O’Hara, T.A., (1980), in “Quick guides to the evaluation of ore-bodies”, estimates capital and operating costs for underground and open pit mines with suitable graphs. Hustrulid, W., and M. Kutchta, (1995), in “Open pit mine planning and design” describe O’Hara’s work as “the classic paper”, and have updated the 1980 material. O’Hara is also the co-author of Chapter 6.3, Costs and cost estimation in Volume 1, SME mining engineering handbook (1992) and has updated his material there.

From O’Hara (1980,1992) costs are simply formulated as;

\[ Q = K T^X \]

where \( Q \) represents cost, \( T \) the tonnage rate, and \( K \) and \( X \) are constants for a particular set of conditions.

O’Hara minimized deviation from the known values taken from his personal experience of some 15 years, and added complexity to the formulations as required.

The cost of a combined mine/mill open-pit project is given by O’Hara (1980) (figure 5.2.d) as;

\[
\text{Cost US$} = 400,000 * (\text{daily tons milled})^{0.6}
\]

\[
400,000 * (24,000)^{0.6} = 400,000 * 425 = $170 \text{ million}
\]

For the Brenda project, 1980 costs are $170 million. Using the ENR index (figure 5.2.a) in 2013 this would be about $170*7500/3000 or US$425 million. 1969 costs would be 170*1000/3000 or US$56 million.

All these figures are of the right order of magnitude, and would form a basis for a capital cost estimate which would produce more detailed item by item costs.
O’Hara, T.A. and S.C. Suboleski (1992) in the SME Mining engineering handbook, 2 ed. Vol 1, H.L. Hartmann editor., p 416-417, also give costs for shovels, trucks and drills as shown in Figure 5.2.e in 1980-92 US$. Using 1986 as the ENR index for 1980-92, the index then is 4295 and index now 9100.

Shovel cost = No. 20 Queen Street West, Suite 1903

Shovels * 510,000 * Bucket Size (yd$^3$) $^{0.8}$

Truck fleet cost = No. Trucks * 20,400 * Truck Size (short tons) $^{0.9}$

Drill cost = No. Drills * 20,000 * Hole Diameter (inches) $^{1.8}$
For an operation with three 10yd$^3$ shovels, twelve 100 st trucks, and two 12 ¼ inch drill 1980-92 US$ costs are:

- 3 Shovels Cost = US$9,654,000 (US$20 million in 2011)
- 12 Trucks Cost = US$15,446,000 (US$33 million in 2011)
- 2 Drills Cost = US$3,637,000 (US$8 million in 2011)

Note that shovels of less than ~20yd$^3$ are no longer made, being replaced by front end loaders and/or hydraulic front shovels/backhoes.

Cost of this equipment for a 24,000 stpd ore and 26,000 stpd waste (50,000 stpd total) are given in Figure 5.2.e (SME Mining engineering handbook, 2 ed., Vol 1, H.L. Hartmann ed., p 416-417) and indicate the following costs in 1981-92 US$, and using the ENR index for 1986/87 of 4351;

- Shovels 3 $20,000,000 $42 million 2011
- Trucks 12 $18,000,000 $38 million 2011
- Drills 2 $3,000,000 $6 million 2011
- Total $41,000,000 $86 million in 2011

The example used is from Brenda Mines Factsheet (1992) and shows the number of shovels (indicated by shovel cost) at Brenda to be inadequate, which was the case. A fourth shovel (12 yd$^3$) was purchased about half way through the mine life. O’Hara notes that at the early stage of feasibility, costs are typically 20% +/- actual.

Knowing the ore tonnage (with consideration for such as porphyry, epithermal or flat lying tabular deposits), the mine throughput can be estimated and then capital costs estimated. Operating costs follow, providing the mine feasibility and planning engineer with a starting base case scenario. Such a scenario is
then examined item by item using various discounting methods to improve the valuation of the project prior to a decision as to whether to go ahead with the mine or not.

The initial study, from which a range of correct (at least based on the present industry practice) plant size and capital costs are estimated, can now be optimized. Halls, Bellum and Lewis (1969) describe means whereby the optimum ore reserves and plant size can be determined by incremental financial analysis. They describe how each increment of extra capital employed to increase the plant size must yield at least the minimum corporate acceptable rate of return. The plant size impacts the mine life, ore reserve and the cut-off grade, and incremental analysis can also be applied to existing plants in order to increase their throughput and size. Thus corporations can compare the starting of new mines or increasing capacity at older operations. Although the authors (Halls et.al.) claim that incremental plant size analysis replaces the use of a series of cut-off grades, there is still the work of assessing each increment of plant size capital and operating cost. The method will, however, show the optimal increment before and after which the discounted cash flow (DCF) reduces, and is another optimizing technique to employ in mining assessments.

Sabour, S.A.A. (2002) also uses incremental cash flow analysis as does Hajdasiński, M.M. (1988). However Hajdasiński compares present day “managerial and engineering judgment” with some decades of academic research and concludes that academia is at best equal to industry experience.
5.3 Operating Costs

Although operating costs may have a greater or lesser effect on project feasibility, they should be one of the best estimated parameters by experienced mine planners. In order to complete this section the author needed operating costs from existing mining companies. A Canadian mining company was approached by Professor Blackwell and provided cost data under the condition that it remains confidential. The costing data comes from several mines and are referred to as “Mine A”, “Mine B” and “Mine C”.

All of the earlier section 5.2 on capital costs has an impact on operating costs and productivity which, when handled with experience and commitment, can reduce operating costs significantly. This can be translated into higher financial returns, early repayment of capital costs, reduced cut-off grade or all three.

Mining costs are reported on both a cost per tonne of rock mined basis, a cost per tonne milled basis and a cost per equipment operating hour basis, and are covered in summaries such as the cost of drilling, blasting, loading, haulage, support equipment and general plus administration (G&A). Many U.S. and other nationality corporations and mines report on an imperial unit basis. Values for operating costs can be found in O’Hara and Suboleski, 1992 and updating with such as Engineering News-Record (ENR) is only acceptable for preliminary costing. The “World mine cost data exchange” and “Global info mine”, offer the latest data for a fee which is acceptable to companies and consultants working in the mine costing field.

A typical percentage cost breakdown (based on information from Mine A, though not accurate on a mine by mine basis) is as follows:

| Drilling | 8 |

105
General and Administration (G&A) may be on a tonne mined or tonne milled basis as long as it is accounted for appropriately. The allocation of G&A may be quite different between companies and must be examined carefully. Again from personal experience (Blackwell, personal communication) found costs at one mine (Mine A before closure and re-opening) to be lower than Brenda’s. On inspection it was found that the other mine had moved costs such as electrical cable moves from the mine to G&A.

General and administrative costs are calculated and assigned to each cost center: mining, milling and concentrating. Milling cost is on a per tonne milled basis and includes all costs from primary crushing to conveying through rod/ball mills and concentrating stages plus the tailings dam construction and maintenance and detail is beyond the scope of this thesis. In section 4.3 smelting and refining cost are conventionally reported on a cost per ton of concentrate basis and converted to a cost per pound of product (e.g. copper and/or gold), and usually include freight, losses and selling costs. In the case of leach pad extraction, concentrating is replaced by solvent extraction and electro-winning (SX-EW) costs. In the case of heap leach gold or copper, leaching cost is reported on a cost per tonne of ore and covers the cost of material placement (dozing), leaching setup (pipes and sprinklers), and acid or cyanide consumed, etc. It is common to recover gold and silver as by-products after smelting and during the copper refining stage.
as sludge, and to recover silver and other precious metals from dore bars. The by-product credits are reported on a gram (or troy ounce, kilogram or pound) of metal basis. (Mine B)

Major cost centers such as explosives (Slurries, Ammonium Nitrate Fuel Oil (ANFO), Aluminized ANFO and blasting accessories), drill bits, wearing steel, tires, engine overhauls, electrical motor rewinds and etc. are often contracted individually to one suitable supplier for a period of a year or so, and then re-tendered to find a cheaper supplier where possible. Test units from other suppliers are paid for at the average unit cost, e.g., drill bits in $/m. If a test bit fails prematurely the supplier is paid for the meters drilled. If a test unit lasts longer than the site average, the supplier is also paid for the bit at the average unit cost. The number of test units allowed is part of the contract, and allows competitors to make informed bids when called for.

The cost of mining includes all major items including the hauling of waste and stockpile material and the cost sum results in an eventual cost per tonne mined. This is converted to cost per tonne milled using the ratio of tonnes mined/tonnes milled which can be estimated based on the strip ratio. Table 5.3.a shows typical costs for an open copper porphyry mine.

Table 5.3.a: Typical costs for Mine C, a porphyry copper operation for 2011 are discussed below

(Personal communication with G. Blackwell, Confidential Mine C costs)

<table>
<thead>
<tr>
<th>Tons mined</th>
<th>57,517,000</th>
</tr>
</thead>
<tbody>
<tr>
<td>Tons milled</td>
<td></td>
</tr>
<tr>
<td>Waste &amp; Stockpiles</td>
<td>15,180,000</td>
</tr>
<tr>
<td>-------------------</td>
<td>------------</td>
</tr>
<tr>
<td>Strip Ratio (Waste &amp; Stockpiles/Ore)</td>
<td>2.80</td>
</tr>
</tbody>
</table>

**Unit Costs per ton milled**

<table>
<thead>
<tr>
<th>Category</th>
<th>Mine Operations</th>
<th>Mine Maintenance</th>
<th>Mine Engineering</th>
<th>Total Mine</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>3.54</td>
<td>1.75</td>
<td>0.24</td>
<td>5.53</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Category</th>
<th>Mill Operations</th>
<th>Mill Maintenance</th>
<th>Total Mill</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>2.89</td>
<td>1.38</td>
<td>4.27</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Category</th>
<th>Administration</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>0.91</td>
<td>10.82</td>
</tr>
</tbody>
</table>

**Unit Costs per ton mined**

<table>
<thead>
<tr>
<th>Category</th>
<th>Mine</th>
<th>Maintenance</th>
<th>Total</th>
<th>Percent</th>
</tr>
</thead>
<tbody>
<tr>
<td>Drilling</td>
<td>0.032</td>
<td>0.025</td>
<td>0.057</td>
<td>4</td>
</tr>
<tr>
<td>Blasting</td>
<td>0.232</td>
<td>----</td>
<td>0.232</td>
<td>16</td>
</tr>
<tr>
<td>Loading</td>
<td>0.058</td>
<td>0.101</td>
<td>0.159</td>
<td>11</td>
</tr>
<tr>
<td>Hauling</td>
<td>0.387</td>
<td>0.148</td>
<td>0.535</td>
<td>38</td>
</tr>
<tr>
<td>Roads, Dewatering &amp; Portable crusher</td>
<td>0.021</td>
<td>0.003</td>
<td>0.024</td>
<td>2</td>
</tr>
<tr>
<td>Utility vehicles</td>
<td>0.17</td>
<td>0.081</td>
<td>0.251</td>
<td>18</td>
</tr>
<tr>
<td>Services, Electrical</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>---------------------</td>
<td>---</td>
<td>---</td>
<td>---</td>
<td>---</td>
</tr>
<tr>
<td>&amp; General</td>
<td>0.09</td>
<td>0.119</td>
<td>0.209</td>
<td>15</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td>0.953</td>
<td>0.463</td>
<td>1.416</td>
<td>100%</td>
</tr>
</tbody>
</table>

**Equipment Productivity**

<table>
<thead>
<tr>
<th>Productivity</th>
<th>Availability</th>
<th>Use of Avail.</th>
<th>Efficiency</th>
</tr>
</thead>
<tbody>
<tr>
<td>Drills</td>
<td>100 ft/hr</td>
<td>94</td>
<td>90</td>
</tr>
<tr>
<td></td>
<td>(15,648 hr/yr)</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>(1,565,000 ft/yr)</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Shovels</td>
<td>6700 tons/hr</td>
<td>92</td>
<td>90</td>
</tr>
<tr>
<td>(P&amp;H 2800 and 4100)</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Trucks</td>
<td>600 tons/hr</td>
<td>89</td>
<td>92</td>
</tr>
</tbody>
</table>

“Use of Availability” is a measure of equipment redundancy. “Efficiency” is the hours worked of those available where available hours are about 22 hours/day, the remainder being lunch times, blasting delays and break times. The drilling footage is 30.5 m/hr (100 ft/hr) with 3 drills operating 15,648 hr/yr and drilling 477,000 m/yr (1,565,000 ft/yr) using BE 49R or equivalent units. The 4 shovels average 6080 tonnes/hr (6700 tons/hr) for P&H 4100 and 2800 shovels with 55 m³ and 30 m³ buckets respectively. The truck fleet comprises mainly Unit Rig 4400 with some 4000 and 3700 diesel electric units plus 930E and 830E Komatsu units, and truck capacities vary from 200 to 300 tonnes.
5.4 Incremental Cost per Bench

The incremental cost per bench is an additional haulage cost covering the cost for mining deeper in the pit. This is principally the cost of fuel and extra truck haulage time required to haul rock up the ramp and is reported as $/tonne/Bench or $/tonne/m. It should also account for any extra trucks required as a capital cost conversion to operating cost.

Because haulage is the main component of the mining cost, it is very important to estimate the increase in haulage cost as mining progresses deeper, and stockpile and waste lifts increase in height and distance from the pit. The incremental cost covers trucking costs loaded uphill from one bench to another on a (usually) 10% incline, and empty downhill and may be converted to per meter of vertical distance. Table 5.4.a represents a sample incremental cost per bench.

Table 5.4.a: Sample calculation of the incremental cost per bench.

<table>
<thead>
<tr>
<th>Item</th>
<th>Value</th>
<th>Units</th>
</tr>
</thead>
<tbody>
<tr>
<td>Truck Operating Cost</td>
<td>300</td>
<td>$/HR</td>
</tr>
<tr>
<td>Truck Capacity</td>
<td>250</td>
<td>Tons/Load</td>
</tr>
<tr>
<td>Cycle Time on Ramp</td>
<td>1.2</td>
<td>Min/Bench</td>
</tr>
<tr>
<td>Cost per minute</td>
<td>5</td>
<td>$/min</td>
</tr>
<tr>
<td>Cost on ramp per truck load</td>
<td>6</td>
<td>$/Bench</td>
</tr>
<tr>
<td>Cost per ton per bench</td>
<td>0.024</td>
<td>$/Ton/Bench</td>
</tr>
</tbody>
</table>

However, this is not the whole cost. To achieve the same production per loader, as depth increases so does the number of trucks required. As a rule of thumb, the number of loaders in the deepest parts of the pit must decrease as the working area decreases and depth increases. In the past it was assumed that as
one pit deepens and mines only ore, the freed up loaders (and any extra trucks) will be moved to the next pit expansion and that finally all of the trucks will be assigned to, and adequate for, hauling ore from the ultimate pit bottom. The number of loaders required will reduce significantly as a mine comes to completion at its ultimate depth.

Some loaders will be “cannibalized” for parts or rebuilt and moved to other mines. The total number of trucks may remain the same (+/-) in order to traverse each incrementally deeper bench and mine close to surface push backs. The assumption that all the remaining trucks will be sufficient for ore mining at the end of pit life is not proven, and if numbers of trucks are inadequate, second hand units will be purchased to maintain production. With the adoption of truck dispatch, the number of trucks required in the final year of operation can be estimated with some precision, enabling the mine to feed the mill at full capacity.

5.5 Equipment Selection

Equipment is purchased once the design and feasibility studies have been approved and corporate go-ahead given. “Corporate Culture” will determine whether the equipment will be required to last the mine life where that life does not exceed say 25 years, or if replacement with new equipment will be possible at every say 5 to 10 years. Most companies prefer proven standard equipment over new designs to avoid having to work with manufacturers to fix design flaws. Some companies purchase one of each of cable shovel, front end loader and hydraulic front shovel/backhoe. From the perspective of spare parts inventory, operating crew training and mechanical/electrical/hydraulic maintenance personnel training, this is not efficient. At the time of writing, delays in delivery of new equipment may make such multi-loader type decisions unavoidable if early mine start-up is demanded. For large companies buying a
loader every year regardless, the maintaining of an inventory of used units might be advantageous from an overall cash flow perspective. Many such mining companies are shipping rebuilt units worldwide based on corporate requirements.

Equipment used at porphyry/epithermal type open pit mines for blast hole drilling generally consist of large electrical powered blast-hole drills using pull-down pressure on an air cooled tungsten carbide inset rotary bit. Rotating rods with a hole widening reamer/stabilizer carry the pull-down and rotation to the bit. The shovels are conventional electric powered with a wire rope hoist for the bucket. The bucket crowd may be rack and pinion or dual rope. Hydraulic shovels and front end loaders are also used, especially at smaller operations. Such units have a shorter life (5-10 years) than cable shovels (15-25 years). Both drills and shovels can be diesel powered resulting in lower maximum power and additional maintenance, but cannot be avoided in very remote locations or where electrical power is unreliable. Trucks are off-highway units of 100 to 400 tonne capacity and may be diesel mechanical or diesel electric. Ancillary equipment includes tracked and rubber tired dozers, road graders, electrical cable pick up and lay units, road gravel spreaders and water trucks.

Decisions regarding which types to purchase are determined by any number of factors including:

- Suitability for purpose
- Cost of equipment
- Immediately available equipment at dockside abandoned by inability of another company to raise funding
- Parts availability and cost
- On-going maintenance costs based on experience at other operations
- Maintenance and parts contracts from manufacturer
- Personal preference based on experience
- Delivery time (may force use of alternates or two or more types/manufacturers)
- Reputation of manufacturer in the mining industry
- Observations at similar types of mines
- Discussion with peers at similar types of mines
- Software availability and usefulness (e.g. truck dispatch, on board maintenance indicators)

A general summary for drills, loaders, trucks and auxiliary equipment is essential in understanding the feasibility and planning processes.

Drills should have a mast tall enough to drill one hole in one pass including sub-grade (no steel changes) while equipped with a reamer/stabilizer above the drill bit. The blast-hole drill is initially chosen on the basis of unit cost per hole which favours large sized drills and large hole sizes of 251, 270, 311, 381 and 445 mm (9.875, 10.625, 12.25, 15 and 17.5 inch) diameter are typical. With a 15m (50ft) bench height, 251 to 381 mm is typical depending on drill pattern size, wall slope protection, rock hardness and need for more drill holes for cuttings assays and grade control. For lower bench heights of 5m (16 ft) high a hole size of 98, 114, 127, 159, 194 and 216 mm (3.875, 4.5, 5, 6.25, 7.625, 8.5 inch) diameter is used.

If grade control is a problem due to an erratic ore body, then the hole diameter may be reduced to 100 mm (4 inch +/-) and the bench height to 5m (15 feet), with a decrease in drill pattern size from 10m to 2m (33 to 7 ft) with commensurate increase in unit cost, lower productivity, and many extra drills. At the San Cristobal mine near Antofagasta, Chile, the initial mine equipment to drill an extremely erratic gold
deposit were two 250mm hole rigs. These proved totally inadequate when the operation was forced to mine 5m (16ft) benches at a 7m drill spacing for grade control. Small diameter drilling and blasting were contracted out and the large drill rigs sold. Blackwell, G.H., M. Anderson and K. Ronson (1999) describe this application. Without the modifications to bench height and drill spacing the mine would not have been viable because of ore dilution and poor recovery.

Although the primary use of a drill hole is to load explosives, it has a secondary use in providing a cone of cuttings on the surface close to the top of the hole for sampling. Such sampling may not provide the necessary unbiased and representative sample (Gy’s sampling line, Chapter 4, section 4.2.4). The effectiveness of the cutting of the initial sample is inadequate for cost reasons, and often a lower bench height and/or reduced pattern size and hole diameter are required to reduce the initial cuttings pile amount. The improved grade control will most probably pay for the increased drilling, sampling and assaying costs.

The choice of electric or diesel electric drill depends on circumstances such as electric power availability and reliability, and how often and far the drill has to be moved. Moving a diesel drill requires no cables to be laid out or electrical switchgear to be moved/opened/closed, but maximum power at the drill will be less. Alternatively, a dozer may tow the drill but the drill mast has to be lowered, and towing downhill is not recommended.

The bench height chosen is one of the major decision criteria in loader selection. Generally Mines Act regulations demand that the bucket can reach almost to the bench above to clear overhangs. This usually limits the choice to cable shovels or the largest tracked hydraulic shovels or front end loaders. Cable
shovels for hard rock applications where blasting may be less effective, can have more hoist/crowd power and less swing speed. Well blasted softer rock would have less powerful hoist/crowd but a fast swing part of the digging cycle. There seems to be little difference between rack and pinion versus wire rope crowd regarding power of the crowd, but changing cables and tightening them over the front and rear of the round stick, and feeding from the motor at the base of the boom (a point where damage from rock falls is prevalent) is a matter of personal preference. Rubber tired front end loaders (FEL’s) have the advantage of mobility at short notice to anywhere in the pit.

The FEL does not have the power of a cable shovel and cannot dig hard rock that has not been lifted and well fractured by blasting. The hydraulic shovel can operate in ‘backhoe’ or ‘front shovel’ configuration making it ideal for selective digging in erratic gold deposits. This is particularly true when digging half bench heights as a backhoe, a technique which also satisfies the digging bench height regulations.

Other considerations include bucket teeth (points) and adapters. Teeth are part of the “wearing steel” cost sector, one of a great number of cost sectors of shovel/loader maintenance. The teeth are placed on the adaptor with “C” shaped metal holders, and the adaptor placed similarly on the bucket lip structure. The five or more teeth can then be changed or switched easily, making maintenance simpler.

At the operating stage of a mine, the types and numbers of equipment are more or less fixed. Changes can occur if the present equipment becomes obsolete, wears out, or if an increase in production (expansion) requires additional equipment. The additional equipment may be new from the manufacturer, with possible delays in manufacture, and requiring upgrading of maintenance and operating personnel. Used equipment has the problems of ensuring that there is still sufficient life in the equipment, or whether to
rebuild/overhaul. A general rule is to keep additional equipment similar to present equipment unless costs of parts or inadequacy for purpose make the selection of different equipment advisable.

Changes have occurred in equipment purchasing, maintenance, repair and parts inventories (McKay, P., 2011, VP (retired) P&H shovels, personal communication). In the 1990’s many mining equipment manufacturers had their plants shut down on a care and maintenance basis. With the emergence of economies such as China which require raw materials, the mining industry has had a resurgence which started in the early 2000’s and has continued to the present. After re-starting their factories when orders materialized, equipment manufacturers now have insufficient capacity. This has led to takeovers such as Bucyrus Erie (BE) purchasing Marion in the shovel business, followed by Caterpillar purchasing BE to make a mining equipment company servicing nearly all the mining industry needs for equipment. There will be further acquisitions and mergers internationally as competition grows in the mining equipment manufacturing industries. The lack of availability of new equipment has also allowed alternate less well known equipment manufacturers to expand sales. With unfilled order books, these suppliers can supply equipment immediately, and can improve their products with the help of the mining industry, e.g. Liebherr trucks, loaders and etc., and become future suppliers of choice.

Other changes in contracted maintenance, and manufacturers on-site parts warehouses have occurred where the use of such services and parts consumption are sufficient, e.g. Brasil (iron ore), USA (coal and copper), Canada (oil sand) and Australia (bauxite, iron ore and coal).

For new equipment, manufacturers typically ask for a 20% down payment to place the order and hold a "slot" for the manufacture of a piece of equipment. Once the major components are completed another
20% payment is required. A further 15% is made at time of shipping and another 10% when major components arrive at site. The remaining third +/- is paid when the equipment operates for an agreed time without breakdown or “availability”.

Mining companies have, when appropriate, contracted maintenance to equipment manufacturers and maintenance specialists. Working with non-union labour under better conditions reduces costs to the contractor and enables fast call-out at night and weekends. Typically parts involved in 5% or more of down time, or having a long (months) procurement time, are stored in local warehouses under agreements between the manufacturer and mining company. Co-operation and openness between manufacturer and consumer has improved considerably. Guarantees of better than 80% availability on an hourly basis are typical stages where the maintenance contractor pays the mining company for production losses over periods of several years. These guarantees are relatively simple for trucks and drills, but not for shovels. Where one piece of equipment is involved, costs of such maintenance are high, but are reduced significantly where three or more similar units are involved. The maintenance contract forbids the use of “pirate” replacement parts, or use of parts on equipment from other manufacturers.

Shovel track shoes for example are an ongoing wear part and made in Scotland for P&H. Other items such as hoist and other motors can be rebuilt by the manufacturer and again become part of the contract. Modern electronics applications on shovels and trucks are complex but easily upgraded or replaced. The use of the “Wi-Fi internet” has enabled the manufacturer to centralize sophisticated diagnostics and supervise repairs on parts subject to cracking and breakage (frames, gantries, sticks and booms), or loss of power in motors. Semi-automatic digging cycles (Blackwell, G.H., 2013), bucket tooth loss detectors (teeth and adapters falling into crushers can result in severe jamming and breakage of the crusher), bucket
weighing (Veigroup srl, Vicenza, Italy, http://www.veigroup.com/home.html) and many other diagnostics are typical additional electronic functions.

Capital costs for a typical 50 cubic meter (65 cubic yard, 90 tonne) shovel are presently US$25 million and delivery 1 to 2 years. Matching trucks (350 tonne, 400 ton) cost nearly US$10 million each and the availability of new truck tires is so poor that China is starting to enter the tire and equipment market. (McGarry, 2007). The increases in capital costs will have a major effect on mine profitability as shown by such as “spider” diagrams (Chapter 2), and may make transport of units intercontinentally, or repair and overhaul, a better option than purchasing new units.

Ancillary equipment must be selected and made for mines, not for farm or municipal use. The major piece of ancillary equipment is the dozer which must be large enough to complete any required task. Rubber tired dozers keep the area around the shovels clear of rocks which will damage truck tires. Crawler dozers are used for reducing bench heights, pushing rock toward the shovel from the bench above (thereby reducing the bench height), and for ripping “hard toes” (possibly from miss-fires of explosives) around shovels. Dozers are also required to safely manage rock dumps, leaving a flat area suitable for truck tires and a berm (ridge) of rock to prevent trucks from backing over the dump.

A new mine may obtain new or used equipment, often from its own operations if a corporation operates many mines. This requires start-up to closure planning of many mines with each mine producing at corporate planned quantities of products. Consequently, at the corporate level, knowledge of when equipment will be required or available at the many mining operations under management is essential. Such corporate decision making may affect equipment selection. An awareness of productivity and
maintenance is a requirement for mine planning at the feasibility stage and also for short or long term planning.

5.6 Equipment Productivity

For the long term planner and the feasibility study, precise detail of what equipment goes where is not of importance except when a pit or pushback is almost at full depth. Shovels should produce at rates which are similar for similar equipment. Thus when the deep pit is nearing completion or a ramp is being driven in a pushback, there is little room to dig and operate trucks, and any extra shovels will have to be moved to an upper bench or pushback. There is now a major change in quantities mined from pit floor and pushback(s), and the long term and short term planner must ensure that sufficient ore (of lower grade) is available in the upper benches, and that average grade is sufficient to meet corporate profitability goals prior to moving shovels. Not only is it important that ore is present in the pushback, the available grade must also be considered at this important stage of shovel allocation.

The feasibility and long term planner uses two criteria in an attempt to ensure that equipment is placed correctly to maintain long term goals of ore grade, stripping ratio and production. These are “availability” and “utilization” of equipment. These are broadly defined as follows by Golder Associates (Marston Consulting), 2013, (http://benchmarking.smartmines.com/Articles/SMART%20Brochure%20Electronic%2020111003.pdf) and in better detail by Lukacs, Z.W. (2013), (http://benchmarking.smartmines.com/Articles/SMART%20Brochure%20Electronic%2020111003.pdf)

Fundamental definitions

- Calendar Hours: Actual Hours in a given time period, 8760 per calendar year (except leap year)
- Scheduled Hours: Hours in a given time period equipment is scheduled to operate, or [calendar hours – scheduled delays (e.g. holidays) + unscheduled delays (e.g. strikes, weather, power outs etc.)]
- Available Hours: Hours equipment is scheduled to operate.
- Utilized (Operating) Hours: Hours equipment is actually working at full production.

**Derived definitions**

- Mechanical Availability = 100 x (Hours Worked) / (Hours Worked + Repair Hours)
- Physical Availability = 100 x (Hours Worked + Standby Time) / (Scheduled Hours for Pit)
- Use of Availability = 100 x (Hours Worked) / (Hours Worked + Standby Hours)
- Effective Utilization = 100 x (hours Worked) / (Total Hours)

A study by [www.smartmines.com/benchmark/EqpBenchmark.htm](http://www.smartmines.com/benchmark/EqpBenchmark.htm) showed that different mining operations had the same name for different definitions and/or the same definition with different names. The reality is that “availability” is a term used by maintenance and “utilization” a term used by operations to justify a perception of their excellence/incompetence without causing a rift between the two groups which must work together to provide the best productivity.

As a general rule, drills work about 4500 hours out of the 8760 which are available each year. This is because drills, at least for small to medium sized operations, must move continually to drill patterns on various pushbacks with the mast placed down. Within pits drills can be towed by dozers. Some operations have flat-bed trailers to load the drill on and tow to other pushbacks using a large rubber tired dozer. Drills also have to move to accurately locate drill hole patterns. This causes wear on tracks and takes
time. Using “rule of thumb”, although a large drill may complete a hole in 20 minutes, the actual productivity (drilling utilization) would be about one hole per hour.

In mid-life, shovels may work 5500 hours per year and trucks (which can usually be driven to a central maintenance shop) 6000. These values represent utilizations of 52 (drills), 63 (shovels) and 69% (trucks) respectively, (Blackwell 1999). Such a definition of hours actually worked out of 8760 per year leaves no room for manipulation by different mines and mine cost centers, and should be preferred.

Newly designed equipment often has flaws which appear very quickly, reducing the utilized hours. This is usually compensated for by the manufacturer reimbursing the operating company at the cost per tonne of similar equipment. Truck tires and drill bits are charged in a similar manner when a manufacturer requests a trial of a new design. For example, drill bits in use at a mine cost $5000 each and drill 1000m. A bit from another supplier might drill 500m, so the mining company will pay the test supplier $2500.

Utilization above 75% (6500/8760) is problematic as the equipment does not have enough down time for scheduled maintenance (730 hours are lost from lunch breaks etc. leaving 1530 hours for maintenance, refueling, etc.). Another writer might assume that with shift change and lunch etc. breaks making 22hr/day (8030hrs), the maximum might be 81%, so being aware of the definition of utilization is essential. Another example has equipment with 98% availability and 75% utilization based on a 22 hour day. This means that in 24 hours, equipment operates (is utilized) for 16.5 hours/day (6022.5 hrs/year). Of the 22 hours, 2% or 0.44 hrs/day are spent in maintenance (161 hrs/yr). This leaves 7 hours per day (2577 hrs/yr) to be accounted for by meal breaks, refueling, moving to/from shop, weather and
temperature delays. Seen from this perspective, 98% availability and 75% utilization are hardly satisfactory, but are precisely the values (along with tonnes per operating hour) the mine planner needs.

This leads to a discussion on what “operating hour” really means. It could be the hours the operator notes on the time card, the hours the shovel or drill is turned on (compressors for brakes and flushing air), the hours the wheel on a truck is turning or the truck engine hours, etc. Dispatch records, where available, are probably the best “hours” to use but the electronic and radio transmissions are often compromised meaning trucks etc., “disappear” for several hours.

For long term planning, equipment can be scheduled on a utilization basis using the actual tonnes or holes per “hour”. Experience with one unit which best represents the operating hours would be recommended. The long term planner can then estimate the tonnes to be moved per period and then use utilization and tonnes/hour to estimate the number(s) and types of equipment required.

Unfortunately trucks become less productive with depth and haulage ramp length and gradient. This requires the use of further factors based on ramp velocity or some other method, Mining INFO (2013), Mining Technology > Materials Handling > Truck Haulage < Haul Road Guidelines, (https://sites.google.com/site/mininginfosite/miners-toolbox/materials-handling/truck-haulage/haul-road-design-guidelines). Table 5.6.a lists recommended gradients based on downhill empty braking.
Table 5.6.a  Recommended gradients based on required braking when travelling downhill empty
From Mining Info > Mining Technology > Materials Handling > Truck Haulage > Surface Mining.
(https://sites.google.com/site/mininginfosite/miners-toolbox/materials-handling/truck-haulage/haul-road-design-guidelines)

**Recommended Gradients** – Off highway large haul trucks

<table>
<thead>
<tr>
<th>Road type</th>
<th>Maximum grade</th>
</tr>
</thead>
<tbody>
<tr>
<td>Permanent surface haul roads</td>
<td>7%</td>
</tr>
<tr>
<td>Permanent in pit haul roads</td>
<td>10%</td>
</tr>
<tr>
<td>Temporary surface haul roads</td>
<td>7%</td>
</tr>
<tr>
<td>Temporary in pit haul roads</td>
<td>10%</td>
</tr>
<tr>
<td>On-bench roads</td>
<td>10%</td>
</tr>
<tr>
<td>Light vehicle roads</td>
<td>20%</td>
</tr>
<tr>
<td>Major bends/Switchbacks</td>
<td>0%</td>
</tr>
</tbody>
</table>

5.7 Summary

This chapter concludes the review of the profitability parameters in the spider diagram (Chapter 2). The key findings are:

- Taylor’s rule and its modifications are adequate for use as a starting point for mill throughput and mine life determination. Taylor’s rule will be tested on the Brenda deposit in Chapter 7.
- O’Hara’s formulas for estimating CAPEX are presented and are adequate as a starting point. The historical CAPEX can be adjusted for inflation to today’s dollars using the ENR or other index. This method will be utilized in Chapter 7 to estimate CAPEX for various mill sizes.
The long range planning process requires the input of operating costs. The discussion in the chapter demonstrates the complexity of summarizing these numbers to a figure that feeds into mine planning. A real life example of cost breakdown is demonstrated and discussed.

To ensure that equipment is placed correctly to maintain long term goals of production, the long term planner can use two criteria: “availability” and “utilization” of equipment, or more effectively, use the estimated operating hours per year based on age of equipment.
Chapter 6

Important Parameters Left Out of Spider Diagrams

The objective of this chapter is to review two parameters that are not traditionally included in the spider diagram but are important value drivers in a mine plan:

- Wall slope angles.
- Pushback selection and design.

6.1 Interfacing Geo-Mechanical Models, Pit Ramps, Berms and Push Backs

A geologic model of rock strength and discontinuity properties is essential at the feasibility stage. Such a model is re-assessed and updated for each long term plan, and problems dealt with in the short term by working on alternative short term mine plans. The geologic modeling of the rock which will eventually become the sloped pit wall face is critical to mine planning. It is essential that overall descriptions of the district geology (Carr, J.M. 1967), surface geology (Soregaroli, A.E., 1968), and mine geology (Soregaroli, A.E., 1974) be available and studied. Too steep a wall might result in long delays (6 – 12 months) or even mine closure if a wall fails. This is especially true when the failure involves the haul ramp. Too shallow a wall slope results in increased waste stripping and delayed ore mining.

The pit ramp is the most sensitive wall structure as all ore production will usually travel over this ramp. By placing a ramp, the overall wall slope is reduced and this might stabilize a failing wall. If such a placement does not achieve the stability objective, the wall slope problem will recur as pushbacks are mined. Having a “cork screw” ramp traversing all-round the pit wall perhaps several times will have problems whichever part of the wall becomes unstable. Consequently well designed switchbacks are
preferred where such a design does not materially reduce the mines value, and where truck haulage safety and mechanical integrity can be assured at the reversal of direction of travel, a `hairpin` curve.

Ramps cause a reduction in the overall wall slope and generally lead to substantial increases in waste mined and concurrent decreases in profitability. Ramps are a necessary part of mine design and as such must be placed using knowledge of the location of better grade material to ensure it is mined and not left below the ramp. Ramp design is a complex process and conducted well can generate substantial value for a mining operation and vice-versa.

6.2 Wall Slope Angles

In an effort to rationalize slope stability analysis, “Acceptance Criteria” have been developed which utilize Factor of Safety (FoS) and Probability of Failure (PoF), (Wesseloo, J., and J, Read, 2009). The FoS is the ratio of resisting force/driving force expressed as such as FoS of say 0.9 (dangerous if not already failed) or say 1.2 where the wall structure is deemed stable. The PoF enables the reliability of driving and resisting forces to be taken into account and establishes a level of confidence in a wall design. If a wall has a PoF of 10% then it also has a reliability of 90%. The reliability value should increase from say 50% at the conceptual phase of mine design to at least 75% during operations.

The common practice (observed by the author at Freeport McMoRan and Kinross Gold) is to have a geo-mechanical department at each mine-site and a geo-mechanical group at corporate office. In addition, a geo-mechanical consultant is usually engaged on an “on-call” basis and visits the mine regularly to report on the wall slopes. A full analysis of wall slope design is beyond the scope of this thesis, but the causes of instability have to be understood by the mine planning engineer in the design of ramps and pushbacks.
Lorig, L., P. Stacey and J. Read (2009, p. 239), describe slope design methods including the use of seismic records as a potential hazard for which the only remedy is overdesign. Their material is up-to-date as of 2009, and as such describes the many computer design techniques available.

The initial input to a slope stability study is an examination of outcroppings at the site and cuttings from reverse circulation drilling, followed by an examination of diamond drill core as this becomes available. Often the strike and dip of structures in the core is estimated using core orientation techniques (Holcombe, R., T. Coughlin and N. Oliver, 2013, (http://www.holcombecoughlinoliver.com/downloads/HCO_oriented_core_procedures.pdf ) Once the most prominent structures have been identified and measured, similar structures can be identified and a picture of the discontinuity orientation and strength (after testing or back analysis) built. Stereo density plots of strike and dip can then be made and analyzed and results assigned to various volumes and potential pit walls (sides) of the proposed open pit and surrounding area. Figure 6.2.a shows a typical stereographic plot (Keaton, J.R., 2011).
Figure 6.2.a: Typical stereographic projection showing lower hemisphere pole plots which, when the pit wall line (azimuth and dip) is placed on the plot, show structures affecting the stability of a particular section of wall. (diagram courtesy MACTEK for Ohio Dept. of Transportation Workshop, 2011 - Keaton, J.R., AMEC, Los Angeles, CA)

However, as shown in figure 2.1.a and figure 6.2.b, it is often a lone extensive discontinuity missed or undocumented in the core analysis that causes major problems for the pit slopes in the vicinity of the structure.
Figure 6.2.b: Left photo, the presence of the slip plane was unknown until it appeared on the expansion phase wall. Right photo, a drill sits below the same potential wedge failure, an extension at depth of the plane in the right hand photo. The right hand plane of the wedge is clearly visible, and no berms are left to catch rock from 120m (400ft) above the drill. Below 50m there was no infilling or slip clay, and it is assumed that the structure is not a fault with a recognizable displacement. (Photographs courtesy G. Blackwell)

When such discontinuities as figure 6.2.b are found in the drill core, the mine design can be amended to mine out the material which might fail. The purpose of diamond drilling is to provide assay data. In the process of handling the core it must be inspected by mining, structural and economic geologists and petrologists who may or may not discover such features. A typical drill log is shown in Figure 6.2.c (photo courtesy Golder Associates, Diavik mine data 2013 picture 14, http://muwww-
Figure 6.2.c: A typical drill log is shown. Rock type, strata and various properties are logged electronically to form a data base. No orientation is possible unless the core dip/strike and relationship to the core barrel is known (courtesy Golder Associates, Diavik mine data 2013 picture 14).

A more informative (and manual) drill log is shown in figure 6.2.d and described by Walton, G. and T. Atkinson (1978) in “Some geotechnical considerations in the planning of surface coal mines”, Trans. IMM, Sec. A, Vol. 87 ppA147-171., their figure 6, pA156, “Diagramatic log of range of soil and rock...
strengths on a sub-bituminous coal prospect with application ranges for stripping equipment related to corresponding material strengths”.

Figure 6.2.d: Drill log core used to examine sub-surface material prior to moving very large draglines on top. (after Walton, G. and T. Atkinson (1978), their page A156)
There is still the option of stabilizing such a structure which was carried out successfully in the (figure 6.2.b) case. The benches were supported by mill rods drilled vertically on the bench to intersect the plane as described by Walker, R.A., G.H. Blackwell, T. Javorsky and J. Schelten, (1989). Here the exposed face of the discontinuity allowed accurate measurement of strike and dip by direct total station distance and angle and by photogrammetry.

To orient the core when this is all that is available, the barrel orientation must be known from marker strata or better from “over-coring” and making an impression of strata at the bottom of the hole using some form of putty. The drill rod must then be removed without rotating, and along with values for the dip and strike of the hole, the true orientation of the strata calculated. Analysis without over-coring is more difficult and is shown in figure 6.2.e, courtesy GeoSpark Consulting Inc. Nanaimo BC, 2013, http://geosparkconsulting.com/gallery/?album=1&gallery=1 )
Figure 6.2.e: Drill core orientation showing the strike/dip plane perpendicular to the hole (b), ellipse long axis (a), and dip of the hole (g). (Diagram and software display courtesy GeoSpark Consulting Inc 2013.)

Figure 6.2.f: Laubscher Table 1, p A2 a “score sheet” or “quality designation” used to identify rock properties that are strong or weak in order to develop wall slope angles. Given the inaccuracies of measurements of structure orientation from drill core (+/- several degrees), an empirical method is adequate.

Such data forms the basis for examining the pertinent drill logs on section through the sloping wall of a pit. Figure 6.2.g. (Blackwell, G., 1989) shows a typical set of drill log datum used in wall slope design.
Figure 6.2.g: A section showing the poor quality rocks and their location with respect to the proposed pit wall. On the left, a potential failure is noted where moderately strong rocks (not shown as these are filtered out in this example) overly weak rocks. (Blackwell, G., 1989 unpublished report for Roscoe Postle Associates)

It is not to be expected that wall slopes are constant both around the pit and up/down the pit wall. When there are a number of slope domains that appear in the open pit (i.e. mine planning layout in example figure 6.2.g), it is necessary to analyze each separately and communicate this information to all mine planning personnel. The common practice is for the slope stability engineer to hand the planning engineer a set of inter-ramp slope angles varying with azimuth and occasionally with depth, as shown in figure 6.2.h where there are 8 different wall slopes depending on location, wall geometry and the strength of
various rock types found there (Manuel Contreras U., 2006, Chuquicamata Mine, Codelco Norte, direccion de geotecnia, superintendencia geotecnia operativa), geotecnia mayo 2005.ppt, (their page 26)

Figure 6.2.h: Slope Design Domains. There are 8 specific domains based on the types of rock which in turn have their own geometric pattern of discontinuities. Such patterns, depending on the 3D geometry of the pit wall, have different stability criteria resulting in variable wall slopes depending on domain.
6.3 Wall Slope Definitions

Wall slope definitions start with the individual bench face slope which is usually sloped as dug by the loader. Wall control blasting can be engineered to make the slope steeper or shallower using waxed cardboard “sono tubes” found in construction projects. The tube diameter must exceed that required for continuous propagation of the blast and small enough to fit in the large diameter blast-hole. The normal size is 115 mm (4.5 inch). The tubes are inserted and a small toe load of explosive is placed in the blast-hole. The tube is then loaded with explosive. Results for the two typical explosives used, ANFO (or AlANFO) and slurries are different. With ANFO or AlANFO loaded tubes, rock breakage is greater than for tubes loaded with slurry because the minimum blasting diameter of slurry is higher than that for ANFO. In some rock conditions it is possible to see individual “half holes” on the face where near vertical bench faces have been achieved.

Table 6.3.a: Definition of slope terms used in open pit mine planning. Angle $\alpha > \beta > \gamma$

<table>
<thead>
<tr>
<th>Slope</th>
<th>Definition</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bench Face Slope $\alpha$</td>
<td>The angle between the horizontal and the line connecting the crest of a bench to its toe.</td>
</tr>
<tr>
<td>Overall Slope Angle (OSA) $\beta$</td>
<td>The angle between the horizontal and the line connecting the upper most crest to the lowest most crest (or toe to toe), and includes the horizontal component (usually width) of any ramp(s) or extra catchment berms.</td>
</tr>
<tr>
<td>Inter-ramp Slope Angle (ISA) $\gamma$</td>
<td>The angle between the horizontal and the line connecting a bench crest (or bench toe) and the crest (or toe) of the</td>
</tr>
</tbody>
</table>
bench below it, but not including the horizontal displacement of any ramp(s).

Table 6.3.a defines the primary slope angles required by the planning engineer to complete a plan, and provides a visual explanation of the common open pit bench design parameters. From the bench floor a berm is left between the bottom of the bench face and the top of the next lower bench termed a catchment. Catchments should be as wide as the bench face stability will allow, catching material raveling over time.

Figure 6.3.a: Section showing pit wall definitions and Bench Face Angle $\alpha$.
The inter-ramp slope angle is made up of a series of bench faces and berms, Figure 6.3.b. Often double (or even triple) benching is practiced such that berms are placed on every other bench. The two adjacent benches then have twice the berm to catch raveling rock. Wall steepening can only be accomplished by reducing the double (or triple) bench berm. This type of operation demands Mines Inspectorate approval as the shovel cannot reach to the top of the double berm.

The overall slope is reduced significantly by haul roads (Figure 6.3.c), especially when these are switched back on one or two arcs of the wall. Often the mine is forced to place a special wide catchment berm to stop rock raveling down onto haul roads. The haul ramp should not fail as this will possibly close the mine for a period and the mine planner should always have a plan (including ramp access) for such an event.

Figure 6.3.b: Section showing important Inter Ramp Slope Angle (ISA) $\beta$
Additionally, in deeper pits, the slope engineer, operations, or even the planning engineer might require a step-out in the high wall for a specified horizontal increment in order to provide a catchment for any large rocks falling from upper benches and to improve stability. Although benches might be accessible all the way round the pit when installed, minor wedges and general raveling usually make all round access impossible when the pit has progressed a few benches below. The immediate catchment step out berm and face can be cleaned off using a dozer pulling a chain over the face below.

Table 6.3.b demonstrates how to determine the overall slope angle from the inter-ramp slope angles. The mine planning engineer will utilize this data and formulate an overall slope angle for each section in the pit. This overall slope angle is one of the inputs in the pit optimization algorithm.
Table 6.3.b: Example of estimating the Overall slope angle OSA from Inter-ramp angle ISA

<table>
<thead>
<tr>
<th>Item</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Top Bench Elevation</td>
<td>915m</td>
</tr>
<tr>
<td>Bottom Bench Elevation</td>
<td>350m</td>
</tr>
<tr>
<td>ISA (degrees)</td>
<td>42°</td>
</tr>
<tr>
<td>Step Out Width</td>
<td>40m</td>
</tr>
<tr>
<td>Vertical Step Out Spacing</td>
<td>155m</td>
</tr>
<tr>
<td>Vertical Height</td>
<td>565m</td>
</tr>
<tr>
<td>ISA Horizontal</td>
<td>625m</td>
</tr>
<tr>
<td>Number of Step Outs</td>
<td>3</td>
</tr>
<tr>
<td>OSA Horizontal</td>
<td>745m</td>
</tr>
<tr>
<td>OSA (degrees)</td>
<td>37°</td>
</tr>
</tbody>
</table>

6.4 Pushback Selection and Design

As most porphyry and epithermal deposits usually have high grade areas surrounded by decreasing lower grade envelopes, it is economically important that the high grade be mined first. The advantages are that the high grade produces greater profits which can be applied to capital cost write-downs. In jurisdictions where early profits are taxed at lower rates, after tax profits can be substantially increased. The NPV is also increased by early mining of the high grade core of the deposit as higher cash flows at the start of operations are discounted less than similar cash flows later in the mines life.
The general method of deciding where and how large pushbacks should be located consists of two initial goals:

- **Practicality** – the pushback must be wide enough to be mined at full productivity and produce sufficient ore for the mill for long enough to make the pushback worthwhile.
- **Safety** – the pushback must not disrupt traffic in the previous (lower) pits and pushbacks. Any following pushbacks must not disrupt traffic in the pushback being considered.

The first stage involves one or both of two strategies using a “pit optimizer” and the block model of grades in the deposit. The pit wall slope used by the optimizer is the “wall slope plus ramp(s)” or “overall wall slope” shown in figure 6.3.c and table 6.3.b, and is an approximation until detailed design is attempted. Each pushback uses the previous pit or pushback as a new starting topography in the following process:

- Run a “pit optimizer” at a series of increasing metal prices. As the product price rises, the overall pit increases in size.
- Run a “pit optimizer” at a series of decreasing total operating “costs” where “costs” includes part or all of the mine capital costs plus interest plus overheads, etc. The difference between the real cost per tonne and the increased cost can be viewed as the amount required to pay part or all of the initial capital, return on equity, and etc.

From this stage, the differential tonnage of ore, waste and cash flow can be determined for pit 1 and later pushbacks pit 2, 3, etc., up to “26, Z” as the long term planners familiarize themselves with the ore-body and extraction sequence. This can be quite time consuming when several products are mined.
simultaneously, e.g. copper, molybdenum and gold, and a differential in the price increase/decrease must be investigated.

The results are examined in plan at several elevations, the more important of which are first, the elevation just below collar and second, the elevation at the bottom of the initial pit 1. The ore grades, tonnages of ore and waste and total cash flow are graphed against the pit number (name) using the grade tonnage curves and the results examined. A mid pit (e.g., a pit 3.5 between pits 3 and 4) is added to the list of metal prices or costs where a pushback (in this case pit 4) has mined an overly large amount of material. Where a metal price or cost produces a pit increase too small to be considered, the pushback is left out of the list. Even the initial pit might be removed and a new initial pit with an higher metal price or lower operations costs analyzed. The overall wall slope, section 6.3 and figure 6.3.c is also amended to include the ramp or ramps if the haul-road “corkscrews” or “switch-backs”.

Eventually, a manual sketch of an initial pit and series of pushbacks can be produced based on the important pit crest and initial pit bottom elevations as well as any others where visualization of the design requires clarification. Outlines are drawn at the mid-bench elevations, halfway between one floor elevation and the next, and converted (smoothed) to most closely match the outlines of the blocks in the block model. The “mid-bench” outline gives an area closely approximating the true area. The small amount of material raveled (spalled or failed) from the upper bench floor is approximately equal to that left against the pit wall on the bench below, forming the angle of repose of the berm (or batter) face.

Such a mid-bench technique provides better estimates of volumes removed especially on ramps which have a mid-bench half way up/down. This manual process can be made far more efficient using
AutoCAD. Mid bench lines are assigned their true elevation, a distinctive colour, and a layer name in the form (pit31540) where pit3 refers to pushback 3 and 1540 meters the floor elevation that the mid-bench contour refers to. Using a bench height of 15 meters, the elevation of this line would be 1547.5 meters. For the “imperial” numbering and elevation as used at the mine, the layer name is Pit35060 (Pit 3 and bench floor 5060) and elevation 5085, half way up a 50ft bench.

With a suitable AutoLISP routine run within AutoCAD, this process can be made “manumatic” (figure 6.5.a below) and could be made an automatic process as will be discussed in section 6.5.

A group of pushbacks is shown in figure 6.4.a with the initial pit A, pushback B and pushback C. The remaining surface topography is also shown. By dropping the pits (A, B and C) into the “hole” in the topography, the original ground contours are restored.
Figure 6.4.a: Brenda topography looking south east. Top shows the initial pit a lifted up 2000m. The first pushback (pit B) is lifted 1300m, and the final pushback (pit C) is lifted 700m to produce the 3D display. The colors represent elevation. Pit B is an expansion to the north and east and pit C an expansion all round. This makes the ramps for A and B pits simple to define, but pit C is more difficult leading to a switchback ramp on the east wall. Pit depths for pits A, B and C are 90, 130 and 230m below collar respectively, and deeper by an extra 150m below the highest ground around the mine.

Unfortunately this is not the end of the pit design process. Haul roads or ramps must be inserted into the pit design at their correct locations and gradients as described in the following section 6.5 on ramps.
6.5 Ramps and Economic Ramifications

Ramps, or inclined up-hill haulage roads, are the life blood of an open pit mining operation. With conventional pit haulage, the ramps are usually designed three truck widths wide. Even if uphill in-pit conveying is installed, truck ramps will probably still be necessary but not as wide. If the in-pit conveying system is “ore only” then full width ramp truck haulage of waste will still be required. If ore and waste conveying is installed, and trucks are used anywhere in the pit without an in-pit maintenance shop (e.g. hauling to the base of the conveyor or returning to the maintenance shop), they must have an access ramp wide enough for at least single truck access. The ramp gradient for large off-highway trucks is usually between 8 and 12%, with the majority of mines at 10%. This would appear to be the optimum between haul speed, rate of climb and maintenance of drive train systems on the trucks. However in Mining Technology, Mining INFO (2013), the anonymous authors state that it is the safety of return empty braking on the ramp that determines the choice of a 10% gradient.

The term “rolling resistance” is used to define the total resistance to motion that the truck engine and transmission have to overcome. Travelling uphill, the rolling resistance is determined by the load carried, the road conditions and tire pressure. The rolling resistance of the road surface to be added to the ramp gradient is typically as follows:

- Frozen ice packed 1%
- Dry gravel 2%
- Wet gravel 3%
- Dry sand 4%
- Wet muddy 5% plus
Tire flexure results in another 2% rolling resistance. Consequently the truck on a 10% ramp is actually working on a 14 to 15% ramp uphill and a 5 to 6% negative (plus air resistance) ramp downhill.

The layout of a ramp is shown in figure 6.5.a (with a ~ conversion to metric from the original imperial units). There are many semi-automated systems in software to avoid the tedious repetitive task of drawing ramps. The use of “mid-bench” as the reference allows for such as bench face and ramp tonnage calculations. By using the mid-bench, errors of overestimating and underestimating balance.

Figure 6.5.a: Typical ramp layout showing mid-bench outlines and bench floors/crests. As the ramp deepens, it moves in towards the excavation. The half ramp allows the crest – mid bench – floor contours to be placed for the ramp. On tight curves, quarter or eighth ramps can be used. The ramp shown on the right is in 3D and has object snapping points on the corners and mid-way across the ramp, making it easy to semi-automatically and quickly complete a ramp from pit crest to bottom using AutoCAD.
When large tonnages have to be hauled, it may be necessary to install two ramps to maintain a safe distance between the down empty trucks to ensure safe braking. Some trucks may be faster or slower depending on braking (electric motor and mechanical transmission heat dissipation), and they may have to reverse back up the ramp to avoid clean up equipment and large rocks, endangering the following truck. The up-haul trucks travel slowly and drive systems can fail. The trucks cannot see directly behind them, and the right hand mirror does not provide a clear view on the right side rear. Consequently the amount of ramp traffic must be monitored especially in large pits with high stripping ratios.

The cost of a second ramp is the cost of removal of a strip of material from the ramp to the surface. A 10% ramp of vertical lift of 100m and 30m wide requires the removal of 1.5 million m$^3$ of rock, or 4 million tonnes costing $6$ million. This is a small investment of (say) 2 or 3 new trucks to provide a better level of safety. The planning alternatives made possible to the mine planning engineer by utilizing double ramps cannot be ignored.

Safety can be further improved (and may be mandated by Mines Acts) by having safety lanes at intervals. Such lanes consist of a 20% uphill ramp with sand piles or tires to further slow the run-away truck. Safety belts must be worn by drivers who would otherwise be thrown from the cab in an emergency.

Ramps are “sunk” from the lowest main working elevation in the initial pit or pushback, with several stages of drill, blast, load and haul depending on rock “blast-ability”. For internal “temporary” ramps in pits with stable walls, the holes can be drilled full depth and the more permanent ramp “raised” from the new bench or sunk from the bench above. In both cases using a small number of more efficient blasts with a “free face” is better than blasting with only one free face, straight upwards. For permanent haul roads
and pits with unstable wall slopes, ramps are drilled to ramp depth plus subgrade. As the ramp is deepened, the area blasted is increased to include full bench material, improving loader productivity which would otherwise be reduced by loading on one side only to avoid damage to the shovel electrical cable (and sump pump installation). The geometry of a ramp cut is shown in Figure 6.5.a and photograph Figure 6.5.b shows an initial ramp blast ready to be sunk on the way down to the next bench elevation.

Figure 6.5.b: A shovel is preparing to start a ramp. The small initial blast is dug out as quickly as possible while fractures in the surrounding rock absorb water. This small section will probably reach 3 to 7 m deep and a deep hole will be dug in the incomplete ramp on the right hand side to place a submersible electric pump. Water is always a problem as the electric shovel might “ground out”. In winter water just above freezing continues to flow in only to freeze in the pipelines if the pump is left on the “snore”, making the work even more difficult.
Maintaining ramp gradients used to be a constant irritant between operators and engineering. With modern GPS equipped shovels this is no longer the case.

Fully automated ramp positioning and design has been commercialized by Gill, T., (2004, Road Planning for Open Cut Pits) via MineMap Mine Planning Software, Gemcom and Whittle). Earlier work by the same author rotated a full ramp around the pit bottom until a maximum economic value had been reached.

In “ExpressRoad Routing: The Application of an Optimal Haul Road Generator to Real World Data”, Gill, T., (Optimum Planit Ltd.), has claimed to have accomplished the resolution of when to corkscrew and when to switchback. The number of alternatives must be modified by wall slope considerations, and the ensuring that good grade ore is not overlain by the ramp. The author has no experience working with the software, and cannot judge the efficacy of the software against semi-automated procedures. Given the inaccuracy of all the inputs, a system such as “Express”, even if less optimal than semi-automated and manual procedures, offers major benefits to the mine planner.

6.6 Summary

The wall slope design inputs to mine planning have been discussed and the following conclusions are drawn:

- Wall slope angles are among the primary inputs into mine planning. Pit optimization requires the input of many slope angles. It is common to have overall slope angles changing by azimuth and occasionally by depth.
- The inter-ramp slope angle among other bench design parameters are input into pushback design.
- The general process for pushback selection and design from a wall slope perspective is described.
- The inclusion of slopes and ramps in the “spider” diagram involves the development of at least five alternatives for each of these two important parameters.
With regard to mine planning at the feasibility and routine long term planning upgrade stages, slope design requires a thorough understanding of geology, diamond drill core sampling and analysis, geotechnical and geo-mechanical concepts.
Chapter 7

Practical Application of Long Term Planning

In this chapter the open pit long range mine planning process is demonstrated on an actual mine. The objective is to produce a life of mine schedule that optimizes the net value of the project and is achievable (operable). First, an economic model is developed for a mid-size copper porphyry open pit operation based on benchmark data. Second, the application of Taylor’s rule in selecting the plant size and mining rate is investigated.

The pit sensitivity to the profitability parameters is tested and presented in a spider diagram. The techniques for ultimate pit selection were evaluated for the chosen mill rate. Given the ultimate pit, a set of pushbacks were designed to bring value forward in time and defer waste stripping. The pushbacks were scheduled at constant and variable cutoff grade strategies and the net present value of the project is estimated.

7.1 Description of the Deposits and Available Information

The Brenda deposit exploration diamond drilling data was made available for research by Mr. Ron Bradburn (last mine manager) in a letter to Blackwell in 1996. Brenda is a low grade copper-molybdenum porphyry deposit located in south central British Columbia, 235 kilometers east of Vancouver. The geology is discussed at length by Soregaroli (1974). Noranda Inc. undertook a feasibility study in 1967 and production commenced in early 1970. The mine continued to produce for approximately twenty years. In June 1990 due to depleted ore reserves and a large west wall failure plus a full tailings dam, the mine officially shut down. Table 7.1.a is a simple fact sheet for the deposit showing key metrics.
It should be noted that mines which started production before (1980+/-), kept imperial measures, only using metric where simple to implement. For example converting 4960 and 5010 named 50ft high benches to 1511.7 and 1527.0 named 15.2 m high benches was impractical, whereas loading a blast-hole with 635 Kg of explosive instead of 1400 lbs was quite acceptable to both mine and supplier.

Table 7.1.a: Brenda Mines Quick Fact Sheet (http://www.brendamines.ca)

<table>
<thead>
<tr>
<th>Brenda Mine Fact Sheet</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mine Opened</td>
</tr>
<tr>
<td>Mine Closed</td>
</tr>
<tr>
<td>Rock removed from pit</td>
</tr>
<tr>
<td>Ore processed</td>
</tr>
<tr>
<td>Avg. number of employees</td>
</tr>
<tr>
<td>Total number employed (includes employees who resigned)</td>
</tr>
<tr>
<td>Benefits Paid (Salary)</td>
</tr>
<tr>
<td>Construction costs</td>
</tr>
<tr>
<td>Taxes paid</td>
</tr>
<tr>
<td>Mine elevation</td>
</tr>
<tr>
<td>Open pit diameter</td>
</tr>
<tr>
<td>Open pit depth</td>
</tr>
<tr>
<td><strong>Metals Produced</strong></td>
</tr>
<tr>
<td>278,000 tonnes</td>
</tr>
<tr>
<td>66,000 tonnes</td>
</tr>
<tr>
<td>125 tonnes</td>
</tr>
</tbody>
</table>
7.2 Economic Model

The revenue model is based on net smelter return. The net smelter calculation for this analysis was based on benchmark data (several NI43-101 reports) in addition to the authors experience in net smelter return terms for North American copper mines. The Huckleberry and Gibraltar 43-101 reports were selected for the following reasons: report date, location of the deposit, metals mined, and deposit type (copper porphyry).

The mining OPEX (operating expense) for the case study was based on the Huckleberry reported costs because of the similarity in mining fleet requirements, i.e. 90 to 110 tonne trucks loaded by 9 to 19 m$^3$ loaders and shovels, (100 to 120 Ton trucks loaded by a 12 to 25 cu. yd. Shovels). The processing OPEX was adjusted upwards because the Huckleberry throughput is 18ktpd while the case study throughput was expected to be between twenty five and forty thousand short tons per day. Table 7.2.a shows the case study NSR (net smelter return) relative to the benchmark data with G&A (general and accounting), S.C. (sustaining capital) and C/O (cutoff grade) where noted.

Table 7.2.a: NSR Benchmark Data. Not all underlying data was available in the 43-101 reports.
<p>| | | | |</p>
<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Copper Price</td>
<td>2.7</td>
<td>2.25</td>
<td>3.0</td>
</tr>
<tr>
<td>Mining Cost incl. G&amp;A and S. C.</td>
<td>1.9</td>
<td>1.05</td>
<td>2.2</td>
</tr>
<tr>
<td>Downstream Cost ($/lb)</td>
<td>0.54</td>
<td>0.38</td>
<td>0.63</td>
</tr>
<tr>
<td>Con Grade, %</td>
<td></td>
<td></td>
<td>28.00</td>
</tr>
<tr>
<td>Freight $/short ton, Concentrate</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>(Dry)</td>
<td></td>
<td></td>
<td>120.00</td>
</tr>
<tr>
<td>Smelting $/short ton, Concentrate</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>(Dry)</td>
<td></td>
<td></td>
<td>125.00</td>
</tr>
<tr>
<td>Freight on copper $/lb</td>
<td></td>
<td></td>
<td>0.085</td>
</tr>
<tr>
<td>Refining $/lb, Payable</td>
<td></td>
<td></td>
<td>0.085</td>
</tr>
<tr>
<td>Smelter Recovery</td>
<td></td>
<td></td>
<td>96.00</td>
</tr>
<tr>
<td>C/O grade in Report</td>
<td>0.2</td>
<td>0.2</td>
<td>N/A</td>
</tr>
<tr>
<td>Calculated C/O grade (using report unit costs)</td>
<td>0.25</td>
<td>0.14</td>
<td>0.26</td>
</tr>
</tbody>
</table>

### 7.2.1 Metal Price

The economic model requires the input of three metal prices that will exist in the copper concentrate: Copper, Gold and Silver. The molybdenum was converted to copper equivalent by multiplying
molybdenum grades by a factor of 3.45 and the grade in the model is an equivalent copper grade percent. The NSR calculations were based on the prices in Table 7.2.1.a, (Sinclair, A.J. and G. Blackwell, 2002).

Table 7.2.1.a: Metal Prices Chosen for the Economic Model.

<table>
<thead>
<tr>
<th>Mineral</th>
<th>Price</th>
</tr>
</thead>
<tbody>
<tr>
<td>Copper ($US/lb)</td>
<td>3.00</td>
</tr>
<tr>
<td>Gold ($US/oz)</td>
<td>1,200</td>
</tr>
<tr>
<td>Silver ($US/oz)</td>
<td>15.00</td>
</tr>
</tbody>
</table>

7.2.2 Mining Cost

Mining cost was estimated at $2.20/Ton including mining sustaining capital. These costs are summarized in Table 7.2.2.a

Table 7.2.2.a: Mining Cost Model, based on benchmark data. Mining Cost Model, (Base mining cost from 43-101 Technical report on the main zone optimization Huckleberry Mine). The author added sustaining capital based on his experience in Freeport McMoRan Arizona)

<table>
<thead>
<tr>
<th>Mine Operating Costs</th>
<th>Units</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Base Mining Cost</td>
<td>$/ton</td>
<td>1.90</td>
</tr>
<tr>
<td>Incremental Cost per Bench below pit edge lowest elevation (approximately the ore crusher elevation)</td>
<td>$/bench/ton</td>
<td>0.04</td>
</tr>
<tr>
<td>Mining Sustaining Capital</td>
<td>$/ton mined</td>
<td>0.30</td>
</tr>
<tr>
<td><strong>Mining Cost</strong></td>
<td>$/ton mined</td>
<td><strong>2.20</strong></td>
</tr>
</tbody>
</table>
7.2.3 Processing Cost

Processing cost was estimated at $7.75/Ton. This cost includes crushing and conveying to the mill, milling cost, tailing management cost, sustainable capital to maintain the mill components and site general and administrative cost. The G&A cost was assigned to the milled tons and not to the total mined tons. In other words assigning the G&A to the milling cost will increase the internal cutoff grade and produce a slightly conservative result, Table 7.2.3.a.

Table 7.2.3.a: Processing Cost Model

<table>
<thead>
<tr>
<th>Milling Costs</th>
<th>Units</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Crushing Cost</td>
<td>$/ ton milled</td>
<td>0.55</td>
</tr>
<tr>
<td>Milling Cost</td>
<td>$/ ton milled</td>
<td>5.00</td>
</tr>
<tr>
<td>Tailings Pumping costs</td>
<td>$/ton milled</td>
<td>0.20</td>
</tr>
<tr>
<td>Sustaining Capital Allowance</td>
<td>$/ton milled</td>
<td>0.50</td>
</tr>
<tr>
<td>G&amp;A Assigned to Mill</td>
<td>$/ton milled</td>
<td>1.50</td>
</tr>
<tr>
<td><strong>Milling Cost</strong></td>
<td>$/ton milled</td>
<td><strong>7.75</strong></td>
</tr>
</tbody>
</table>

7.2.4 Downstream Costs

The concentrate grade was fixed for Copper at 28%, Gold at 0.32oz/Dry Ton, and Silver at 1.25oz/Dry Ton. The Brenda deposit is notable for its lack of oxidation, and produced only copper and molybdenum sulphides. Silver and gold content in the ore was so low as to make drill core and pit blasthole assays meaningless, and consequently Ag and Au content was only measured in the concentrate, and assumed to be the average recovered for the previous year in the ore (Personal communication with G. Blackwell).

Concentrate penalties for lead, arsenic, antimony and mercury are not included as these materials are preferentially passed to the molybdenum circuit. A batch hot pressure leach system removes impurities from the molybdenum concentrate making high purity clean molybdenum sulphide. This is used for roasting to oxide for direct feed to iron and steel processes making typical low temperature strength alloys, and penalties have been neglected. High lead areas of the ore deposit are mined along with cleaner ore to keep the lead head grade to less than 0.01 % as part of the short term planning process. The historical data does not indicate the presence of these deleterious metals in the separate copper and molybdenum concentrates, as they have been preferentially sent to the molybdenum circuit rather than the copper circuit. In the process of leaching the molybdenum post flotation, all these deleterious substances are removed making a high purity molybdenum concentrate.

The concentrate transportation costs are trucking, rail haulage, (stereotypical) and ocean freight. The total transportation cost of US$120 /SDT was used in the NSR calculation. The smelting costs include sales commission, deductions, treatment and refining costs and losses in transit. A $US125/SDT for treatment was used in the NSR calculations. Refining costs were fixed at $US0.085/lb. of Copper, $US3/oz. of Gold and $US0.3/oz. of Silver. The downstream costs are presented in Table 7.2.4.a.
Table 7.2.4.a: Downstream Processing Cost Model.

<table>
<thead>
<tr>
<th>Downstream Costs</th>
<th>Units</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>a. Concentrate Transportation</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Conc. Grade % Cu</td>
<td></td>
<td>28.0%</td>
</tr>
<tr>
<td>Moisture Content</td>
<td></td>
<td>7.0%</td>
</tr>
<tr>
<td>Ocean Freight</td>
<td>$/ wet ton</td>
<td>52</td>
</tr>
<tr>
<td>Inland Freight</td>
<td>$/ wet ton</td>
<td>40</td>
</tr>
<tr>
<td>Stevedoring</td>
<td>$/ wet ton</td>
<td>20</td>
</tr>
<tr>
<td>Concentrate Transportation Cost</td>
<td>$/ dry ton</td>
<td>120</td>
</tr>
<tr>
<td><strong>b. Smelting and Refining</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Smelting</td>
<td>$/ dry ton</td>
<td>125</td>
</tr>
<tr>
<td>Refining</td>
<td>$/ lb</td>
<td>0.085</td>
</tr>
<tr>
<td>Smelter Recovery including transportation loss</td>
<td></td>
<td>96%</td>
</tr>
<tr>
<td><strong>c. Sales and Taxes</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Freight to Market &amp; Sales Costs incl. Taxes</td>
<td>$/lb</td>
<td>0.085</td>
</tr>
<tr>
<td><strong>Net Smelter Cost including credits</strong></td>
<td>$/lb</td>
<td><strong>0.63</strong></td>
</tr>
</tbody>
</table>

7.2.5 Cutoff Grade

Based on the economic model, two cutoff grades were calculated; the internal cutoff grade (block value is adequate to cover the processing cost but not the mining cost) and the breakeven cutoff grade (block value pays for mining and processing), Table 7.2.5.a. The breakeven cutoff grade is calculated for a
surface block, as mining progresses deeper the incremental cost per bench will be added to the mining cost and gradually increases the cutoff grade.

Table 7.2.5.a: Cutoff Grade Calculation.

<table>
<thead>
<tr>
<th>Item</th>
<th>Units</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Copper Price</td>
<td>$/LB</td>
<td>3</td>
</tr>
<tr>
<td>Downstream Cost</td>
<td>$/LB</td>
<td>0.63</td>
</tr>
<tr>
<td>Mining Cost</td>
<td>$/Ton</td>
<td>2.2</td>
</tr>
<tr>
<td>Processing Cost</td>
<td>$/Ton</td>
<td>7.75</td>
</tr>
<tr>
<td>Mill Recovery</td>
<td>%</td>
<td>85%</td>
</tr>
<tr>
<td>Smelter Recovery</td>
<td>%</td>
<td>96%</td>
</tr>
<tr>
<td>Breakeven Cutoff</td>
<td>%</td>
<td>0.26</td>
</tr>
<tr>
<td>Internal Cutoff</td>
<td>%</td>
<td>0.20</td>
</tr>
</tbody>
</table>

Because of the increase in copper prices, the cutoff grade has significantly dropped (<0.3%) from values used in 1970-90. Although this increases resources, it adds to the effort required to determine the operational cutoff grade. Should the cutoff grade be lowered to defer waste stripping and accept low grade material (generate less profit at lower cost), or elevated, incurring more cost but generating a higher profit? A starting value has to be assumed to progress through the early “scoping” work and a 0.35% cutoff grade is selected. The cutoff grade will be revisited when scheduling in section 7.8.
7.3 Planning Model

The geological model was available in text format with a block size of 100x100x50ft (30.5mx30.5mx15.2 m). A planning model was built in MineSight software to include items for recoverable copper, density and percent of the block inside topography (usually 100% unless intersecting the ground surface) among other items. The model limits are presented in Table 7.3.a.

Table 7.3.a: Brenda Planning Model Limits.

<table>
<thead>
<tr>
<th>Axis</th>
<th>Minimum</th>
<th>Maximum</th>
<th>Size</th>
<th>Number of blocks</th>
</tr>
</thead>
<tbody>
<tr>
<td>Easting (X)</td>
<td>8450</td>
<td>17650</td>
<td>100</td>
<td>92</td>
</tr>
<tr>
<td>Northing (Y)</td>
<td>9450</td>
<td>18650</td>
<td>100</td>
<td>92</td>
</tr>
<tr>
<td>Elevation (Z)</td>
<td>3360</td>
<td>5760</td>
<td>50</td>
<td>48</td>
</tr>
</tbody>
</table>

Brenda copper recovery was assumed to be linear and was fixed at 85%. The geological model did not include multiple rock types as the deposit host rock was a fresh graniodiorite/quartz monzonite, and the in-situ density for all rock was set at 2.67 Kg/m³ (12ft³/Ton). The initial topography was available in DXF format. The topography was coded into the block model in the TOPO item as a percentage of the block below topography, Figure 7.3.a.
Figure 7.3.a: Plan view of Bench 5210 showing how initial topography (dashed line) is coded in the block model as a percent. This percent is multiplied by the block tonnage and is reflected during pit optimization and scheduling.

7.4 Pit Optimization

The pit optimization was completed using the Lerchs-Grossmann algorithm for pit limit definition. The LG optimization program requires the following input:

- Net Block Values
- Wall Slopes
- Rock Density
- Surface Topography

The MineSight planning model was imported into NPV Scheduler software to run the pit optimization. First all blocks were valued and routed to the best value process based on the NSR model. Table 7.4.a shows the values used in the block valuation.
Table 7.4.a: Pit Optimization Parameters. The economic model values form the basis for the optimization parameters.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Units</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Copper Price</td>
<td>$/lb</td>
<td>3.00</td>
</tr>
<tr>
<td>Downstream Costs</td>
<td>$/lb</td>
<td>0.63</td>
</tr>
<tr>
<td>Mining Cost</td>
<td>$/ton</td>
<td>2.20</td>
</tr>
<tr>
<td>Incremental Cost per bench below pit exit point</td>
<td>$/ton/bench</td>
<td>0.04</td>
</tr>
<tr>
<td>Processing Cost including G &amp; A</td>
<td>$/ton</td>
<td>7.75</td>
</tr>
<tr>
<td>Overall Slope Angle (includes ramps)</td>
<td>Degrees</td>
<td>40</td>
</tr>
<tr>
<td>Density</td>
<td>Kg/m³</td>
<td>2.67</td>
</tr>
</tbody>
</table>

Figure 7.4.a is a plan view at 4960 showing block grades, and figure 7.4.b is the same bench but showing the net block (30.5x30.5x15.2 m or 100x100x50 ft) value populated in each block.

It can be seen that blocks with grades of 0.6% generate a net block value of approximately five hundred and fifty thousand. On the other hand blocks with grades less than 0.25% results in a negative value. Table 7.4.b gives an example of several blocks; a positive block routed as direct feed to the mill, a positive block routed to the high grade stockpile, a positive block routed to the low grade stockpile, a negative block routed to the low grade stockpile because it generates marginal value, and a waste block routed to the waste dump. Note that the original imperial units are used to make bench calculations simpler e.g. 100x100x50 = 500,000 ft³ or 41,666.67 short tons rather than 30.5x30.5x15.25 = 14,186 m³ or 37,877.45 tonnes.
Figure 7.4.a: Plan view of bench 4960; blocks are color coded by equivalent copper grade.

Figure 7.4.b: Plan view of bench 4960; blocks are color coded by net value.
Table 7.4.b: Block Value for several sample blocks, each block is routed to the process that provides the highest profit or the lowest negative value.

<table>
<thead>
<tr>
<th>Item</th>
<th>Sample Block1</th>
<th>Sample Block2</th>
<th>Sample Block3</th>
<th>Sample Block4</th>
<th>Sample Block5</th>
</tr>
</thead>
<tbody>
<tr>
<td>Grade</td>
<td>0.6</td>
<td>0.3</td>
<td>0.26</td>
<td>0.25</td>
<td>0.15</td>
</tr>
<tr>
<td>Block Volume</td>
<td>500,000</td>
<td>500,000</td>
<td>500,000</td>
<td>500,000</td>
<td>500,000</td>
</tr>
<tr>
<td>Block Tonnage</td>
<td>41,666.67</td>
<td>41,666.67</td>
<td>41,666.67</td>
<td>41,666.67</td>
<td>41,666.67</td>
</tr>
</tbody>
</table>

**Revenue**

<table>
<thead>
<tr>
<th>Item</th>
<th>Sample Block1</th>
<th>Sample Block2</th>
<th>Sample Block3</th>
<th>Sample Block4</th>
<th>Sample Block5</th>
</tr>
</thead>
<tbody>
<tr>
<td>Copper Price</td>
<td>3</td>
<td>3</td>
<td>3</td>
<td>3</td>
<td>3</td>
</tr>
<tr>
<td>Downstream Costs</td>
<td>0.63</td>
<td>0.63</td>
<td>0.63</td>
<td>0.63</td>
<td>0.63</td>
</tr>
<tr>
<td>Realized Copper Price</td>
<td>2.37</td>
<td>2.37</td>
<td>2.37</td>
<td>2.37</td>
<td>2.37</td>
</tr>
<tr>
<td>Recovery</td>
<td>85%</td>
<td>85%</td>
<td>85%</td>
<td>85%</td>
<td>85%</td>
</tr>
<tr>
<td>Smelter Recovery</td>
<td>96%</td>
<td>96%</td>
<td>96%</td>
<td>96%</td>
<td>96%</td>
</tr>
<tr>
<td><strong>Block Revenue</strong></td>
<td><strong>966,960</strong></td>
<td><strong>483,480</strong></td>
<td><strong>419,016</strong></td>
<td><strong>402,900</strong></td>
<td><strong>241,740</strong></td>
</tr>
</tbody>
</table>

**Cost**

<table>
<thead>
<tr>
<th>Item</th>
<th>Sample Block1</th>
<th>Sample Block2</th>
<th>Sample Block3</th>
<th>Sample Block4</th>
<th>Sample Block5</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mining Cost</td>
<td>2.2</td>
<td>2.2</td>
<td>2.2</td>
<td>2.2</td>
<td>2.2</td>
</tr>
<tr>
<td>Processing Cost</td>
<td>7.75</td>
<td>7.75</td>
<td>7.75</td>
<td>7.75</td>
<td>7.75</td>
</tr>
<tr>
<td><strong>Block Mining Cost</strong></td>
<td><strong>91,667</strong></td>
<td><strong>91,667</strong></td>
<td><strong>91,667</strong></td>
<td><strong>91,667</strong></td>
<td><strong>91,667</strong></td>
</tr>
<tr>
<td>Block Processing Cost</td>
<td>322,917</td>
<td>322,917</td>
<td>322,917</td>
<td>322,917</td>
<td>322,917</td>
</tr>
<tr>
<td>Re-handle Cost</td>
<td>0.9</td>
<td>0.9</td>
<td>0.9</td>
<td>0.9</td>
<td>0.9</td>
</tr>
<tr>
<td><strong>Block Re-handle Cost</strong></td>
<td>37,500</td>
<td>37,500</td>
<td>37,500</td>
<td>37,500</td>
<td>37,500</td>
</tr>
</tbody>
</table>

**Profit**

165
The resulting optimized pit shell is shown in Figure 7.4.c. A 50ft spacing contour map was generated, Figure 7.4.c. Note that this is the mid-bench contour (example say 4985) which would be referenced as 4960 bench floor. The size of the pit shell is approximately 4000ft long (~1219m), 3800ft wide (~1158m) and 1000ft deep (~305m). The pit inventory is shown in Table 7.4.c. The grade tonnage curve of this pit is in figure 7.4.d.
Figure 7.4.c: Plan View of the Pit Optimization Shell - 50ft mid-bench contours. Note ramps not yet included and wall slope is 40 degrees to allow for ramps with an inter ramp slope of 45 degrees.

Table 7.4.c. Ultimate Pit Inventory. Ore is calculated at a cutoff grade of 0.35%.

<table>
<thead>
<tr>
<th>Item</th>
<th>Units</th>
<th>Values</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total Tons</td>
<td>Short Tons</td>
<td>508,133</td>
</tr>
<tr>
<td>Ore Tons</td>
<td>Short Tons</td>
<td>240,076</td>
</tr>
</tbody>
</table>
### Table

<table>
<thead>
<tr>
<th>Waste Tons</th>
<th>Short Tons</th>
<th>268,057</th>
</tr>
</thead>
<tbody>
<tr>
<td>Strip Ratio</td>
<td></td>
<td>1.12</td>
</tr>
<tr>
<td>Cu Grade %</td>
<td></td>
<td>0.4915</td>
</tr>
</tbody>
</table>

Figure 7.4.d: Grade-Tonnage curve of optimized pit with marker ore tonnes (red) and resulting ore grade (blue).

**7.5 Mine Life, Mill Throughput and Capital Cost**

The use of Taylor’s rule to estimate the mine life and mill throughput was discussed in Chapter 5. As mentioned before, it is important to realize that the “expected ore tonnes” is a function of cutoff grade among other parameters, which in itself needs to be optimized. As such Taylor’s modified (1984) rule
Table 7.5.a: Estimated mine life and mill throughput using Taylors modified rule, Long & Singer (2001), and Long (2009) for a range of cutoff grades. Ore and waste tonnage is calculated for each cutoff grade. Based on the ore tonnage the life of mine and throughput is calculated.

<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>M t/y</td>
<td>M t/y</td>
<td>Life of Mine</td>
<td>Throughput</td>
<td>Life of Mine</td>
</tr>
<tr>
<td></td>
<td>Years</td>
<td>kt/day</td>
<td></td>
<td></td>
<td>Years</td>
</tr>
<tr>
<td>0.50%</td>
<td>82</td>
<td>426</td>
<td>17</td>
<td>13</td>
<td>17</td>
</tr>
<tr>
<td>0.45%</td>
<td>118</td>
<td>390</td>
<td>19</td>
<td>17</td>
<td>15</td>
</tr>
<tr>
<td>0.40%</td>
<td>169</td>
<td>340</td>
<td>21</td>
<td>23</td>
<td>16</td>
</tr>
<tr>
<td>0.35%</td>
<td>230</td>
<td>279</td>
<td>22</td>
<td>28</td>
<td>17</td>
</tr>
<tr>
<td>0.30%</td>
<td>294</td>
<td>214</td>
<td>24</td>
<td>34</td>
<td>18</td>
</tr>
<tr>
<td>0.25%</td>
<td>361</td>
<td>147</td>
<td>25</td>
<td>40</td>
<td>19</td>
</tr>
<tr>
<td>0.20%</td>
<td>417</td>
<td>91</td>
<td>26</td>
<td>44</td>
<td>20</td>
</tr>
</tbody>
</table>

One can see that a lower cutoff grade will increase the ore tons and will sustain a large mill (>40ktpd). In other words a larger mill will be hungry for ore and will require a cutoff grade as low as the breakeven.
cutoff grade. On the other hand selecting a higher cutoff grade (0.3-0.35) will sustain a mill size between (30-35ktpd).

To investigate the selection of throughput, a more detailed study was completed. The project CAPEX (capital expenditure) shown in figure 7.5.a was estimated for several mill sizes using O’Hara ‘s (1980) formula for estimating combined mine/mill open-pit projects (Figure 5.2.d in section 5.2) and adjusted to 2011 dollars using the ENR index chart (Figure 5.2.a in section 5.2).

![Figure 7.5.a: Estimated project total CAPEX versus mill throughput mine using O’Hara’s formula.](image)

The processing OPEX was estimated for the different throughput (Figure 7.5.b) based on benchmark data; Huckleberry mines for 18ktpd (Page 78, 43-101 report, 2011), Mount Milligan for 60ktpd (Page 16-56, 43-101 report, 2009) adjusted to 2011 dollars, and Gibraltar for 85ktpd (Page 71, 43-101 report, 2011). Also one should consider that the processing cost is a function of numerous factors including ore characteristics, logistics (G&A), tailings management, etc. Nevertheless, for the purpose of this analysis
these other factors are neglected (small in magnitude) and the benchmarked data was adequate to complete the throughput study.

![Graph: Estimated total processing OPEX versus mill throughput using benchmark data.]

Figure 7.5.b: Estimated total processing OPEX versus mill throughput using benchmark data.

For each throughput the following steps were completed:

- Run Pit Optimization using the given OPEX to determine the pit limits.
- Generate a set of phases (pushbacks with no ramps) to be used for scheduling.
- Schedule the pushbacks to satisfy the given mill capacity at several mining rates
- Calculate the NPV for each scenario.

It is a common mistake in the industry (observed by the author at several mining companies) to test different throughputs on the same ultimate pit. The change in throughput is reflected in two main inputs, OPEX and CAPEX. As OPEX decreases, the pit limits will start to decrease the pit size because more blocks become uneconomical (move from ore to waste). Running a schedule with a higher OPEX in a larger pit will count waste blocks as ore resulting in a misleading NPV. It is important to note that
running the schedule in a smaller pit will make waste of the low grade material and could benefit or hurt the NPV (see next section 7.6 for discussion on ultimate pit selection).

Figure 7.5.c shows the present value (CAPEX not included) of the tested mill sizes. For each mill size, the mining rate was varied to investigate the mining rate required to feed the mill. The mining rate was kept constant over the mine life. The mining rate will be revisited in the following section 7.6 to test the value of variable mining rate for the chosen mill size.

![Figure 7.5.c: Present value versus mining rate for varying mill throughputs. For each mill throughput, the mining rate was escalated and a life of mine schedule was generated at 0.35% cutoff grade. The discounted cash flow for each schedule was calculated and the resulting present value was plotted against the mining rate for each mill throughput. For each throughput the maximum present value was selected to incorporate CAPEX. (Values are in short tons)](image-url)
The best present value for each option was selected ($550M at 10mtpd for the 20ktpd option, $757M at 15mtpy for the 30ktpd option, and $950M at 20mtpy for the 40ktpd option). The NPV was calculated for each case by deducting the corresponding CAPEX and is presented in Figure 7.5.d.

The analysis above shows that a mill rate of 40,000 short tons per day (36k metric t/day) generated a higher NPV than the other cases. Even when evaluating the options at a lower copper price ($2.80) the 36ktpd mill continues to generate the best NPV. It also common to have factors other than deposit size affecting the mill throughput selection; for example, a step change in capital due to increased power requirements, a project being limited by the size of the tailings storage facilities, availability of water, permitted footprint, geographical (country) risk, and availability of financing. These factors usually play a major role in guiding the selection of the mill size by directly changing the NPV, or by forcing a strategic decision to be made. It is also common to carry two or more alternative project sizes through the prefeasibility stage and finalizing the decision at the feasibility stage. For example, Kinross Tasiast prefeasibility carrying both 30ktpd and 60ktpd options into a prefeasibility study (Kinross 2012 third-quarter results, [http://www.kinross.com](http://www.kinross.com)). For the purposes of this research a mill size of 36k metric tons per day is selected for further analysis. Remarkably, the detailed analysis is in line with Taylor’s modified formula (see Table 7.5.a) at a 0.25% cutoff grade (breakeven cutoff grade in Table 7.2.5.a).
Figure 7.5.d: Net Present Value versus mill throughput (Values are in short tons). The bars represent the NPV for each throughput evaluated under two copper prices ($3/lb and $2.80/lb). The CAPEX for each mill is shown in a pink shaded box.

7.6 Ultimate Pit Selection

The pit by pit graph and the discounting by bench methods in selecting the ultimate pit were tested on the Brenda project. The Brenda pit by pit graph was generated by running a set of L-G’s at incremental copper prices. A total of 68 pits were generated at prices starting at $3.00/lb and down to $0.93/lb seen in figure 7.6.a.
Figure 7.6.a: Cross section at 13650N looking north showing the incremental price shells from $2.00 to $3.00. The blocks are color coded by equivalent copper grade value.

The non-discounted pit by pit graph of copper price by cumulative PV and by million tons of rock removed (ore + waste) for Brenda is shown in figure 7.6.b. The graph shows that the present value continues to increase steadily until reaching the $2.20 pit (230M total tons moved) after which the increase in value starts to diminish and is seen in the flat top part of the curve. The worst case schedule is also generated by mining the pit top down without any intermediate phases, and show an inflection point at the $2.20 pit. That the “curve” for worst PV is flat indicates that it is not sensitive to copper price (or cut-off grade), agreeing with Hustrulid’s conclusions.
Figure 7.6.b: Brenda pit by pit graph, the present value is on the primary vertical axis and the tonnage is on the second vertical axis. (Values are in short tons).

One can clearly see that the PV in the pit by pit graph is directional and is based on mine schedules that have not demonstrated their practicality. To better understand the value of the pits, a set of shell based phases (which do not include ramps but respect the minimum mining width and overall wall slope including allowance for ramps) were generated and scheduled. The objective is not to generate a final life of mine schedule but to generate a more realistic one because the pit by pit graph has no regard to minimum mining width and the LG shells could be just a few meters apart. The present value for each scheduled pit is presented in figure 7.6.c.
Figure 7.6.c: Brenda pit by pit graph, the present value is on the primary vertical axis and the tonnage is on the second vertical axis. (Values are in short tons).

Figure 7.6.c shows an optimum lying between the 2.50 and 2.80 pits. This corresponds to a total material mined (dotted line) of about 300mt and 410mt respectively. Increasing the pit size beyond a total of 410mt shows no increase in value, and on the contrary, the value decreases.

To help understand the change in NPV as one progresses from one pit to another, the incremental value for each pit was calculated. Table 7.6.a shows the inventory of the $2.00, $2.20, $2.40, $2.60, $2.80 and $3.00 pits and the increment between the pits.

Table 7.6.a: Ore is calculated @ a CO of 0.35%, (Values are in million short tons).

<table>
<thead>
<tr>
<th></th>
<th>2.20 Pit</th>
<th>Increment</th>
<th>2.40 Pit</th>
<th>Increment</th>
<th>2.60 Pit</th>
<th>Increment</th>
<th>2.80 Pit</th>
<th>Increment</th>
<th>3.00 Pit</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rock</td>
<td>229</td>
<td>51</td>
<td>281</td>
<td>54</td>
<td>335</td>
<td>74</td>
<td>410</td>
<td>98</td>
<td>508</td>
</tr>
<tr>
<td>Ore</td>
<td>157</td>
<td>22</td>
<td>179</td>
<td>18</td>
<td>197</td>
<td>21</td>
<td>218</td>
<td>21</td>
<td>240</td>
</tr>
<tr>
<td>Waste</td>
<td>72</td>
<td>29</td>
<td>101</td>
<td>36</td>
<td>137</td>
<td>53</td>
<td>191</td>
<td>76</td>
<td>268</td>
</tr>
<tr>
<td>Strip Ratio</td>
<td>0.46</td>
<td>1.33</td>
<td>0.57</td>
<td>2.00</td>
<td>0.70</td>
<td>2.55</td>
<td>0.87</td>
<td>3.60</td>
<td>1.12</td>
</tr>
<tr>
<td>-------------</td>
<td>------</td>
<td>------</td>
<td>------</td>
<td>------</td>
<td>------</td>
<td>------</td>
<td>------</td>
<td>------</td>
<td>------</td>
</tr>
<tr>
<td>EQCU</td>
<td>0.52</td>
<td>0.45</td>
<td>0.51</td>
<td>0.45</td>
<td>0.50</td>
<td>0.42</td>
<td>0.49</td>
<td>0.42</td>
<td>0.49</td>
</tr>
</tbody>
</table>

It can be seen that each increment contains about 20M tons of ore at an average grade above 0.4%. Nevertheless, the amount of waste associated with each increment continues to increase (strip ratio increases from 1.33 to 3.60). The cost of mining and processing the final increment reduces the discounted revenue and as such reflects negatively on the project present value. Therefore, by selecting the smaller pit the low quality ore (low grade material or better grade material associated with a high strip ratio) is excluded.

The bench discounting method was tested for Brenda and compared with the pit by pit graph approach in selecting the ultimate pit. Figure 7.6.d is a section showing the non-discounted pit, the 15% discounted pit and the 30% discounted pit as DR=0, DR=15 and DR=30 where DR is discount rate. It is very clear that the discounting by bench does not have a great effect on Brenda because of the relatively moderate strip ratio (about 1:1). This approach does not provide much guidance in selecting the ultimate pit and is worthy of further discussion (outside the scope of this thesis) because it is very popular in the mining industry. The commercial pit optimization software packages have included the discounting by bench as part of their options (Whittle, NPV Scheduler and MineSight MSEP).
Figure 7.6.d: Cross section at 13650N looking north showing the effect of bench discounting on the ultimate pit limit. DR 0 is the $3.00 LG pit in figure 7.6.a. DR 15 is the $3.00 LG pit at 15% discount. DR 30 is the $3.00 LG run with a 30% discount rate. As expected the discount by bench decreases the size of the pit. This discounting technique does not have a significant effect on the Brenda ultimate pit. This is because of the relatively low strip ratio. The blocks are color coded by equivalent copper grade value.

The above analysis shows that the pit by pit approach provides a better method for selecting the most valuable pit. Nevertheless, it is preliminary and is based on scheduling a set of shells that are certainly not operable. The better approach is to select a set of price shells that are reasonably spaced (respect the minimum mining width) and schedule these shells to produce figure 7.6.c, and table 7.6.a. This requires slightly more effort by the planning engineer but generates a better understanding of the value associated with each increment.

A valid question is “Why should we focus on the final pushback given that the conditions (price, cost, slopes…etc.) will change by the time this final pushback is mined? ”.
There are two main reasons:

- Annual Reserve.
- Project Valuation

Mining companies are required to declare reserves on annual basis and complete a fair value analysis (Impairment Test). Selecting a large pit shell will increase the stated reserves. On the other hand a larger pit might decrease the asset value which potentially will limit a corporation’s ability to raise capital for a project or force an impairment charge for an existing deposit. As such, selecting the ultimate pit is a strategic corporate decision; a company interested in maximizing reserves will select the largest economical pit while a company looking for financing for its project will be searching for every dollar that increases the NPV.

For the purpose of this research the $2.80 pit is selected to guide the design of the ultimate pit.

7.6.1 Pit Sensitivity

The sensitivity of the project to the following input parameters was tested: copper price, mining cost, processing cost, recovery, slopes and G&A. The testing methodology included the following:

- Base case; optimized pit in section 7.4.
- Change one variable at a time to capture the effect of each parameter independently of the others.
  At 10% increments from -30% to +30%.
- Update the block valuation and run the pit optimization.
- Run a high level schedule to see the effect on the project PV.
Although this PV is not valid as a final project NPV, it is acceptable for relative comparison. Therefore, the percent change in the PV is reported as relative versus absolute values.

Figure 7.6.1.a: Project sensitivity to input parameters. The project is most sensitive to copper price.

Figure 7.6.1.a, shows a spider graph for ±30% variations in each parameter. The project is less sensitive to mining cost because of the relatively moderate strip ratio. As discussed in Chapter 2, a mine with a high strip ratio (for example a gold deposit with a 4/1 to 6/1 strip ratio) will be severely negatively impacted by a change in the mining cost.

Brenda is most sensitive to copper price. A 10% decrease in price resulted in a 30% drop in PV.

Processing cost behaves closely to the reverse of copper price as they are multiplied to produce revenue. The effect of recovery is less than price because the copper price has selling costs subtracted prior to realizing the revenue.
7.7 Design

The final (ultimate) pit design was based on the $2.80 optimized pit shell, figure 7.7.a. The mill is assumed to be located on the south east side of the pit (lowest topography to exit at), and the pit exit is conveniently located to allow for easy access to the mill. The ramp starts on the east side of the pit and spirals down counter clockwise to the bottom of the pit at 4160 Elevation.

Figure 7.7.a: Ultimate Pit design based on the $2.80 pit. Crest in pink and toe in blue. Ramp starts at 5130elev and spirals down counter clockwise to 4160elev at bench bottom.
Upon completion, the quality of the design was reviewed by comparing the ultimate pit inventory against the optimized pit shell. The results are summarized in Table 7.7.a. The tolerances of the final designs were well within the expected deviation of 5% generally accepted by industry.

Table 7.7.a: Ultimate pit design quality review, key items are within 5% indicating an acceptable design. Ore is calculated at a 0.35% cutoff grade. Values are million metric tons (values in brackets are million short tons).

<table>
<thead>
<tr>
<th></th>
<th>2.8 Pit</th>
<th>Design</th>
<th>Difference</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total Tons</td>
<td>373 (410)</td>
<td>361 (397)</td>
<td>-3%</td>
</tr>
<tr>
<td>Ore</td>
<td>198 (218)</td>
<td>196 (215)</td>
<td>-1%</td>
</tr>
<tr>
<td>Waste</td>
<td>174 (191)</td>
<td>166 (182)</td>
<td>-5%</td>
</tr>
<tr>
<td>Strip Ratio</td>
<td>0.88</td>
<td>0.84</td>
<td>-4%</td>
</tr>
<tr>
<td>Cu Tons</td>
<td>992k (1,090k)</td>
<td>978k (1,075k)</td>
<td>-1%</td>
</tr>
</tbody>
</table>

7.7.1 Pushbacks

Pushback design defines the way in which the deposit is going to be mined; indicating where mining is going to start and how it is going to progress. The value of the project is dependent on the pushback design and sequence because of the time value of money degrading the value of ore.

As such, the drivers for pushback value are:

1. Grade.
2. Strip Ratio.
3. Mineral distribution within the pushback.
4. Time of mining.

In order to define areas of higher grade material, the copper price was incremented and LG’s were run for each price (alternatively the mining cost could be escalated). Although this approach is sound it has the following limitations:

1. A significant number of LG product price shells will produce pushbacks that have too small a mineable shape or are delaying deeper ore mining because the pushback is too large, i.e. wide. This is mitigated by selecting shells at no more than operable widths for efficient mining (room to maneuver shovels and trucks).

2. The shells may increase in different directions which is preferred as this aids ramp design. Material may not be able to be routed under upper bench mining especially when spillage falls onto the ramp. Adept design allows parts of previous ramps to be used by later pushbacks. A "donut" (or onion ring) shaped form of expansion in multiple directions is problematic. It is most difficult to mine earlier shells and lower benches safely unless the expansion is carried out in “orange segment” pieces. This is the main risk associated with the designed pushbacks for Brenda.

3. The LG pit assumes the material is mined in one instance, and does not account for the time value of money unless post LG analysis by shells and time of mining is carried out.

As such, a set of shells were selected based on the following criteria:

- Bring revenue forward and spread out waste mining over the life of mine.
- Each pushback must contain sufficient ore tons to maintain production for a period of time that allows the completion of the waste stripping of the next pushback.
• Satisfy the minimum mining width, set at 60m for the majority of mining (other than ramp development)

Figure 7.7.1.a: Cross section at N13750 through pushback design and LG price shells. Dotted brown lines are the LG shells used to guide the pushback design (no ramps, shallow wall slope is overall plus ramps). Solid black lines are the designed pushbacks including ramp detail. Blocks are color coded with fading to emphasis pushback lines.

Figure 7.7.1.a is a cross section the shows the LG shells selected to guide the pushbacks design. The starter pit design was focused on producing high grade ore with minimal pre-stripping requirements. This starter pit contained approximately 33 million short tons of ore at 0.57% Cu equiv., and a low strip ratio (0.1:1) which is enough to feed a 40ktpd mill for two years. Pushback 2 also contains higher grade material and remains at a relatively low strip ratio. Waste is deferred to the last two pushbacks which also include lower grade and mill feed material stored in stockpiles as part of the ore supply to the mill.

The pushback inventory is presented in Table 7.7.1.a. This pushback design will facilitate generating a schedule that can satisfy the mill at a higher grade in the early years of the project and allow for a faster capital recovery. Figure 7.7.1.b is a plan view of the designed pushbacks.
The pushback method in this case is to mine the pushback ramp and expand it. There is then a short period when trucks cannot move material from the prior high grade pushback (or original starter pit) as the bench must be blasted at the top of the old ramp. At the elevation of the bottom of the new ramp a flat
road crosses the bench to the new top of the old ramp. Loaded trucks from the prior pit (usually carrying good ore) then come up the old ramp, cross the flat bench and then travel up the new ramp, figure 7.7.1.c.

Table 7.7.1.a: Pushbacks inventory, (Values are in Short Tons)

<table>
<thead>
<tr>
<th>Units</th>
<th>Pushback #1 (Starter Pit)</th>
<th>Pushback #2</th>
<th>Pushback #3</th>
<th>Pushback #4 (Ultimate Pit)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ore</td>
<td>Million tonnes</td>
<td>30 (33)</td>
<td>49 (54)</td>
<td>65 (71)</td>
</tr>
<tr>
<td></td>
<td>(short)</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Waste</td>
<td>Million tonnes</td>
<td>3 (4)</td>
<td>19 (21)</td>
<td>50 (56)</td>
</tr>
<tr>
<td></td>
<td>(short)</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Total</td>
<td>Million tonnes</td>
<td>33 (37)</td>
<td>69 (76)</td>
<td>115 (127)</td>
</tr>
<tr>
<td></td>
<td>(short)</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Grade</td>
<td></td>
<td>0.57</td>
<td>0.52</td>
<td>0.49</td>
</tr>
<tr>
<td>Strip Ratio</td>
<td></td>
<td>0.1</td>
<td>0.4</td>
<td>0.8</td>
</tr>
</tbody>
</table>
Figure 7.7.1.c: Top view showing the material flow from the bottom of the starter pit.

7.8 Scheduling

The purpose of this step is to generate a life of mine plan that sustains ore delivery to the mill. The challenge is at what cutoff grade and what mining rate, both of which are interdependent. The following approaches were tested on the designed pushbacks:

1. Schedule at a constant cutoff grade.
2. Schedule at a variable cutoff grade.
7.8.1 Constant Cutoff Grade

Three schedules feeding a 40ktpd mill were completed; the first at a 0.30% cutoff grade, the second at 0.35% cutoff grade, and the third at a 0.40% cutoff grade. In all cases waste is deferred as long as possible and no stockpiles were available to store the material at a grade less than the cutoff. A mill production ramp-up of 50% in year 1, 75% in year 2, and full capacity at year 3 was assumed for all schedules.

Figure 7.8.1.a: Material extracted; 40ktpd mill throughput, 0.30% cutoff, 0.35% cutoff, and 0.40% cutoff grade. Bars show the total tons moved for each case on the left axis. The top lines show the head grade per year on the right axis.

Figure 7.8.1.a shows the cutoff grade and material moved by year. The relationship between mining rate and cutoff grade can be seen; at a 0.3% cutoff a 20mtpy mining rate is adequate; at 0.35% the mining rate
must be increased to about 35mtpy; and at 0.4% the mining rate must be increased further to 40mtpy. The primary question is whether elevating the cutoff grade facilitates a faster capital recovery. To answer this question the cumulative discounted cash flow is calculated for each schedule, figure 7.8.1.b. The figure shows that increasing the cutoff grade does not accelerate capital recovery.

Figure 7.8.1.b: Cumulative present value by period in millions of US dollars. No acceleration in capital recovery is realized as the cutoff grade is increased. The 0.40 cutoff schedule requires a noticeable increase in total tons mined in order to maintain the targeted cutoff. This results in a lower NPV relative to the other schedules.

Upon completion of the above schedules the following conclusions were drawn:

- It is important to include low grade material to increase the NPV of the schedule; the key point is to delay the processing of this low grade material. It has already been delayed by stockpiling.

- The head grade may require some smoothing. Although this usually reduces the NPV, it may improve the performance of the mill process in terms of recovery and throughput.
- The total tonnage mined requires smoothing. This will smooth the truck requirement if the inclusion of capital for extra trucks (new or used) reduces the NPV.

### 7.8.2 Variable Cutoff Grade and Stockpiling

Based on the conclusions from the constant cutoff grade schedules, a new mine plan was generated with the following adjustments:

- Two additional stockpiles were introduced; a low grade stockpile (holding material below 0.35%) and a high grade stockpile (for material higher than 0.35 and lower than 0.4 % Cu).
- Use a variable cutoff grade strategy coupled with the mining rate and stockpile utilization to generate a smoother grade profile.

![Figure 7.8.2.a: Material movement. 40ktpd mill throughput, variable cutoff grade with stockpiles, and smoothed head grade.](image)

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The updated schedule is seen in figure 7.8.2.a. The previous schedules were used to guide the cutoff grade strategy. A mining rate of about 15mtpy was required to achieve a 0.6% grade in the first two years. The grade was dropped to 0.5% up to year seven and the mining rate had to be increased to about 28 mtpy. Mining from the pit is complete in year 21, and mining from the stockpiles continues until year 25. Prior to year 21, the stockpiles were used to provide part of the mill material if the grade from the stockpile is higher than material coming from the pit.

Copper production and cumulative net present value by period for the variable cutoff grade schedule is shown in Figure 7.8.2.b.

Figure 7.8.2.b: Copper production versus period, and cumulative NPV on secondary axis. Copper production in the first two years is low reflecting the mill ramp up. Capital is assumed as a lump sum in year one. The non-discounted cash flow continues to increase and flattens towards the end of mine life as copper production is coming from the low grade stockpile. The discounted cash flow shows that it takes approximately 10 years to recover the capital (at 8% discount rate).
The sinking rate by year was checked in order to evaluate the practicability of the schedule. Figure 7.8.2.c shows that the maximum sinking rate is 5 benches per year or 1 every 2 months which is reasonable. At 1 per month the operation would be continually sinking, an inefficient low productivity mining function, which would only happen if the push-back were too small.

Figure 7.8.2.c: Pushback development, the bottom elevation at the end of each period is shown for each pushback. Depth is in feet relative to 5435 Elev. Maximum sinking rate 5 benches per year. The final pushback does not reach the previous pit bottom as the incremental mining cost at depth is not supported by the grades available.

7.9 Summary

It was shown that the selection of mill throughput is not a straightforward problem, but rather an iterative process. Optimal mill throughput is related to the reserve size, which in itself is a function of the cutoff grade. At a high metal price the cutoff grade drops resulting in the examination and choosing of a wider...
range of possible operational scenarios. Furthermore, the mining rate and availability of stockpiles has a significant role in the selection of a cutoff grade. Nevertheless, given all these moving criteria, Taylor’s rule (and its adjustments) is a sound starting point especially at the scoping study stage where most variables still have unknown values.

This agrees with Hustrulids (2006) comments (p 493) “Even with certain knowledge of everything, optimizing theory yields different answers depending on what quantity is selected to be maximized. The maximum quantity might be total profit, total cash flow, the net present value or the internal rate of return. Furthermore, the peaks of such curves are rather flat. Thus when allowing for the practical inaccuracies of data, the calculated results cannot be considered critical. Hence, although valid, a highly mathematical approach to mine life determination is seldom of practical use. Other ways must be found to provide a reasonable first approximation for mine life”, and in the Hustrulid text, leads to Taylor’s rule.

It was also shown that the Brenda project is most sensitive to copper price and mill recovery followed by slopes, processing cost, and mining cost. This is in line with Doggett and Mackenzie spider diagram figure 2.5.3.a.

The selection of the ultimate pit is better guided by a modified approach to the pit by pit graph rather than the increasingly popular discount by bench method. This requires further investigation and is a recommended topic for future research.
The use of stockpiles and elevated cutoff grade strategy adds value to copper porphyry deposits. Even in the case of a relatively low grade deposit, such as Brenda, where a large amount of marginal material is available, this is true. Given a high metal price and availability of tailings storage facilities at the end of the mine life, this marginal material could be milled at an attractive profit.
Chapter 8

Conclusions and Recommendations

A common quote in the mining industry, attributed to Mark Twain, says:

“A mine is a hole in the ground with a liar on top”

Although this statement has its merits in some cases, the industry has matured in many ways in an effort to minimize (or at least quantify) the risk associated with mining projects. Analysts spend hours in attempting to guide investors regarding the direction metal prices taking, only to realize several months later how inept their estimate was. This is because of the lack of knowledge of individual and combined factors and their interaction on an international stage. The thesis examines what drives the price of the product mined, and the options the mine planning engineer has to mitigate a price downtrend, and take advantage of increases in product value. The problems associated with increased operations costs when product prices remain depressed are also addressed.

The aim of this research was to identify the variables driving profitability for open pit mines, investigate the importance of each variable and determine how the planning engineer can utilize this knowledge to improve stakeholder value throughout the mine planning process.

This chapter discusses the conclusions that can be drawn from this work, and proposes several lines of research to investigate issues that materialized from the study.
8.1 Conclusions

The research showed that the key variables driving profitability can be summarized as the following: metal price, grade, processing recovery, operating cost (OPEX), initial capital (CAPEX), sustaining capital, dilution, and reserve size.

The sensitivity of the project value to each of these variables was confirmed, and an understanding of the level of predictability achievable at the different planning stages indicated and summarized in Table 8.1.a.

Table 8.1.a: Ranking of the profitability parameters based on effect on NPV and predictability.

<table>
<thead>
<tr>
<th>Order of Importance</th>
<th>Very Unpredictable</th>
<th>Unpredictable</th>
<th>Somewhat Unpredictable</th>
<th>Somewhat Predictable</th>
<th>Predictable</th>
<th>Very Predictable</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Metal Price</td>
<td>Head Grade</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>After Blasting</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>Process Recovery</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>2</td>
<td></td>
<td>OPEX</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>After Production</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>3</td>
<td></td>
<td>CAPEX</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>4</td>
<td>Wall Slopes</td>
<td></td>
<td>After Mining</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>5</td>
<td>Reserve Size</td>
<td>Mine Recovery</td>
<td>After Mining &amp; Processing</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>6</td>
<td>Dilution</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>7</td>
<td>Sustain. Capital</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
**Metal Price**

Mining projects, especially large low grade open pit epithermal and porphyry deposits, are most sensitive to metal price. When coupled with the difficulties of predicting metal prices 7 to 10 years into the future (when the mine is an established producer at mid-life), the mining industry is “hostage” to factors beyond its control. From a stakeholder point of view, this conclusion is particularly onerous.

For copper prices one can conclude the following: if a more serious recession occurs then copper prices may reduce to approximately US$2000/MT (US$1.00/lb). This price level would demand operating mine re-design, reduction in mine life from “high grading”, or even shutting down or “mothballing”, along with postponements in the opening of new mines. If the copper price remains near US$8000/MT (US$3.60/lb), investments in new mine discoveries and corporate acquisitions would be prudent provided studies show that total supply will not greatly exceed consumption. Such a strategy can give a corporation a dominant position in the future, as the consumption of product must continue to deplete operating mines unless substitution, scrap or new technologies significantly reduce primary demand.

**Grade Estimates**

Grade estimates and locations within the mineralized ground are not verifiable until the short term planning stage of mining when close blast-hole sampling can confirm the grade model. In the study of grade estimation it was confirmed that models from exploration drilling can match production. The grade distribution and grade/tonnage curves had some slight differences, but were considered quite adequate. Public disclosure of grade/tonnage curves in such as SEDAR is essential to understanding the value of a mining asset (cut-off grade is equivalent to metal price) and shows a lack of transparency in the mining industry.
Further, estimates can include the classification of measured, indicated and inferred using verifiable geostatistical methods. Financial analysis of the estimated grades (resource block model) forms that basis of mine planning.

It was also realized that the mining industry has been reluctant to accept the maxim of block size being related to drill spacing, preferring to use small blocks whose grade estimates are almost meaningless swathes of ore or waste seldom found in practice. This in return leads to the selection of large loading equipment that cannot mine selectively. Having a good understanding of the definition of an SMU is critical. A good definition of an SMU is given by Leuangthong, Neufeld, and Deutsch and states “the block model size that would correctly predict the tonnes of ore, tonnes of waste, and diluted head grade that the mill will receive with anticipated grade control practice”. When followed, this statement aids in applying the right amount of ore dilution and recovery where applicable, and includes such in mineable reserve declarations. Costs are also an issue as small equipment can dig more selectively but has higher operating costs and vice-versa. With an excellent grade SMU model indicating the short range fluctuations in grade, equipment can be selected such that extra mining costs (more loaders, trucks and personnel) can be more than compensated for by an increase in head grade.

**Mine Life and Mill Throughput**

It was shown that the use of Taylor’s rule (and its modifications) is a sound starting point for estimating the mill throughput especially at the scoping study stage were most variables are still unknown. The research also shows that the optimal mill throughput is related to the reserve size, which in itself is a function of the cutoff grade. At a high metal price the cutoff grade drops resulting in a wide range of possible operational choices. Furthermore, the mining rate and availability of stockpile material has a
significant role in the selection of a cutoff grade. The selection (optimization) of mill throughput is a function of several interdependent variables that require iterative analysis and do not lend themselves to a simple computer model but rather a series of graphical iterative processes.

However, the measures of profitability were not overly sensitive to the choice of a cut-off grade. Further, much academic effort has been spent to “optimize” the cut-off grade, and the input parameters are not sufficiently well defined to do this. Hustrulid’s comments regarding cut-off grade optimization have been verified. The cut-off is an insensitive parameter in measures of profitability.

The study also demonstrated that mine production is affected by depth of mining because haulage up ramp loaded utilizes more truck hours. It is possible that the mine plan is not achievable in practice because the majority of available truck hours are spent hauling from depth. Stripping ratios cannot then be met and ore from the next push-back delayed. The alternatives are purchase (new or used) trucks or accept the decrease in head grade. By “Murphy’s Law” the decrease in head grade will be matched by a decrease in metal price, which reduces cash flow such that investing in even used trucks is unlikely.

Wall Slopes

The study demonstrates the importance of having a good understanding of the slope design criteria and to incorporate this knowledge in the mine planning process. The overall slope by domain is required for the pit optimization process. The inter-ramp slope angle is required for the ultimate pit and pushback design. Under-design will most likely result in failures, if major could result in mine closure. Over-design is very costly because it increases the strip ratio, which in return could render an operation uneconomical.
**Pushback design**

The general design methodology was presented and demonstrated on Brenda deposit. Pushbacks must be designed to bring value forward and delay stripping. Furthermore, pushbacks must practical, wide enough to be mined at full productivity and produce sufficient ore for the mill, and operable under safe conditions, not disrupt traffic in the previous (lower) pits and pushbacks.

**Ultimate pit Selection**

The literature research shows that the selection of the ultimate pit is better guided by a modified approach to the pit by pit graph rather than the increasingly popular top down discounting by bench method. It was demonstrated that selecting the last increment generated by the LG algorithm will result in a lower NPV when scheduled because of the effect of discounting. Further, this last ultimate pit push-back has to be wide enough for efficient mining. The decision on the size (width) of the penultimate pit push-back influences the decision to mine the ultimate pit as much as any other input.

**Cutoff grade, block routing, and elevated cutoff grade strategy**

It was demonstrated that the use of (high grade mineralization waste or low grade ore) stockpiles and an elevated cut-off grade strategy adds value to the epithermal/porphyry type deposit. This is true even in the case of a relatively low grade deposit, such as Brenda, where a large amount of marginal material is available.

Soaring copper prices have resulted in a significant drop in the cutoff grade. Although this has increased the amount of ore tons in the pit, it has widened the range of operational cutoff grade selection. This is a complex problem that requires numerous iterations to achieve the strategic targets of an organization.
Scheduling the material at a low cutoff grade results in reduced stripping but decreases the amount of copper production. Throwing away the low grade material and running the mill at a high cutoff grade resulted in a lower overall value of the Brenda project. There is a contradictory argument that the high metal prices are a “window of opportunity” and the shareholders should receive the resulting excess profit in the form of dividends. A complete analysis of the tax ramifications of such a decision are beyond the scope of this thesis, as tax abatement is best left to skilled accountants.

The best NPV was achieved by allowing the cutoff grade to fluctuate over the mine life and adding the better grade material to stockpiles. This schedule was refined to generate a more achievable schedule by running the mill at a more constant head grade coupled with a planned increase in the mining rate (extra trucks) followed by a decrease in mill throughput and mining rate. This end of life decrease may not materialize if milling efficiencies in throughput and recovery materialize as they did at Brenda. Here the use of new milling technology (analyzing mill noise versus throughput, huge 3x3 x 10m float cells) enabled cut-off grades to be reduced, reducing the stripping ratio, and maintaining the original planned mine production.

8.2 Recommendation

The research in this thesis has generated the following recommendation for future work:

Demonstrate the advantage of generating several mine plans using a series of five values based on intuitive guesses and historical metal prices.

The research concluded that the planning engineer must generate a series of plans based on five choices of metal prices; worst conceivable >> expected low >> most likely >> expected high>> highest
conceivable. One needs to demonstrate this concept on several open pit deposits. For example: take year 5 in the Brenda mine life, after the starter pit is mined, and generate the five plans. Show at each price what would be the strategic decision.

**Analysis of the discount by bench method in pit optimization.**

A number of mining professionals use the top down bench discounting technique to emulate the time value of money during pit optimization. This technique has been incorporated in numerous open pit mine planning software packages such as NPV Scheduler, MineSight and Whittle. Furthermore, several mining companies have adopted this as part of their standard in long range mine planning. This thesis showed that bench discounting does not have much effect on the Brenda deposit. One needs to investigate the effect of this approach on numerous types of open pit deposits and understand the benefits and limitations.

**SMU from the mine planning prospective**

This research discussed the difference between the block size provided by the geologist in the block model and the operational definition of an SMU. One needs to develop a process of determining the SMU to be used in the mine planning process. Is it based on aggregating the blocks into larger ones? Is it based on polygons? Can the process be automated by a computer algorithm? How to incorporate dilution and ore loss into this process? Is it a mine planning function or a geology function?

**Schedule with truck hours**

The scheduling in chapter 7 was based on tonnes moved with no regards to the effect of truck hours on the plan. One needs to incorporate truck hours during the development of schedules. Demonstrate when to
purchase new trucks and the opportunity cost associated with this investment. Furthermore, it might be more economical to park some of the trucks under certain conditions.

**Using other economic evaluation metrics other than NPV**

The analysis in chapter 7 was based on net present value as the main metric to compare options. There are other metrics used in finance to evaluate projects. One needs to study and demonstrate the pros and cons of using these different tools at each stage of the mining study and during the mine planning process.


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Appendix A

Chapter 4 Expanded
Sampling of Deposit Grades

Grade estimation is a continuous process starting with regional and local geology, exploratory surface mapping and various electronic or ground vibration techniques which locate a potential target. Typically a junior mining company or exploration group from a larger mining company drills the area by the least expensive method available, often with reverse circulation drill rigs employing a down-hole hammer drill. Using this method, drill cuttings and dust are sucked up inside the hollow drill rods (drill stem). A typical drill rig is shown in figure 4.1.a and note the mast is down ready to move the drill. Hollow drill stem is stored on the mast and rig, and the amount of sample collected is reduced by pouring the drill cuttings into a series of two rotating sample bins with compartments for dumping or collecting samples.

Figure 4.1.a  Down-hole reverse circulation percussion drill rig. Note mast is down ready to move drill, and hollow drill stem is stored on mast and rig. The amount of sample collected is reduced by
pouring the drill cuttings into a series of two rotating sample bins on rear with compartments for dumping or collecting samples. (photo courtesy G. Blackwell)

In figure 4.1.b, cuttings are produced by the tungsten carbide insert drill bit driven by a down-the-hole compressed air drill. The cuttings are removed by pushing cuttings and air up outside the drill stem, or (preferred) sucking the cuttings up the centre of the drill stem. Minimal water is used as a dust suppressant. Contamination of the samples from material washed in from higher up the hole can be problematic and is usually observed when a sudden increase in payable grades slowly decreases as the hole deepens. Drilling below the water table is also problematic due to the washing in of such material which is forced up inside the drill stem as drill sludge. The junior company will drill and assay as much ore as possible because drilling waste can deter further investment necessary to continue proving up the deposit.
Figure 4.1.b  Percussive drill bit with tungsten carbide inserts and holes to suck gravel and dust up inside the drill stem and on to the sampling bins (preferred) or outside the drill stem and deposited as cuttings around the top of the hole (photo courtesy G. Blackwell)

Once the deposit can be identified and grades are indicated to be sufficiently good, diamond drilling is the preferred method of proving up the deposit, figure 4.1.c. Drilling for cores (short solid cylinders of unbroken rock as found in-situ) from 25mm to 70mm (1 in to 2.5in) is directed across the dip to ensure that cores are representative of the deposit and surrounding waste and low grade. Assay lengths can be determined by geological inspection such that similar types of material are assayed independently, or a fixed length of core can be used. Geologists examine, log and photograph the cores prior to assaying, figure 4.1.d.
Figure 4.1.c  Diamond drill rig. The 45 degree dipping hole is being deepened by the drill stem being added. The hydraulic pushers to force the drill bit forward are shown around the yellow hydraulic rotating mechanism, and the stem is rotated as it is pushed forward. (photo courtesy G. Blackwell)

The amount of material actually drilled and sampled in an individual 50mm diameter hole drilled down at 45 degrees and 425m long (300m vertical depth) might be 2.2 tonnes using a density of 2.6 tonnes/cubic meter. A deposit might cover an area of 1000m x 1000m and 300m deep with 100 holes at 100m intervals on a square pattern in plan, making 220 tonnes of sample. Sample reduction, if representative, would reduce this tonnage to (say) 22 tonnes for actual assay. The deposit measuring 1000x1000x300 cubic meters might contain (say) 800 million tonnes, and is represented by 220 tonnes of sample of which 22 tonnes might be assayed. Extra in-fill drilling in “5 on a dice” pattern might double the assay samples to 44 tonnes. Trusting that mines can estimate the deposit grades so well given the huge volume represented by such a small sample base is extraordinary!
The core is pushed out of the barrel after releasing the core catchment mechanism which sits inside the front of the barrel. The barrel is pulled out using a wire-line and then returned down the hole. This significantly speeds drilling as the drill stem and diamond drill bit do not have to be removed until the bit wears out (photo courtesy G. Blackwell).

The mitigating factor is that the sampling of each hole might be conducted on consecutive 1.5m segments making 300 individual assays per hole. Each sample of 8 kg might be represented by assaying 0.8 kg of material which in turn represents 2.7 million tonnes of material.

Samples are repeatedly crushed then cone and quartered to reduce the sample size to a volume which can be assayed using an X ray on-stream analyser (or similar equipment), or for gold, usually by fire assay. The fixed segment length depends on hole diameter as the sample size should initially be small. This allows the sample to be crushed to small size particles prior to “cone and quartering” to reduce the
volume of sample. This process of grinding to a suitable size before cone and quartering maintains Gy’s “safety line” (Gy, P., 1979 in “Sampling of particulate materials”, Developments in geomathematics, 4, Elsevier Scientific, Amsterdam) for repeated crushing and/or grinding and cone and quartering until a small representative sample is produced for assaying. This process will be described shortly in section 4.2.4 on production blast-hole cuttings sampling where the safety line cannot economically be followed and poor repeatability of sample grades results.

In the case of gold, the fire assay procedure is used on an assay tonne or fraction of an assay tonne. The sample is “fired” in a crucible with fluxes until a bead of gold remains. The bead is then dissolved in acids and the solution on-stream analysed for gold. In the case of diamond drill cores this can be a problem as short lengths of core may contain a gold vein. Often the assay laboratory will be unable to accurately assay very high or low grades and will return a grade of “greater than” some high value or less than some low value, or simply “below threshold”. For the mine planner, the “less than” low is noted as zero, and the greater than high noted as that high value. Such high values are termed “outliers” and can have severe and erroneous effects on calculated variances. Often such highs represent the upper 1 to 5% of the population of assay results and may be representative of a geologically different form of deposit or “outlier” within the main ore zone. (Sinclair, A.J. and Blackwell, G, 2002 pp 168-179)

For copper, the process is similar up to the point where the final “cut” of crushed sample fines (consistency of flour) is achieved. The correct dry weight of these fines are then treated with acids to liberate the metals, and an on stream analyser used to estimate the metal content.
For well managed assay laboratories, the repeatability and agreement with “standards” (provided by Government Agencies) and with the “same” sample sent to a Government or an independent nearby laboratory (e.g. another mine) is usually excellent despite the very small amount of gold or copper material assayed from a large initial sample.

The process of drill hole location, sample length and composite length is subjective, and no two practitioners will arrive at the same values. Corporate budgets will also form part of the decision process. Sinclair, A.J. and G.H. Blackwell, (2002) in “Applied mineral inventory estimation”, Cambridge University Press, U.K., provide further material on sampling. The reader is encouraged to refer to this book for explanations regarding sampling which are beyond the scope of this thesis.

**Mineral Deposit Grade Estimation**

**Grade Continuity or Variography**

The hole spacing at the grade estimation stage is, to some extent, dependent on the deposit type and size. The preliminary data is then used to test deposit continuity using the (semi) variogram (shown in figure 4.2.1.a), relative variogram (relative to mean grade squared), or some other related investigative tool (e.g. covariogram) describing increasing variability with distance between known sample data. The best, but not commonly used measure of continuity is the correlogram. It is conventional to refer to the more often used semi-v variogram simply as “variogram”.

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Figure 4.2.1.a  Idealized (semi) variogram and terminology. The axes are X distance and Y variance. At zero distance the Y intercept or nugget describes the variability between many collected and assayed samples of the “same” material. The maximum variance is the sill or population variance for the deposit (or a “local” volume within the deposit). The distance at which the variogram reaches the sill is the range beyond which samples are not correlated. For distances below the range, samples grades are more or less similar depending on distance. It should be noted that the sample data making up the variogram must be of the same size (support) and often of the same orientation.

Initially, with minimal diamond drill hole assays completed, the drill spacing is so large that the variogram at short distances merely describes continuity along the hole. The data is usually composited by weighted averaging of adjacent core assays to such as a possible bench height. If the bench height is changed later, the data can be re-composited and variograms revised. When the drill spacing (usually in a large test area representative of both ore and waste) is small enough, the range of the relative (to grade squared) variogram can be determined. The drill hole spacing can then be set as some fraction of the variogram range. Figure 4.2.1.a shows the nugget (the variance between the collection and assay of material from the same core), population (deposit) variance between grades from similar sized samples (support), and the range beyond which sample to sample variance is the same as that of the parent variance. The prediction of grade from samples more distant than the range is not appropriate, as the
variance between samples is the population (or “local”) variance. In this case the best estimate of grade is the average grade of the deposit (or locality).

Figure 4.2.1.b Relative Variograms for a typical porphyry copper deposit (Brenda Mines) using all data (all directions or omni-directional) within grade limits of 0.05 to 1.25% Cu. Two data sets have been used, one bench of blast-hole cuttings (BH) assays, and all the diamond drill (DDH) data. The lower grade limit (0.05%Cu) is defined by lack of mineralization which makes the variogram appear better than it is. The upper limit (1.25%Cu) removes extremely high grades which define high grade pods too small to be mined in practice and which make the variogram difficult to interpret (saw-tooth). The axes are X distance and Y relative variogram value (unit-less). The support for BH is nominal 15m vertical 310mm diameter blast holes. The support for DDH is 15m composites of 45° dip south, 45° dip north, vertical, plus two horizontal holes. Because BH samples cannot follow Gy’s safety line when the initial sample is collected, the nugget is twice as high and range 200m less than DDH samples where the small well-handled cores follow Gy’s safety line. Both data sets have a sill at 0.25, the typical proportional effect of ¼ for copper. Mathematical models of the data curves are made using “nested spherical models” (see Sinclair, A.J. and G.H. Blackwell, 2002) for use in grade estimation using the kriging process.
The model typically used to describe variograms is the “nested spherical”, and can be used to describe almost any type of continuity model due to the nesting of two structures (C1 and C2) and using the nugget (C0) as the starting point where X= 0 and y=nugget.

The model is described as follows;

\[
\gamma(h) = C_0 + C_1 \left[ (1.5 \cdot \frac{h}{a_1}) - 0.5 \cdot \left( \frac{h}{a_1} \right)^3 \right] + C_2 \left[ (1.5 \cdot \frac{h}{a_2}) - 0.5 \cdot \left( \frac{h}{a_2} \right)^3 \right]
\]

Where

- \( \gamma(h) \) is the variogram value at distance \( h \)
- \( C_0 \) is the nugget
- \( C_1 \) and \( C_2 \) are the sills of the first and second structures
- \( C_0 + C_1 + C_2 \) is the sill of the total variogram
- \( a \) refers to the range of the structures
- \( h \) refers to the distance
- if \( h > a \) then the term \( (1.5 \cdot \frac{h}{a}) - 0.5 \cdot \left( \frac{h}{a} \right)^3 \) = \( C_1 \) or \( C_2 \)

With a computer modelling tool such as VGM3MODL (Blackwell, G.H. (1998) “Geostatistics Course Notes”, Dept. of Mining Engineering, Queen’s University at Kingston), the interactive movement of the computer mouse up/down and left/right allows the relative values of \( C_1 \) and \( C_2 \) plus \( a_1 \) to be adjusted to fit the real (experimental) variogram as \( C_1 + C_2 \) is (sill-nugget) and \( a_1 \) is (max range – \( a_2 \)).

A typical study variogram is shown in figure 4.2.1.b. In this case the blast-hole (BH) and diamond drill (DDH) data are compared using the pairwise relative variogram which reduces the proportional effect consequences (very hard to model scattered “saw-tooth” graphs) and gives a variogram that can be used for any grade range (if the range is the same for all grades – not usually found in practice). Of note is the better lower nugget and longer range of the DDH data which has been crushed and cone-and-quartered.
with care to follow Gy’s safety line. Detailed information on the “nested spherical or Matheron model” of the variograms used to provide an accurate mathematical model can be found in Sinclair, A.J. and G.H. Blackwell (2002) and is beyond the scope of this thesis.

**Interpolating (Kriging) the Block Model**

For convenience, the mineralization is described in terms of a block model. The block size (the dimensions of a regular 3D shape, usually a cube on a regular 3D grid of multiple blocks) describes the ore-body, (see Section 4.2.3, SMU’s) and grades are assigned to each block using a kriging procedure based on the co-variogram which is commonly the inverted variogram. The co-variogram (figure 4.2.2.a) is used to simplify matrix inversion such that the co-variance at distance zero is the sill, and at distances greater than zero the co-variance reduces from (sill minus nugget) to co-variance zero at the range.

![Co-Variogram Diagram](image)

**Figure 4.2.2.a**  Co-variogram. The variogram is inverted and placed such that the co-variogram is zero at the range. The co-variance at zero distance is the sill, and at finite distances the co-variogram is (sill – variogram). Matrices using the co-variogram are simpler to invert in the kriging process.
Samples close to the block are used to estimate the block grade by making a vector of sample to block co-
variances, and a matrix of sample to sample co-variances. Often a search of each octant (upper and lower
quadrants around the block centre) for the closest sample(s) is included. In the software used for this
analysis a “ninth octant” is included where samples located within the block are stored. The number of
samples per octant search is not affected by the samples stored as the ninth octant. Any sample falling
within the block must be included prior to the octant search. A vector of weights to apply to each sample
to estimate the block grade is the required result. It is possible to generate “negative weights” when a
sample falls behind another in a line of sight to the block. The output includes this information for any
block affected and the user can assess whether the estimate should be repeated with the offending sample
removed.

The inverse of the matrix of the sample to sample co-variances is multiplied by the vector of sample to
block co-variances to provide the vector of weights to apply to each sample, as shown in figure 4.2.2.b.
The sample to block co-variance describes the size and shape of the block by placing a grid over the block
and finding the co-variance from sample to grid points included within the block and averaging. The
sample to sample procedure automatically accounts for samples close together by their larger co-variance
values in the matrix. The method uses the (co-) variogram which is a good estimator of sample/sample
and sample/block similarity and geology. When the sum of weights is forced to unity (or 100%) by
augmenting the matrix and vectors, the process is referred to as “Ordinary Kriging” as opposed to
“Simple Kriging”. It should be noted that sometimes negative weights may result from the augmenting in
which case values for distant samples are removed from the matrix if their negative weight is problematic.
In testing the grade estimates, one bench, 4860 at Brenda Mines will be used. It is felt that this bench, some 45m to 75m below surface and some 1000 m across is representative of material in the pit.

Figure 4.2.2.b “Simple” kriging matrix used to apply weights to samples close to the block. The block size and shape can be approximated using co-variance values from each sample to a grid of points describing the block. The method uses the co-variogram which describes similarity of samples within the ore-body. By augmenting the matrix with “1;s” and “0’s” the “Ordinary” kriging, matrix results and the sum of weights is forced to unity.
Isaaks, E.H. and R.M. Srivastava, (1989) in “An introduction to applied geostatistics” provide basic material on the subject of geostatistics. The reader is encouraged to refer to this book for further explanations regarding grade estimation which are beyond the scope of this thesis.

The Resource Block Model - Estimating Grades of Selective (or Small) Mining Units (SMU’s)

David, M., (1977), in “Geostatistical ore reserve estimation”, Elsevier, Amsterdam, Netherlands, p 283, recommends that the block size be no smaller than a quarter of the drill spacing. If the rule is ignored and blocks are too small, adjacent estimates will be of similar value, in which case why make so many small block estimates whose grades are not usually found in practice? Modern geostatistical techniques such as probability kriging or simulation (Journel and Isaaks 1984), which are beyond the scope of this thesis, can also be used to estimate the grades (and tonnages) of all the blocks making up the orebody, and may be capable of estimating smaller parts of individual blocks referred to as selective (small) mining units or SMU’s. If the variogram has a long range and a very small nugget, grades of small SMU’s may be reasonably well estimated. A pictorial presentation of SMU’s, blocks and equipment size is shown in figure 4.2.3.a.

A better definition of the SMU is given by Leuangthong, O., C. Neufeld and C.V. Deutsch (2003) (http://www.uofaweb.ualberta.ca/ccg/pdfs/2003%2027-smuselect.pdf) “the block model size that would correctly predict the tonnes of ore, tonnes of waste, and diluted head grade that the mill will receive with anticipated grade control practice”.

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Blocks and SMU’s and their relation to selective mining. The SMU’s are shown as smaller cubes 15x15x15m (9000 tonnes) which can be dug in a shift by a small cable shovel using “double back-up” of trucks. The black square on right represents a block containing 4x4 or 16 SMU’s, and the block size is 60x60x15m (140,000 tonnes). Assuming exploratory drilling on 90m spacing, the larger block grade can be well estimated, but not the grade or location of SMU’s within the block unless more sophisticated kriging estimation procedures are applied. This is because the SMU’s are exactly 1/6 of the drill spacing, breaking David’s quarter spacing rule for estimation. The production drill is shown on the left, and the shovel is operating in “double back-up” mode with room for a truck left and right of the shovel. The figure is to scale.

The basis for SMU grade estimates, the volume-variance relationship, is described by Parker, (Parker, H. M., (1980), “The volume-variance relationship; a useful tool for mine planning”, Eng. and Min. Jour., October, pp.106-123). A block has a large internal variance because of its size and because it spans both high and low grades. An SMU has a small internal variance because it is small and is located in material of more similar grade. The block or SMU variance is described by the average variogram value within the volume. In a block there are several SMU’s and the variance “between” individual SMU’s would be large when a large block contains high and low grades i.e. when the variogram has a short range. As variance is
proportional to the square of the grade (the Proportional Effect, Isaaks & Srivastava, (1989), pp 49-50), variance also increases when the block is of higher grade. Therefore the block grades (from David, (1977)), are well estimated as long as they are dimensioned greater than ¼ of the drill spacing. Because the drill spacing is based on some fraction of the variogram range (say ½), this “¼ rule of thumb” would appear to be appropriate. Many smaller mining companies use large drill spacing as a cost cutting measure. As will be seen, the money spent using obscure computer data processing methods to determine “bankable” resources and reserves would be better spent on the extra drilling.

The variance within a block (or SMU if/where appropriate) can be found from the variogram by placing a regular grid of points (pseudo-samples) in the block or SMU. From the variogram, the average variance of samples in blocks (and variance of samples within SMU’s) can be found by averaging the variances between all the (pseudo) samples in the particular volume. The average variance within blocks and average variance within SMU’s can then be estimated. By the volume/variance relationship, the variance between (or among) the SMU’s within a block can then be found. It is not necessary to drill the pseudo samples as it is variance, not grade, which will determine the distribution of SMU grades within a block, and variance is known from the variogram and “proportional effect”. It is important to distinguish “within” as opposed to “among or between”.

\[
\text{Variance (samples in Block)} = \text{Variance (samples in SMU)} + \text{Variance (SMU’s in Block)}
\]

Expressed as average variogram values (denoted by the bar \( \bar{\gamma} \)) and using “s” for (pseudo) samples, B for block and SMU as itself,

\[
\bar{\gamma}_{s \text{ in B}} = \bar{\gamma}_{s \text{ in SMU}} + \bar{\gamma}_{\text{SMU in B}}
\]

Where \( \bar{\gamma} \) comes from the sample variogram (or similar) which describes variability and ‘geology’.
The variance of SMU’s in Blocks \( \gamma \) can now be found using the variogram, but variances will change with SMU size/shape and block (B) size/shape. The grade distribution of the SMU’s in blocks will often be positive right skewed (and even lognormal for gold) to some degree. The variance of SMU’s in Blocks will vary depending on the SMU size and the proportional effect and is dictated by the volumes equipment can mine selectively (e.g. 2 m\(^3\) front end loaders versus 60 m\(^3\) cable shovels).

\[
- \gamma \text{ SMU in B} = - \gamma \text{ s in B} - - \gamma \text{ s in SMU}
\]

From the variogram and (pseudo) samples, the average gamma (gamma bar) of (s in B) and (s in SMU) are known and if SMU and/or Block sizes change, (SMU in B) will change. The proportional effect (variance is proportional to grade squared) can also be included once the proportionality constant is known (from plotting grade\(^2\) vs variance for samples within large overlapping volumes in the deposit. Typically the variance of copper is proportional to \(\frac{1}{4}\) of the grade squared. For gold the variance is typically equal to the grade squared. From these relationships, given that the block size is 60x60x15 m and the 16 contained SMU’s are 15x15x15 m and the variogram is that for 15m composites in the ore-body under study, 16 factors can be estimated (figure 4.2.3.f) which when multiplied by the block grade provide the grade distribution and grades of the 16 contained SMU’s.
Figure 4.2.3.f  SMU factors for 16 15x15x15m SMU’s in a 60x60x15 m Block. SMU’s are placed within the block on a regular 4x4 grid. The location of each of the 16 SMU’s within the block is unknown, but will be present when production drill cuttings sampling defines the SMU locations. The average grade of the SMU’s must be the same as the block grade, or contained product (grades) will be biased, a situation which cannot be allowed and is adjusted for by multiplying by the ratio of block grade and average SMU grade. The copper data comes from Bell Copper mine, Babine Lake, B.C. and the gold data from Barrick Goldstrike in Nevada.

The above theory is described as “change of support” where the block and SMU are of very different sizes (support), and can be proved in practice using exploration and production blast hole data (Blackwell, G.H. (1998) “Geostatistics Course Notes”, Dept. of Mining Engineering, Queen’s University at Kingston). Typical example results for copper with a block grade of 0.5 % Cu would have the lowest grade SMU as (0.6x0.5) or 0.3 % Cu, the highest grade SMU as (1.4x0.5) or 0.7 % Cu, and the mean grade of the 16 SMU’s as 0.5 % Cu. If the cut-off grade were 0.6 % Cu, the block would be waste. By changing the support from 60x60x15m to 15x15x15m, three SMU’s in the waste block are ore which can be selectively mined. The variance of SMU’s in blocks is slightly higher using actual data because extra
variance is introduced by the nugget of blast-hole and to a lesser extent, exploration data used to confirm the theory. Khosrowshahi, S., R.L. Gaze and W.J. Shaw (1999) in “Change of support for recoverable resource estimation”, SME Annual Meeting, Denver, Colorado, March 1-3, provide a description of alternatives for recoverable reserve estimation.

Despite gold grades being traditionally difficult to estimate, results using the same method with Barrick Goldstrike data are also good (figure 4.2.3f). Although in the case of copper, and with the block and SMU dimensions chosen, the grade distribution of SMU’s in blocks can be approximated as a normal distribution. The problem with gold is that the distribution is skewed, approximately log-normally, and this is difficult to duplicate using the variogram and change of support method. Using a 20 gram/tonne block grade the lowest grade SMU would be 10 g/t and the highest 40 g/t. The average SMU grade is equal to the block grade despite the pronounced skewness of the distribution shown in figure 4.2.3f.

For feasibility and long range mine planning purposes the SMU’s and their grades are randomly placed as a grid inside the block. This is not accurate as the SMU’s also have a variogram based on (usually) 6 individual cuttings assays per SMU. Methods such as “turning bands”, Emery, X. (2007), (http://www.captura.uchile.cl/bitstream/handle/2250/6994/Emery_Xavi.pdf) can be used to better locate the SMU’s. However, for long term planning and feasibility the knowledge of the existence of the SMU’s given they are only a maximum of (say) 60m from their true location is sufficient. The blast-hole drill cuttings will be used as the ultimate decision making data.

The mining industry has been reluctant to accept the maxim of block size being related to drill spacing, preferring to use small blocks whose grade estimates are almost meaningless swathes of ore or waste.
seldom found in practice. This leads to inappropriate selection and small numbers of large equipment which cannot mine selectively. The selective mining unit (SMU) is required for tonnage and grade purposes in long and short term planning. A typical definition of an SMU can be the tonnage which equipment can dig in a shift, usually 15m x 15m in plan for most large loading equipment. This definition implies correctly that in blocks, “ore” will contain some waste and “waste” some ore, and the loader must be sized to differentiate the ore/waste boundaries. Costs are also an issue as small equipment can dig more selectively but have higher operating costs and vice-versa. The solution is to select equipment such that selective mining and better grade control of small SMU’s is balanced by higher operating costs.

**Confirming the Accuracy of the Resource Block Model**

The efficacy of the grade model must be tested against the mill production. This is usually carried out on a monthly basis, assuming the error from material in transit through the mill stockpiles is small. The material in the stockpile may be a month or so old and of lower grade. This may be the case where there is a grade/rock-size relationship. This relationship means that softer more friable ore is transferred directly to the mill grinding circuit and harder larger gangue passes over screens and is placed on a last-in/last-out stockpile for secondary crushing. Sizing the blast-hole cuttings shows that this may be the case, larger cuttings having smaller grades.

The grade reconciliation consists of finding the average grade of all blast-holes sent to the mill versus the mill head grade. Extremely high blast-hole grades (defined as the grade where the grade probability plot of individual blast-holes deviates from the general trend or slope) are reduced to the threshold where the departure from the general slope occurs. The blast-hole grade control will now be discussed.
The reason that blast-hole sampling and assaying procedures are inadequate is that each blast hole produces a few tonnes of cutting, depending on the hole diameter, which cannot be adequately sampled in a cost effective manner, figure 4.2.4.a. The first sample cut from a few tonnes of mixed gravel, sand and fines to say 2 kilograms breaks the sample reduction criteria of Gy (1979) and such samples and the assay values derived will never be adequate. This inadequacy appears as an increased “nugget” when the diamond drilling and blast hole sampling are compared as variograms. It is also of note that should reverse circulation drilling be used in the place of diamond drilling a similar but less severe problem becomes apparent, especially where the drill intersects the water table.

Figure 4.2.4.a  Left is the BE 60R drill at work. A hole and cuttings pile is located in the foreground right. The sampling boxes and riffle box are located foreground centre. Cuttings are placed in one box, riffled to halve the sample and repeated until about 1 kilo of sample is bagged for assay. Right hand side
shows the filled sample cutter (a 100mm diameter steel pipe sharpened at one end with a handle on the other) being emptied into the box. Twelve “cuts” are taken from each hole, and the indentations in the cuttings pile are evidence that the sample was taken (or a boot was used to make the indentation!)

Typical Gy safety lines are shown in figure 4.2.4.b. Water encountered in blast-holes can also wash much of the sample away in the cuttings removal process. In drilling below the water table with reverse circulation drilling, the sample retrieved is a mud and not easily cut. The water also washes soft ore bearing material from higher up the hole as the hole is cleared of water in the cuttings clearing process.

Because porphyry copper and epithermal gold deposits are often sub-vertical, material from short term planning after close blast holes have been drilled and sampled is used to extend this data for a few benches downward along the sub-vertical axis (Norrish and Blackwell, 1986). Such data, if well collected and sampled, can be added to the data base and grade models revised at regular short intervals. Unfortunately this new data, although numerous and concentrated, is not generally as reliable as diamond drill data.

Given the problems described above, the planning engineer must still provide a logical and accurate plan for the operation. Fortunately the short term planner has the blast hole assay data to work with, either as drilled or as interpolated from the benches immediately above. The inexperienced long term planner, although working with limited data, has the grade simulation tools described by Anderson (1999) to use. The long term planner is more concerned with the grade distribution and tonnages on a bench being correct, the short range planner also depends on this information but is more interested in precisely where the grades are located and at what tonnages.
Figure 4.2.4.b  Typical Gy sampling safety lines for top left diamond drilling, top right reverse circulation drilling, bottom left production blast-hole drillings, bottom right pilot plant. Only the diamond
drilling and pilot plant produce sample cutting procedures that are adequate for copper porphyry deposits. For gold deposits further sampling and cutting require a more rigid procedure.

When mine and mill tonnages and grades differ it was historically common to use such as a “mine call factor” (say 90-95%) to multiply the mine head grade to match the mill value (Truscott, 1937, 1962). Grade and tonnage factors are generally in this range in porphyry and epithermal deposits. The diluent is material in close proximity to ore and has a grade close to that ore. The tonnage is estimated from surveyed volumes mined and truck counts from which the density is estimated. The usefulness of compiling such data on the mine side of the accounting procedure is to ensure that no “blunders” have been made. In the Brenda, Barrick Goldstrike and Bell cases there were no serious discrepancies in mine/mill grade and tonnage balance.

Blackwell (personal communication) relates a serious positive molybdenum grade discrepancy after the mill month end material balance was completed. This was due to a clean-up below the molybdenum float cells increasing molybdenum production. The mine/mill personnel relationship was excellent, and the clean-up was carried out intentionally to see if the mine would complain that the mill was inventing grade.

Such excellent correlation between mine and mill is not common, and can result in difficult relationships. However it is the mill weightometers on the conveyor belts that determine tonnage, and short of having mine personnel present when the weightometers are calibrated (using heavy rolling chains), the mill tonnages must be accepted. The mill grade can be incorrect if the “metal balance” is not conducted properly. The tailings are a huge tonnage of slimes containing very little metal. The product is a small tonnage of high grade concentrate. The mill “metal balance” can be manipulated to improve recovery (a
value on which the mill is judged), but operations mining low grades such as those used as examples in this thesis have near constant tailing grades. Recovery is then better with higher head grades. The concentrate is measured by the smelter/refiner and it is these values that the mill in turn must accept within the bounds of the smelting contract described in section 4.3.

The remaining reconciliation is that the estimates of grade (and tonnes) from the resource model based on exploration data match those of the blast-holes. The first model to be compared is that for 60x60x15m blocks, and the second for 15x15x15m SMU’s. To accomplish this task, four graphical methods can be used, and these four have some built in redundancy,

- grade probability distributions
- X/Y plot of grades
- comparative grade/tonnage curves
- comparative plan plots of grades

Grade Probability Distributions
In the first plot (Figure 4.2.4.b) for 60x60x15m blocks, the blast-hole data has a higher variance, possibly due to the addition of the higher nugget. However about 50 blast-holes are averaged to find a block grade so the nugget effect should be minimal (÷ 50 data points). The mean grades are 0.446 and 0.420 for blast-hole and kriging of copper grades respectively. Such an error (0.023%Cu) is extremely large, about equal to the tailings grade. Johnston, T. and G. Blackwell (1986) quote a grade comparison during the feasibility stage of the Brenda operation. Two exploration drill holes were placed to intersect an adit from underground workings at the site dating from the early 1940’s. The drill holes were sampled and assayed and a raise driven using the holes as a center. The raise material was run through a pilot mill for metallurgical testing and assayed accordingly. The raise material averaged 0.459% Cu, and the diamond
The loss is probably due to washing out of soft material including mineralised gouge zones common throughout the Brenda deposit, during diamond drilling. This would imply a 4% loss of copper from drilling. The raise and drill hole data is shown in Figure 4.2.4.a. However it is not possible to add a 0.04 percentile (4%) to grades. Such a process would render a project “un-bankable” and contravene NI 43-101 securities regulations.

Figure 4.2.4.a  Copper grades, diamond drill hole 15, repeat drill hole 15 RP and pilot mill data from raise assays. Raise assays taken during pilot plant milling are 10% higher than diamond drill holes would suggest. Unfortunately adding 0.04 percentile to grades is not “bankable” and should never be practiced. The loss is probably due to washing out of soft material including mineralised clay gouge zones common throughout the Brenda deposit. Assays of drilling mud indicated very high values of copper, but no process to use such data would be acceptable in a feasibility study.
Figure 4.2.4.b  Probability plot using normal scale. The two curves have a similar shape, but the increase in blast-hole grade is clear (0.42 versus 0.45). This implies a loss of copper in exploration data from washed out gouge zones.

The test using 15x15x15m SMU’s with an average of 4 blast-holes per SMU, shows a more serious problem when compared with ordinary kriging of the SMU’s (no volume-variance adjustment applied).

The blast-hole mean for blocks is 0.44 and for SMU’s 0.42. Only blocks with all 16 SMU’s present were used in the study, and blocks on the lower grade edge of the deposit would not conform to this constraint. Hence blocks should have a higher grade. The kriged mean for blocks is 0.42 and SMU’s 0.40 % Cu for the same reasons as the blast-hole data, blocks tending to be located in the higher grade centre of the ore-body.
Figure 4.2.4.c  Copper grades, 15x15x15m SMU’s. Blastholes are simple averages of holes falling inside the SMU. The kriging uses an octant search, 1 sample per octant, and no volume-variance correction. The octant search finds the same waste holes, often smoothing the grades.

Other points to note in the kriging of blastholes and SMU’s are the numbers of samples used in the kriging process. For blocks, the two closest samples per octant were used, plus the “ninth octant” for samples within the block. For SMU’s this was reduced to 1 per octant, otherwise the smoothing of grade becomes a major problem. It should be noted that the samples used in all cases are 15m composites, not individual small core lengths. There is no literature on the effect of number of samples chosen (large numbers smooth grades badly) or the effect of compositing. The 2/octant for blocks and 1/octant for SMU’s are based on experience and the respective volumes being estimated. The maximum distance sample-block is set at the variogram range for the estimates made, and the number of “pseudo-samples”
per block set at 4x4 and SMU, 2x2, again literature on the definitive values for these values is not available.

Figure 4.2.4.d  Probability plot comparing blast-hole SMU composites with kriged blocks (60x60x15m) converted to their sixteen “child” SMU values (15x15x15m) by the volume-variance distribution. There is some overestimation of low grades (<0.2%Cu) and underestimation of grades from 0.4 to 0.9 % Cu. The mean grade of the SMU’s (0.4%Cu) is lower than the blasthole (0.42%Cu) probably from core losses in clay gouge zones. The results are not perfect, but there are several factors involved in the systematic shape of the error.

The blast-hole SMU’s are compared with the SMU’s derived from kriging 60x60x15m blocks and applying the SMU volume-variance factors. The hoped for perfection where the two distributions are almost the same is not achieved. There is systematic error from overestimation of low grades (<0.2%Cu) and underestimation of grades from 0.4 to 0.9 % Cu. However, the results are the best achieved and can
be accepted. Further analysis of the problem is beyond the scope of this thesis, but producing a grade distribution which predicts the blast-hole data would be is a goal for future research.

The X/Y plot of blasthole and kriged grades

The X/Y plot for 60x60x15m blocks has a correlation coefficient of 72% which means, in simple terms, 72% of the variability between kriged blocks and blasthole averages can be explained by a simple X=Y equation. The Y intercept is close to zero and the slope to unity. This confirms David’s ¼ rule and that it is easy to estimate the average grade of bigger blocks. The graph is shown in Figure 4.2.4.d.
Figure 4.2.4.e  Copper grades. X/Y plot of blasthole averages of 60x60x15m blocks and their kriged equivalent, the correlation is excellent for natural resource data, and the slope and intercept imply that the kriged grade approximates the blast-hole averages very well (72% explained).

The grade X/Y plot for 15x15x15 SMU’s in Figure 4.2.4.e is far more dispersed with a percent explained by X=Y of 55%. The equation X=Y still holds, but deviation from the equation can be expected to be large.

![BLASTHOLE vs KRIGED GRADES](image)

<table>
<thead>
<tr>
<th>BLASTHOLE vs KRIGED GRADES</th>
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</thead>
<tbody>
<tr>
<td>% Cu 15x15x15m SMU’s</td>
</tr>
<tr>
<td>Brenda Mines 4860 Bench</td>
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</tbody>
</table>

Figure 4.2.4.f  The scatter cloud for predicting 15x15x15m blast-hole SMU’s from kriged SMU’s is far larger than that for 60x60x15m blocks, confirming David’s rule that blocks should be no smaller than \( \frac{1}{4} \) the drill spacing (in this case about 100m).

Even more scattered, with a correlation coefficient of 38%, is the blast-hole grade vs. SMU grade from using the volume-variance relationship and randomly placing SMU’s inside their parent block. This is not
surprising as the placement could result (in the worst case) in the highest grade blast-hole SMU being compared with the lowest of the 16 SMU grades and vice-versa.

**The Grade-Tonnage Curves**

The next comparison involves the Grade/Tonnage (GT) curve. This curve provides the user with the fundamental worth of a mine (or bench in this case). The X axis is cut-off grade which can be transformed to revenue or cash flow quite easily. At cut-off zero, the grade is the “deposit” bench average and the ore tonnage is the total tonnes in the bench. By mining material below cut-off as waste, the average grade for “ore” on the bench increases, and the tonnage of ore decreases. If this curve is incorrect, then the mine will be under or over-valued significantly.
Figure 4.2.4.g Grade tonnage curve for copper, 60x60x15m blocks. The grade curve shows the kriging grade for this block size is less than the blast-hole grade for the same 60x60x15m block. This loss of grade is probably due to core losses, but reduces to almost no difference at 0.5 % Cu cut-off. The tonnage curve shows a small loss of ore tonnes which is insignificant at a 0.35 % Cu cut-off. Generally it can be stated that bigger blocks, less than ¼ drill spacing, are better predicted.

Figure 4.2.4.h Grade Tonnage curve for 15x15x15m SMU’s, blast-holes versus kriged SMU’s. The grades show that the kriged SMU’s, with no volume-variance correction, always underestimate grade by about 0.05% Cu. Using the raise test (Figure 4.2.4.a) a 0.1 percentile (10%) increase in Cu at 0.4 cut-off increases 0.5%Cu to 0.55%Cu which is approximately the case. The tonnage appears to be biased high below 0.33 % Cu and low thereafter which is where the cross-over occurs in the probability plots.
Figure 4.2.4.i  Grade Tonnage curve for 15x15x15m blast-holes versus 15x15x15m Child SMU’s from volume-variance with 60x60x15m blocks. The grades show that the child SMU’s, after the volume-variance correction, always underestimate grade by about 0.04% Cu. Using the raise test (Figure 4.2.4.a) a 0.1 percentile (10%) increase in Cu at 0.4 cut-off increases 0.5% Cu to 0.55% Cu which is approximately the case. The tonnage appears to be biased high below 0.35% Cu and low thereafter which is where the cross-over occurs in the probability plots. This plot is slightly better than the plot for kriged 15x15x15m SMU’s with no volume-variance application (figure 4.2.4.h)

The 60x60x15m kriging block model prediction of grades and tonnes is adequate given the input diamond drill assays. This estimation is referred to as the “parent” block model. The 15x15x15m model using kriging is inadequate. By using volume-variance factors based on the variogram, the 15x15x15m “child” grades based on the parent block model are better, but require more investigation. There is no readily available information in the literature on such as sample/block maximum distance as a proportion of variogram range, the number of samples chosen per octant, and the number of “pseudo samples” used to describe the block.
The Typical Plan of Grades on a Bench

Information on what sort of patterns of grade, and how large and dig-able areas of various grade distributions (waste/stockpiles/ore) are is invaluable to the mine feasibility and planning engineer. If grades occur as large swathes of individual products, planning is easier and small numbers of large equipment will be suitable. If, however, the products are mixed together and adjacent SMU’s are of very different material types, e.g. waste adjacent to ore, then grade control will be more difficult, recovery low and dilution high. In this case large numbers of small equipment are required, and unit costs will be higher and bench heights lower. This results in reduced productivity and redundancy of digging equipment such that if expected material is not available, it can be dug from other benches.

Figure 4.2.4.j (left) shows the 15x15x15 m blast-hole SMU’s made up of averaging the (usually) 4 contained and assayed blast-holes. This is the reality that faces the mine planning engineer when designing blasts and preparing the shovel digging plans and moves to ensure the required production. These grades are shown in plan on the left hand side of the figure. The centre of the figure shows the result of kriging the SMU’s directly, a process very common in the mining industry assuming the exploration hole spacing is small enough. The right of the picture shows the result of the process of;

- finding the volume-variance relationship (factors) of sixteen contained child SMU’s using the variogram (or similar)
- kriging 60x60x15m parent blocks
- finding the grade of each of the 16 SMU’s from the kriged block grade
- placing the 16 child SMU’s geometrically within the block

The north arrow is shown and the north arrow line is 500 m. The grade legend is shown for 0.1% Cu intervals, with the final interval greater than 0.5 % Cu. The small gaps in the blast-hole layout are the
result of the shovel “free digging” softer material without drilling and blasting, and the small number of blocks left out of the study in all cases.

![Diagram of Cu grades on 4860 bench](image)

**Figure 4.2.4.j** Plan plots of Cu grades on 4860 bench. The plots represent (starting on the left) the raw blast-hole assays (usually 4 averaged) for each 15x15x15 m SMU. Centre is the kriging of the same SMU’s. Clearly the mixed grade mosaic of the blast-holes is not replicated by kriging SMU’s. Using such a kriged output at the feasibility stage would indicate small numbers of large equipment. The right hand plot shows the result of kriging appropriate size blocks (60x60x15 m) which are sized greater than ¼ the exploration drilling spacing of 100m. The volume-variance relationship is used to find 16 SMU factors to multiply by the kriged block grade to find the SMU grades which are randomly placed within the parent block.

The conclusions from the application of the volume-variance relationship are clear. Under no circumstances should SMU’s be kriged directly when the exploratory drilling is conducted on a large spacing. The solution is to krig suitably sized (> ¼ drill spacing) blocks and apply the SMU factors to the block grade. There are other ways to accomplish this such as probability or indicator kriging, both beyond the scope of this thesis. The placement of the SMU’s should also reflect the variogram of the
SMU’s, again a task beyond the scope of this thesis involving such as “turning bands”, (Emery, X. (2007)).

The best solution is to diamond drill at a suitable small spacing prior to estimating resources.

The left and right plots are clearly similar given that the presence, but not the location of SMU’s in the block is found. By randomly placing the child SMU’s in the parent block, one per spot in the sixteen places in the block, a true picture of the grade plan is clear and enables the feasibility study to consider smaller equipment and greater unit costs per tonne mined. The raw blast-hole data plot indicates that the block plus volume-variance SMU’s are underestimates of the grade. The recovery of cuttings and sampling procedure (see Chapter 4, section 4.2.4) are good but have a high nugget (the Gy crush and cut cycle for samples cannot be economically carried out). Variance of the blast-hole data is therefore higher than reality. Only when excess water is present (usually during sinking in spring run-off) are the cuttings unrepresentative due to the washing out of fines (better grade material).

The exploratory diamond drilling assays are subject to continual inspection and check sampling and analysis by other laboratories, and can be assumed to be excellent. However, core losses from washing out of clay gouges reduce sample grades. These two effects (blast-hole and exploration results) combine to make the right hand predictor of lower grade than the left hand target to be predicted. Not much can be done about such problems which would not be “bankable” and would fail securities regulations including NI 43-101.

**Classification of Grades and Tonnes of Resources**
In the mining industry the definition of ore is contentious. Good thought provoking definitions of ore include "Ore is rock that may be, is hoped to be, is or has been mined and from which something of value may be extracted" (Taylor, H.K. 1991) and "the valuable solid material that is sought and later extracted from the workings of a mine for the hoped or expected (though not always achieved) advantage of the mine operator or for the greater good of the community" (Taylor, H.K., 1986). Perhaps "dug up for money" is equally fitting.

The classification of mineral resources generally has two components, increasing level of geologic knowledge and increasing confidence in economic considerations. These concepts are combined in the "McKelvey Box", McKelvey, V.E. (1972) on which most is classification systems are based.

<table>
<thead>
<tr>
<th>Cumulative Production</th>
<th>IDENTIFIED RESOURCES</th>
<th>UNDISCOVERED RESOURCES</th>
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<tbody>
<tr>
<td></td>
<td>Demonstrated</td>
<td>Inferred</td>
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<tr>
<td></td>
<td>Measured</td>
<td>Indicated</td>
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<tr>
<td>ECONOMIC</td>
<td>Reserves</td>
<td>Inferred Reserves</td>
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<td>Marginal Reserves</td>
<td>Inferred Marginal Reserves</td>
</tr>
<tr>
<td>SUB-ECONOMIC</td>
<td>Demonstrated Subecon.</td>
<td>Inferred Subecon. Reserves</td>
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</tbody>
</table>

Other Occurrences: Includes nonconventional and low-grade materials

Figure 4.2.5.a The McKelvey Box with geologic knowledge as (X) and confidence in economics as (Y). The "ECONOMIC IDENTIFIED measured, Indicated and Inferred" categories are common to nearly all natural resource classifications including coal, industrial minerals and oil as well as metalliferous mines and are fundamental to the NI 43-101 regulations.
The common categories into which resources are placed are “measured, indicated and inferred”. Given
the need to protect investors from enthusiastic over-reporting to outright fraud, the Ontario Securities
Commission (OSC) and Canadian Securities Administrators, with the input of mining interest groups,
developed National Instrument 43-101 covering most aspects of disclosure for mineral properties.

The National Instrument also defines who may prepare reports and draw conclusions, and the format of
such reports. The reports are made available to the general and investing public in “SEDAR” which is the
electronic system for the official filing of documents by public companies and investment funds across
Canada (and often internationally). Its purpose is “to provide protection to investors from unfair, improper
or fraudulent practices and to foster fair and efficient capital markets and confidence in capital markets”.

The report must be compiled by or under the direction of a “qualified person”. The definition of
"qualified person" requires a university degree or equivalent accreditation in an area of geoscience or
engineering rather than a membership designation in a professional association. A qualified person (QP)
must also be subject to continuing professional development within the professional associations.

The placing of material into measured, indicated or inferred is still the opinion of the QP which is
subjective. Generally for porphyry or epithermal deposits, the variogram (or similar) range is used to
determine drill spacing. The resource category is then determined by such as

- the number of drill assay intersections used in the kriging estimate
- the number of intersections closer than (say) half the variogram range
• the requirement of close intersections in a majority of octants (upper and lower quadrants)

Only the last of the above requires data around the unknown volume. The number of intersections required is subjective, and using a number which provides the required result is common. This information is made public via SEDAR and the reader (investor or investor’s adviser) can draw their own conclusions. The intersections used for a particular estimate may, in the worst case, be from one octant, so the need for a measure of the geometrical dispersion, closeness of samples and number of samples is required. This is analogous to the DOP (dilution of precision) of co-ordinates found from GPS (global positioning system) when satellites are low on the horizon and bunched in one area resulting in poor precision of co-ordinate data.

A solution to the problem is given by Blackwell (1988) where the kriging error (a natural product of the kriging process) is used in a similar manner to the DOP. It was found that misclassification due to sample geometry, distance away and number of intersections was reduced. By using kriging standard deviation (the square root of kriging variance) and making variogram sill values unity regardless of metal (copper porphyry or epithermal gold) it was found that these RKSD values could also be used to define measured (RKSD<=0.3), indicated (RKSD>0.3 and <=0.5) and inferred (RKSD>0.5).
COPPER SMU RKSD vs # EXPLORATION DATA (BELL COPPER)
RESERVE CLASSIFICATION BASED ON RKSD

Figure 4.2.5.b  Plot of the relative kriging standard deviation versus number of samples the block or SMU grade was estimated using. The RKSD is a far better indicator of certainty that grades will be found as it is based on number of samples used, relative geometry of sample/sample and sample/block, and distance sample/block. The RKSD values of 0.3 and 0.5 are arbitrary but seem to fit both copper porphyry and epithermal gold. Note the “relative” is relative to a sill of unity and the variogram (or similar) is scaled (in X variance) to fit.

The relative kriging standard deviation (RKSD) can be seen on a drill section of the Brenda operation. Blackwell, (1998) discusses RKSD at length, and the figures that follow are based on this paper. On section 12650 east (looking west) the areas where lots of holes intersect are shown lighter. Where holes are sparse, the lower RKSD values follow the hole closely, and the lone hole in the south (left) drilled at 40 degrees dip to the north is a good example.
Figure 4.2.5.c  Section showing diamond drilling on drill “line” (top) and resulting RKSD values below. The better RKSD’s follow the drill path (lighter shading) except where there are fewer holes in a particular vicinity.

In plan a similar plot, figure 4.2.5.d is used to highlight areas of high RKSD and (in this case) material near cut-off. This plot can then aid in drill layouts to improve the knowledge of the ore-body and can be applied to sections or to a three dimensional block model showing the mine geologist/engineer the best drill hole locations.
Figure 4.2.5.d  Top left is a plan of the diamond drilling. Holes are mainly drilled to the south at 45 degree dip or vertical. Top right are shown the RKSD values on 4860 bench. The better estimates follow the north-south drill lines. Bottom left are the grades on the bench. The objective is to be more certain of the material around the cut-off grade shown in the bottom center. Superimposing the top right and centre plots gives the plot bottom right. This shows material near cut-off with a high RKSD, the target areas for more drilling.
The presence of drill holes which are not correctly located can also be determined. Individual diamond drill holes are sequentially removed from the data base and the grade at the each composite location kriged. A score for each hole is made summing the absolute value of the difference between composite grade and “back-kriging” of the location. Analysis of the absolute value of the error can then be used to find the worst back-kriged individual composite locations, and individual hole locations. Analysing the error values as a normal probability plot to find which back-kriged values are greater than two or three standard deviations from the mean provide a sense of which errors are important. Using three dimensional displays may show the presence of clay gouge zones, aquifers or even which drill crew is providing the most representative core material.

**Conclusions from the Resource Block Modelling Process**

At this stage it is possible to produce a “resource” as defined in CS 43-101, or at least be in a position to determine the extra drilling, sampling requirements and methodology (variography, kriging, block and SMU definition) to produce one. Improvements to the categories of grades not included in the “measured” category of an existing resource should also be sought. It should be clearly understood that a “resource” no matter how good, is not a “reserve”. To become an “ore reserve” there must be a mining plan and if implemented, resulting profits. It is unlikely that all the “resource” will be mined because of such as depth, stripping ratio and etc. Profitability has an element of risk which is highest in the “inferred resource” category. The interpolation/extrapolation method used for each block and SMU is again subjective, and typically estimates of grade and tonnage and their grouping into such as measured, indicated and inferred (decreasing certainty of existence) might be +/- 5% between various practitioners in the best case.
Given the sensitivity of “grade” to profitability, excellent estimates of grade and tonnage are essential to the development of mines, and on-going operations must continually compare “estimates” with the reality of material mined and processed and revise estimates accordingly. It might be prudent to finance extra drilling rather than consulting fees and software and computer costs to utilize obscure methods of improving resource and reserve accuracy.

Although it is possible to estimate the grade distribution of SMU’s in blocks, i.e. the recoverable or mineable resources, it is not possible to define the location of individual SMU’s other than within their parent block. This is quite acceptable for most blocks in the potential pit, but there may be “edge effects” where a block is cut by the pit wall. There is no guarantee that the proportion of ore SMU’s in the edge block will be duplicated in the part of the block inside the pit. Again, grade simulation (Anderson, J.M., 1999), might provide a suitable estimate. Rossi (1993) describes a method of using probability to define the most likely wall location by applying “least financial loss” (see Anderson, 1999).

Much research into estimation methods has been conducted because of differences in opinion as to the “mineable” grade and tonnage, and also into which certainty grouping estimates of grades and tonnes fall. Care has to be taken to quantify the “SMU grades and tonnes above cut-off”, a difficult value to estimate. If the average grade above some cut-off (often a grade which will just produce a cash flow or profit) is lower than expected, this may result in an unprofitable operation which would close in a short time. This is especially true for gold mines. Tonnage values (number of ore grade SMU’s times density and volume) may also be substantially lower resulting in expected annual cash flows but over a much shorter mine life. The volume (area) allocated for various stockpiles may be substantially smaller or larger.
Other aspects include bulk density estimates commonly to 3 figures (e.g. 2.64 tonnes/m$^3$) which are then used to convert say 123,456 m$^3$ to 325,924 tonnes. The “924” term is meaningless until bulk density values of 6 figures are available. Minor changes in waste and ore minerals in the rock make such 6 figure values meaningless. The “correct” tonnage term is 123k tonnes. Another aspect which should be considered is the sampling of a “constant grade and density ore-body”. Just the act of sampling where the variogram has a nugget and short range suggests areas above and below the constant grade which are not there in reality. If the “constant grade” is the cut-off grade, the ore portion is not there and the mine is not feasible. In well estimated resources, differences between practitioners are expected to be +/-5% (percentile). This is an indication of how far the “art” and regulations regarding resource estimation (e.g. Canadian Standard NI 43-101) and their publication in “SEDAR” have progressed.

The most important conclusion is that grade estimates can be made and verified prior to production. Once mine production starts, the grade modelling processes can be upgraded and lessons learned applied to future feasibility studies.