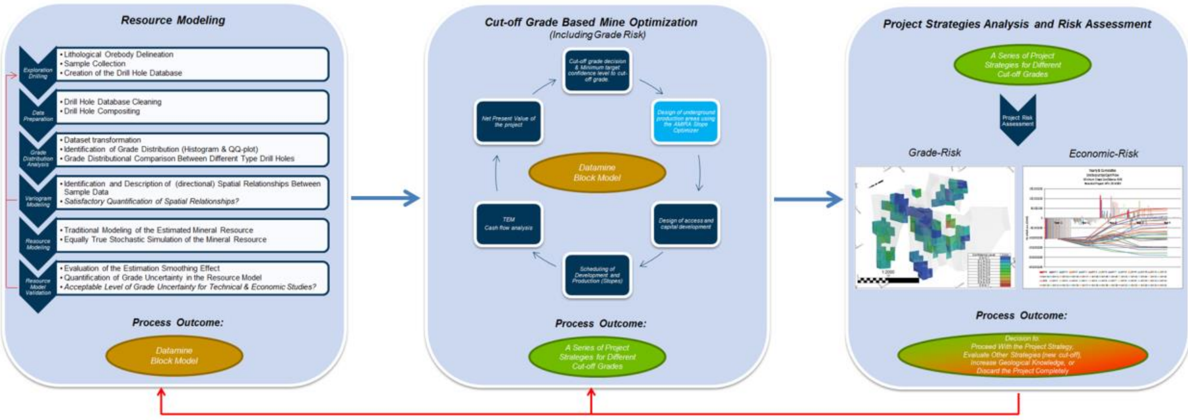
AES/RE/13-44 Cut-off Grade Based Sublevel Stope Mine Optimization.

Introduction and evaluation of an optimization approach and method for grade risk quantification

18-12-2013 M.T. Bootsma



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Title	:	**Cut-off Grade Based Sublevel Stope Mine Optimization.**
		Introduction and evaluation of an optimization approach and method for grade risk quantification

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I. ABSTRACT

The aim of this research project was to introduce and evaluate an optimization approach for sublevel stope mine optimization using the AMIRA Stope Optimizer that includes grade-risk quantification and can be used by Boliden engineers for future mine planning and economic assesment.

It has long been realized that mining simply every part of an orebody with a grade higher than 0 is not economical and will not lead to a successful mining operation. By selecting a cut-off grade it is decided what part of an orebody is economical to extract. In its most basic form the cut-off grade is a break-even grade which makes sure that each block of ore pays for its own mining, processing and refining cost resulting in zero gains and losses for a tonne of ore containing this grade (zero profit). As the aim of most mining companies is to maximize the Net Present Value of its mining projects it becomes clear that the break-even grade in many cases does not accomplish this goal.

Research in the field of cut-off grade optimization has proven that a relationship exists between the selection of a cut-off grade and project NPV. Although mathematical equations exist to optimize the cut-off grade and prioject NPV, they contain several simplifications of reality. The mathematical equations are therefore difficult to apply in real world problems and as a result the underground mine planning process is still a mainly manual time consuming process.

Boliden Mineral AB (Boliden) is a Swedish mining company that invested in the development of the AMIRA Stope Optimizer. The stope optimization software assists in the optimization of underground stopes at user defined cut-off grades and was developed to accelerate the time consuming underground optimization process. The AMIRA Stope Optimizer was evaluated to define its capabilities and limitations and it was concluded that the software is very valuable in strategic mine planning studies.

The stope optimization software was implemented in the mine optimzation process and succesfully applied to optimize a project strategy for one of Boliden's mineral deposits (Älgträsk). It was found that the optimum gold cut-off grade for this deposit is 1.8 g/t, resulting in a project NPV of 31MSEK. This is a 35% increase in project NPV compared to the break-even grade of 1.6 g/t.

The spatial grade uncertainty was identified as a major risk in underground stope design and therefore the optimization process was further extended to account for grade risk in mineral resources and subsequent stope optimization. Grade risk can be assessed by comparing estimated block models of a mineral resource (e.g. Kriging) with stochastic simulated block models of the same mineral resource. Because the simulations provide equally true, but different, interpretations of the mineral resource it is possible to quantify grade risk involved in Kriging estimation model based optimized mine designs.

The optimization process was adapted to account for grade risk early in the design process. Instead of optimizing the mining project at a certain cut-off grade based on the estimated model only and subsequently back-analyze the optimization outcome using the simulations, the simulated block models are used in the optimization process itself. By the application of a target confidence level to the desired cut-off grade, the underground production areas are designed to meet the cut-off grade in the estimated model, as well as a percentage of the simulated models. This will reduce the risk of a stope not meeting the cut-off grade which would result in a loss of money. By optimizing the mining project at different confidence levels (eg. 20%, 40%, 60%, 80%) the project economic risk can be quantified and used in decision making.

The implementation of grade risk resulted in the final underground mine optimization process as presented in Figure 1.

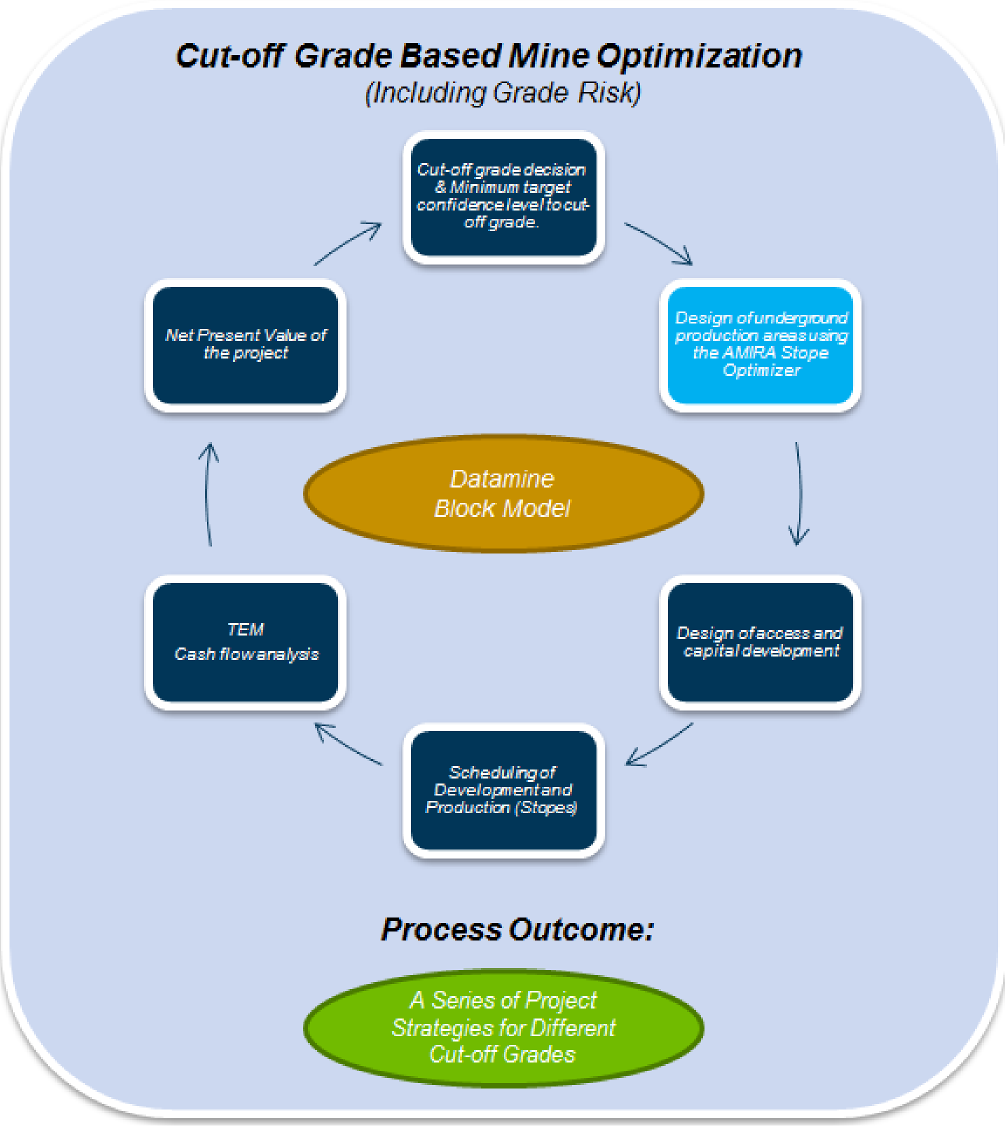

Figure 1 - Cut-off grade based mine optimization (including grade risk).

II. ACKNOWLEDGEMENT

This thesis research project completes my studies in Resource Engineering at Delft University of Technology. I would like to thank the many people who assisted me to complete this project.

First of all I would like to thank Stephen Manning of Boliden Mineral AB for providing the possibility to perform this research project, his guidance and especially for inviting me to Sweden and to work in the mine planning department. Performing part of the research 'on-site' certainly helped to keep up my progress and proved valuable at times when I thought the research was at a dead end. I would also like to thank the many other people in the office for their time and effort to clarify company approaches to mining projects, help me with software related problems and joining the presentations and discussions that resulted in new ideas to continue this research project.

Furthermore I would like to thank Mike Buxton (TU Delft) for supervising my thesis research project. I appreciate our meetings about my performed research and the discussions on how to extend it in order to bring my thesis to the next level. This resulted in a geostatistical study on grade risk in mineral resources and mine plans. I would like to thank Joerg Benndorf for his valuable advice on geostatistics as this was a topic of which I had little knowledge on forehand.

Special thanks go to Chris Alford. As the consultant behind the stope optimization software he has been my main contact for optimization software related questions and I am very grateful for the time he took to answer my e-mails as well as the two days we sat down together in order for me to understand the optimization principles. With a user manual of the software yet non-existing it is clear that this project would not have been possible without his help.

Last, but certainly not least I would like to thank Saskia for her reminding me that sometimes things that seem simple, can in fact be (!) as well as her patience whilst I was working on my thesis. Especially when I moved to Sweden, subsequently extended my stay with another month and finally when I returned home but spent day and night working to complete this study.

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1 INTRODUCTION & PROBLEM STATEMENT

Mine optimization is an important factor in the success of a mining project. It can be described as the decision of what part of a mineral resource should be mined (and what part should be left in place) in order to maximize the return on a project.

It has long been realized that mining simply every part of an orebody with a grade higher than 0 is not economical and will not lead to a successful mining operation. By selecting a cut-off grade it is decided what part of an orebody is economical to extract. In its most basic form the cut-off grade is a break-even grade which makes sure that each block of ore pays for its own mining, processing and refining cost resulting in zero gains and losses for a tonne of ore containing this grade (zero profit). As the aim of most mining companies is to maximize the Net Present Value of its mining projects it becomes clear that the break-even grade in many cases does not accomplish this goal.

One of the most well-known theories in the field of cut-off grade optimization to maximize the NPV of a project is the one proposed by Lane (1988). He derived several equations to find the optimal cut-off grade that maximizes the NPV of a project. However, these equations assume certain parameters to be constant such as mining costs, fixed costs and production rate.

In open pit mining many optimization packages exist to assist in the design of the ultimate pit limit and development of production plans (cutbacks, mining rate). Once the resulting mining plan is completed it is relatively easy to calculate the resulting costs and hence calculate the optimal cut-off using for instance Lane's theory (Alford and Hall, 2009).

Mine optimization tools for underground mining do not exist to the extent of surface mining tools. This has to do with the fact that whilst changing the cut-off in surface mining may lead to the decision of sending a ton of ore to the waste dump instead of the mill (or leaving it in place), in underground mining the decision of a cut-off grade has a direct impact on the whole mining operation.

By selecting a cut-off grade the shape and size of an orebody changes. The shape and size of the orebody together with geotechnical considerations result in the selection of possible mining methods. Depending on the selected mining method the required development is designed, an appropriate production rate is chosen and equipment is selected. With the mining plan in place, it is relatively easy to calculate the profitability of the project through cash-flow analysis.

The underground mine optimization process is iterative and by selecting a new cut-off grade, the subsequent mine design and life of mine schedule may change as well resulting in a different project Net Present Value.

The process of finding the best mine plan can be summarized as the evaluation of multiple mine plans based on a range of cut-off grades and selecting the best strategy. Due to the lack of automized optimization tools the underground mine planning process can be very time consuming. Especially the design of stopes at different cut-off grades can take a significant amount of time. As a consequence of the process being very time consuming, underground mine optimization is in many cases not carried out properly resulting in sub-optimal project performance and loss of potential profit.

The AMIRA Stope Optimizer software package aims to assist the engineer in the rapid design of stopes at different cut-off strategies. Based on a cut-off grade and stope size limitations the optimization algorithm analyzes a geological block model of a mineral resource to find the economic zone and 'designs' appropriate mineable stopes that maximizes stope profit. Because the software uses the same stope size criteria at each cut-off evaluation, a consistent set of potential stoping evenlopes over a range of cut-off grades can be generated. These stope designs can be used, together with infrastructure design, for scheduling and economic analysis.

Because the design of underground production areas (stopes) is automated, more scenario's can be evaluated in the same period of time increasing the likeliness that the optimum strategy is found to optimize the project Net Present Value.

This research project focuses on the AMIRA Stope Optimizer and how it can be succesfully integrated into a sublevel stope mine optimization process for future mine planning at Boliden Mineral AB.

1.1 MOTIVATION

Boliden Mineral AB invested in the AMIRA P1037 Stope Optimizer research project that resulted in a tool for automated cut-off based stope design. A first version of the software package was recently released for commercial use and the main question for Boliden is whether the tool is an added value to the planning process in Sublevel Stoping currently applied within the company.

1.2 AIM AND OBJECTIVES

The aim of this thesis was defined as:

To introduce and evaluate an optimization approach for sublevel stope mine optimization using the AMIRA Stope Optimizer that includes grade-risk quantification and can be used by Boliden engineers for future mine planning and economic assesment.

To achieve this the following objectives were defined:

1. Evaluate the AMIRA Stope Optimizer capabilities and limitations
2. Develop an underground mine optimization process in which the AMIRA stope optimization software is integrated and test the process on one of the company's mineral deposits.
3. Investigate the possibility to quantify grade-risk in the optimized stope design and evaluate its' impact to the profitability of a mining project.

1.3 RESEARCH QUESTIONS

The objectives as stated in section 1.2 were investigated according to their respective research questions

1. *Objective: Evaluate the AMIRA Stope Optimizer capabilities and limitations*

- How does the AMIRA Stope Optimizer work?
- How does the stope optimization software perform on a simple test model with a known solution?
- Wat is the best way to setup an optimization scenario?
- How does the Stope Optimization software perform compared to manually designed stopes at the Lappberget 1250 level of the Boliden Garpenberg mine?

2. *Objective: Develop an underground mine optimization process in which the AMIRA stope optimization software is integrated and test the process on one of the company's mineral deposits.*

- What is the general approach to underground mine optimization?
- What is the current state-of-the-art approach to underground mine planning at Boliden?

- How can the AMIRA Stope Optimizer be integrated in a new underground mine optimization process for future mine planning at Boliden Mineral AB?

3. *Objective: Investigate the possibility to quantify grade-risk in the optimized stope mine design and evaluate its' impact to the profitability of a mining project.*

- How can grade-risk in a mineral resource model be quantified?
- Can the mine optimization process be adapted to account for grade-risk and reduce the risk of poor project performance?

1.4 REPORT OUTLINE

The report is composed of 6 parts:

1. Introduction
2. Theoretical Background
3. Stope Optimizer Evaluation
4. Case Studies
5. Conclusions
6. Recommendations

After the introduction to the thesis project, the theoretical background of the project is presented in Chapter 2. First an introduction to stope mining methods is presented with related geotechnical design constraints after which the AMIRA Stope Optimizer is introduced.

Subsequently, the theory to the Net Present Value and its relation to cut-off grade (Lane's theory) is presented. Based on Lane's theory an approach to mine optimization is proposed which is compared to the current state-of-the-art mine optimization process at Boliden Mineral AB. Required changes to the current mine optimization process as carried out at Boliden Mineral AB are presented to succesfully implement the AMIRA Stope Optimizer in the mine planning process.

Subsequently, the approach to quantify quantify the grade risk in a mineral deposit is introduced and a modified version of the optimization process is proposed to account for grade risk during Sublevel Stope mine design.

Chapter 3 covers an evaluation of the AMIRA Stope Optimizer and describes its capabilities and limitations.

Chapter 4 presents several case studies in which the concepts introduced in Chapter 2 are applied to real mineral resources.

Chapter 5 and 6 present overall conclusions and recommendations based on the performed research. The conclusions and recomendations are subdivided based on the objectives as stated in section 1.2.

1.5 RESEARCH SCOPE AND LIMITATIONS

This thesis introduces an optimization approach for sublevel stope mining operations and all research has been performed with this method in mind. Most of the mine optimization theory is generally applicable to different mining methods. The proposed optimization approach was tested on a tabular, vein type deposit but the practicality of the method was not tested on other type deposits. Further important details about included and excluded topics are presented in the table below.

	Included	**Excluded**
Considered Mining Method	Sublevel Stope Mining	All other mining methods
Stope Optimizer Evaluation	Sublevel Stope Mining methods	All other mining methods
	Performance comparison to manually designed stopes	Mathematical performance analysis of the optimization algorithm
Älgträsk Underground Study	Optimization of underground stopes	Optimization of mine infrastructure
	Cash Flow Analysis using standard costs and costs from comparable operations	Detailed mining cost analysis
	Single cut-off grade based stope optimization	Blending of stopes with different cut-off grades to increase total stope tonnage
	Basic sequencing to obtain a reasonable life of mine schedule	Stope sequencing to optimize the ore stream and improve project economics
Älgträsk Grade Uncertainty	Estimation and Simulation of the Älgträsk Area of Interest based on the Boliden block model	All other ore lenses of the Älgträsk Mineralization

2 THEORETICAL BACKGROUND

2.1 AN INTRODUCTION TO SUBLEVEL STOPE MINING METHODS

Underground stope mining can be described as the removal of (parts) of an orebody by creating underground openings (stopes) separated by pillars. The stopes can, depending on rock conditions and resource value, be left open which means that the pillar is left in place or they can be backfilled in order to mine out the pillar (pillar recovery).

Stope mining methods can be divided into three separate classes. Namely unsupported (e.g. sublevel stoping), supported (e.g. cut&fill stoping) and caving type methods (e.g. longwall or sublevel caving). These three classes depend on ore and rock strength. Further distinction of mining method is based on orebody geology and geometry (Tatiya, 2005).

Due to the geometry of the orebodies considered in this study (steeply dipping deposits) and the desire to move from supported to unsupported stoping methods, only sublevel stoping methods are discussed.

2.1.1 SUBLEVEL STOPING

In sublevel stoping (Figure 2) the to be mined orebody is divided into ore blocks separated by rib- and sill-pillars. At the bottom of the ore block the haulage level is constructed from which the ore will be drawn from the stope. The ore block is now divided vertically into sublevels from which production drilling can take place. Whilst the loading crosscuts are driven perpendicular to the strike of the orebody, the blasting sublevels are driven inside the orebody along strike. Blasting or the use of a raise borer creates an initial opening slot. This opening slot is required to create a free face for blasting. Production progresses along the strike of the orebody whilst ore is blasted into the open void. The size of the final open stope (or open void) is depending on the rock geotechnical conditions.

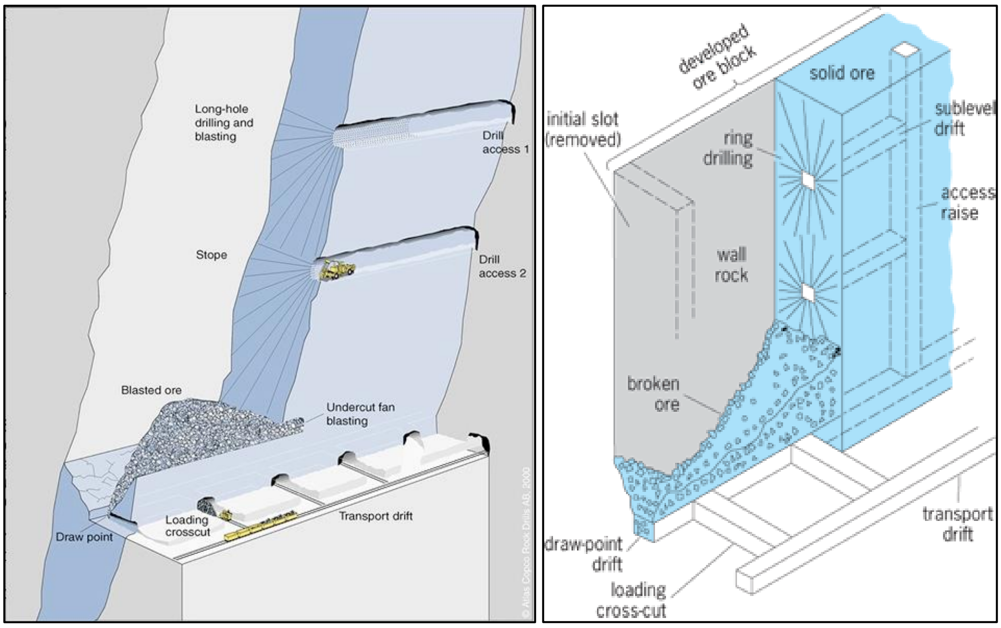

Figure 2 - Sublevel Stoping (tunnelbuilder ltd for Atlas Copco Rock Drills AB, 2007).

2.1.2 VARIATIONS IN SUBLEVEL STOPING

Several variations based on Sublevel Stoping have been developed over the years and their main goal has been to tackle some of the limitations involved in Sublevel Stoping. The main limitations of Sublevel Stoping are:

- Due to the large size of stopes in Sublevel Stoping the selectivity of mining is very low. Blast rings are blasted over the full vertical height of the ore block resulting in a single pile of ore so there is no potential for blending of the ore.
- Because a stope covers the full thickness of the orebody there is a limitation on the applicability of the mining method. The stability of the stope is affected by its span. How to calculate the maximum stable stope size (including the span) will be discussed in section 2.2.
- Rib and sill pillars are left in place to support the open stopes in Sublevel Stoping. Valuable ore may have to be left in place and the mining method has been adapted to avoid the use of these pillars or to be able to recover them.

2.1.2.1 TRANSVERSE STOPING

Transverse stoping is a variation on the sublevel stoping method in which the stopes and crosscuts are situated perpendicular (transverse) to the strike of the orebody. The mining method is applied in situations where the rock mass quality of the hanging wall limits the size of the open stope (or ore block). Because of the smaller stope size (stope size covers the height of 1 sublevel) more development drifts and crosscuts are necessary. Development usually takes place in the footwall where drifts are developed parallel to the orebody. The orebody is accessed from these drifts by the development of crosscuts at designed vertical intervals that coincide with the top and bottom of a stope. Production drilling and blasting takes place from the top of a stope and mucking of the ore takes place at the bottom.

The biggest differences compared to sublevel stoping are the fact that production progresses across strike (in the transverse direction) rather than along strike and the use of primary and secondary stopes to avoid the use of rib- and sill-pillars and increase selectivity of mining.

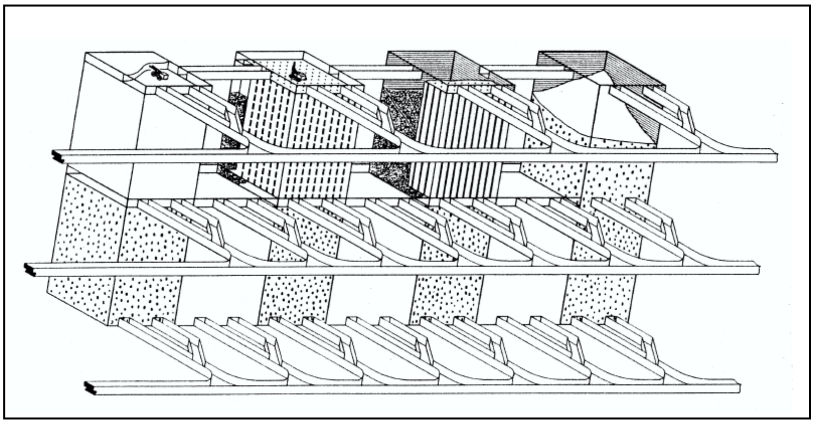

Figure 3 - Transverse Stoping (Hustrulid, 1988).

A typical Transverse Stoping layout is shown in Figure 3. From left to right it shows the full production cycle starting with the opening of the upper drilling level followed by production drilling, blasting and mucking. The final step of the sequence is the backfilling of the stope. By backfilling of the mined-out stopes it becomes possible to recover the rib pillars (secondary stopes) which are also backfilled making a sill-pillar unnecessary. The mining sequence of primary and secondary stopes is not only possible along strike, but can also be applied across strike making this method very useful in massive orebodies. This principle is shown in Figure 4 (Agnico-Eagle Mines Limited, 2012).

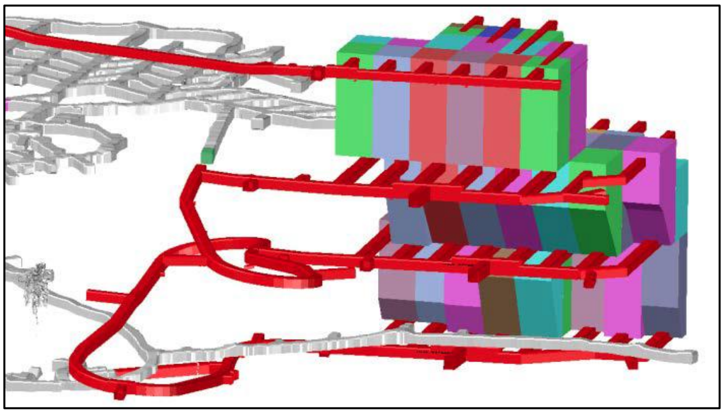

Figure 4 - Transverse stoping in a massive orebody (colours depict different stopes) (Agnico-Eagle Mines Limited, 2012).

2.1.2.2 LONGITUDINAL STOPING

Longitudinal Stoping is more similar to Sublevel Stoping than Transverse Stoping. In longitudinal stoping the direction of mining is in the same way as sublevel stoping along the strike of the orebody (longitudinal direction). The method can be applied in orebodies where the footwall dip exceeds the angle of repose (the ore can flow towards the drawpoint by means of gravitational force). Longitudinal stoping is designed for orebodies with a thickness in the range of circa 5-20m (Tatiya, 2005) although the number of sublevels that make op the stope void can be varied depending on stability. Because most of the development drives are constructed within the orebody, the method is significantly cheaper compared to transverse stoping.

Multiple variations of Longitudinal Stoping exist with the simplest being a retreat sequence from the orebody end towards the access point of the ore drive in a production cycle that starts with opening up the ore drive at the top and bottom of the sublevel. The next step involves the drilling, blasting and mucking of the maximum stable stope size after which the open void will be backfilled and left to cure. Once the backfill has cured, the adjacent stope can be mined, resulting in a retreat type mining sequence (Figure 5).

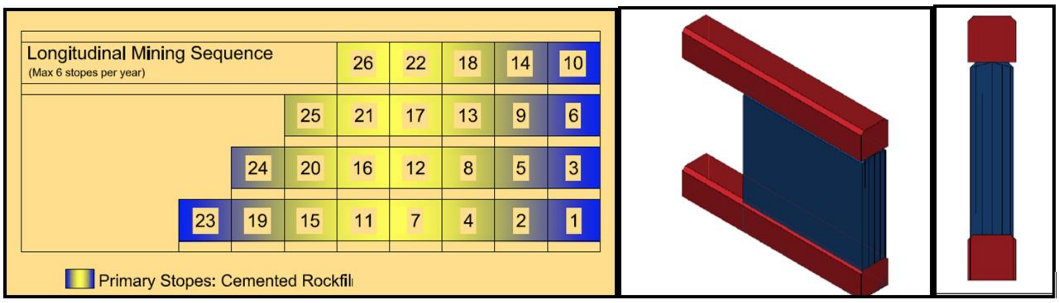

Figure 5 - Longitudinal Stoping in a retreat type sequence (Agnico-Eagle Mines Limited, 2011).

Multiple variations exist within longitudinal stoping which all differ slightly depending on rock conditions. Variations with and without backfill such as Longitudinal Uphole Stoping (Figure 6) or AVOCA (

Figure 7), which is a continuous operation of mining and backfilling but requires access to the ore drive from two sides therefore increasing development costs.

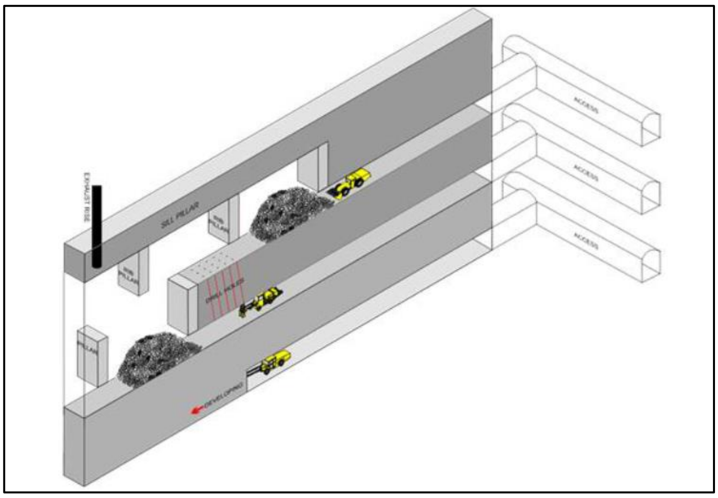

Figure 6 - Longitudinal Uphole Stoping using rib and sill pillars (Agnico-Eagle Mines Limited, 2012).

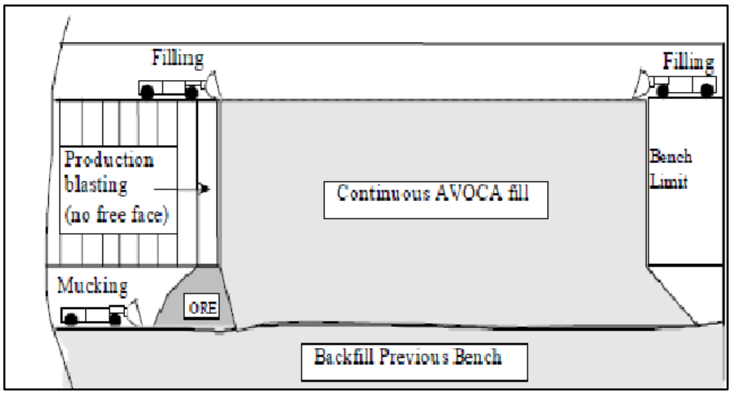

Figure 7 - AVOCA Longitudinal Stoping (Queen's University, 2012).

2.2 GEOTECHNICAL STOPE DESIGN

Sublevel stope mining techniques are characterized by the fact that at a certain point in time, the stope walls are unsupported and exposed. Because access to the stopes is limited to the sublevel ore drives, there is only a limited possibility for support of the exposed walls. This means that the method can only be applied successful if the stope walls are inherently stable. Experience has resulted in the graph as presented in Figure 8 (Villaescusa E. , 2000). It shows that in most cases stable stopes can be achieved by designing stopes having high vertical and short horizontal dimensions or stopes having short vertical and long horizontal dimensions.

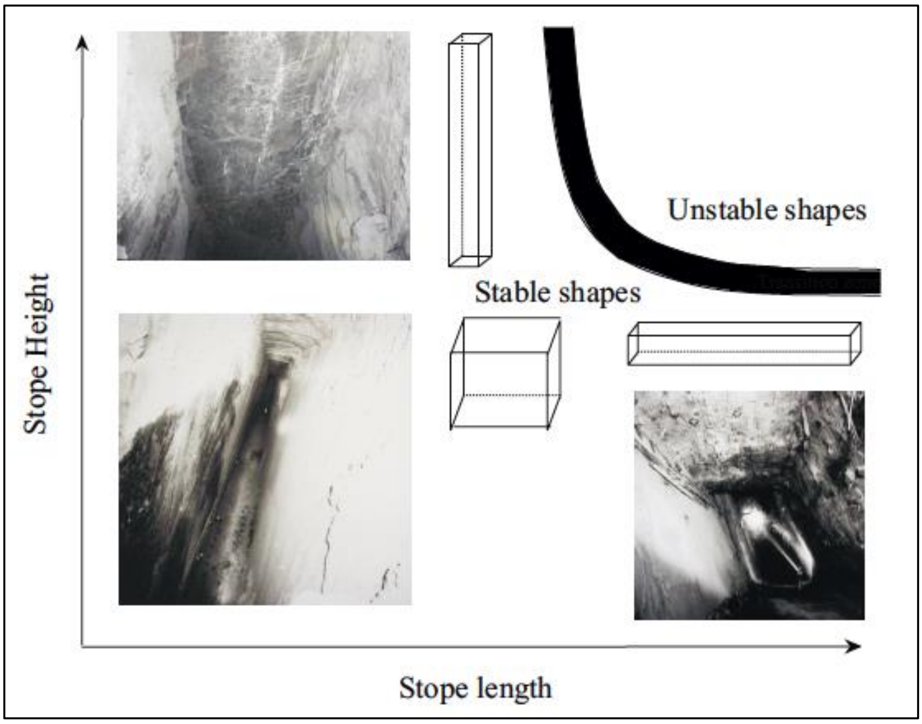

Figure 8 - Stable Stope Shapes in Sublevel Stoping (Villaescusa E. , 2000).

A widely applied empirical method to design stable stopes is the Modified Stability Graph Method (Mathews, 1980 & Potvin, 1988). The method is based on the Tunneling Quality index Q and takes into account the most important factors affecting stope design. The rock mass strength and structure, the stresses around the stope and the size, shape and orientation are used to determine if a stope design will be stable (Hoek, 1993).

Barton's Tunneling Quality Index Q is defined as:

$$Q = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF}$$

$RQD = Rock\ Quality\ Designation$

$J_n = Joint\ set\ number$

$J_r = Joint\ roughness\ number$

$J_a = Joint\ alteration\ number$

$J_w = Joint\ water\ reduction\ factor$

$SRF = Stress\ reduction\ factor$

Hoek describes the three quotient components as follows:

1. $\frac{RQD}{J_n}$: The Rock Mass Block size
2. $\frac{J_r}{J_a}$: The Inter-block shear strength
3. $\frac{J_w}{SRF}$: The Active stress and Water Pressure

Mathews uses a Modified Tunneling Quality Index or Q' defined as:

$$Q' = \frac{RQD}{J_n} \times \frac{J_r}{J_a}$$

In the modified version of Q, the last quotient which takes into account the stress conditions and water pressure is removed and will be replaced by a factor A when calculating the modified stability number N'. The modified stability number represents the ability of the rock mass to stand up under a given stress condition. It is calculated by multiplying the modified tunnelling quality index (Q') by factors A, B and C:

$$N' = Q' \times A \times B \times C$$

in which;

Factor A = The ratio of the intact rock strength to induced mining stresses, degrading the overall N' value.

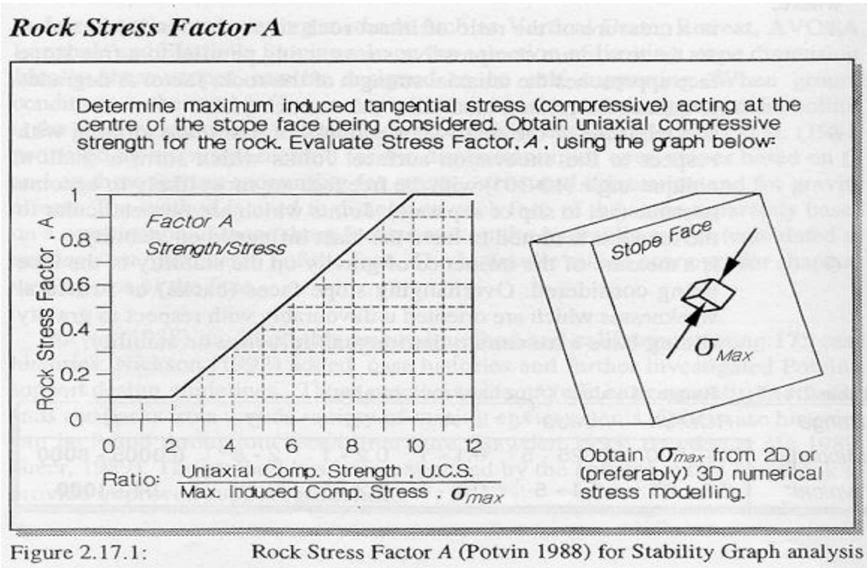

Figure 9 - Modified Stability Graph Rock Stress Factor A (Hutchinson & Diederichs, 1996).

Factor B = A measure of the effect of the least favorable oriented joint set affecting the stope surface.

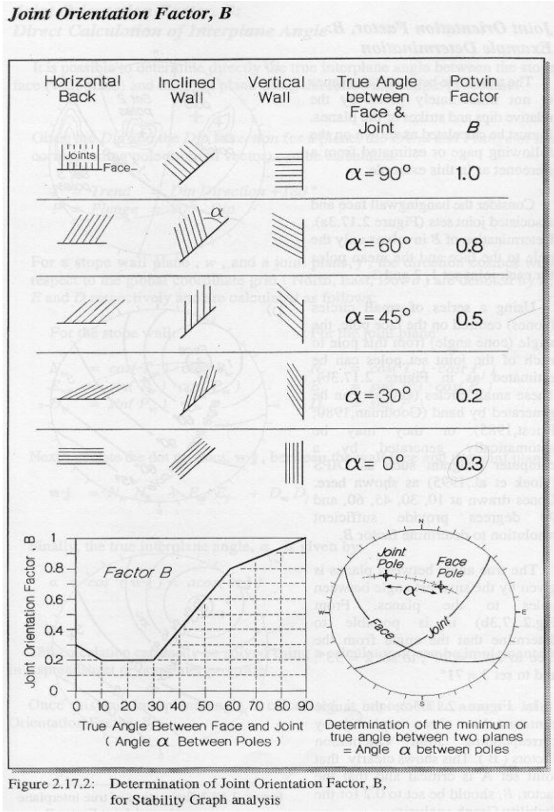

Figure 10 - Modified Stability Graph Joint Orientation Factor B (Hutchinson & Diederichs, 1996).

Factor C = A measure of the effect of gravity on potential failure modes(slabbing, sliding)

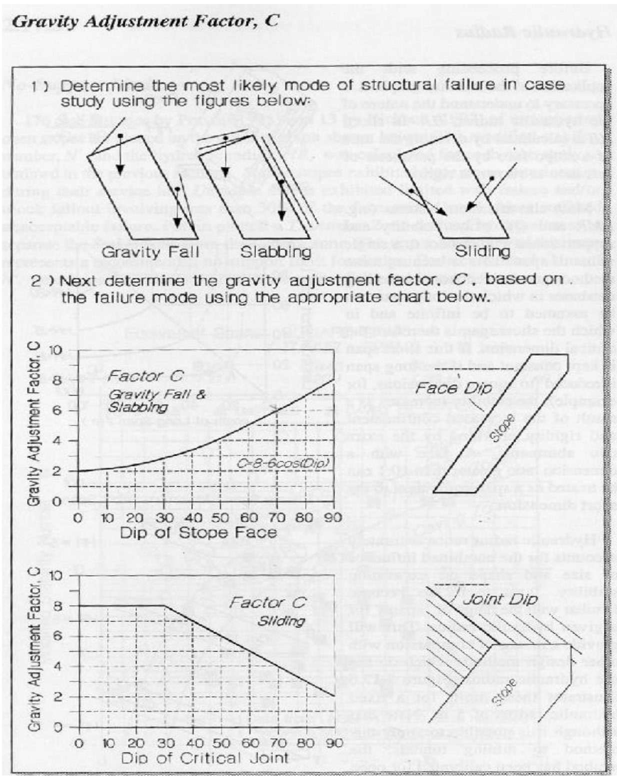

Figure 11 - Modified Stability Graph Gravity Adjustment Factor C (Hutchinson & Diederichs, 1996).

The modified stability number N' is used, together with the hydraulic radius, to assess stope stability by plotting the modified stability number in the chart as presented in Figure 12. The different zones of the graph (stable, transition and caving) are based on regression of many observations in active mines. Depending on the location plotted, the assessed stope shape is estimated to be stable (stable zone), unstable (caving zone) or potentially stable or unstable (transition zone).

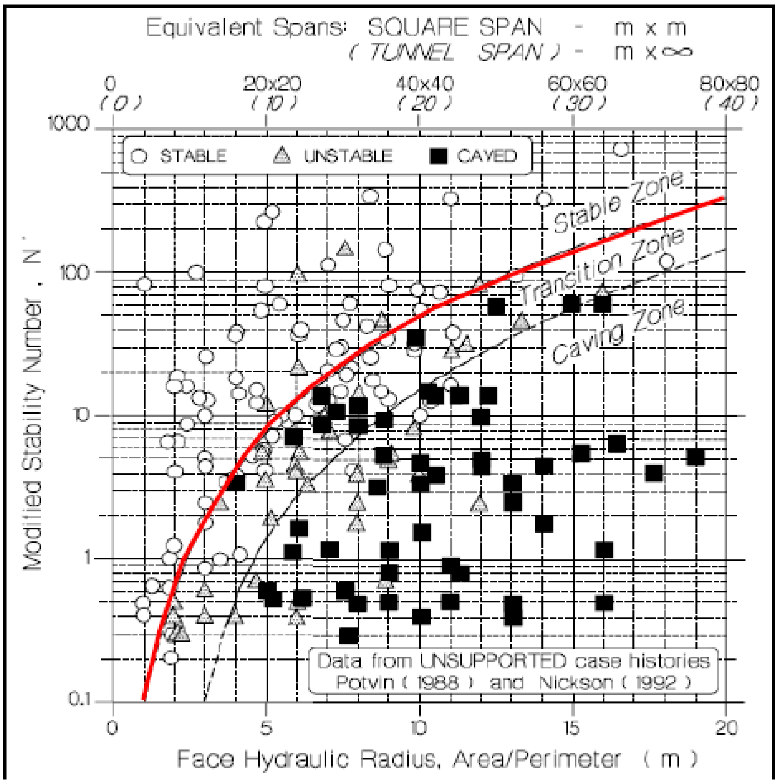

Figure 12 - Modified Stability Graph (Hutchinson & Diederichs, 1996).

2.3 NET PRESENT VALUE AND ITS RELATION TO CUT-OFF GRADE

One of the best-known research projects in the field of cut-off grade optimization in mining operations is the work of Lane (1988).

The work of Lane is based on the Net Present Value (NPV) theory, which is a mathematical way of defining the difference between the present value of future cash inflows and cash outflows. The Net Present Value theory is used in capital budgeting to analyse the profitability of an investment project (Investopedia, 2013). The Net Present Value takes into account the time value of money based on the principle that money today is worth more than money tomorrow. When calculating the NPV we want to know the net difference of all future cash inflows and all cash outflows in today's money.

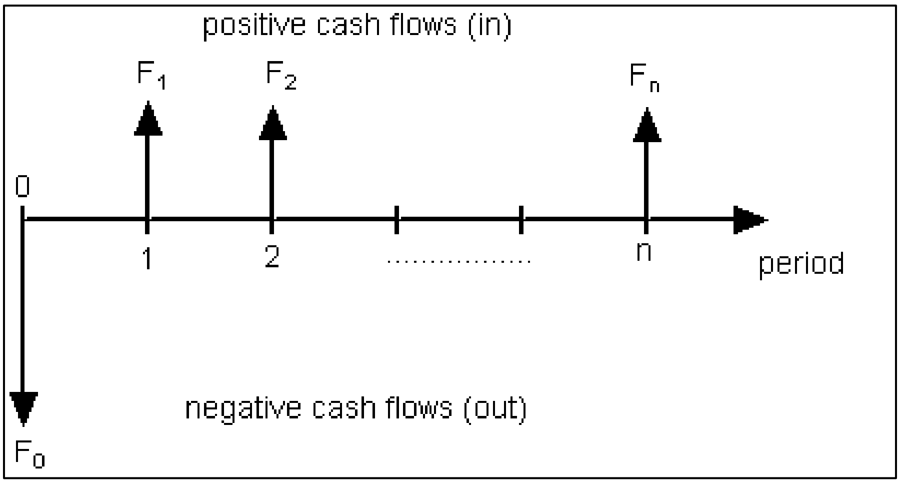

Figure 13 - Cash inflow and outflow over n Periods.

In the mining industry, investments in new mining projects are often large before mining can start and the return on this investment takes place over long periods of time, this means that the future returns (cash inflow) on the capital investment of today (cash outflow) have to be discounted in order to calculate the profitability of the project in today's money.

Figure 13 shows typical cash in- and outflows of a mining project over n periods of time. To calculate the Net Present Value, the present value (PV) of all future cash flows should be calculated by using the present value equation.

$$PV = \frac{Fn}{(1+i)^n} \qquad (1)$$

in which;

$F_n = Cash\ flow\ in\ period\ n$

$i = Discount\ rate$

$n = period$

The Net Present Value is the sum of the present value of all future cash flows and is defined as:

$$NPV = -F_0 + \sum_{N=1}^{n} \frac{F_n}{(1+i)^n} \qquad (2)$$

in which;

$n = Life\ of\ project$

$F_0 = Initial\ capital\ investment\ (cash\ outflow)$

$F_n = Future\ cash\ flow\ in\ period\ n$

$i = Discount\ rate$

It was Lane who first introduced the theory that a relationship exists between cut-off grade and NPV and there is a cut-off grade for which the Net Present Value is optimized. Lane's theory is clarified by the function defining F_n, the cash flow in period n.

The cash flow function for a period n is defined as:

$$F_n = M_n[(s-r) \cdot \bar{g}_n y - m - p - o] \qquad (3)$$

in which;

$M_n = Amount\ of\ material\ mined\ in\ period\ n\ (t)$

$s = Selling\ price\ per\ unit\ of\ product\ (USD/unit)$

$r = All\ smelting\ and\ Refining\ costs\ (USD/unit)$

$\bar{g}_n = Average\ grade\ of\ ore\ in\ period\ n\ (g/t)$

$y = Yield\ or\ recovery\ (\%)$

$m = Mining\ Cost\ (USD/t)$

$p = Processing\ Cost\ (USD/t)$

$o = Overhead\ Cost\ (USD/t)$

By substituting the cash flow function F_n(equation 3) into the NPV equation (2), the NPV equation is rearranged into:

$$NPV = -F_0 + \sum_{N=1}^{n} \frac{M_n[(s-r) \cdot \bar{g}_n y - m - p - o]}{(1+i)^n} \qquad (4)$$

If the average grade of the ore stream is a direct consequence of the chosen cut-off grade (Figure 14), a first link between cut-off grade and project NPV is obtained. The second link between cut-off grade and project NPV is the relationship between cut-off grade and the size of the mineable reserve. If the mining capacity is assumed constant (regardless of the chosen cut-off grade and the resulting size of the deposit) then the life of mine (n in the NPV equation) is defined as:

$$n = \frac{Q_m}{M} \qquad (5)$$

in which;

$Q_m = size\ of\ the\ mineable\ reserve\ (t)$

$M = Mining\ Capacity\ (t/a)$

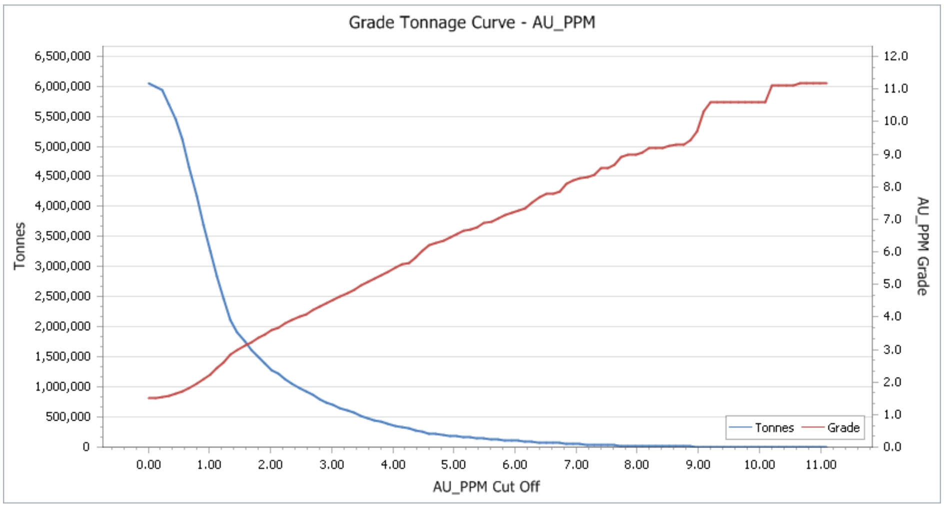

Figure 14 - Grade Tonnage Curve showing the relation between average grade and cut-off grade.

Lane proved that a cut-off grade exists for which the average grade and life of mine is optimized, which results in a maximized project NPV (Figure 15).

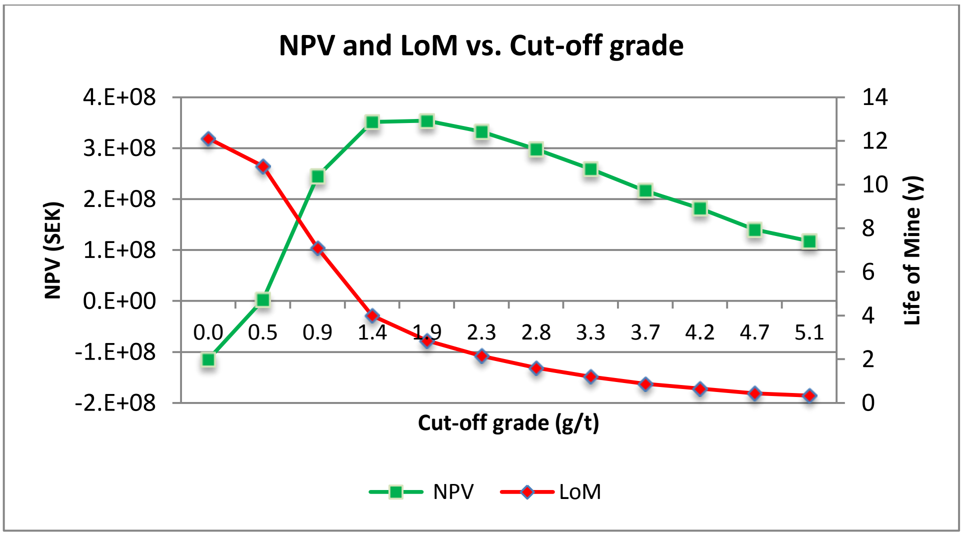

Figure 15 - NPV and LoM vs. Cut-off grade.

Lane however, proved his optimization theory by making use of the grade tonnage curve only and although his theory is acknowledged in the mining industry today, there are several problems when using the grade tonnage curve for cut-off grade optimization and project valuation in underground mining.

- The grade tonnage curve is purely a measure of tonnes of ore with a grade above a certain cut-off. It does not take into account the spatial position of the ore within the deposit. If for example high-grade ore is scattered around within the total mineral deposit and is surrounded by below-cut-off material, it may be impossible to extract the ore because the individual high-grade ore zones are too small to offset development costs.

- The grade tonnage curve purely describes the tonnage and grade distribution but fails to show orebody geometry. It may be impossible to 'just mine the ore' and some waste may have to be mined in order to develop a feasible mine layout (for example a stable stope shape). This will increase the mining cost and reduce the average ore grade resulting in a reduced profit.

- Lane's theory is based on the fact that all variable costs per tonne of ore mined are known. In underground mining, the variable costs are often based on the mining capacity, which by itself is again based on a set of variables that result from a cut-off grade decision (e.g. the shape and depth of the deposit, production constraints, etc.). Furthermore there are development costs that vary when the cut-off grade is changed due to the changing shape and location of production areas.

It is because of these complex relationships that cut-off grade optimization to optimize project NPV has not progressed much in underground mining since Lane introduced his theory. Many operations are designed based on a break-even cut-off grade (Hall, 2009), which is defined as the minimum grade that a tonne of ore should have in order to pay for its own mining processing and refining (Rendu, 2008). The break-even grade equation is defined as:

$$g_c = \frac{(m + p + o)}{y(s - r)}$$

in which g_c is the break-even cut-off grade. It should be noted that the break-even cut-off grade *is not* the average grade, as this average grade will in practice be higher as material with a higher grade than the break-even grade is present.

The break-even grade equation is, when comparing it to the rearranged NPV equation, very similar to part of the cash flow function F_n. However, by using the break-even grade as the cut-off grade it is questionable if the NPV is optimized. Lane has shown that the optimal NPV is based on a combination of cut-off grade and the life of mine (as a function of production rate) due to the implementation of a discount rate. In other words, using the break-even grade as the cut-off grade is too simple and does not take into account complex relationships involved in underground mining.

The next chapter will introduce a mine planning process that tries to implement Lane's theory in underground mine optimization.

2.4 THE MINE OPTIMIZATION APPROACH

This chapter will introduce an optimization approach for underground mining based on the theory as introduced by Lane to optimize project value. First the general approach to mine planning will be discussed followed by a description of the existing planning approach as used by Boliden. The chapter will end with conclusions on gaps in the existing optimization process that could be improved.

2.4.1 GENERAL CONCEPT

The planning process for an underground mine is summarized in Figure 16 (Poniewierski, MacSporran, & Sheppard, 2003). The underground planning process is an iterative process where a decision on cut-off grade has a direct impact on the subsequent optimization steps. In other words, if the initial cut-off grade decision is not optimal, all subsequent design and planning steps will be sub-optimal as well, resulting in a sub-optimal Net Present Value.

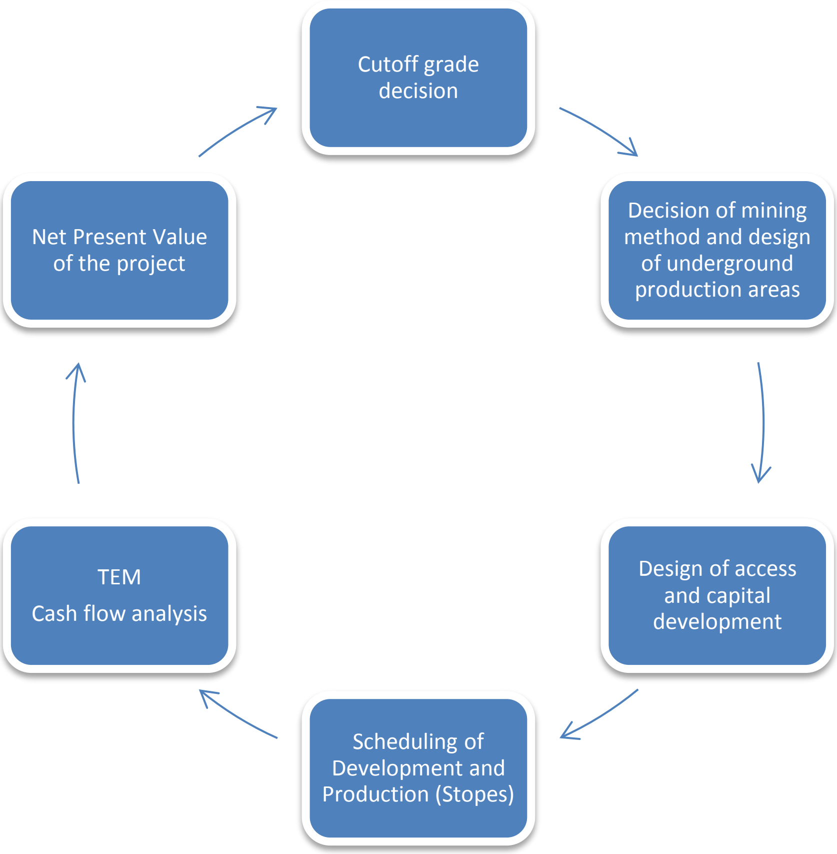

Figure 16 - A typical approach to underground mine planning.

The approach shown in the figure above involves all evaluation steps to calculate the Net Present Value of a project based on a cut-off grade selection followed by all subsequent optimization steps. The approach is in line with the theory as introduced by Lane, which states that project NPV is a controlled by a combination of cut-off grade and life of mine production scheduling. A brief explanation of each planning step will be provided here and the theory is applied during a case study and reported in section 4.3.

Cut-off grade decision

The mine planning process starts with an estimation of the break-even grade for the deposit and although it is unlikely to be the optimal cut-off, the break-even grade is often used as a firsts cut-off grade assumption.

Decision of mining method and design of underground production areas

Once a first cut-off grade is applied to the *mineral resource*, the geometry of the *mineral reserve* can be visualized. Based on the orebody shape, orientation and geology together with geotechnical rock conditions a suitable mining method has to be selected. It is common practice to use mining method selection tools such as the Nicholas Method or the UBC Mining Method Selection Tool.

The production areas of the deposit (e.g. stopes) are designed based on geotechnical conditions and the mining method of choice.

Design of access and capital development

With the production areas of the mine in place, the next step is to design the access development (shafts, declines, level development and ore drives) necessary to access the production areas. It is important to compare the value of the ore contained in the production area to the cost of the access development required to reach the production area.

Scheduling of Development and Production

The designed Development works and Production areas (stopes) are loaded into a scheduling package to produce a life of mine schedule for the operation. This schedule is based on a feasible mine sequence, development rate and production rate. The result of this stage is a production schedule that can be fed into a Technical Economic Model (TEM).

TEM Cash flow Analysis

The imported production schedule coming from the scheduling package forms the basis for the cash flow analysis. The cash flow model contains all revenues and costs calculated on a monthly basis and based on the production schedule. It also includes all other costs (e.g. Capital Expenditures) related to the mining operation. The resulting cash flow can be discounted to obtain the project NPV.

Net Present Value of the project

The calculated project NPV is the value based on the life of mine schedule which itself is based on the designed production areas and access development. The production areas and access development are ultimately based on the cut-off grade.

The theoretical background as introduced by Lane clearly returns in the described optimization approach making the next step in the process an obvious one. As already indicated in Figure 16, the optimization process is iterative and the next step would be to change the cut-off grade and re-analyse the subsequent steps to find a new project NPV. Once a range of NPV’s has been found, the optimal strategy that maximizes NPV can be selected.

2.4.2 MINE OPTIMIZATION AT BOLIDEN

This section will introduce the state of the art mine optimization practice as used by Boliden Mineral AB. The approach differs from the concept introduced in the previous section in that it consists of two separate phases that are carried out by two different departments within the company.

The mine optimization steps at Boliden are visualized in Figure 17.

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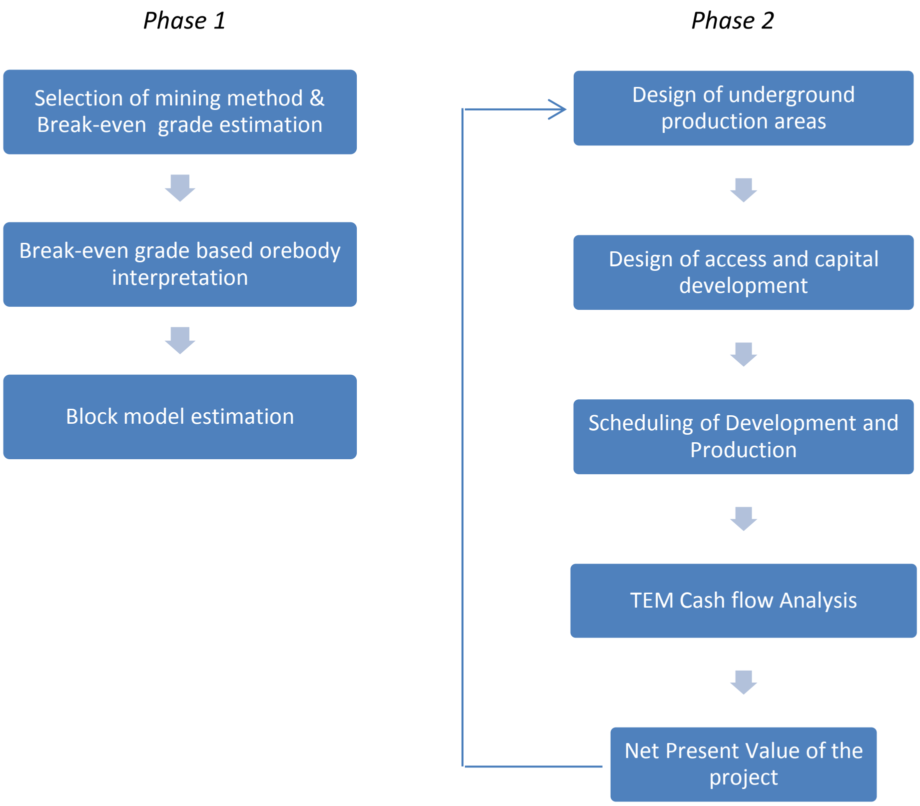


Figure 17 - The Mine Optimization Approach at Boliden

Phase 1 – Carried out by the Resource Geology Department

During this phase a geological block model is created that represents the mineable part of the mineral resource.

A mining method for the deposit is chosen based on the interpretation of geological maps that result from field mapping and exploration drilling. Based on historical data from similar type operations a break-even cut-off grade is determined. The resource geology department will then use this cut-off grade to delineate the mineable part of the resource within the raw drill hole data. The resulting wireframes are therefore not a representation of the true lithology but rather a grade shell representing ore that is equal or above the chosen cut-off grade. The grade shell wireframes are further modified and areas that are not mineable because the minimum thickness is not met, are removed. The finalized wireframes are used as boundaries during grade estimation and only grades within these wireframes are estimated and included in the block model. This approach assures that (almost) all the estimated blocks in the final block model are equal-to or above the selected cut-off grade.

Phase 2 – Carried out by the Mine Planning Department

The block model created by the Resource Geology Department is used by the Engineering Department to design the production areas and access development. With the cut-off grade being implemented into the block model, cut-off grade based mine design optimization is limited because all the ore is considered economical to mine. The subsequent steps of scheduling and cash flow analysis follow the same approach as the previously described optimization approach.

2.4.3 CONCLUSION

Based on the comparison between the general concept of mine optimization and the optimization process at Boliden the following is concluded.

Because the cut-off grade optimization is carried out by the Resource Geology Department and the resulting block model only represents the economic part of the orebody, further optimization is limited. On the one hand, this makes mine design relatively simple as stopes can be created by simply splitting the orebody on a regular grid.

The downside of the used approach is that valuable information about the deposit is lost. Whenever first assumptions change (e.g. change of mining method, metal prices, mining costs, processing costs etc.), the cut-off grade will change as well. In case the cut-off grade will drop, this means that the whole planning process has to be re-executed starting from the first step of *Phase 1*.

The current approach especially proved to be problematic whilst working on the Kankberg case study (more information to be found in section 4.2) where a change of mining method was considered resulting in a lower break even cut-off grade.

Possibilities

Based on the gaps in the mine optimization approach at Boliden the following changes to the process are proposed in order to change the optimization approach towards the concept described in section 2.4.1.

Ore boundaries should, whenever possible, be defined on lithology. This assures that all the ore is estimated and represented in the block model. Discussions with the Resource Geology Department however, indicated that it is not always possible to define ore boundaries based on purely lithology and assay based cut-off grade interpretations are necessary. If this is the case, it is proposed to lower the cut-off value by assuming a positive scenario with positive metal price forecasts and/or lower mining costs. This lowers the risk that once first assumptions change, a full re-interpretation and estimation of the mineral resource is required.

As a result of the new way of creating block models not all the ore blocks in the resulting block model are automatically mineable. This means that the complete optimization of the mining operation is now shifted to the Mine Planning Department.

As the author is aware of the fact that cut-off grade based mine design can be a time consuming process, the AMIRA Stope Optimizer is proposed to assist the engineers in this process.

The Stope Optimizer can be used to find the economic zone within a block model based on a chosen minimum cut-off grade. The Optimizer will not only find the economic zone, but will also design mineable stopes within the economic zone. The design of these stopes is based on design constraints such as minimum and maximum stope sizes, pillar sizes and internal waste percentage (planned dilution) that can be supplied by the engineer. With the assistance of Stope Optimizer to design mineable stopes at multiple cut-off grade scenarios, it should be much easier to find the mining scenario that maximizes the Net Present Value of the mining project.

By automating the process of cut-off based stope design, the need for continuous repetition of *Phase 1* whenever first assumptions change is avoided. This means that the Mine Optimization Process at Boliden is transformed into the mine optimization process as described in section 2.4.1.

2.5 IMPLEMENTING GRADE RISK IN UNDERGROUND MINE PLANNING

With mining projects requiring large capital investments and the fact that return on these investments can take years, it is obvious that risk mitigation is an important part of the mine planning process. Thus far, a method was proposed to find the optimal mine plan based on an estimated mineral resource model. As the optimum mine plan is completely founded on this model, the success of the project is largely based on the accurateness of the resource model. The accurateness of the mineral resource can be considered a risk and is often referred to as 'grade-risk'. It is important to have a sound understanding of the uncertainty within the resource model before any decisions are made to proceed with a project.

The main source of information about a mineral resource is drill hole data resulting from exploration campaigns. Due to the cost of drilling and for reasons of practicality it is impossible to drill an endless array of holes to literally 'pinch through' the whole orebody. For this reason holes are usually drilled and sampled on a regular grid resulting in local knowledge of the orebody. With the local orebody shape and mineral grades known, the unknown parts of the orebody are interpolated manually (the shape of the orebody) and by means of so-called estimation algorithms (the mineral grades inside the orebody shape). Multiple estimation algorithms exist including Nearest Neighbor, Inverse Distance Weighting and Kriging which are all well described in literature such as (Goovaerts, 1997).

Most simple methods (including Inverse Distance Weighting) interpolate grades based on surrounding 'measured' grades that are given a weighting factor based on their distance from the to be estimated point. Together these surrounding grades (multiplied by a weighting factor) result in an estimated grade at an unknown location.

Although the distance weighting methods are a correct first approach to grade interpolation, these simple methods fail to take directional relationships between spatially separated samples into account. There might be a strong down-dip or along-strike relationship between the measured grades that needs to be respected when interpolating grades. The (semi-)variogram is a mathematical way of describing this (directional) relationship between measured points and can especially be useful in deposits that show a strong directional component. The constructed variogram models can be used in conjunction with Kriging interpolation methods to estimate grades within a mineral resource more accurately.

A relatively new development (yet not applied by Boliden Mineral AB) within the minerals industry is the application of stochastic simulations in resource evaluation. By simulating multiple 'equally true' realizations of the mineral resource it becomes possible to quantify the spatial grade uncertainty in a mineral resource. These simulated resource models can be a valuable source of information in the mine optimization process.

The theory behind grade estimation and grade simulation is introduced in this chapter and two modified versions of the mine optimization approach as introduced in section 2.4 are proposed to account for grade-risk. The proposed approach were validated in a case study and the results of this study are presented in section 4.4

2.5.1 THE RESOURCE ESTIMATION AND SIMULATION PROCESS

The required steps in resource estimation and simulation are presented graphically in Figure 18. The data preparation and spatial description steps (also referred to as Exploratory Data Analysis) are the same in both estimation and simulation and can be described as the steps required to determine the characteristics of the orebody. This section will describe all steps in the estimation and simulation process and includes the theoretical background of Kriging Estimation as well as Stochastic Simulation.

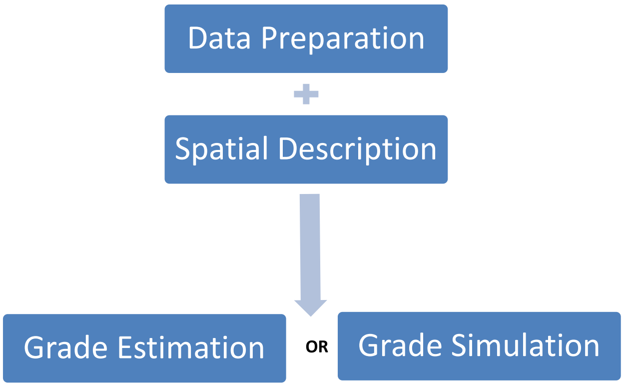


Figure 18 - The resource estimation and simulation process.

2.5.1.1 DATA PREPARATION AND SPATIAL DESCRIPTION

Data preparation can be considered the transformation of raw drill hole data into a dataset suitable for statistical analysis. Important steps are the cleaning of the drill hole database to remove faulty data (e.g. measurement errors or typing errors introduced whilst the database was created) and the application of a top-cap on the measured grades to reduce the influence of high grade outliers. Subsequently the corrected drill hole data should be regularized into equal-length sections (drill hole composites). By regularizing drill holes into sections with a constant length (and averaging the grades measured within each section), equal weight is given to each measurement which allows for geostatistical analysis of the spatial relationship between drill holes.

The created cleaned composited drill hole data can be loaded into a geostatistical analysis software package to investigate spatial relationships between the measured data in the drill holes. The goal of this step is to investigate whether or not there is a (directional) trend present in the measured data that should be honored when estimating or simulating grades.

Variogram models are used to mathematically describe the (directional) variance between datapoints measured in and between drill holes. The theory and process of variogram modeling is well described in the work of Goovaerts (1997) and is referred to for further reading. A practical example of variogram modeling is presented in section 0 of this report.

2.5.1.2 GRADE ESTIMATION – LINEAR REGRESSION MODELS

In this process description it is assumed that geostatistical analysis of the composited drill hole data showed directional trends between spatially separated measured datapoints in multiple directions and variograms were modeled in the major, semi-major and minor direction of grade continuity. Because directional (grade) trends are present in the dataset, local estimation by means of least-square linear regression algorithms (Kriging) is the preferred method for grade estimation.

All Kriging algorithms are based on the the linear regression estimator which is defined as:

$$Z^*(\boldsymbol{u}) - m(\boldsymbol{u}) = \sum_{\alpha=1}^{n(\boldsymbol{u})} \lambda_\alpha(\boldsymbol{u}) \, [Z(\boldsymbol{u}_\alpha) - m(\boldsymbol{u}_\alpha)] \qquad (1)$$

In which;

$Z^*(\boldsymbol{u})$ = *The estimated grade at the unknown location u.*

$Z(\boldsymbol{u}_\alpha)$ = *The measured grade Z at location* $\boldsymbol{u}_\alpha$ *with α=1,...,n).*

$\lambda_\alpha(\boldsymbol{u})$ = *The weight assigned to the measured grade* $Z(\boldsymbol{u}_\alpha)$ *at location* $\boldsymbol{u}_\alpha$.

$m(\boldsymbol{u})$ *and* $m(\boldsymbol{u}_\alpha)$ = *The expected values (mean) of the Random Variables*[1] $Z^*(\boldsymbol{u})$ *and* $Z(\boldsymbol{u}_\alpha)$.

Because the known and estimated grades at location $\boldsymbol{u}_\alpha$ and $\boldsymbol{u}$ can be considered outcomes of the random variable functions $Z^*(\boldsymbol{u})$ and $Z(\boldsymbol{u}_\alpha)$ it is possible to define the estimation error at unknown location $\boldsymbol{u}$ as a third random variable:

$$Z^*(\boldsymbol{u}) - Z(\boldsymbol{u}) \qquad (2)$$

This means that the closer to 0 this random variable at location $\boldsymbol{u}$ becomes, the better the grade estimation and the smaller the estimation error is.

The expectation in all Kriging estimation methods is based on the fact that

$$E\{Z^*(\boldsymbol{u}) - Z(\boldsymbol{u})\} = 0 \qquad (3)$$

This means that the expectation is that both the estimated value and the measured value, when taken at the same location , are equal and hence the measurement error is 0 (unbiased).

Because grades are not estimated and measured at the same location, equation 2 is only practical in theory. The goal of all Kriging methods is to minimize the estimation variance ${\sigma_E}^2$, which is defined in equation 4.

$${\sigma_E}^2(\boldsymbol{u}) = Var\{Z^*(\boldsymbol{u}) - Z(\boldsymbol{u})\} \qquad (4)$$

[1] *The random variable can be defined as a variable (the grade) that can take a series of outcomes based on a probability distribution (the grade distribution found based on the measured drill hole data).*

Multiple Kriging algorithms can be distinguished such as Simple Kriging, Ordinary Kriging and Kriging with a trend model. The different types are distinguished based on the way $Z(\boldsymbol{u})$ is calculated but all functions are built up by the sum of two components (equation 5).

$$Z(\boldsymbol{u}) = R(\boldsymbol{u}) + m(\boldsymbol{u}) \tag{5}$$

The first component ($R(\boldsymbol{u})$) is the residual component and is modeled as a stationary random function with a mean of zero and covariance $C_R(\boldsymbol{h})$[2]. The second component ($m(\boldsymbol{u})$) is a trend component (the mean) and the way it is defined depends on the Kriging type used. Because the Expectated value of the residual component at a location **u** is 0, the expected value of the random variable $Z(\boldsymbol{u})$ (the grade) at location **u** reduces to the local trend component $m(\boldsymbol{u})$ (equation 6).

$$E\{Z(\boldsymbol{u})\} = m(\boldsymbol{u}) \tag{6}$$

In Simple Kriging the trend component (mean) is assumed to be known and constant for the whole orebody. As this is often not the case for mineral deposits, the Ordinary Kriging method is more widely applied. The Ordinary Kriging method takes into account that the mean grade within a deposit can fluctuate locally and is therefore based on datapoints in the local estimation neighborhood surrounding the to be estimated point. The mean is considered unknown but locally constant within the local estimation neighborhood.

Re-arranging equation 1 yields:

$$Z^*(\boldsymbol{u}) = \sum_{\alpha=1}^{n(\boldsymbol{u})} \lambda_\alpha(\boldsymbol{u})\,[Z(\boldsymbol{u}_\alpha) - m(\boldsymbol{u}_\alpha)] + m(\boldsymbol{u}) \tag{7}$$

After re-writing equation 7 the linear estimation functions of Simple Kriging and Ordinary Kriging thus become:

Simple Kriging:

$$\boldsymbol{Z}^*_{\boldsymbol{SK}}(\boldsymbol{u}) = \sum_{\alpha=1}^{n(\boldsymbol{u})} \lambda_\alpha^{SK}(\boldsymbol{u}) Z(\boldsymbol{u}_\alpha) + \left[1 - \sum_{\alpha=1}^{n(\boldsymbol{u})} \lambda_\alpha^{SK}(\boldsymbol{u})\right] m \tag{8}$$

Ordinary Kriging:

$$\boldsymbol{Z}^*_{\boldsymbol{OK}}(\boldsymbol{u}) = \sum_{\alpha=1}^{n(\boldsymbol{u})} \lambda_\alpha^{OK}(\boldsymbol{u}) Z(\boldsymbol{u}_\alpha) + \left[1 - \sum_{\alpha=1}^{n(\boldsymbol{u})} \lambda_\alpha^{OK}(\boldsymbol{u})\right] m(\boldsymbol{u}) \tag{9}$$

The kriging weight (λ_α) assigned to a measured data point at location $\boldsymbol{u}_\alpha$ is derived from the covariance function or (semi) variogram as to minimize the estimated grade at the unknown location. This is where Kriging

[2] $Cov\{R(\boldsymbol{u}), R(\boldsymbol{u}+\boldsymbol{h})\} = E\{R(\boldsymbol{u}) \cdot R(\boldsymbol{u}+\boldsymbol{h})\} = C_R(\boldsymbol{h})$

methods differ from other distance weighted estimation algorithms (e.g. Inverse Distance Estimation) where weights are assigned based on distance to the estimation location only.

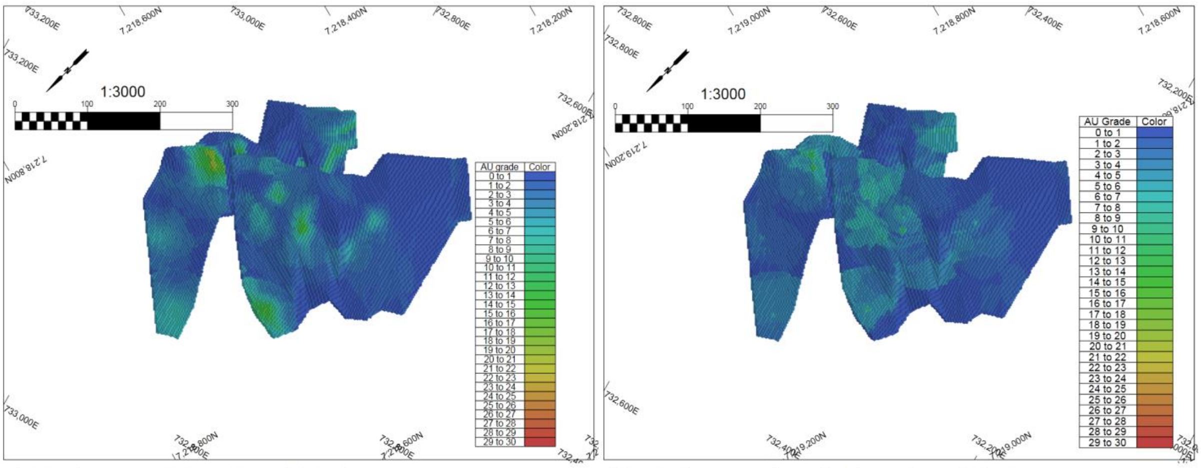
2.5.1.3 THE WEAKNESS OF ESTIMATION ALGORITHMS

The Kriging algorithms as described in the previous section when used in resource estimation result in a block model where each estimated block is interpolated based on its surrounding measured data points. Individually the blocks are the 'best estimate' of the local grade as the Kriging algorithm ensures that the local error variance ${\sigma_E}^2$ at location **u** is minimized with respect to the measured drill hole data surrounding the estimation location (Goovaerts, 1997). However, because surrounding previously estimated values are not considered as estimation progresses, the resulting estimated block model may not be the best representation of the grade distribution within the orebody.

Estimation algorithms such as Kriging type algorithms tend to smooth locally measured grades over the area surrounding the measured point (Figure 19). As a result of this effect, lower grades tend to be over-estimated whilst higher grades are generally under-estimated. Unfortunately, the smoothing effect is not uniform and becomes stronger with increasing distance towards the drill hole (Goovaerts, 1997). This makes it impossible to simply cancel out the smoothing effect by application of a correction factor.

Although the overall grade in the orebody may be interpolated correctly by the kriging algorithm, the smoothing effect is a serious concern when trying to pinpoint high grade areas in an orebody and define areas to mine or leave in place. Especially when exploration drill hole spacing is large and the number of measured data points is limited, the smoothing effect increases which may lead to incorrect identification of high grade mining areas. Because the design of underground production areas (stopes) is based on these high grade areas, the mine once taken into production, may fail to produce ore at the expected head grade.

The next section will introduce Stochastic Simulation, an alternative to grade estimation developed to avoid the introduction of the smoothing error and to account for spatial grade uncertainty in the resource model.



OK Estimate - Directional Variograms

OK Estimate - Omni-directional Variogram

Figure 19 - Grade smoothing effect surrounding high grade drill holes.

2.5.1.4 GRADE SIMULATION – STOCHASTIC SIMULATION

An alternative to estimation is Stochastic Simulation. Stochastic simulation uses measured data and previously simulated values simultaneously whilst creating simulations of the mineral resource. By already considering the previously simulated grades during the simulation process itself, the smoothing effect as introduced in kriging estimation due to interpolation between measured data only is avoided. The resulting simulated model reasonably matches the sample statistics as determined based on the drill hole database (e.g grade distribution, variogram model) and multiple ’equally true’ realizations of the same resource will provide valuable insights in spatial grade uncertainty.

Sequential Gaussian Simulation is a simulation method for which the composite drill hole data is transformed into a Gaussian Random Field by means of normal score transformation. The resulting normalized dataset is used in the simulation process after which the normal score transformation is reversed in order to obtain the simulated grade model.

The Sequential Gaussian Simulation Process can be divided into 3 distinct steps. Namely; Data Preparation, Sequential Simulation and Back Transformation followed by Simulation Validation. Each step in the simulation process will be explained briefly (after Goovaerts,1997).

1. Data Preparation

Sequential Gaussian Simulations (SGS) are based on a Gaussian Random Field with a mean of 0 and variance 1. The first step in simulation is thus to transform the composite drill hole data into the Gaussian random field model by means of a normal score transformation. This process is visualized in Figure 20 and should result in a dataset with a standard normal cumulative distribution function. If the composite dataset is correctly transformed into the normal space, variogram models should be constructed based on the normal score data before simulation can be performed on the transformed dataset.

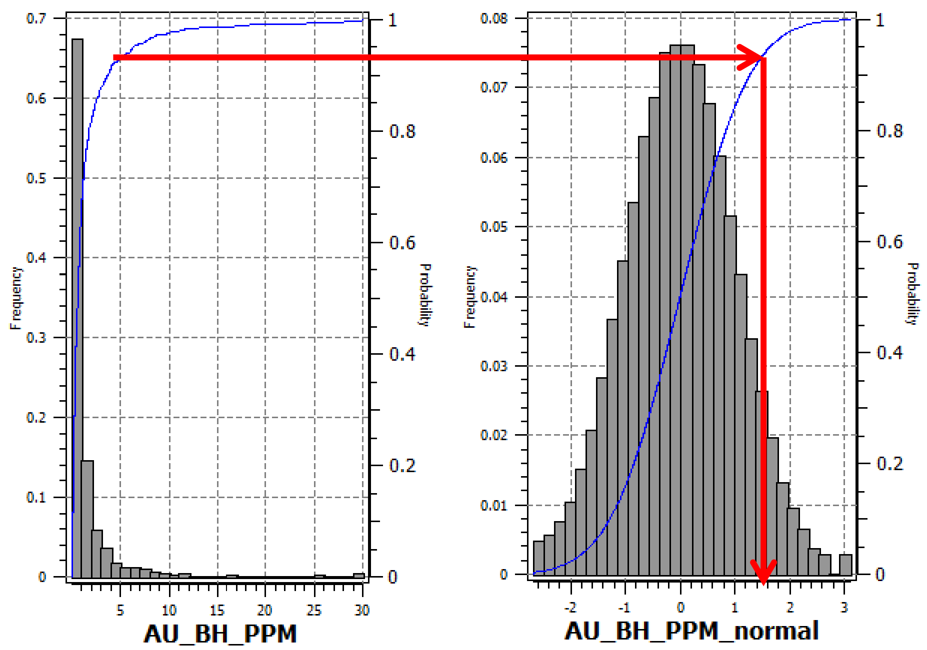


Figure 20 - Normal Score Transformation of composite drill hole data.

2. Sequential Simulation and Back Transformation

The sequential simulation algorithm requires a regular simulation grid (a block model with a single blocksize) for simulation. The measured 'hard data' is placed onto the grid as the respective measured location. This assures that hard data is considered during simulation and measured values are reproduced in the final simulated model.

Based on this regular grid a random path is defined visiting each node on the grid exactly once. At the first node, the mean and variance of the Gaussian complementary cumulative distribution (ccdf) function are determined. Because the data is transformed into a normal space, Simple Kriging in combination with the normal score semi-variogram can be used to estimate this local mean and variance.

A random value is drawn from the resulting ccdf which is the estimated (normalized) grade for the first node and this value is added to the dataset. Each successive node along the random path is subsequently visited and the estimation process is repeated with inclusion of the previously simulated nodes as data values in the kriging process. By including previously simulated values in the Kriging process the spatial variance (supplied by the variogram) is reproduced in the simulated resource model.

Once all nodes along the random path are visited, the simulated normal scores are back-transformed into simulated values for the original variable (the grade).

3. Simulation Validation

Stochastic simulations are based on several assumptions and in order to validate a resource model resulting from simulation, several checks have to be performed (Goovaerts, 1997):

1. Measured datapoints should be honored at their measured locations. The measured values are known to be 'true' and should as such return in the simulated model as hard data.

2. It is assumed that the grade distribution determined based on the measured drill hole data is representative for the grade distribution within the whole orebody. This is an important assumption in conditional simulation of mineral resources and the grade distribution should be honored and reproduced in the simulated model (no grade smoothing effect).

3. The variograms that were modeled based on the drill hole data to describe the spatial relationship between grades, should be reproduced within the simulated model. This assures that the spatial relationship between spatially separated data , as measured in the drill holes, is retained.

By defining multiple random paths it is possible to create multiple realizations of grade distribution within the mineral resource that are all 'equally true' and can when analyzed simultaneously, be used for the quantification of spatial grade uncertainty.

2.5.2 THE MINE OPTIMIZATION APPROACH – INCLUDING GRADE RISK

The mine optimization approach as introduced in section 2.4 is again presented in Figure 21 (blue shapes). The optimization approach is based on an estimated block model for which it is now assumed that a grade smoothing effect has been introduced by the Kriging Estimation algorithm. The optimization approach will result in an 'optimal mine plan' and project strategy based on the Estimated Model that can be evaluated using equally true simulations of the mineral resource. The probability that a stope will meet the estimation based headgrade can be calculated (grade risk) and a range of possible project cash flows and resulting Net Present Values can be determined to quantify the economic risk of the project.

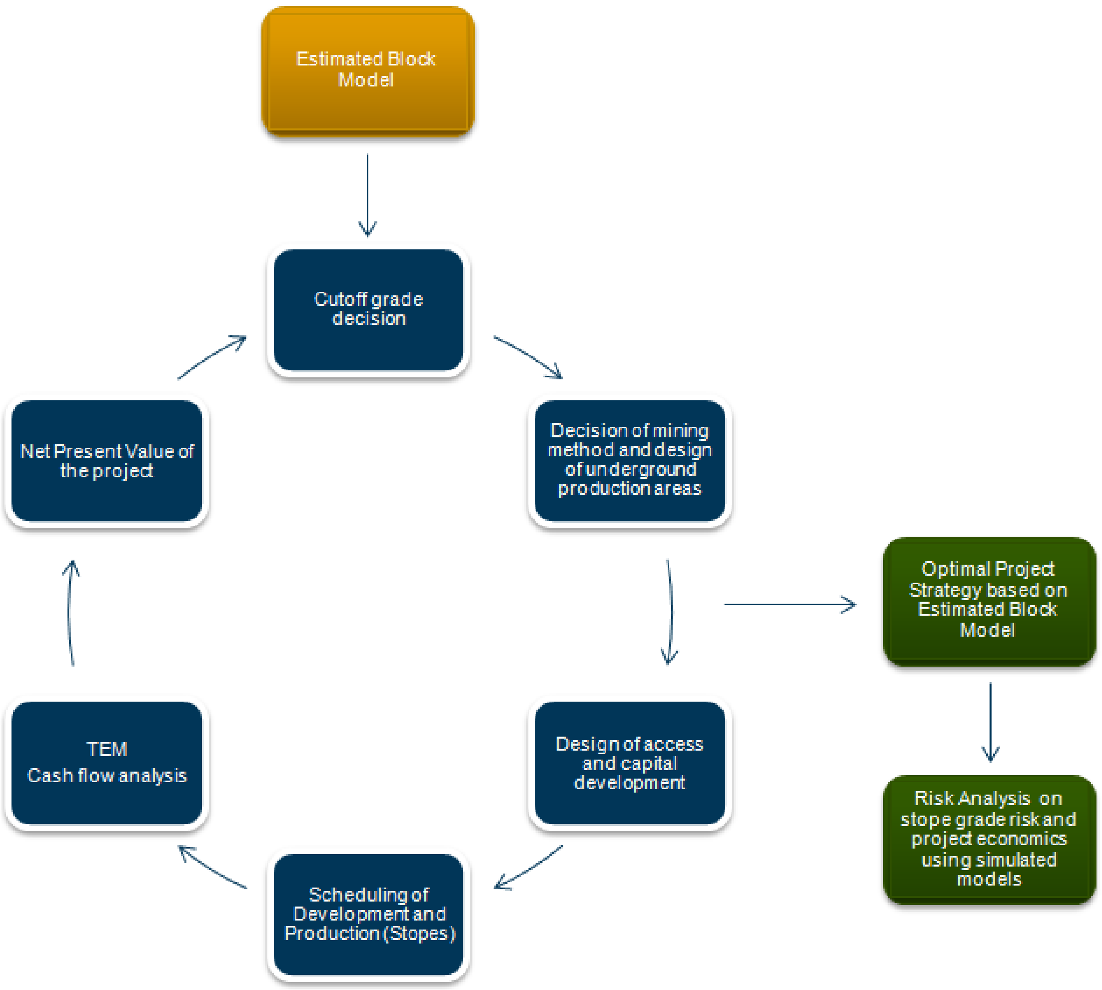


Figure 21 - The mine planning approach and subsequent risk analysis using simulation models.

A second approach to mine optimization whilst using the estimated and simulated resource models simultaneously is presented in Figure 22. Instead of optimizing the mining project at a certain cut-off grade based on the estimated model only and subsequently back-analyze the optimization outcome using the simulations, the simulated block models are used in the optimization process itself. By the application of a target confidence level to the desired cut-off grade, the underground production areas are designed to meet the cut-off grade in the estimated model, as well as a percentage of the simulated models. This will reduce the risk of a stope not meeting the cut-off grade which would result in a loss of money. By optimizing the mining project at different confidence levels (eg. 20%, 40%, 60%, 80%) the project economic risk can be quantified and used in decision making.

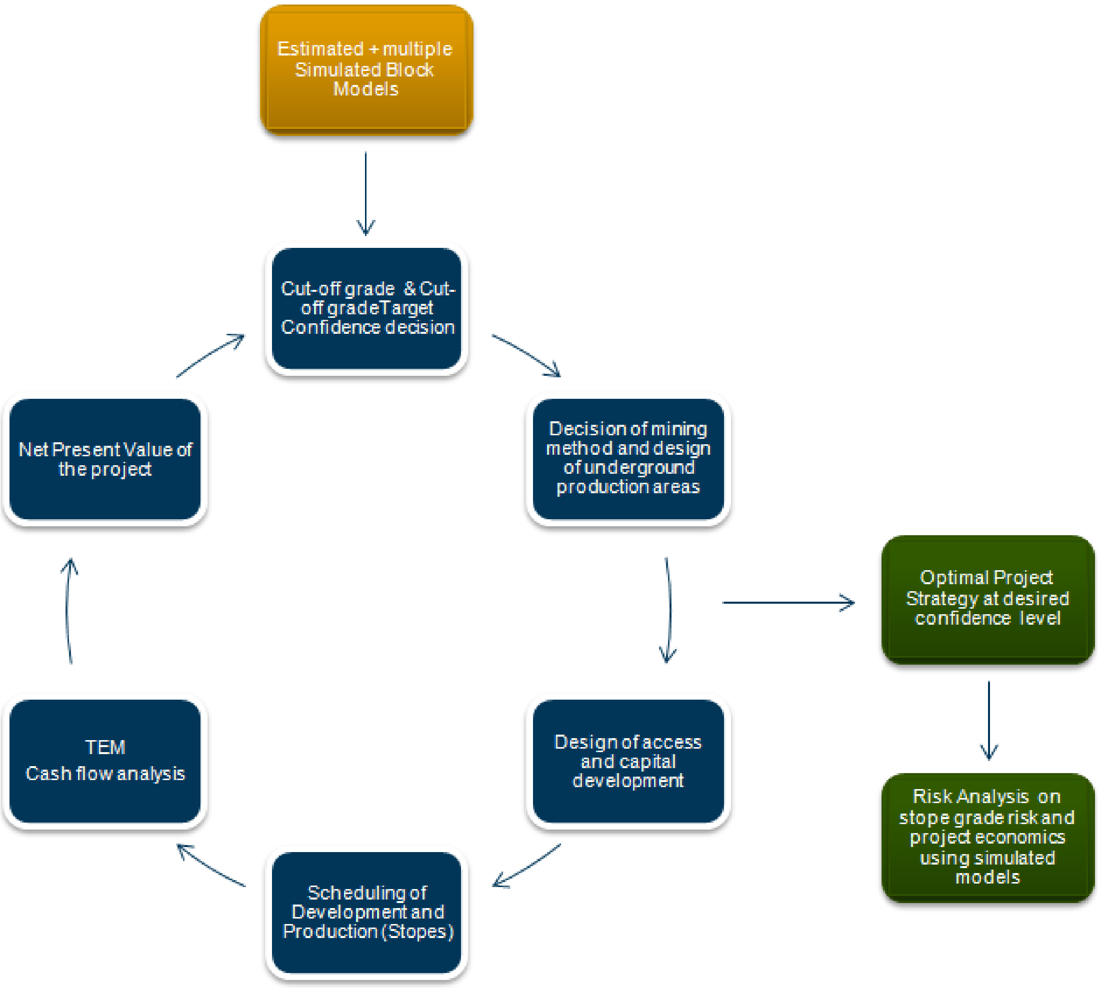


Figure 22 - The mine planning approach using resource estimation and simulation models simultaneously.

2.6 THE AMIRA STOPE OPTIMIZER

Section 2.4 and 2.5 showed that the underground mine planning process is an iterative process that can be very time consuming. Unfortunately no optimization software exists to-date that covers the full optimization process. One of the most time consuming steps within the planning process is the design of the production areas at multiple cut-off grades. The AMIRA Stope Optimizer was developed to automate this step. This chapter will introduce the Stope Optimizer software and the optimization principles used to design optimzed stopes.

2.6.1 GENERAL DESCRIPTION

The Stope Optimizer is a tool that assists a user in the design of underground openings (stopes). It optimizes both the location and shape of the stope to *maximize the total metal above a set cut-off* within the stope boundaries. By optimizing the total metal above cut-off it is the stope profit that is being maximized. The stope optimization is solely economic and no consideration is given to the practicality and technical feasibility of the design. To control the optimization process and ensure that the stope designs are technically viable, thought should be given to factors that control the maximum allowable stope shape. Controling factors can be geotechnical stress field conditions, faults and fractures, orebody characteristics and/or production limitations (e.g. maximum production drill hole lengths). The controling factors together result in constraints to the maximum stope dimensions that are supplied to the optimization software.

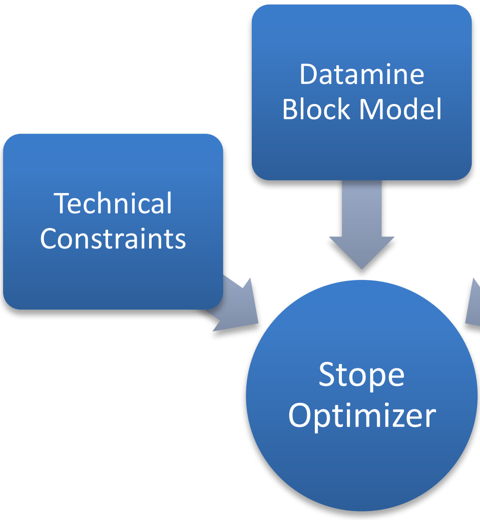


Figure 23 - Inputs into the AMIRA Stope Optimizer.

Figure 23 summarizes the required inputs for stope optimization. The optimization process is block model based as it is the block model that provides the grade (or NSR) and density data of the mineral resource. This means that the quality of the optimization is depending on the quality of the block model. The Stope Optimizer does not consider geological wireframes whilst designing stopes. Technical constraints are supplied by means of stope width, height and maximum span. Limitations on dip and strike angles have to be provided as well.

2.6.2 OPTIMIZATION STEPS

Whilst progressing through the stope optimization process three stages can be distinguished:

Slice Evaluation and Seed Generation

Stope Shape Annealing

Post Processing

During the first stage, the optimum basic stope shapes (the seed shapes) and their respective location in the optimization framework are determined whilst respecting several basic design parameters. A mathematical algorithm is subsequently applied to these seed shapes into the final stope shapes (stope annealing). The final stope shapes can be modified further based on the selected settings in the post processing stage.

This section will deal with each stage of the optimization process in more detail.

2.6.2.1 SLICE EVALUATION AND SEED GENERATION

During the slice evaluation process slices are placed within the XZ-Plane in a regular optimization framework. An example slice pattern for a primary-secondary stoping sequence at a single sublevel is shown in Figure 24. The optimization framework fixes the slice location in the XZ-Plane leaving the Y direction for optimization.

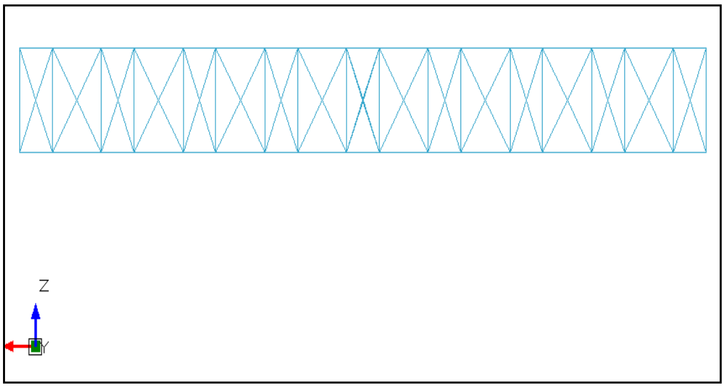


Figure 24 - The Stope Optimizer Optimization Framework.

The slices have the size and shape of the so-called 'smallest mining unit'. It is the smallest volume that can be selectively mined and therefore it is this 'slice' of the stope that is evaluated against the cut-off. The slice has a width and height equal to the stope width and height whilst the thickness can be any number. The thinner the slice, the more accurate the basic stope shape will be.

During the slice evaluation process slices are placed within the XZ-plane starting from the framework origin (either the origin of the block model or a user defined origin). At every slice location the optimizer will cut through the block model whilst creating and evaluating slices against the cut-off grade. The volume, tonnage and average grade of each slice is recorded to be used in the Seed Generation stage. Figure 25 shows a typical output of the slice evaluation process. Slices are stacked and each has a thickness of 1 metre. Only the slices that have an average grade above the chosen cut-off are presented graphically. It should be noted that also the slices below cut-off are recorded to be used in the Seed Generation and Stope Shape Annealing processes (Alford, 2013).

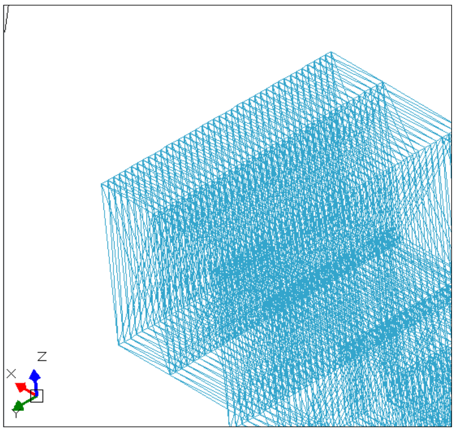


Figure 25 - Slice Evaluation output.

The Seed Generation can be considered the first optimization step. During this step the optimization algorithm aims to find the best combination of slices whilst respecting the design limits as set by the user.

The seed shapes are created whilst respecting the following design limits/constraints:

- **Stope Length**
- **Stope Height**
- **Shape Dip and Strike**
- **Minimal and Maximal Stope Width**
- **Minimal Waste Pillar width between two adjacent stopes**
- **Maximum Internal Waste Percentage**

The idea is to create a basic stope shape (seed shape) that is the optimal combination of slices that meet the design constraints as stated. The slices together form a Seed Shape above the defined cut-off and could as such be mined economically. However, the seed shape, being a very basic shape, is not fully optimized to maximise the stope profit. A seed shape resulting from the optimum combination of slices that meet the design criteria is shown in Figure 26. It is observed that although many slices (blue) are above cut-off, only a few also meet the extra constraints resulting in a smaller Seed Shape (red). The Seed Shape functions as a starting point for optimization in the Stope Shape Annealing stage.

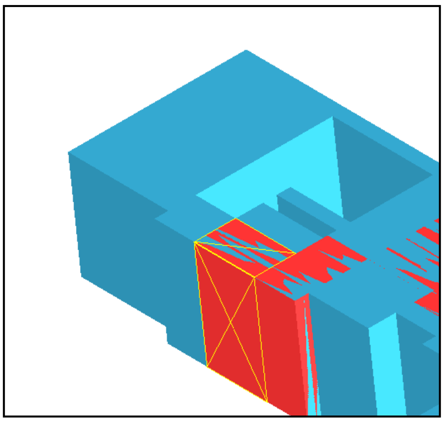


Figure 26 - Resulting Seed Shape (red) from individual Slices (blue).

2.6.2.2 STOPE SHAPE ANNEALING

In this stage a global optimization technique is applied to the Seed Shape. Because the stope height and length are fixed within the final stope shape, optimization is now limited to the corners of the seed shape. They can be moved in the transverse direction to further optimize the final stope shape. The concept of translating seed shape corner points is visualized in Figure 27.

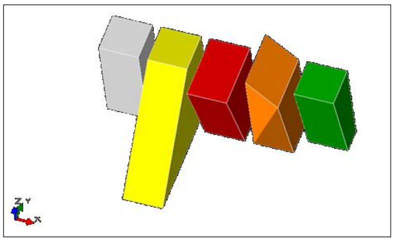


Figure 27 - Transverse corner translation examples.

From the shapes in Figure 27 it can be observed that the Seed Shape (shown here in grey) has 8 corner points that can be moved in the Y-direction whilst respecting stope length and height. A hill-climbing algorithm is now applied to optimize the Seed Shape into the final stope shape. This is an iterative algorithm that starts with an arbitrary solution (the Seed Shape) and then tries to find a 'better solution' (higher total metal above cut-off) by incrementally changing the location of one or more corner points. The hill-climbing technique is comparable to manual rubber bending the stope shape interactively on the screen and evaluating the new stope value but a computer can do thousands of iterations in a short period of time to find the highest value (Alford, 2013). The hill-climbing type optimization again takes place within the limitations as set by the user. To guide the hill-climbing algorithm a user can define the minimum and maximum strike angle as well as the maximum change between the strike angles of the front and back of the stope. The same type limits are applied to the hanging wall and footwall of the stope for which the minimum and maximum dip and dip angle change between the footwall and hanging wall have to be defined.

2.6.2.3 POST PROCESSING

Post Processing is a separate optimization step that can be activated or not. Options exist to split or merge stope shapes in the longitudinal or transverse direction but the main functionality is stope smoothing. By activating the stope smoothing functionality individually optimized stopes are lined-up in order to represent a more technically feasible mine design. Stope smoothing has a large overhead time on the optimization process which is caused by the fact that stope smoothing can involve up to 9 individual stopes when trying to smooth (the stope + all surrounding stopes) and the optimization algorithm will not compromise the boundary conditions as set by the user (e.g. maximum internal waste percentage, cut-off grade, etc.). An example of the influence of stope smoothing on the stope design is presented in Figure 28. The left design is the optimized design for each individual stope (no smoothing) whereas the right design is more feasible due to the fact that the stopes are lined-up better.

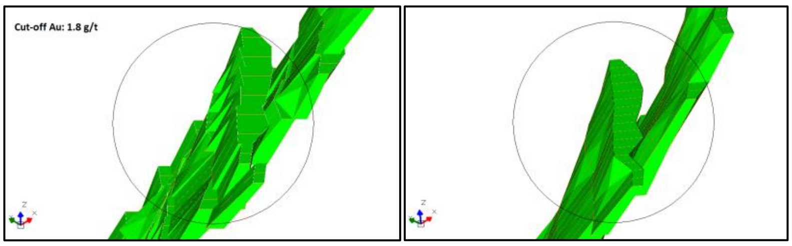


Figure 28 – Optimization output without and with Stope Smoothing.

3 STOPE OPTIMIZER EVALUATION – CAPABILITIES AND LIMITATIONS

As a first step in this research project it was decided to analyze the performance of the AMIRA Stope Optimizer. With the software currently being unknown to Boliden employees and user manuals yet non-existing, a thorough study was undertaken to learn how to use the software and how to set up optimization scenarios. This resulted in a Stope Optimizer user guideline that can be found in appendix A. Subsequently the capability of the optimization software to find the optimal stope shape within simple test block modelswith a known solution was tested. Finally the stope optimization software was compared to manual stope designs of the 1257 sublevel at the Boliden Garpenberg mine.

3.1 PERFORMANCE ANALYSIS ON TEST MODELS

In order for the stope optimization software to have any added value to mine design it is important to analyze if the optimizer is able to find the best stope shape (the shape that maximizes the profit) within the boundary conditions set by the end-user.

To analyze the capabilities of the optimization algorithms several test models were created to check if and how the optimization algorithms find the optimal stope shapes. As grade distributions within real geological block models are very irregular, it is very difficult to manually design the optimized stope and compare the design to the stope optimization algorithm solution. It was therefore decided to create simple block models that aim to test one specific optimization variable at a time or a combination of variables.

Two block models were created with properties as defined in Table 1:

1. **A strike model**
2. ***A HW (hangingwall) and FW (footwall) model***

Block Model Property	Value
Block Size	5x5x5
Ore Grade	1
Waste Grade	0
Density (ore&waste)	2.8

Table 1 - Test block model properties.

3.1.1 ANALYSIS APPROACH

For evaluation purposes, each model was loaded into the Stope Optimizer environment. First the optimization base case scenario was run with ‘loose parameters’ that give the optimization algorithms a lot of freedom to bend the seed shape into the optimum solution. By leaving the settings very loose the aim is to imitate a real-life optimization scenario where strike and dip angles can vary from stope to stope and one wants to find the optimal solution for each individual stope in one single optimization process.

Once the ’loose’ solution is found, combinations of settings that tighten up the limits for seed shape bending are subsequently applied to evaluate the sensitivity of the optimization algorithm to the input parameters. Ideally, the optimization algorithm should return the optimal design whilst keeping the optimization limits as loose as possible.

3.1.1.1 OPTIMIZATION SCENARIO SETTINGS

Each optimization scenario consists of two pairs of settings. The fixed settings stay the same in the base case scenario and all subsequent tightened scenarios whilst the 'loose parameters' as defined in Table 2 are base case scenario settings that are being varied in each subsequent optimization scenario.

Fixed Parameter	Unit	Value
Default Shape Dip	degr.	90
Default Shape Strike	degr.	00
Stope Width	m	5
Minimum Stope Length	m	5
Maximum Stope Length	m	100
Minimum Waste Pillar Width	m	0
Cut-off Grade	-	0.5
Max Side Ratio (Top-to-Bottom)	-	2.5
Max Side Ratio (Front-to-Back)	-	2.5
Shape Slice Interval*	m	1

** Slice intervals of 1, 2.5 and 5 metre were evaluated.*

Loose Parameter	Unit	Value
Minimum Strike Angle	degr.	-45
Maximum Strike Angle	degr.	+45
Maximum Strike Angle Change (Front-to-Back)	degr.	90
Minimum HW and FW Dip	degr.	40
Maximum HW and FW Dip	degr.	90
Maximum Dip Angle Change	degr.	50

Table 2 - Performence Analysis Optimization Scenario Settings.

3.1.2 STRIKE MODEL ANALYSIS

The strike model that was evaluated is shown in Figure 29 with the manual solution shown in black.

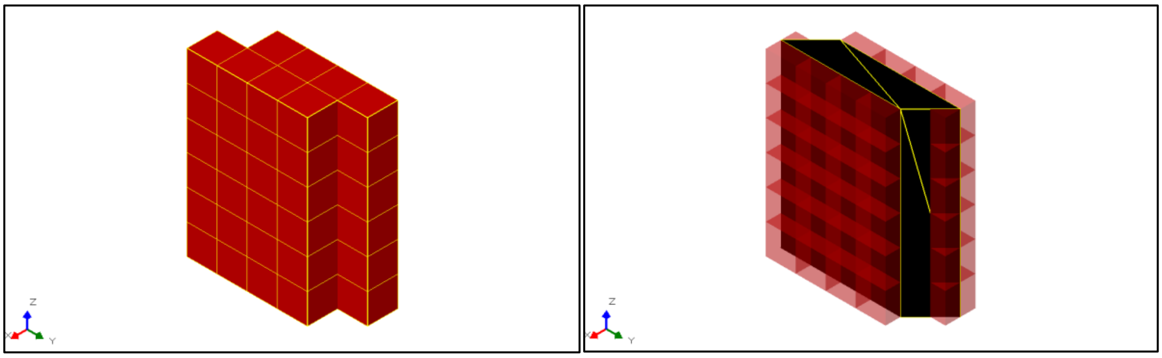


Figure 29 - Evaluated Strike Model (Performance Analysis).

3.1.2.1 MANUAL OPTIMIZATION AND CALCULATIONS

In order to compare the optimization algorithm solution to the manual design the volume, tonnage and grade of the 'black stope' (Figure 29) was calculated. The author assumed this to be a good estimate of the optimal solution and it will be up to the optimizer to find the same design or to beat the author.

The manual design yields:

Strike angle: 45 degrees

HWFW dip angle: 90 degrees

Volume: 3000 cu.m

Tonnage: 8400 tonnes

Average Grade: 0.9375

With a cut-off grade of 0.5 applied, the stope 'profit' can now be calculated by:

$$Profit = tonnes \times (average\ grade - cutoff\ grade)$$

$$Profit = 8400 \times (0.9375 - 0.5) = \mathbf{3675}$$

3.1.2.2 OPTIMIZATION APPROACH AND PRESENTATION OF RESULTS

In order to evaluate the Stope Optimizers' performance the following subsequent steps were performed to find the optimal stope design.

1. Find the optimization algorithm solution to the stope problem using the base case scenario settings.
2. Subsequently reduce the maximum dip change of the HW and FW to tighten the dip change between front and back of the stope.
3. Change the HW and FW dip angle range.
4. Subsequently reduce the maximum strike angle change to tighten the strike angle change between front and back of the stope.
5. Change the strike angle range.

All results were analyzed on a profit base. The base case scenario was assumed to be the optimal design that leads to the highest profit. All subsequent 'tightened' scenarios were compared to this base case solution to see if the base case remains the best solution. The results of these comparisons are presented graphically in the next section.

3.1.2.3 OPTIMIZATION RESULTS

Shape Slice Interval and Base Case Profit

The results of the base case scenario evaluation at different slice intervals are presented in Figure 30 and Figure 31. With a profit of 3684.2 for all three cases it is concluded that the shape slice interval in this example is of little importance. This has to do with the fact that the default slice shape strike and dip fall in line with the orientation of the block model. With a block size of 5x5x5 meters and no sub-blocking it is of little importance if a slice of 5 meters is evaluated against the model or 5 separate 1-metre slices are evaluated against the cut-off. The resulting seed shape will have the same shape. One could argue that the smaller the slice interval the more accurate the result will be, but there is a limitation on the total number of slices that can be evaluated when running an optimization scenario due to the fact that there is a limit on the amount of slice evaluation results that can be stored in the software memory (Alford, 2013).

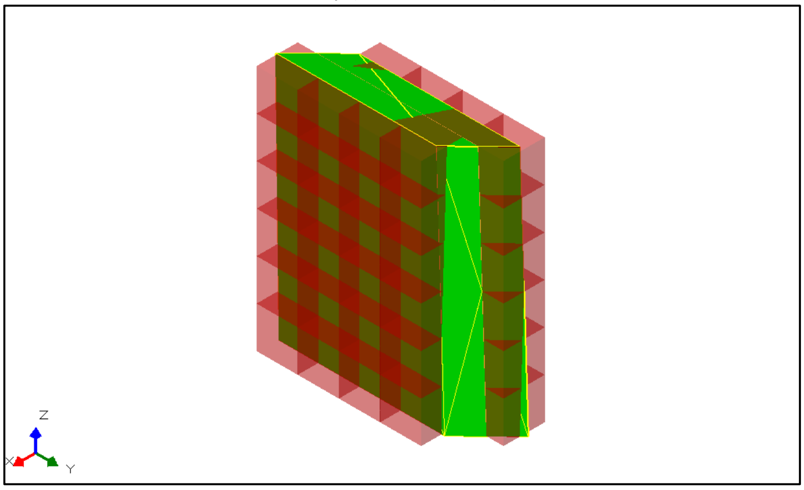


Figure 30 - Stope Optimizer Strike Model Base Case Solution.

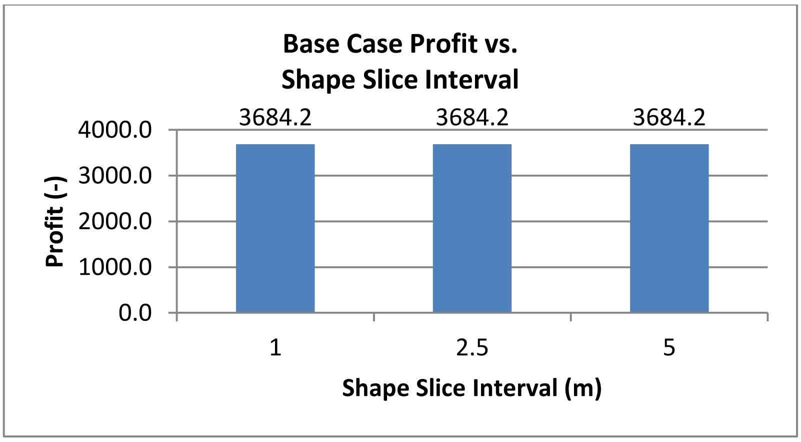


Figure 31 - Strike Model Base Case Profit vs. Shape Slice Interval.

A shape slice interval of 1 was chosen as the base case scenario during this evaluation and the resulting profit is 3684.2 making the Stope Optimizer solution slightly (0.25%) better compared to the manual design. By visual observations it was concluded that the optimized stope has hanging wall and footwall angles in the range of 89.2 to 90.8 degrees when measured in the Deswik.CAD environment where the manual design assumes exactly 90-degree angles. It is concluded that this small difference between the manual design and the optimization algorithm solution are the reason for the slightly better result.

Maximum Dip Angle change between the Hanging wall and footwall

By visual observations it was found that for the base case scenario the optimization algorithm returned hanging wall and footwall angles in the range of 89.2 to 90.8 degrees. Due to this small range, changing the maximum dip angle change has no impact. Only setting the maximum dip angle change to exactly 0-degrees has a negative effect as it leaves the optimization algorithm absolutely no freedom during the stope shape annealing stage and the algorithm therefore returns the seed shape as the most optimal design. This observation was confirmed by C. Alford who states that the hill-climbing algorithm behind the Stope Optimizer needs some freedom during the stope shape annealing stage in order for the hill-climbing algorithm to work (Alford, 2013).

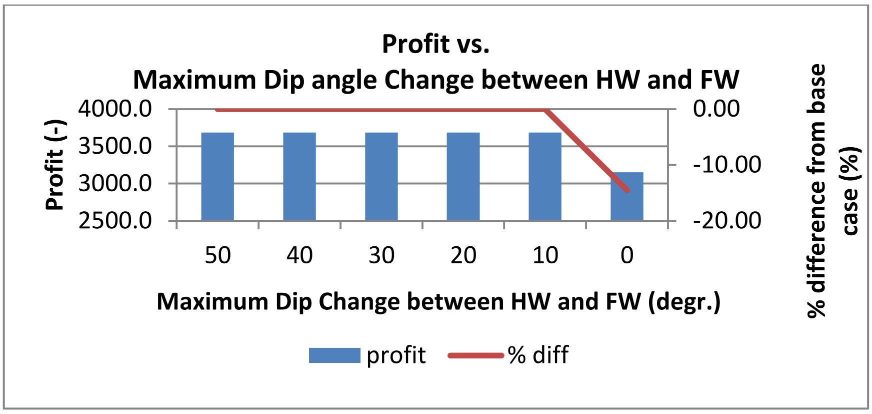


Figure 32 - Profitability Analysis on Maximum Dip Angle Change between HW and FW.

Hanging Wall and Footwall Dip Angle Range

Within the base case scenario the dip angle was allowed to vary within the range of 40 to 90 degrees. The optimal dip was found to be within 0.8 degrees from 90. Tightening of the dip angle range therefore had no influence on the stope profitability.

Maximum Strike Angle Change

Tightening of the maximum strike angle change between the front and back of the stope has no real influence on the profitability of the stope. The optimizer found strike angles that are exactly the same for the front and back of the stope (as is to be expected based on the block model shape). It is concluded that leaving the maximum strike angle change to the maximum allowable value gives the best results as can be seen in Figure 33.

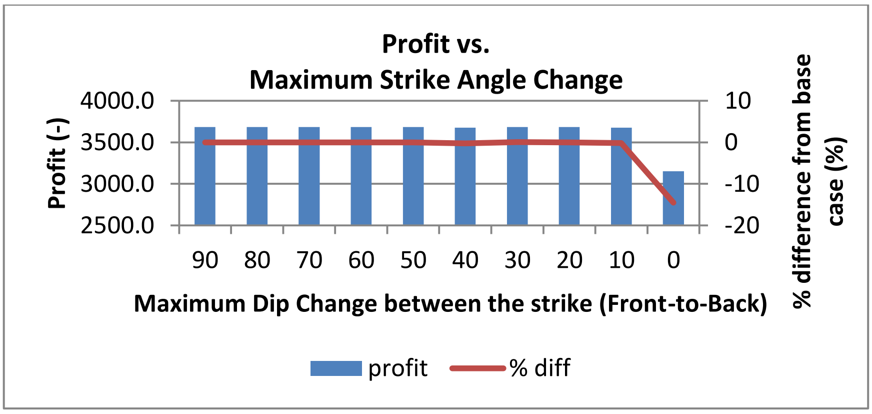


Figure 33 - Profitability Analysis on Maximum Strike Angle Change between the Front and Back of the stope.

Strike Angle Range

The strike Angle Range determines the range in within the stope strike can deviate from the default strike angle (fixed setting). From Figure 34 it can be observed that the profitability decreases whenever the strike angle range is reduced from the base case range of -45 to 45 even though the optimal strike angle is 45 degrees. Profitability increases whenever the strike angle range increases because the optimal strike angle will increase, decreasing the amount of waste. It is however questionable if a strike angle of 50 degrees or more is technically feasible in a stope design.

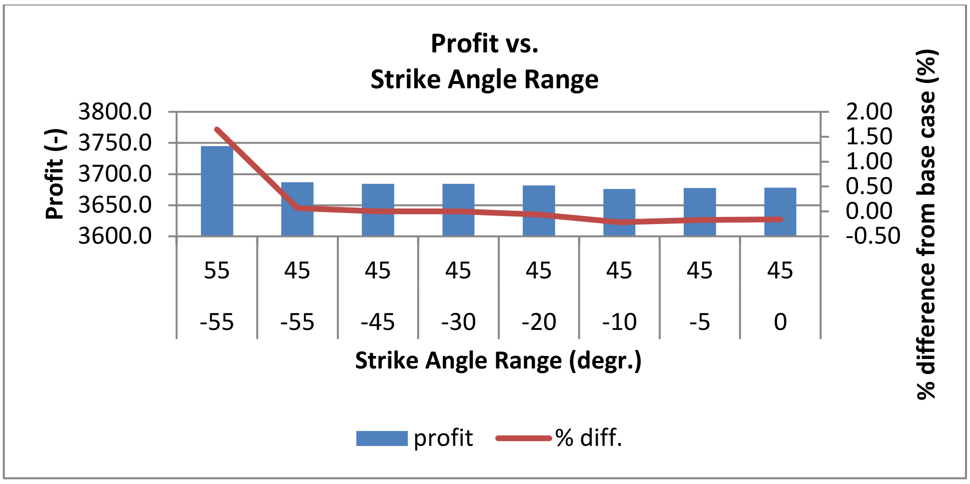


Figure 34 - Profitability Analysis on Strike Angle Range.

3.1.3 HANGING WALL AND FOOTWALL MODEL ANALYSIS

The strike model that was evaluated is presented in Figure 35 with the manual solution shown in black.

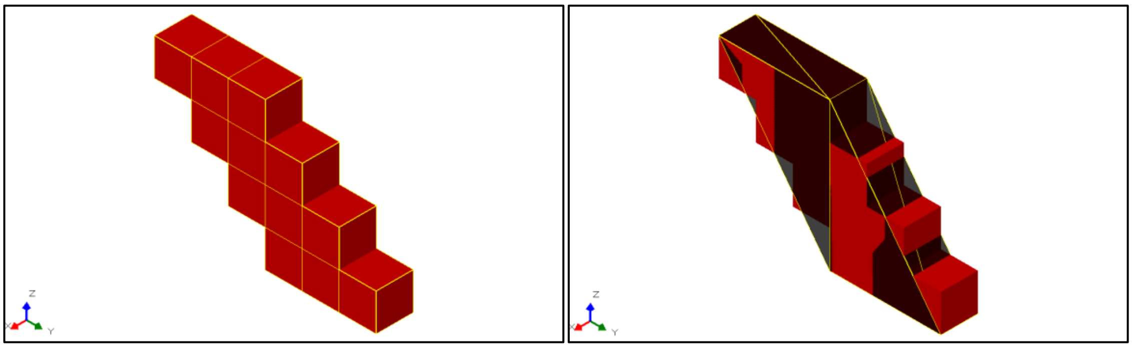


Figure 35 - Evaluated HWFW Model (Performance Analysis).

3.1.3.1 MANUAL OPTIMIZATION AND CALCULATIONS

In order to compare the optimization algorithm solution to the manual design the volume, tonnage and grade of the 'black stope' was calculated. Again the author assumed this to be a good estimate of the optimal solution and it will be up to the optimizer to come up with the same design or to beat the author.

The manual design yields:

Strike angle: 0 degrees

HWFW dip angle: 53.1 degrees

Volume: 1500 cu.m

Tonnage: 4200 tonnes

Average Grade: 0.9030

With a cut-off grade of 0.5 the stope profit can now be calculated by:

$$Profit = 8400 \times (0.903 - 0.5) = \mathbf{1692.6}$$

3.1.3.2 OPTIMIZATION APPROACH AND PRESENTATION OF RESULTS

In order to evaluate the Stope Optimizers' performance the following subsequent steps were performed to find the optimal stope design.

1. Find the optimization algorithm solution to the stope problem using the base case scenario settings.
2. Subsequently reduce the maximum strike angle change to tighten the strike angle change between front and back of the stope.
3. The optimum strike should be about 0. Decrease the strike angle range around this value to verify this.
4. Subsequently reduce the HWFW dip angle range to check if the optimization algorithm found the best hanging wall and footwall angles.
5. Change the HWFW dip angle change to also verify if the optimizer found the best combination of dip angles (front and back).

All results were analyzed on a profit base. The base case scenario was assumed to be the optimal design that leads to the highest profit. All subsequent 'tightened' scenarios were compared to this base case solution to see if the base case remains the best solution. The results of these comparisons are presented graphically in the next section.

3.1.3.3 OPTIMIZATION RESULTS

Shape Slice Interval and Base Case Profit

The results of the base case scenario evaluation at different slice intervals are presented in Figure 36 and Figure 37. There is little difference in the optimization result between the 1 and 2.5-metre shape slice scenarios. Visual observations confirmed that the formed seed shape for both scenarios is the same whilst for the 5 metre slice interval, the seed shape is different. The hill-climbing algorithm in the stope shape annealing stage uses the seed shape as the basis for finding the optimal stope which explains the difference in the found solutions.

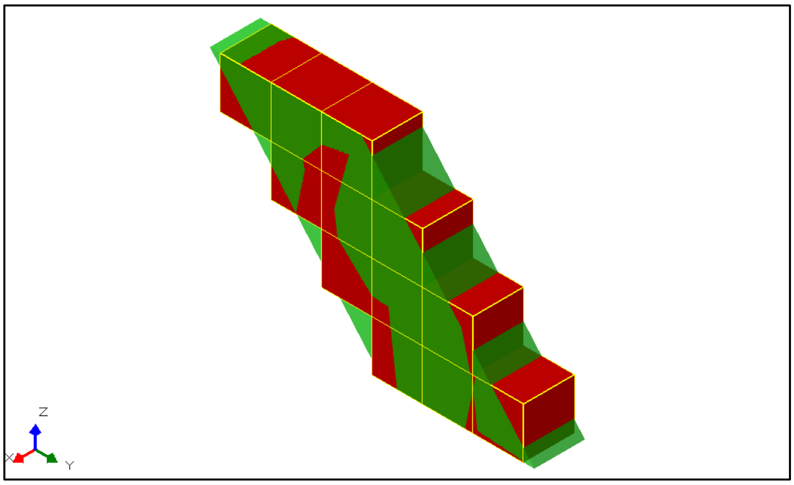


Figure 36 - Stope Optimizer HWFW Model Base Case Solution.

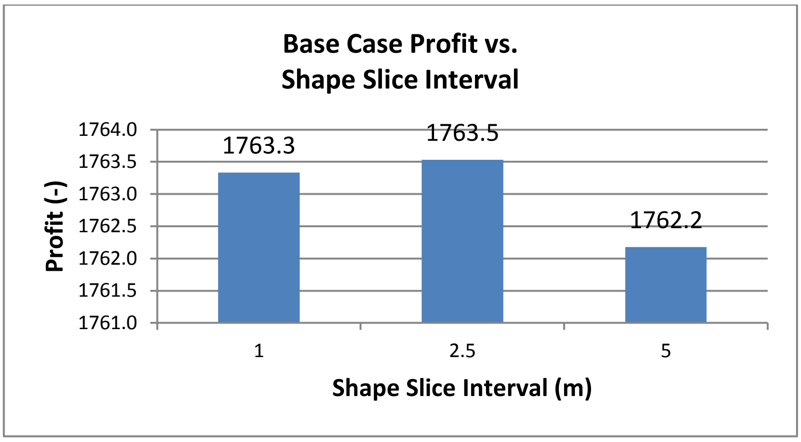


Figure 37 - HWFW Model Base Case Profit vs. Shape Slice Interval.

Due to the small difference between the 1 and 2.5 metre scenario and based on the fact that smaller slices should in general give better solutions, the 1 metre slice interval was chosen for investigation (this is also in line with the evaluation of the strike model).

With a stope profitability of 1763.3 compared to a profitability of 1692.6 for the manual stope design it is clear that the optimization algorithm outperformed the author. The 4.2% increased profit is obtained by slightly moving the stope corners in the transverse direction, which reduces the stope hanging wall and footwall dip angle to 47.6 degrees. Although the total stope tonnage does not change compared to the manual design, the average stope grade is increased from 0.9030 to 0.9205.

Maximum Strike Angle Change

Compared to the Strike model a bigger influence of the maximum strike angle change between the front and back of the stope in order to find the optimal stope is observed (Figure 38). It is likely that this has to do with the stope shape annealing stage and its hill-climbing approach to optimization. The bigger the allowance to have different strike angles between the back and front of the stope, the easier it is for the optimization software to execute the hill-climbing algorithm. Although differences are relatively small (in the range of ~0.1%) it is observed that the optimization algorithms perform best when using the base case scenario 'loose settings'.

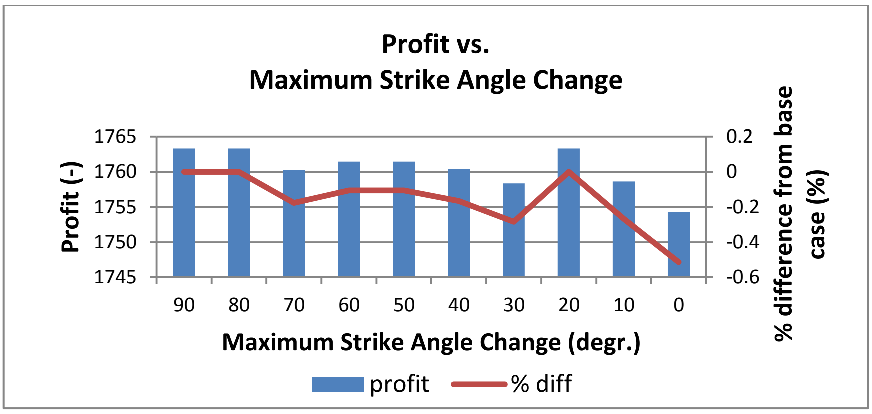


Figure 38 - Profitability Analysis on Maximum Strike Angle Change between the Front and Back of the stope.

Strike Angle Range

Based on the block model that was created for the HWFW evaluation it is concluded that the optimal strike angle will be exactly 0 degrees. This angle was chosen as only the performance of the optimization algorithm as to find the best hanging wall and footwall angle is of interest. The influence of changing the strike angle range is therefore not expected to have much influence, which is confirmed by the results in Figure 39. Again, only when the flexibility is reduced to zero, the algorithm fails to return the optimal design which results in a dramatic loss of profit of circa 60%.

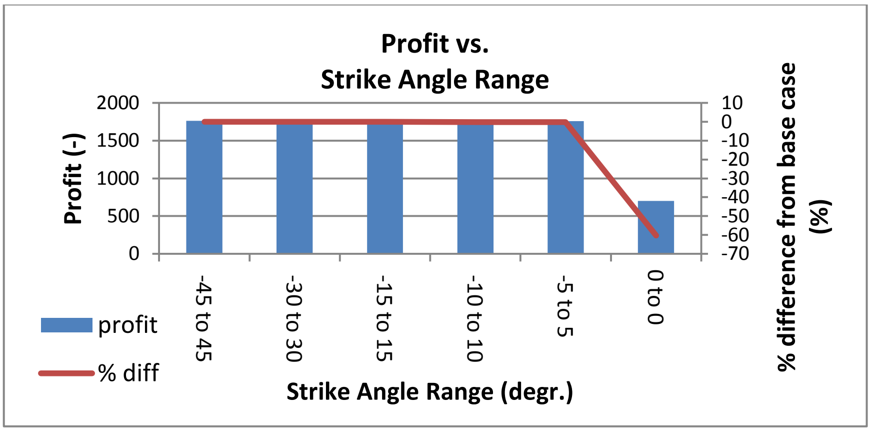


Figure 39 - Profitability Analysis on Strike Angle Range.

Hanging Wall and Footwall Dip Angle Range

In order to analyse the optimization algorithm performance when the maximum dip angle range is subsequently reduced the default shape dip has to be changed to the upper angle of the dip angle range. What can be observed in Figure 40 is that the best profit is obtained when the default shape dip is set to 45 degrees and the range is kept widest at 40 to 90 degrees. This shows the positive influence of using a control wireframe to define the local shape dip instead of a default dip for all stopes in an optimization scenario.

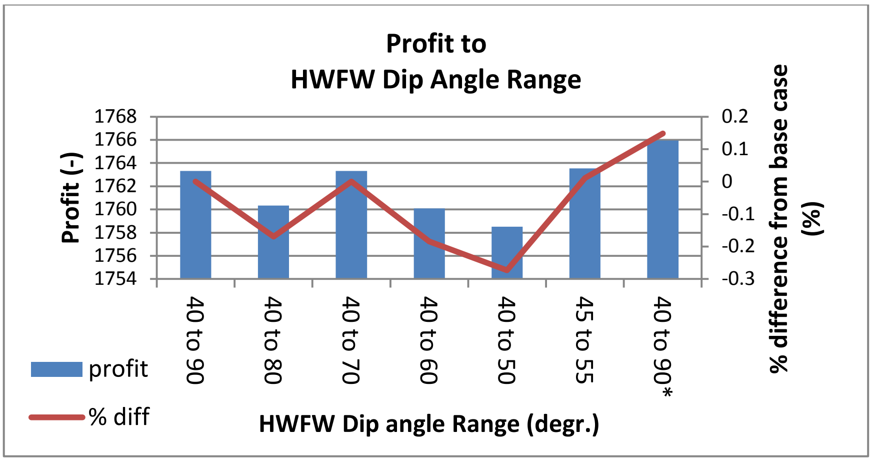


Figure 40 - Profitability Analysis on HWFW Dip Angle Range. (* default shape dip set to 45 degr.)

Maximum Dip Angle change between the Hanging wall and footwall

When analysing the results presented in Figure 41 it can be concluded that reducing the maximum dip angle change between the hanging wall and footwall reduces the optimization algorithm's capability of finding the optimal stope. Since the algorithm found an optimal dip of 47.6 degrees for both the hanging wall and the footwall the 'real' dip change is 0-degrees (base case scenario). The reduction of the maximum dip angle change reduces the freedom of the hill-climbing algorithm and should therefore be kept at maximum allowable limitations. This should, in practice, not be a big problem since hanging wall and footwall angles do not necessarily have to be the same.

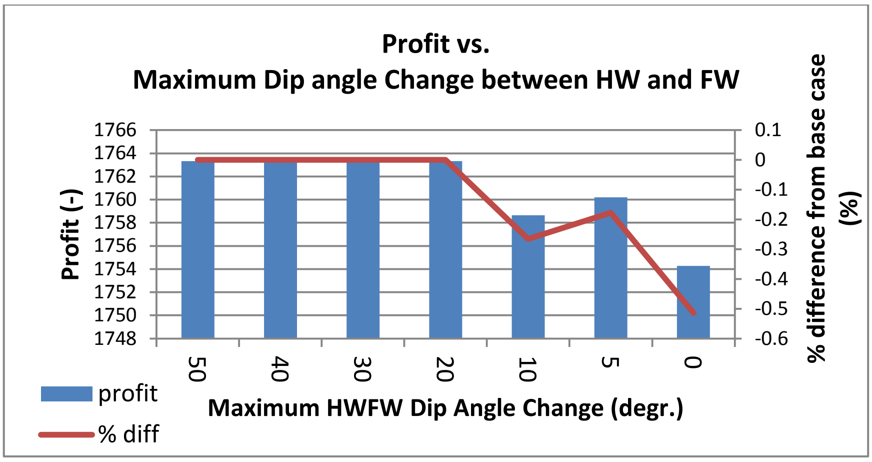


Figure 41 - Profitability Analysis on Maximum Dip Angle Change between HW and FW.

3.1.4 CONCLUSION

In general, it is concluded that the optimization software is able to find the optimum stope solution in manually created block models.

The provided optimal stope solution for the Strike Model is in the same range as the manual design whilst for the Hanging Wall and Footwall Model the optimizer found a better solution to the optimization problem. Based on the optimization approach applied by the AMIRA Stope Optimization algorithm the smaller slice intervals should give more accurate results and it is observed that a too large interval can reduce accuracy.

After evaluating both the test models it is concluded that the best solution is obtained by using the base case scenario 'loose settings'. The strike model stope solution could only be improved when the allowable strike range was increased resulting in an increased optimal strike angle. The Hanging Wall and Footwall Model stope solution could only be improved when the default shape dip was set to 45 degrees (very close to the optimum of 47.6 degrees) showing how the use of a wireframe to define a local dip (e.g. based on geology) at each stope location rather than using a default dip could increase the optimization result.

3.2 OPTIMIZATION COMPARISON TO MANUAL STOPE DESIGN

After the evaluation of the stope optimization software whilst comparing to simple models, this section will evaluate the optimization software performance when applying the stope optimization algorithms to a real-life mineral resource model. The Lappberget orebody (part of the Boliden Garpenberg mine) was chosen for this comparison. Optimized stopes resulting from the stope optimization software will be compared to the manually designed stopes.

3.2.1 GENERAL LOCALITY

The Garpenberg mine (located near the town of Garpenberg, Bergslagen, Sweden) is Boliden's biggest underground mining operation (Zn-Pb-Ag(Cu,Au) sulphide ore) and is the only longhole stoping mine that the company currently operates in Sweden. Acquired by the company in 1957, the operation consists of several orebodies of which the Lappberget orebody is the main source of production (Figure 42). In 2012 total ore production from the Garpenberg operation reached 1,484 ktonnes.

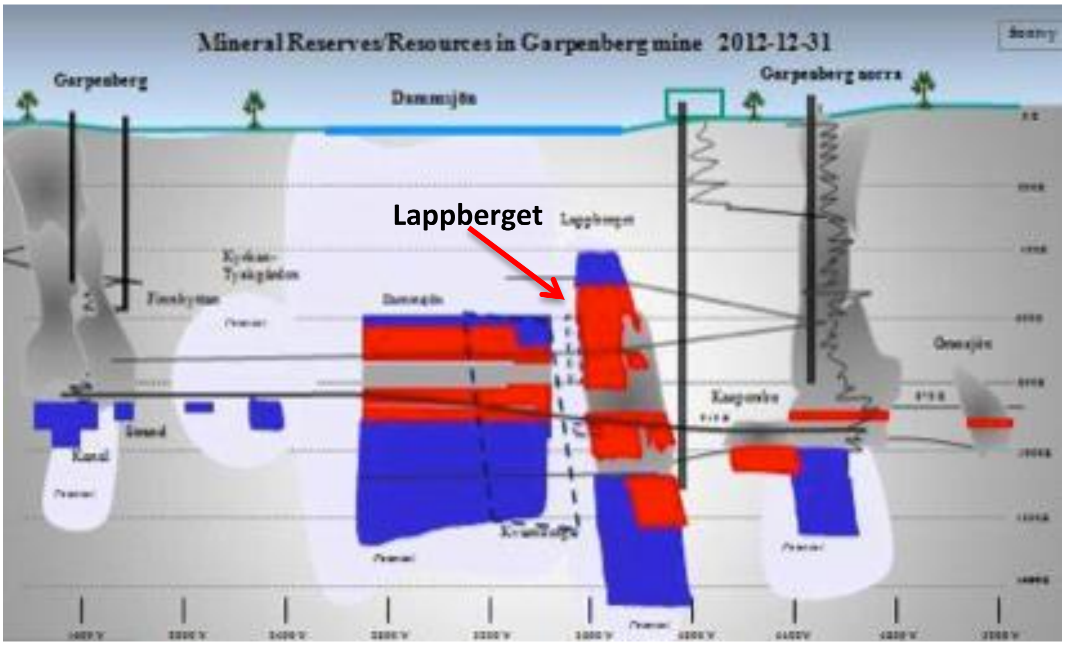


Figure 42 - Garpenberg Mine and Orebody overview (Reserves and Resources shown in Red and Blue respectively).

3.2.1.1 GEOLOGY

The Garpenberg mine is located in the province of Bergslagen in south central Sweden. This province is known for hosting multiple Palaeoproterozoic igneous ore deposits. The most common types of deposits are Fe-oxide and polymetallic sulphide deposits.

The massive sulphide orebodies are surrounded by calcitic marble, which is sometimes altered into dolomite. The Footwall rock consists of phlogopite-biotite-cordierite-sericite-quartz altered felsic volcaniclastic rocks. The Hanging wall consists of relatively unaltered volcaniclastic and sedimentary rocks and dacitic intrusions.

3.2.1.2 MINING METHOD

The main mining method applied within the Lappberget orebody is transverse longhole stoping with a primary –secondary stoping sequence. The decision to use this method is mainly based on the massive shape and near vertical and dip of the orebody. In thin areas longitudinal stoping is applied (mainly at the edges outer of the orebody). The sublevel spacing is 25m (limited by drill hole accuracy) with primary stopes having a width of 10 to 15m and secondary stopes being 15 to 20m wide. Part of the 2013-2014 stoping pattern is shown in Figure 43.

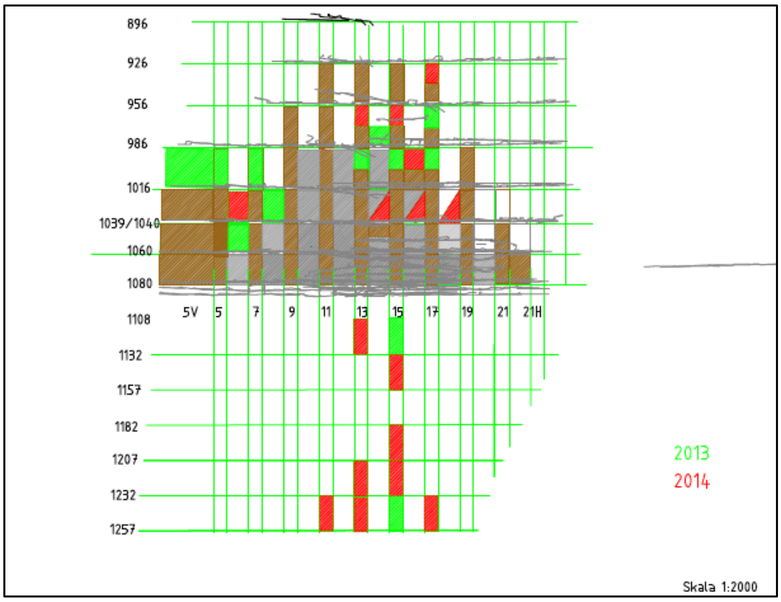


Figure 43 - Part of the Lappberget Stoping Pattern 2013-2014.

3.2.2 EVALUATION AREA OF INTEREST

In order to evaluate the stope optimization algorithm performance in a real-life situation the decision was made to compare the manual stope designs of a sublevel of the Lappberget orebody to the optimal solution as provided by the optimization algorithm for the same level. Because only the latest block model including the 2014 Long Term Prices was made available to the author, comparison was limited to the 1250 level as this is the only level for which designs are available based on the latest block model and price assumptions. Figure 44 shows the most recent (2013) stope designs for the 1250 level.

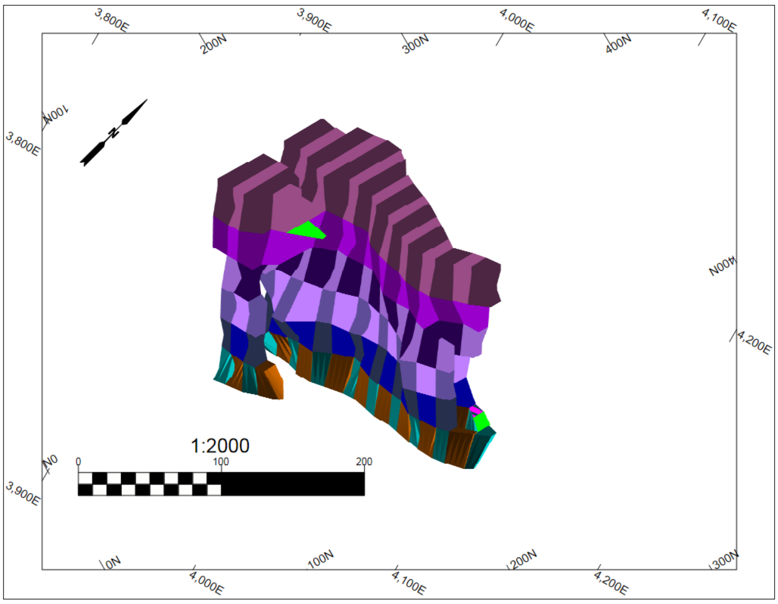


Figure 44 - 2013 Stope Designs for the Lappberget 1250 level.

The 1250 level is a new production level within the Lappberget orebody that will go into production during 2013 and as production will start at the 1257 sublevel (Figure 43) it was decided to specifically compare this level. The 1257 sublevel is presented in Figure 45 with primary stopes shown in turquoise and secondary stopes coloured orange.

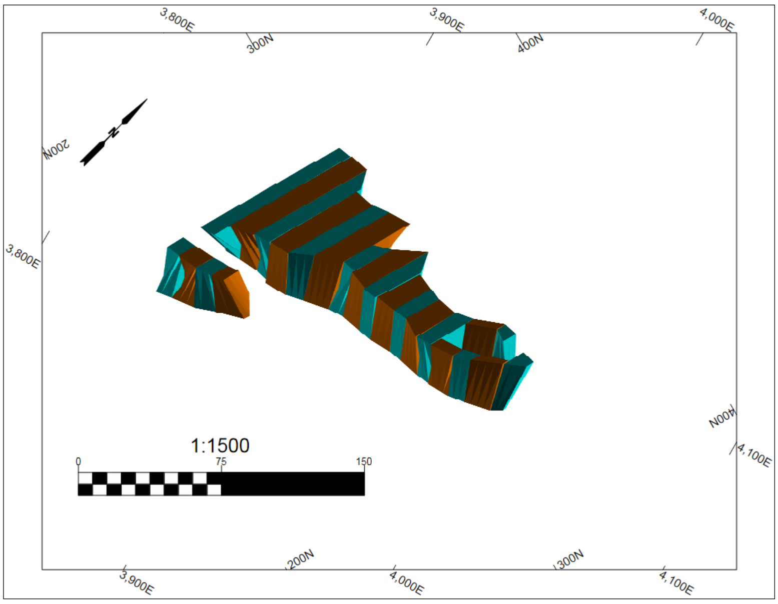


Figure 45 - Lappberget 1257 sublevel.

3.2.3 ASSUMPTIONS FOR COMPARISON

In order to compare the Stope Optimizer stope solution to the manually designed stopes the following is assumed:

- The manual designs are based on the 2013 resource model as was made available to the author.
- The metal prices used to design the manual stopes are the LTP14 prices and all NSR calculations are based on these prices.
- Manual stope optimization is *only* based on the cut-off value of 370 SEK/t.
- The manual stope designs are 'the best' solution that could realistically be found by the engineer.
- Stope profit is calculated as:

$$Profit = tonnes \times (average\ NSR - 370SEK/t)$$

3.2.4 MANUAL STOPE DESIGN

Engineers on site create the stope designs for the Garpenberg mine manually. With the deposit being polymetallic, a Net Smelter Return or NSR value of a block is calculated based on all metal grades. Based on this NSR value and taking into account all operating costs, a cut-off value 370 SEK/t is used to delineate ore reserves. The ore reserves together with geotechnical constraints dictate the stope design. Geotechnical conditions at the 1250 level dictate the use of transverse longhole stoping in a primary- and secondary stoping sequence (Table 3).

Parameter	Primary Stopes	Secondary Stopes
Stope Width (m)	10	15
Stope Height (m)	25	25
Stope Span (m)	40	40

Table 3 - Geotechnical Design Constraints for stopes at the Lappberget 1250 level.

The approach taken for stope design is to create horizontal block model slices at 25-metre intervals and to draw a contour string that follows the 370 SEK/t cut-off to ensure only ore above this cut-off falls within the horizontal slice. Secondly, the horizontal plane is divided by the primary and secondary stope widths on a regular grid. Once this approach is executed at each 25m sublevel, the individual horizontal slices are connected vertically to create stopes. These stopes can be interrogated to find the stope profitability (tonnes x (NSR/t – Cost/t)). Manual optimization (e.g. adapting the stope shape to increase profit) can now take place to try and improve the profitability of the stope. The stope designs are reconciled on a yearly basis and by an engineer whilst doing the blast design before the stope is taken into production.

3.2.4.1 MANUAL DESIGN TONNAGES AND VALUES

The manually designed stope wireframes were imported into the Deswik.CAD environment and transformed into solids that could be interrogated against the 2013 block model. The results of this interrogation are shown in Table 4 where stopes are ordered by their Xmin coordinate. Based on the manual stope designs there is 680 ktonne of mineable ore within the stope boundaries at an average profit of 758 SEK per tonne of ore mined (assuming 370 SEK/t as the total cost).

STOPE (Z_X)	Tonnage (t)	Average NSR (SEK/t)	Total Profit (SEK)	Average profit (SEK/t)
-1257_3693	7.49E+04	839	3.51E+07	469
-1257_3703	1.08E+05	856	5.24E+07	486
-1257_3718	7.40E+04	1218	6.27E+07	848
-1257_3728	8.60E+04	1362	8.53E+07	992
-1257_3743	4.63E+04	1291	4.27E+07	921
-1257_3753	5.38E+04	1023	3.51E+07	653
-1257_3768	2.42E+04	1330	2.32E+07	960
-1257_3778	4.25E+04	1452	4.59E+07	1082
-1257_3793	3.13E+04	1353	3.07E+07	983
-1257_3803	3.55E+04	1437	3.78E+07	1067
-1257_3818	2.04E+04	1455	2.21E+07	1085
-1257_3828	3.02E+04	1380	3.05E+07	1010
-1257_3843	2.12E+04	756	8.18E+06	386
-1257_3853	1.28E+04	425	7.04E+05	55
-1257_3868	9.30E+03	408	3.50E+05	38
-1257_3878	6.17E+03	606	1.46E+06	236
-1257_3893	4.48E+03	724	1.58E+06	354
TOTALS	**6.81E+05**		**5.16E+08**	**758**

Table 4 - Sublevel 1257 Manual Stope Design Tonnages and Values.

3.2.5 AUTOMATED STOPE DESIGN

3.2.5.1 STOPE OPTIMIZATION APPROACH

In order for the stope optimizer to find the best stope solution the following steps were subsequently undertaken:

1. Run the optimization process with loose settings on the 1257 sublevel.
2. Subsequently reduce the maximum internal waste percentage.
3. Apply stope smoothing and include the sublevels above and below the 1257 level in the optimization process for both horizontal and vertical smoothing.
4. Further manual smoothing of the stope optimizer output stopes to create a ‘mineable’ stoping layout.

The Stope Optimizer solution was compared to the manual design after each subsequent step.

3.2.5.2 OPTIMIZATION SCENARIO SETTINGS

Framework Limits

Before optimization of the stopes can take place the starting point and extend of the area for optimization has to be defined. As only the 1257 sublevel is considered in this study, the coordinates of the left end lower corner point of the first stope on the 1257 sublevel has to be provided. This coordinate is used as the starting point for optimization. The other input required is the stope width. Because a primary secondary stoping sequence is required with stopes of 10 and 15 meters wide respectively, the increment of the X-coordinate is set to 25 meters (1 primary and 1 secondary stope). The individual widths of the primary and secondary stopes are specified in the sub-shape control menu. Finally, the required stope height of 25 meters is entered as an increment of the Z-coordinate. The Y increment in this case has to be a large number to make sure that the Stope Optimizer will optimize throughout whole block model (in the Y-direction).

Parameter	Coordinate	Increment	Number of Increments
Origin X	3693.4	25	15
Origin Y	350	500	-
Origin Z	-1257	25	1

Table 5 - Framework Limits Lappberget 1257 sublevel.

Optimization Control Parameters

Table 6 summarizes the optimization control parameters as used in the Lappberget 1257 sublevel optimization. A clarification of each parameter can be found in the Stope Optimizer guideline provided in appendix A.

Parameter	Unit	Value
Default Shape Dip	degr.	90
Default Shape Strike	degr.	00
Stope Width (Primary)	m	10
Stope Width (Secondary)	m	15
Stope Height	m	25
Minimum Stope Length	m	5
Maximum Stope Length	m	200
Minimum Waste Pillar Width	m	20
Cut-off Value	-	370
Max Side Ratio (Top-to-Bottom)	-	2.5
Max Side Ratio (Front-to-Back)	-	2.5
Shape Slice Interval	m	1
Minimum Strike Angle	degr.	-45
Maximum Strike Angle	degr.	+45
Maximum Strike Angle Change (Front-to-Back)	degr.	90
Minimum HW and FW Dip	degr.	50
Maximum HW and FW Dip	degr.	120
Maximum Dip Angle Change	degr.	70
Maximum Internal Waste Percentage	%	100

Table 6 - Optimization Control Parameters Lappberget 1257 sublevel.

3.2.5.3 OPTIMIZATION RESULTS

Loose settings

Figure 46 shows the stopes (green) that resulted from the optimization process whilst using loose settings. The manually designed stopes are shown in turquoise and orange. It is observed that the optimization algorithm returned larger stopes compared to the manually designed stopes. Table 7 summarizes the Stope Optimizer solution and compares it to the manual stope design figures. The optimization algorithm found almost twice the amount of mineable ore whilst the total profit increased by 17%.

Scenario	Tonnes (t)	NSR (SEK/t)	Total Revenue (SEK)	Total Costs (SEK)	Total Profit (SEK)	Av. Profit (SEK/t)
Manual	680,728	1,128	767,903,546	251,869,388	516,034,158	758
'Loose settings'	1,000,512	971	971,643,365	370,189,598	601,453,767	601
%Difference	**+47%**	**-14%**	**+27%**	**+47%**	**+17%**	**-21%**

Table 7 - Loose Settings Optimizer Output compared to the Manual Design.

The difference is caused by the fact that the smallest mining unit (stope width x stope height x slice thickness) is evaluated against the cut-off. The average value is calculated for this 'slice' of stope and if this average value is above cut-off, the slice is included in the stope. Due to this optimisation approach it is possible that a stope slice intersecting a high-grade core surrounded by low-grade waste material, is evaluated to be above cut-off. The stope optimization software will however, report the amount of internal waste. With the maximum internal waste percentage set to 100%, this resulted in 'optimized' stopes containing up to 50% internal waste. From a production point of view this is not an acceptable percentage therefore the maximum allowable internal waste percentage was subsequently reduced in the next series optimizations.

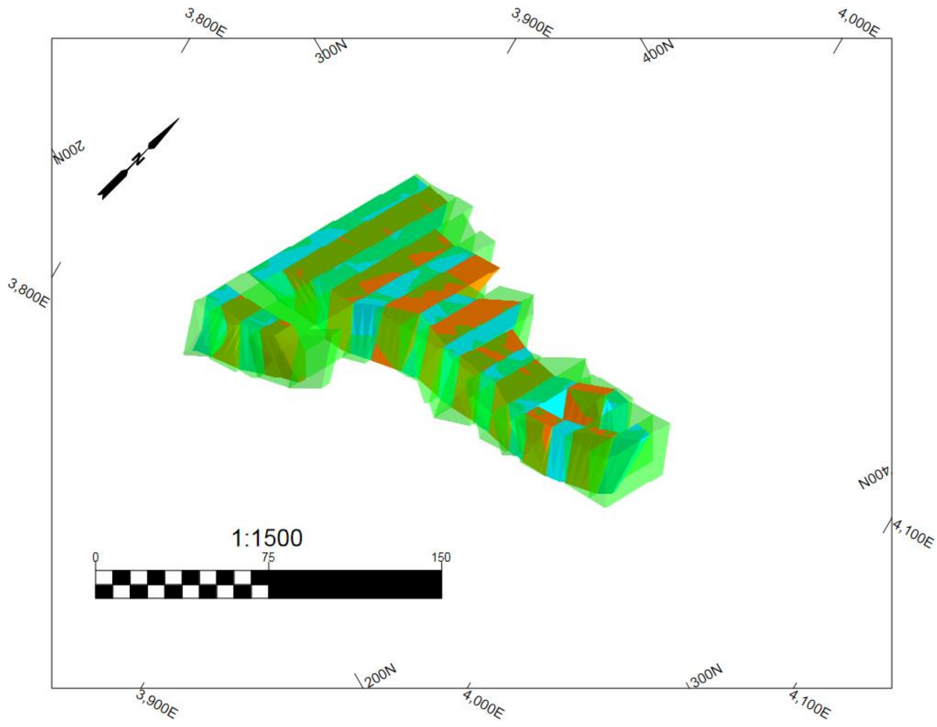


Figure 46 - Stope Optimizer Solution for the 1257 sublevel using 'loose settings'.

Reducing the Maximum Internal Waste Percentage

Lowering of the maximum internal waste percentage reduces the difference in tonnages and value significantly between the stope optimizer stope solution and the manually designed stopes. Slices are first evaluated against the cut-off but during the seed shape generation stage and slices that contain too much internal waste will be discarded. Stope Optimizer outputs for different internal waste percentages are graphically presented in Figure 47 and according tonnages and values are provided in Table 8.

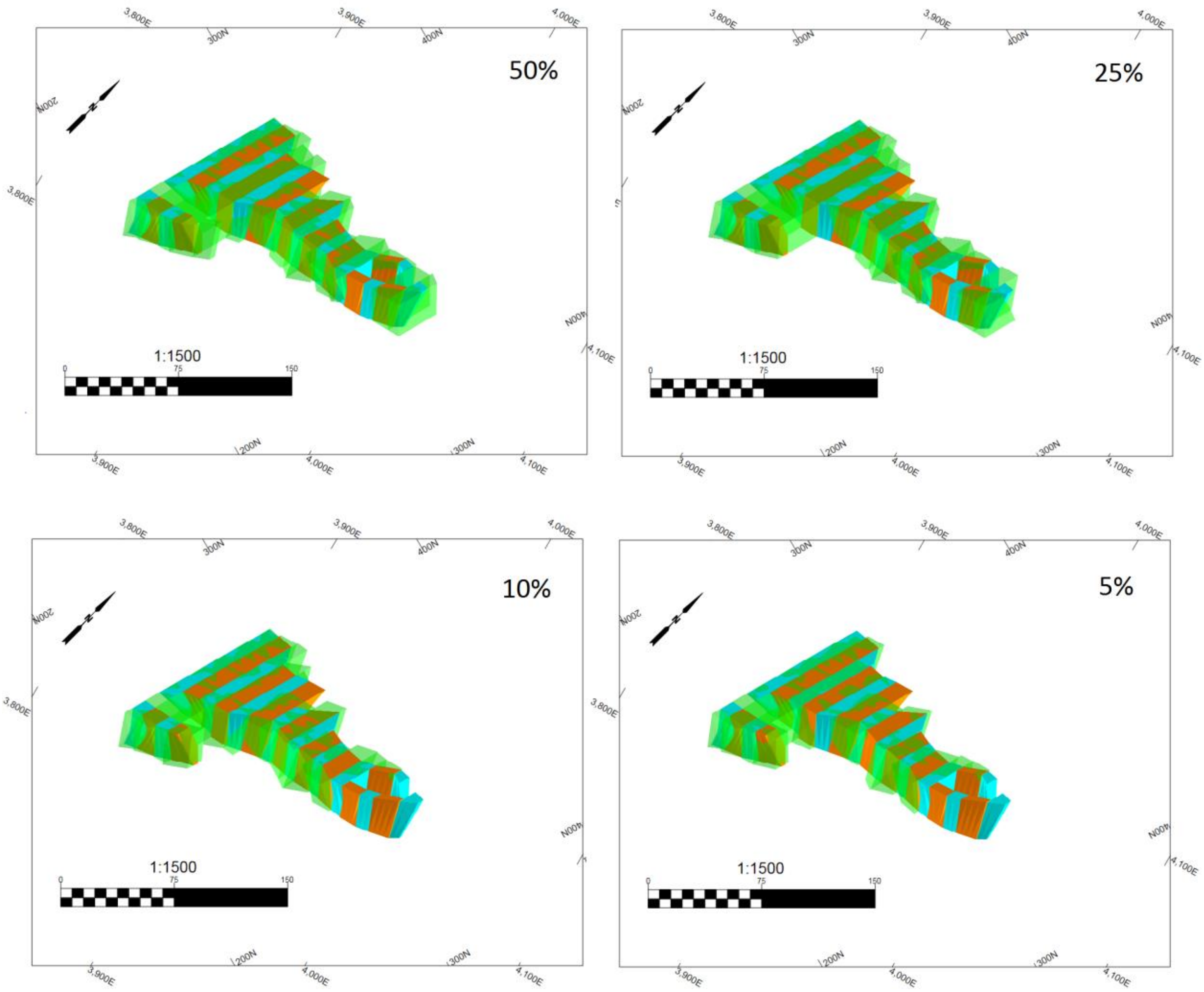


Figure 47 – Stope Optimizer Solution at reduced maximum internal waste percentages.

Scenario	Tonnes (t)	Total Revenue (SEK)	Total Costs (SEK)	Total Profit (SEK)	Av. Profit (SEK/t)
Manual	680,728	767,903,546	251,869,388	516,034,158	758
100%	983,123	971,346,177	363,755,329	607,590,848	618
50%	976,320	968,402,010	361,238,278	607,163,139	622
25%	952,044	953,419,253	352,256,114	601,163,139	631
10%	816,697	875,788,779	302,177,775	573,611,004	702
5%	757,272	845,661,845	280,190,488	565,471,356	747
%Difference to Manual Design (5%)	+11%	+10%	+11%	+10%	-1.5%

Table 8 - Reduced Internal Waste Percentage Optimizer Outputs compared to the Manual Design.

It is observed that for a maximum internal waste percentage of 5%, the stope optimizer designed stopes are approaching the manually designed stopes. For this 5% maximum internal waste scenario the stope optimizer optimized stopes yield 11% more mineable tonnes of ore and a total profit increase of 10%. It should be noted that it is virtually impossible to 're-create' the manual design by using the Stope Optimizer. During manual design one can apply soft constraints (that is, change the design rules slightly from stope to stope to adapt to the local conditions) whilst the Stope Optimizer software will apply the same set of strict rules to the design of each stope. Applying 'Stope Smoothing' further optimized the 5% maximum internal waste scenario.

Stope Smoothing and further Manual Smoothing

One of the drawbacks of using the Stope Optimizer is the fact that the optimization takes place at each individual stope location without taking the surrounding stopes into consideration. The result is a set of optimized stope shapes that are individually the most optimized solution but are technically impossible to mine. One of the stope design criteria for ore extraction from the Lappberget orebody is that the stopes have to be lined-up horizontally and vertically. One way to do this is by activating the stope smoothing algorithm in the Stope Optimizer software. The stope smoothing algorithm will try to connect adjacent stopes both horizontally and vertically whilst still respecting the constraints as set by the engineer. The functionality is still under development and cannot always connect all stopes because the algorithm will never violate the optimization constraints (e.g. cut-off grade, waste percentage, strike angle). It is up to the end-user to finalize the design under such circumstances.

Both horizontal and vertical smoothing was applied during the optimization process by including the sublevels above and below the 1257 level into the optimization process. The influence of stope smoothing as compared to the non-smoothened stope designs is visualized in Figure 48 and Figure 49.

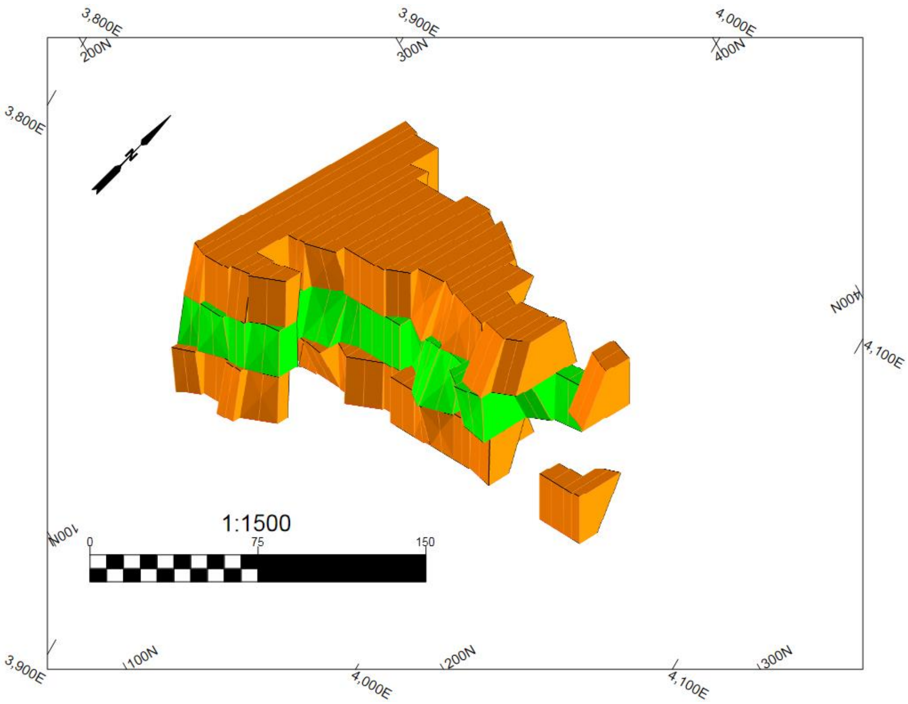


Figure 48 - Stope Optimizer Output without Stope Smoothing (individual stopes optimized).

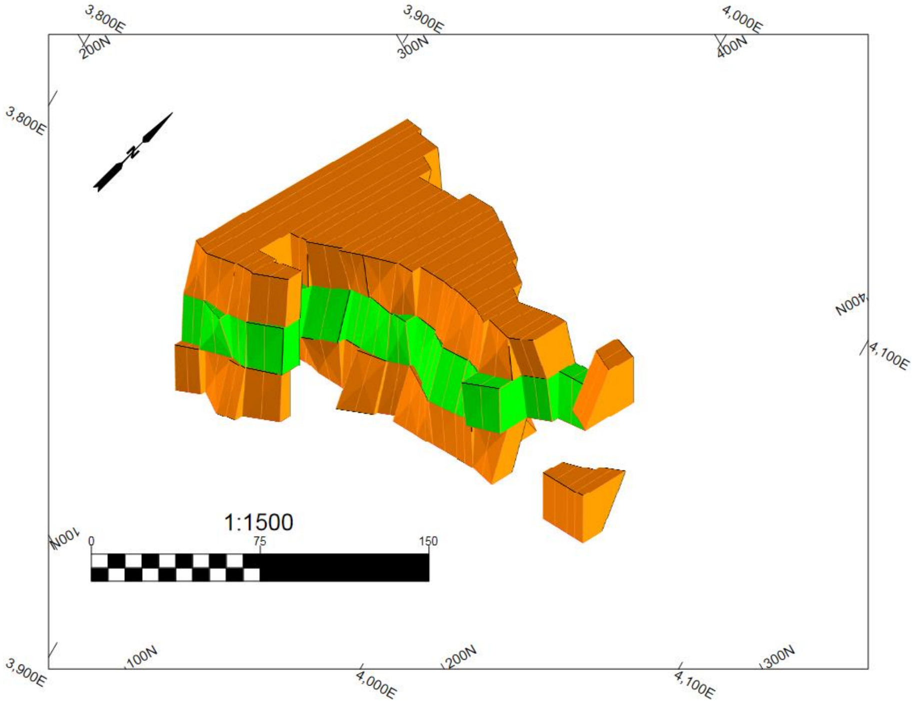


Figure 49 - Stope Optimizer Output with Stope Smoothing (the 'more feasible' design).

It is clearly visible that the optimization process went from 'optimizing the individual stope' towards more feasible optimization where the interconnectivity between the individual stopes is acknowledged and the smoothing algorithm connected stopes both horizontally and between sublevels. When comparing the smoothened stope tonnages and values in Table 9 it is observed that the smoothing of the stopes resulted in a 4% loss in stope tonnage. However, due to the fact that total stope revenues and profit decreased by just 2%, the average profit per tonne of ore mined increased by 2% compared to the non-smoothened case. Compared to the manual stope design the increase is even bigger and falls within the range of 7 to 8%, resulting in an average profit per tonne of ore mined increase of 0.5%.

Scenario	Tonnes (t)	Total Revenue (SEK)	Total Costs (SEK)	Total Profit (SEK)	Av. Profit (SEK/t)
Manual	680,728	767,903,546	251,869,388	516,034,158	758
5%	757,272	845,661,845	280,190,488	565,471,356	747
Stope Smoothing	728,572	824,608,275	269,571,610	555,036,664	762
%Difference to 5% non smooth	-4%	-2%	-4%	-2%	+2%
%Difference to Manual Design	+7%	+7%	+7%	+8%	+0.5%

Table 9 - Stope Smoothing Optimizer Output compared to the Manual Design.

The smoothened output as visualized in Figure 49 was further processed manually to obtain a final design for the 1257 sublevel which is the design as presented in Figure 50. By manual further smoothing some revenue is lost and total tonnage increases slightly which is to be expected. Total profit compared to the manual design however is still up by 7% (Table 10). Total Tonnage and Profit of the final stope design are summarized in Table 10.

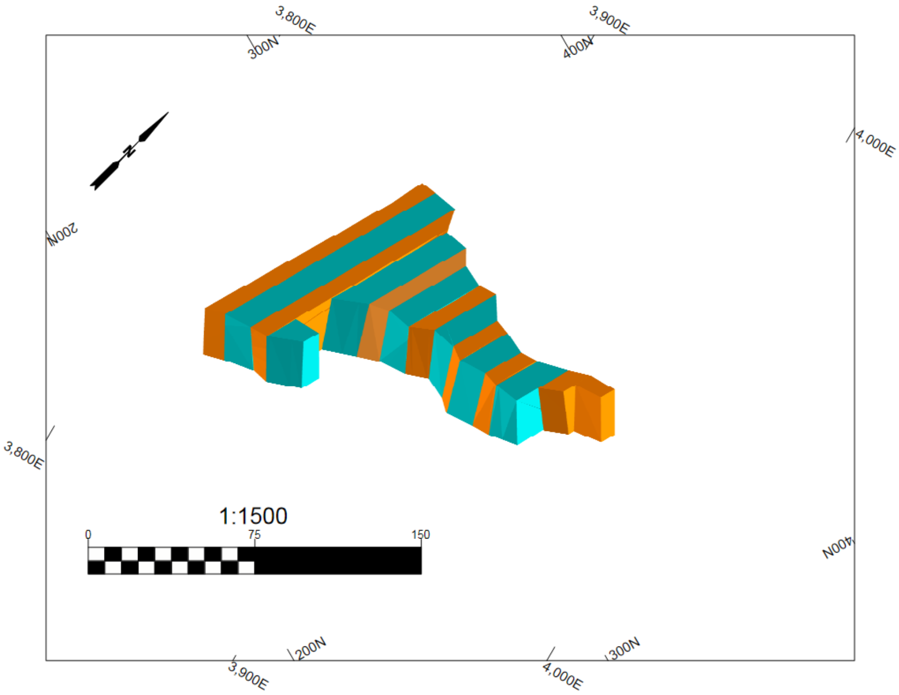


Figure 50 - Final Stope design for the Lappberget 1257 sublevel.

Scenario	Tonnes (t)	Total Revenue (SEK)	Total Costs (SEK)	Total Profit (SEK)	Av. Profit (SEK/t)
Manual	680,728	767,903,546	251,869,388	516,034,158	758
Stope Smoothing	728,572	824,608,275	269,571,610	555,036,664	762
Further Manual Smoothing	733,371	821,599,720	271,347,270	550,252,450	750
%Difference to Manual Design	+8%	+7%	+8%	+7%	-1%

Table 10 - Further Processed Scenario Compared to Stope Smoothing Scenario and Manual Design.

In order to compare the manual design to the optimized design in more detail; tonnages, revenues and profits were analysed on a stope-to-stope basis. When analysing the graphs in Figure 51 and Figure 52 it is observed that in general the Revenue and Profit per tonne of ore curves of both the manual and optimized design follow follow the same trend. However, the Optimizer in many cases found more tonnes of ore resulting in higher stope profits. There are also situations where the optimizer found less tonnes of ore compared to the manual design (for example in stopes 3743, 3753 and 3793), however the average value of the ore within the optimized stopes is higher which compensates for the decreased stope tonnage.

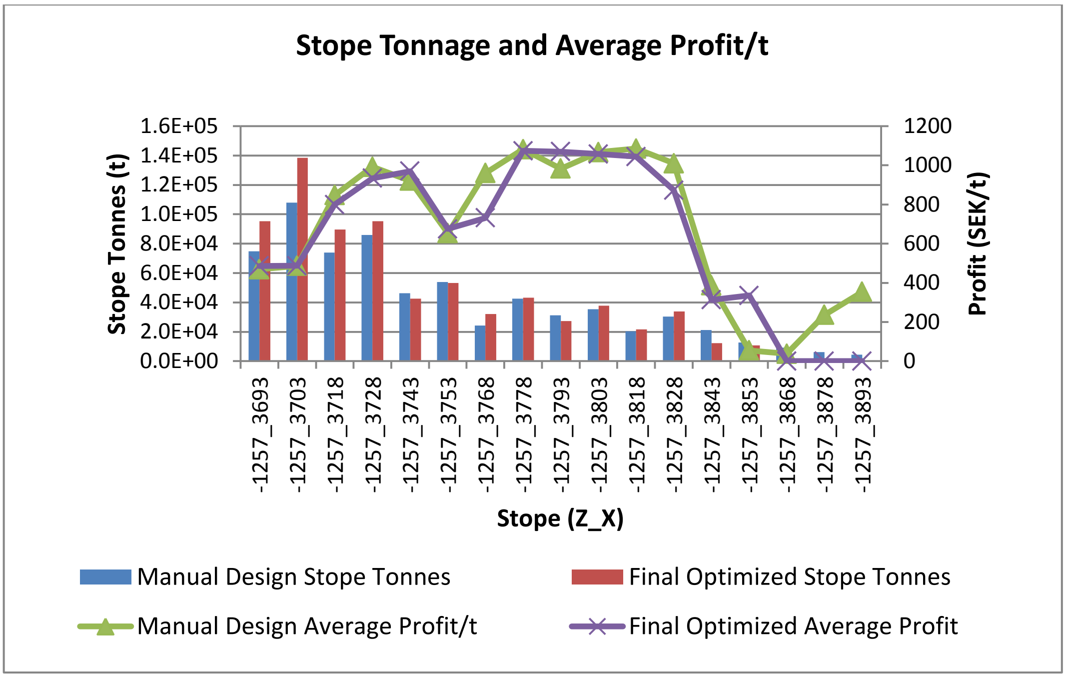


Figure 51 – Stope Tonnage and Average Profit comparison between Manual and Final Optimized Stopes (Lappberget 1257).

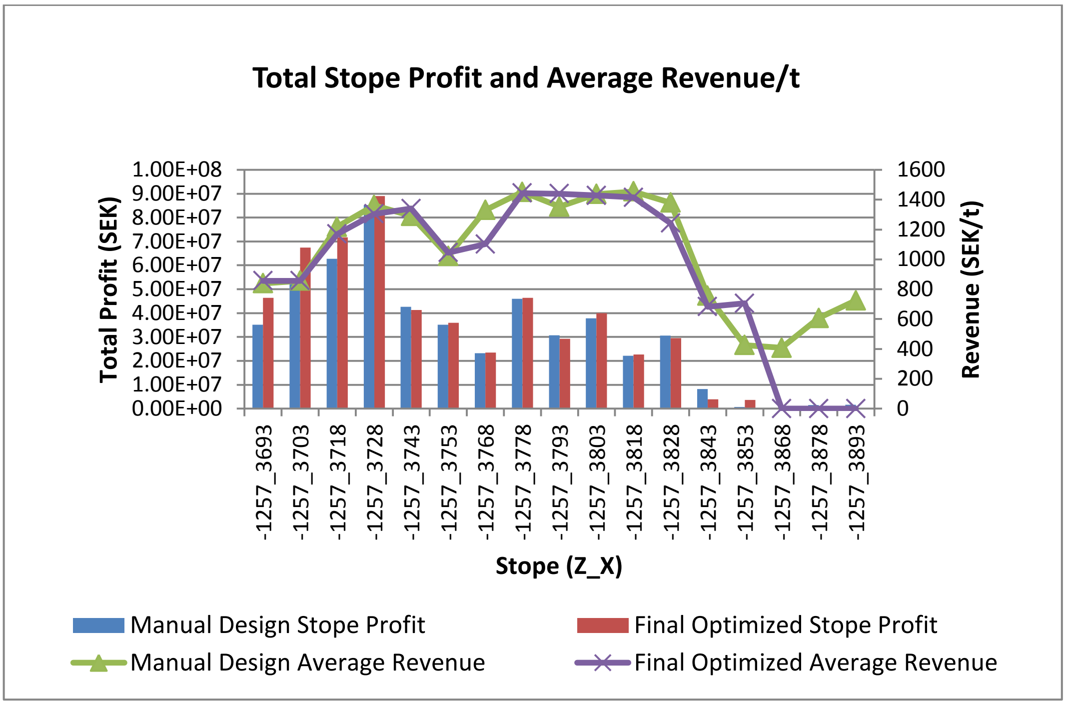


Figure 52 – Stope Profit and Average Revenue comparison between Manual and Final Optimized Stopes (Lappberget 1257).

Outlier analysis

When analyzing the graphs in Figure 51 and Figure 52 significant differences between the manual design and the optimized design are observed in stopes 3768 and 3853.

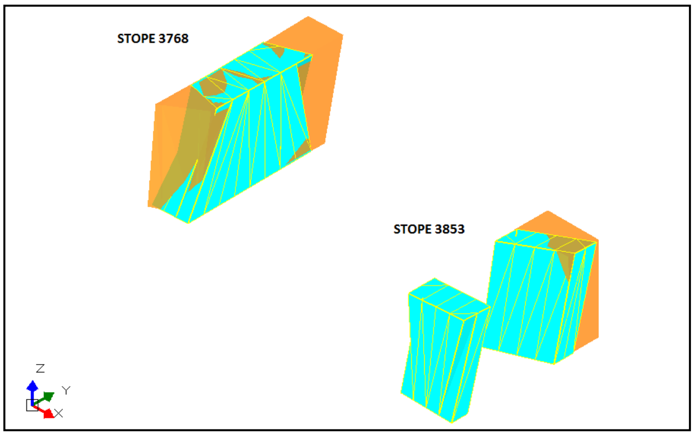


Figure 53 - Stopes 3768 (left) and 3853 (right).

Figure 53 shows stopes 3768 and 3853 with the Stope Optimizer result shown in orange and the manual designs shown in turquoise. The difference in average revenue and profit/t in stope 3768 can be explained by the fact that more ore meets the requirements (Ore value>370, maximum internal waste 5%) than is included in the manually designed stope. In other words, the optimization algorithm returned a better and more accurate stope shape based on the optimization of total metal above cut-off. The result is an optimized stope for which the stope average revenue and profit per tonne may be lower whilst due to the increased tonnage, total profit is the same or higher.

The opposite situation is observed in stope 3853 where less ore meets the requirements than is included in the manual stope design. The optimizer designed a single stope with a totally different shape compared to the manual design. However, the average revenue per tonne of ore for this optimized stope is 66% higher and with the total stope tonnage decreased by 16% this results in a profit increase of more than 400%.

The root cause of the difference between the optimized and manual design is believed to be found in the fact that the optimzation algorithm uses the same strict set of design constraints to optimize each individual stope. Figure 54 shows all the evaluated slices at the 1257 sublevel that have a value >370 SEK/t. However, because there is a second constraint (maximum of 5% internal waste), any slices that do not meet this second constraint are discarded and are not transformed in a seed shape.

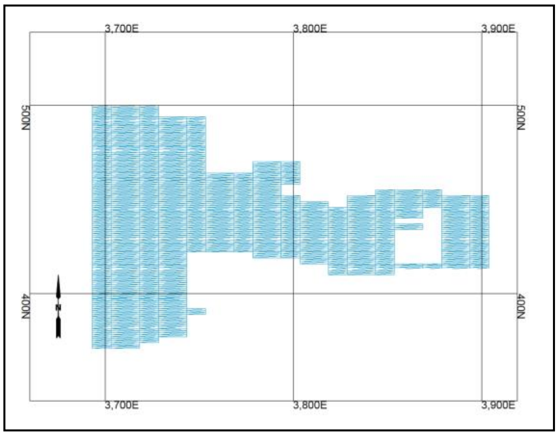


Figure 54 - Slice evaluation of the Lappberget 1257 sublevel.

3.2.6 CONCLUSION

The stope optimization software was extensively tested on the 1257 sublevel of the Lappberget orebody at the Garpenberg Mine. The results are considered satisfying by means of similarity with the manually engineered designs.

As there is no human factor involved in stope optimzation using the optimization software, the designs are not a full match with the manually designed stopes. The optimization algorithm applies the same strict set of rules at each stope location, whereas an engineer can apply slightly different constraints to each individual stope. As a result of this, no stopes were designed by the optimizer towards the eastern end of the orebody because the maximum internal waste percentage limit was not met.

The biggest advantage of using the stope optimization to design stopes is the fact that the optimization algorithm builds up the stope by evaluating the smallest mining unit (a slice) against the cut-off grade instead of the full stope (as would be in manual design). By this approach the resolution is increased as a stope can be build up by many slices that are all above cut-off and meet the design constraints. In manual design one would create an initial full stope design wireframe, convert this wireframe into a solid and interrogate the solid against a block model to find if the stope meets the desired cut-off grade. This means that only the stope head grade is evaluated to be above the cut-off whilst the optimization algorithm will evaluate the head-grade of the smallest mining unit against the cut-off, which increases the optimality of the stope solution.

4 CASE STUDIES

4.1 INTRODUCTION

Three different case studies were conducted to clarify the use of the AMIRA stope optimization software and to apply the proposed approaches to underground mine optimization and grade risk assesment to one of Boliden's mining projects.

4.2 CASE 1: THE KANKBERG UNDERGROUND STUDY

4.2.1 INTRODUCTION

The Kankberg mine is a gold mining operation in the Boliden area. The mine was closed in the late 1980's but renewed exploration campaigns in the period 2004-2010 identified additional mineralization at about 1 km distance from the closed mine (Golder Associates, 2011). Mineralization consists of gold, silver, telurium, bismuth, and copper.

Cut & Fill mining was selected as the mining method of choice as test drilling revealed extreme rock hardness that prevented the drilling of long blastholes required in sublevel stoping methods. After consecutive periods of production it was observed that rock hardness is less and the encountered extreme rock hardness was a local occurence. Because rock conditions in fact do allow the application of sublevel stoping methods, the aim of this study was to use the stope optimization software package in order to develop a new optimized mine plan for the Kankberg mineral resource.

4.2.2 THE KANKBERG RESOURCE MODEL

4.2.2.1 GEOLOGICAL INTERPRETATION

With the Kankberg deposit being a polymetallic deposit, the measured individual grades in raw exploration drill holes were combined to obtain a Net Smelter Return value for each measured sample (similar to an equivalent grade). The mineralization interpretation of the Kankberg deposit was guided by application of a cut-off on the NSR values as measured in the raw drill holes. The applied cut-off was set to 525 SEK/t and is a break-even grade representing the cost of producing and processing a ton of ore by means of cut&fill mining and subsequent processing.

4.2.2.2 RESOURCE STATEMENT

The obtained mineralization interpretation is as such an interpretation of the economically extractable part of the deposit only rather than a geological interpretation of the complete ore zones. Figure 55 shows the discovered ore zones as interpreted in the Kankberg project area and the resulting Mineral Resource statement is provided in Table 11.

RESCAT	TONNES	AU	TE	AG	BI	CU	S	NSR
	t	ppm	ppm	ppm	ppm	%	%	SEK/t
Measured	1,410,000	3.03	175	11.3	133	0.01	0.45	1039
Indicated	3,040,000	4.75	200	16.3	160	0.02	0.86	1538
Inferred	440,000	6.75	196	15.1	118	0.02	0.27	2058

Table 11 - Mineral Resource startement of the Kankberg mineralization.

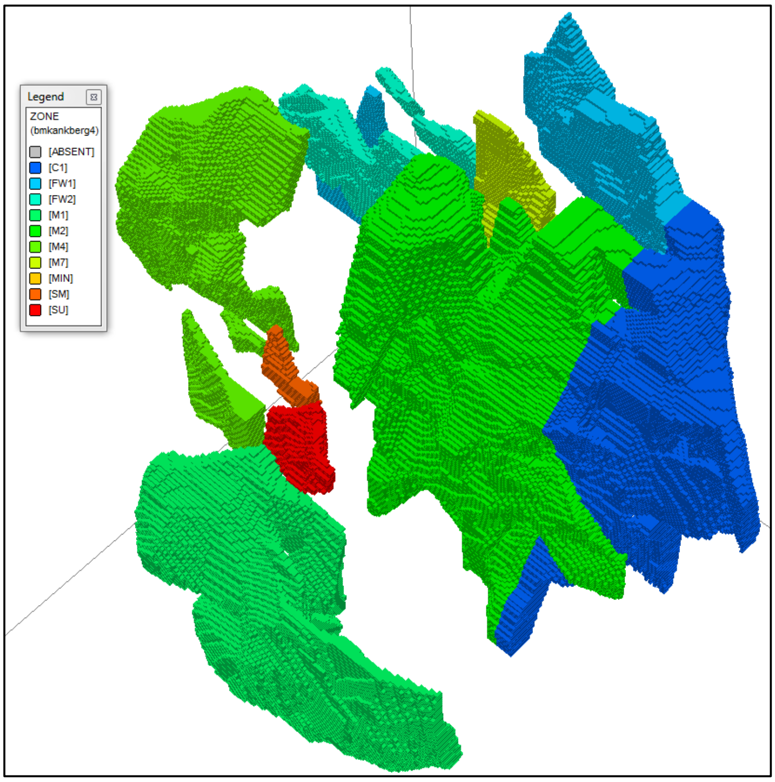


Figure 55 - Kankberg ore zones (Boliden Mineral AB, 2012).

4.2.3 OPTIMIZATION APPROACH AND ENCOUNTERED PROBLEMS

As a first approach to stope optimization for the Kankberg underground mine the M1 zone was selected (Figure 55). Mining has not yet started in this zone which meant that the complete zone could be optimized for sublevel stoping.

The NSR – Tonnage curve of the M1 zone is presented in Figure 56 and reveals a major problem related to stope optimization. Due to the application of a NSR cut-off in the raw drill holes (to assist geological interpretation) the value of ore in the resulting economic mineral resource all at least has a value of 525 SEK/t as is observed in the grade tonnage curve (Figure 56).

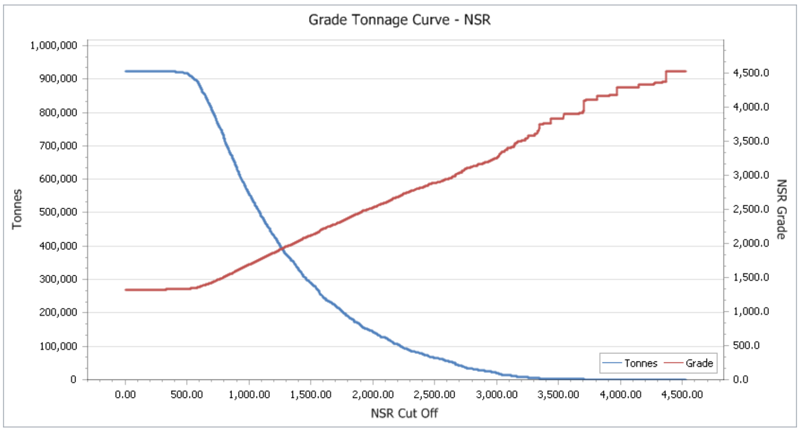


Figure 56 - NSR - Tonnage Curve for the Kankberg M1 zone.

Due to the application of a cut-off value that assumes Cut & Fill mining, valuable information about the resource is lost when considering sublevel stoping methods. Sublevel Stoping is a less costly mining method compared to Cut & Fill mining and therefore the cut-off value is likely to be lower and the economic mineral resource could be much larger than the resource described by the existing model.

By considering the resource model that is interpreted on a higher raw cut-off value than cut-off applicable to the sublevel stoping method, all of the mineralization is part of the economic zone and becomes mineable. Application of the stope optimization software is in that case not useful because it cannot follow the irregular resource model shape (Figure 57) and manual designs would under such circumstances always be more accurate.

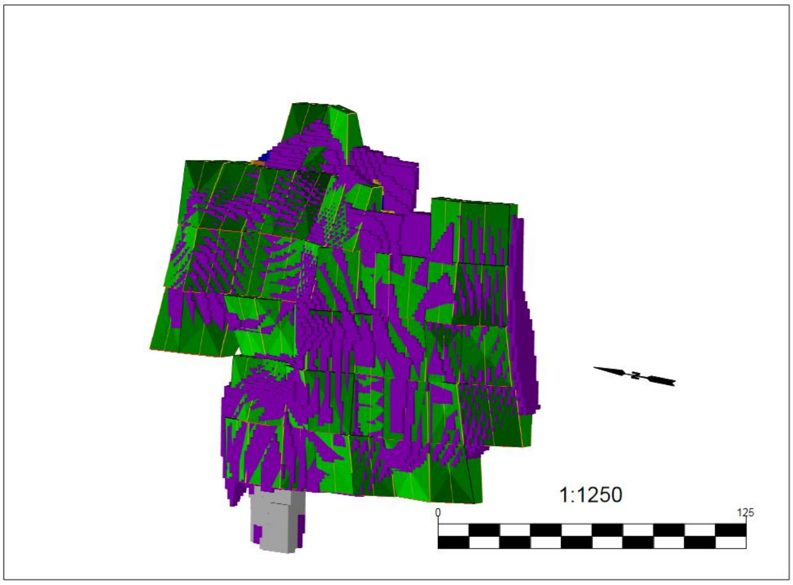


Figure 57 - Problematic stope optimizations as a result of the cut-off based resource interpretation.

4.2.4 CONCLUSIONS

Based on the encountered problems whilst optimizing stope shapes for the Kankberg underground mine the following was concluded;

Virtually all the ore included in the resource model has an NSR value of 525 SEK/t or more. This means that all the ore is economical to mine as long as development costs are off set.

Under such circumstances manual design is always preferred as it gives more flexibility in terms of following the 'true interpreted' ore zone boundaries. The stope optimization software package optimizes stopes by moving the corner points of the stope but cannot truly follow the ore zone boundary (i.e. it cannot adapt the stope shape to the wireframe defining the true ore zone boundary).

The stope optimization software should be used as a strategic planning tool to find the economic zone and design optimized stopes in an unconstraint (in terms of cut-off value in raw drill holes) resource model. By evaluating stope designs at a range of cut-off grades and performing subsequent mine design and economic evaluation it is possible to find the optimal mine design that will maximize project profitability.

Because the Kankberg model turned out not to be suitable for stope optimization and the preparation of a new model would take too much time, it was decided not to continue with the Kankberg underground study.

A suitable model was found to be the Älgträsk resource model and this model was used for further research.

4.3 CASE 2: THE ÄLGTRÄSK UNDERGROUND STUDY

4.3.1 INTRODUCTION

The aim of this case study is to prove the concept of underground mine optimization as proposed in section 2.4. By using the newly availble stope optimization software, production areas (stopes) are optimized at multiple cut-off grades and according infrastructure is designed to connect the production areas and allow for life-of-mine scheduling. A Technical Economic Model (TEM) will be created and used to analyze all mine scenario's to find the optimal project strategy as to maximize project NPV. All steps followed to find the optimal strategy are presented in this case study.

4.3.2 GENERAL LOCALITY

The Älgträsk gold project is located near the town of Älgträsk in the Skellefte mining field circa 30km northwest of the town of Boliden (Figure 58). Both the company head office and processing facilities are located in this town. The area has a long history of mining activity and infrastructure is well developed within the area (roads, railways, airport). The Älgträsk project is located at circa 60km from the city of Skelleftea and with the presence of several smaller towns in the area a local workforce is available.

Figure 58 - Map showing the Älgträsk project location.

The Älgträsk gold project consists of two separate mining concessions that are fully owned by Boliden (Figure 59). Concessions Älgträsk K 1 and K 2 are overlaying the Liden and Nyhem ore zones. Although the mining concessions are currently not connected, Boliden holds the exploration licenses for the areas surrounding the concessions.

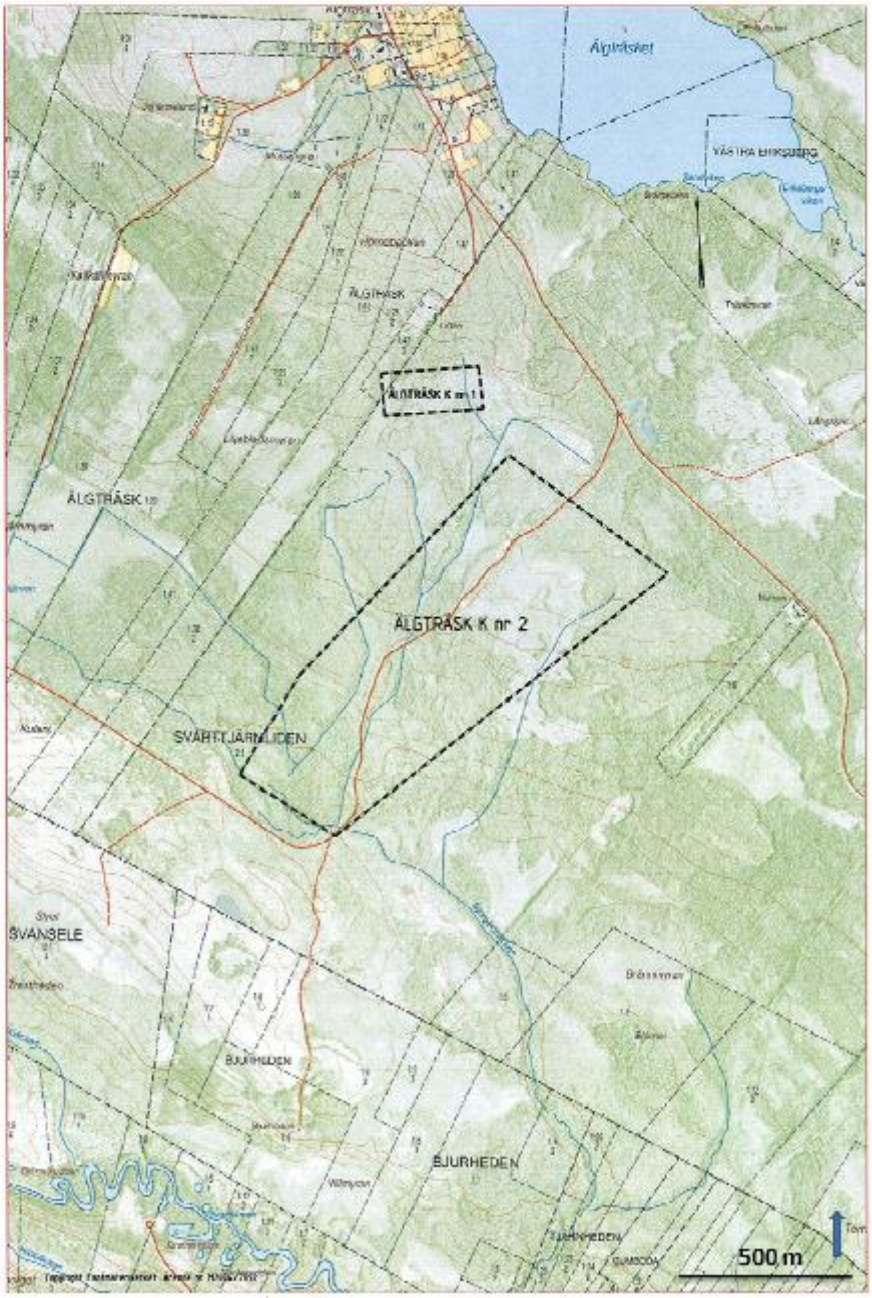


Figure 59 - The Älgträsk K1 and K2 mining concessions.

The Älgträsk project area is located in the south of the Jorn Granitoid Complex and has been explored by Boliden since the 1960's. Gold was first discovered in the mid 80's when till samples were found containing high concentrations of gold. A diamond drilling campaign started in 1987 and reverse circulation (RC) drilling followed together with trenching and geophysics (SRK Consulting – 2010).

4.3.3 GEOLOGY

The Skellefte mining field covers circa 3600 square kilometers and is an important mining area within Sweden. The area contains both pyritic polymetallic volcanic massive sulphide (VMS) deposits as well as vein type gold deposits and low grade porphyry Cu-Au-Mo deposits.

4.3.3.1 REGIONAL GEOLOGY

The Skellefte district is located in the northern part of the Fennoscandian shield and was developed during the early Preterozoic. It is interpreted as the remainings of an ancient volcanic arc behind a northward dipping subduction zone.

Directly north of the VMS deposits within the Skellefte district lies the Jorn Granitoid Complex (JGC). It is interpreted as a synvolcanic and early orogenic intrusion and is generally divided into 3-4 generations based on geochemistry and age (Bejgarn, 2009). The G1 unit of the JGC (ca. 1.88 Ga), in which the Älgträsk deposit is located, hosts several different styles of mineralization:

Tallberg & Granberg: Porphyry Cu-Au-Mo deposit hosted by a tonalite type rock

Algliden: A steeply dipping NE-striking ultramafic-mafic dyke. This dyke, with a length and thickness of circa 3km x 50m, contains disseminated magnetite, phyrrotite, chalcopyrite, pentlandite, minor pyrite and gold.

Nasberg gabbro: Magnetite veins crosscutting the gabbro that were historically mined for iron in the 19th century.

Älgträsk: The only discovered deposit so far that only contains gold. It is located at circa 3km from the Tallberg deposit but is mainly hosted by a coarse grained quartz-porphyritic granodiorite that is not genetically related to the Tallberg tonalite.

4.3.3.2 SITE GEOLOGY

The Älgträsk intrusive gold mineralization consists of several ore lenses that are dipping at an angle of circa 63 degrees in a NW- direction (Figure 60). The mineralization consists of several orezones varying in width and with disseminations and veins of pyrite, locally enriched in chalcopyrite, sphalerite, arsenopyrite, accessory Te-minerals and Gold (Bejgarn, 2009). The gold mineralization is hosted within a coarse grained, quartz porphyritic granodiorite.

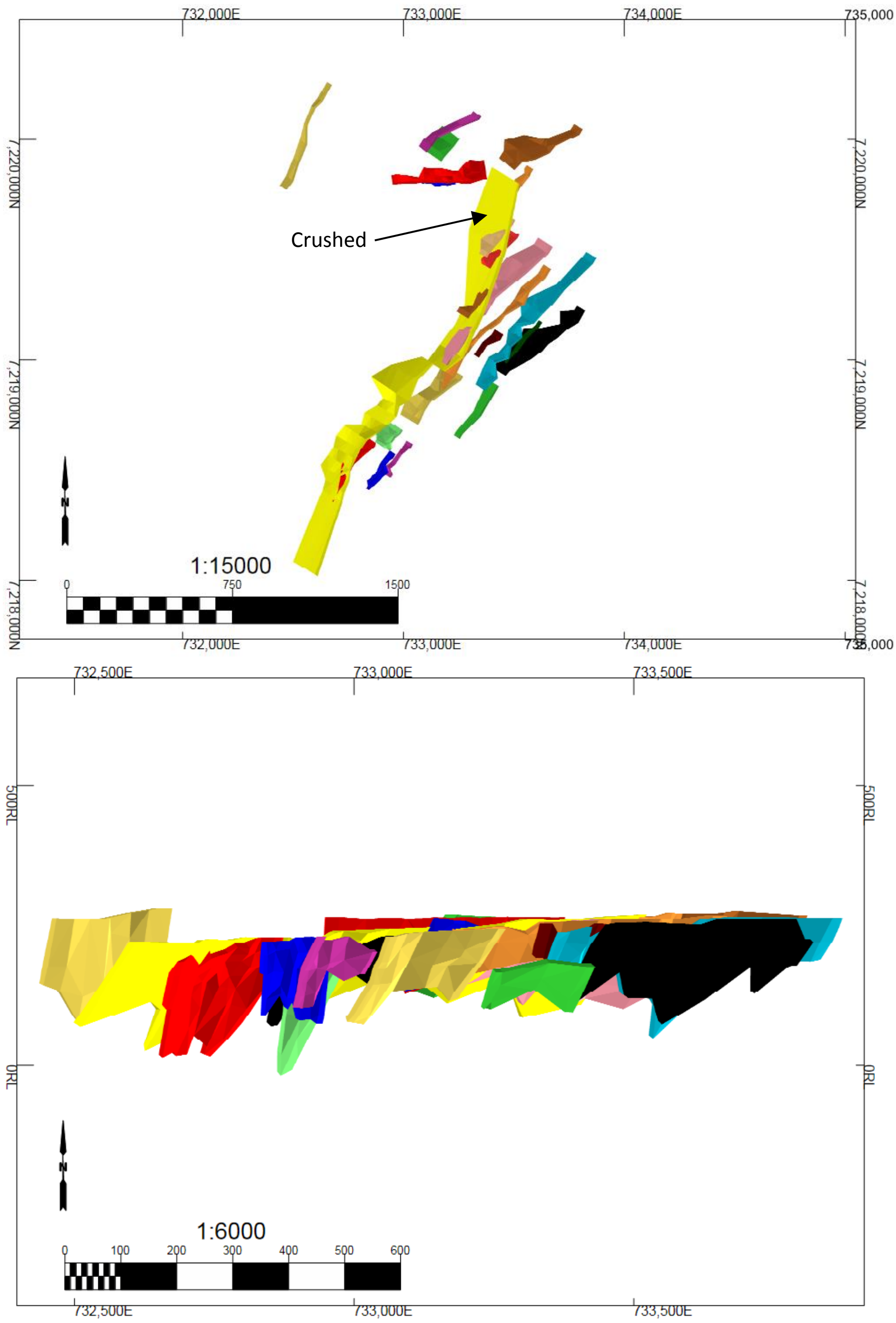


Figure 60 - Nyhem and Liden concessions showing the individual ore zones and crushed zone (yellow)

4.3.4 RESOURCE ESTIMATE

A resource estimate for the Älgträsk project is being updated on a regular basis. The latest resource estimate dates from the 31st of December 2012 and is summarized in Table 12.

Resource Classification	*Cut-off Au (g/t)*	*Tonnage (Mt)*	*Average Grade Au (g/t)*
Indicated + Inferred	0.4	24.4	1.28

Table 12 - Älgträsk Resource Estimate december 2012.

In the process of creating the resource model the assumption has been made that the minimum mining width will be 10m. The consequence of this assumption is that ore sections with widths of less than 10 metres are (regardless of gold grade) removed from the model and hence there is a potential larger mineral resource when a geological interpretation is done based on a minimum mining width of 5 metres (which is reasonable for a small scale underground stoping operation.

4.3.4.1 AREA OF INTEREST RESOURCE ESTIMATE

A conceptual study for open pit mining of the Älgträsk orebody was carried out by SRK in 2010 with an update (based on updated price assumptions) created in 2012. Due to the high strip ratio (8.63:1 at a gold price assumption of 1350 USD/oz (Bradley, 2012)) it was decided to perform a conceptual study for underground mining of part of the deposit containing the highest grade (Figure 61). Three ore lenses are situated within the area of interest and the ore zones extend over a vertical distance of circa 250 metre by a strike distance of circa 450 metre. A resource estimate for the area of interest is given in Table 13.

Ore lens	Tonnage (Mt)	Au grade (g/t)	Ag grade (g/t)	As (%)
1	1.85	1.89	3.36	0.26
2	0.56	2.00	2.44	0.16
3	0.63	2.29	2.47	0.08
Total	**3.01**	**1.99**	**3.00**	**0.21**

Table 13 - Area of Interest Resource Estimate (December 2012).

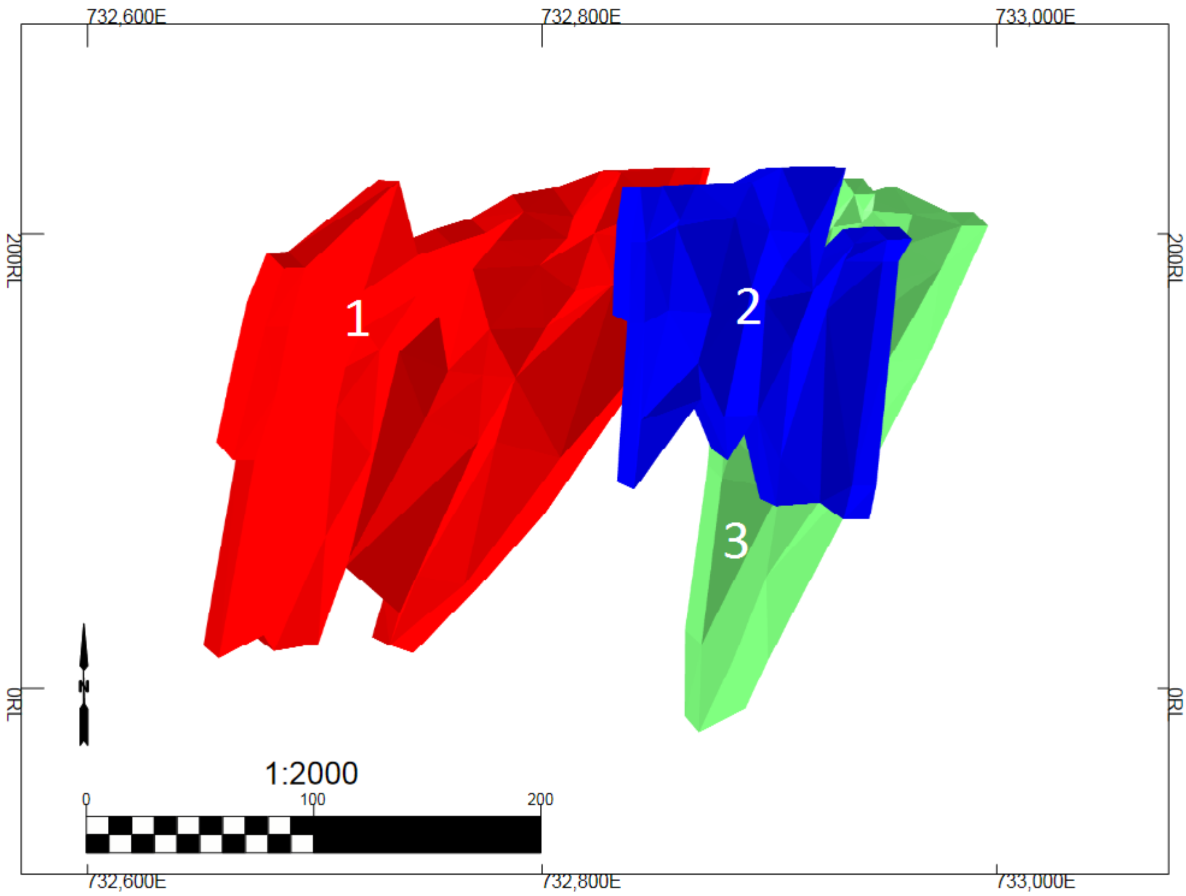


Figure 61 - Älgträsk Area of interest.

4.3.5 GEOTECHNICAL SETTING

As the Älgträsk gold project is still in a conceptual investigation stage and therefore geotechnical information is scarce. No oriented geotechnical drilling has been commenced so all structural data is obtained from ordinary drill core and estimations based on corebox photographs.

4.3.5.1 CRUSHED ZONE

A crushed zone is running through the Nyhem ore zone and is shown in Figure 60. This crushed zone varies in thickness from 0-15 metre and was found to be water bearing. Since the rock within the crushed zone is completely disintegrated the RQD=0. A sample of a core section intersecting the crushed zone is shown in Figure 62. For as long as no further detailed information on the crushed zone is available, an underground opening should not be constructed within 6 meters from the crushed zone (Nystrom, 2013).

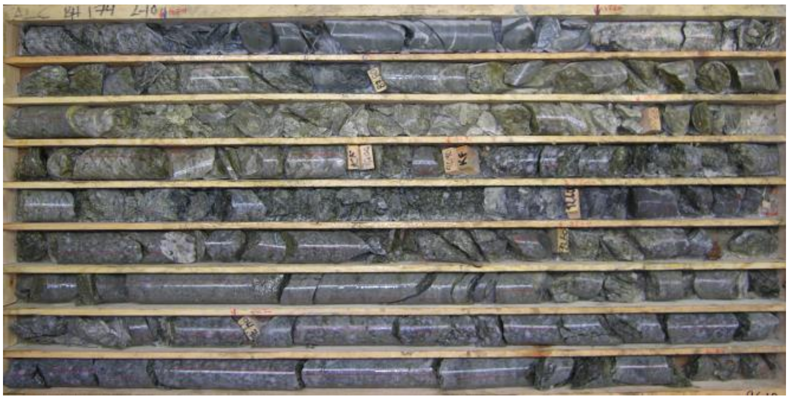


Figure 62 - Corebox with drill core intersecting the crushed zone

4.3.5.2 STRESS REGIME

No stress measurements have been carried out on-site to date. To get an indication of the underground stress regime an empirical formula was used (Amadei & Stephansson, 1997). Linear regression of the maximum and minimum horizontal stress measurements in Scandinavia on both overcoring and hydraulic fracturing measurements resulted in the following rough relations:

Overcoring	Hydraulic Fracturing
$\sigma_H = 10{,}4 + 0{,}0446z$ $\sigma_h = 5 + 0{,}0286z$ $\sigma_v = \rho g z$	$\sigma_H = 2{,}8 + 0{,}0399z$ $\sigma_h = 2{,}2 + 0{,}0240z$ $\sigma_v = \rho g z$
At 500m depth: σH = 33 MPa σh = 19 MPa σv = 13 MPa	At 500m depth: σH = 23 MPa σh = 14 MPa σv = 13 MPa

Table 14 - Stress field estimations in Scandinavia (after Amadei & Sthephansson, 1997)

It should be noted that these relations are indicative and no final mine design should be based on these figures.

4.3.5.3 ROCK MECHANICAL DATA

Initial rock mechanical investigations revealed the following rock mechanical parameters to be used for preliminary geotechnical design (Nystrom, 2013):

Rock Type: Tonalite and Granodiorite (Hard Rock)

Rock Strength Estimate: ~ 200MPa or higher

RMR Estimate: ~ 70

Rock Quality: Good

Fracture frequency (BRQD):

	Hanging wall	Ore zone	Footwall
BRQD	Average: 60-70% Range: 0-80% *	Average: 70-80% Range: 0-90% *	Average: 60-70% Range: 0-80% *

* BRQD = 0% within Crushed Zone

RQD: Positive Estimate: 70%

Negative Estimate: 40%

Joint Set number (Jn): Positive Estimate: 2, One Joint Set

Negative Estimate: 6, Two Joint Sets + random

Joint Roughness (Jr): Positive Estimate: 4, discontinuous joints

Negative Estimate: 3, Rough and irregular, undulating

Joint Alteration number (Ja): 1.0, unaltered joint walls, surface staining only

4.3.6 MINING METHOD SELECTION

The selection of a mining method largely depends on rock conditions, orebody geometry as well as mineral economics. A well-known mining method selection tool was developed by Nicholas in the 1980's. The UBC mining method selection tool was developed by Miller et al. (Miller et al., 1995) and is a modified version of the Nicholas selection method that emphasizes more on stoping methods. Due to the nature of the Älgträsk orebody the UBC method is most suitable to determine (a) feasible mining method(s).

The following selection criteria were applied for the Älgträsk orezone:

GEOMETRY AND GRADE

Orebody Shape: Platy/Tabular

Orebody Thickness: Intermediate (10-30m)

Orebody Plunge: Steep

Orebody Depth: <100m, 100-600m

Ore Grade: Low

ROCK MASS RATING

RMR Hanging Wall: Strong (60-80)

RMR Orezone: Moderate (40-60)

RMR Footwall: Strong (60-80)

ROCK SUBSTANCE STRENGTH (UCS/PRINCIPAL STRESS)

RSS Hanging Wall: Moderate (10-15)

RSS Ore Zone: Weak (5-10)

RSS Footwall: Moderate (10-15)

For each mining method included in the UBC mining method selection tool, the suitability of each of these properties is evaluated and rated numerically. The ratings for each property are added up and highest score ranks the methods. For the Älgträsk area of interest the ranking of the mining methods is presented in Table 15.

Depth	*<100m*		*100-600m*	
Ranking	**Method**	**Points**	**Method**	**Points**
1	Sublevel Stoping	38	Sublevel Stoping	39
2	Open Pit	35	Cut and Fill	32
3	Cut and Fill	31	Open Pit	31
4	Shrinkage Stoping	31	Shrinkage Stoping	31
5	Sublevel Caving	29	Sublevel Caving	28
6	Block Caving	23	Block Caving	24
7	Top Slicing	17	Top Slicing	16
8	Square Set	11	Square Set	11
9	Longwall	-25	Longwall	-25
10	Room and Pillar	-27	Room and Pillar	-27

Table 15 – UBC Mining Method Selection Tool – Results for the Älgträsk Orebody.

From Table 15 it is observed that sublevel stoping is rated as the best mining method up to 600m of depth followed by cut & fill mining. At shallow depths, open pit mining becomes an option as well.

When relating the outcome of the UBC mining method selection tool back to the site geology it is concluded that the outcome was to be expected. With the orebody running virtually from surface open pit mining is a logical approach for mining. However, due to the limited thickness of the orebody, strip ratios can increase to uneconomic numbers rapidly making underground mining perhaps a more feasible approach. The shape and dip of the orebody makes that sublevel stoping or cut and fill are the most feasible mining methods and with sublevel stoping being the cheaper alternative, it is the preferred method of choice for a low grade deposit.

4.3.7 STOPE STABILITY EVALUATION USING THE MODIFIED STABILITY GRAPH METHOD

Due to the limited availability of geotechnical data for stope size determination the stope size was based on experience from other mining operations, first assumptions and stress field estimates.

Experience from the Garpenberg operation suggests that a 20 metre vertical drill hole length is feasible. With inclusion of a 5 metre high ore drive this means that from a production point of view, 25 metre is a workable sublevel interval.

A stope width of 15 metre was chosen based on the drill hole diameter vs. drill hole length and stope size rough relationship as provided by Snowden (Snowden Consultants, 2004) and presented in Table 16.

Hole Diameter (mm)	Max. Hole Length (m)	Min Stope Width (m)	Max Stope Width (m)
57	15	1.5	6
64	20	6	15
76	25	8	20
89	30	12	25
102	40	15	30
114	45	20	40
152	50	20	40

Table 16 - Hole Diameter vs. Hole Length and Stope Size.

By combining a sublevel interval of 25 metre with a stope width of 15 meters it is possible to use 64 or 76mm drill holes. The use of smaller sized drill holes means that the amount of explosives contained per metre of hole is kept at a low level which is especially useful in small geometry stopes as excessive dilution is minimized by blasting smaller charges.

The selected stope size was evaluated for stability using the Modified Stability Graph method using the stress field estimates and rock mechanical estimates as provided by the Geotechnical Department. A positive as well as a negative scenario was evaluated based on RQD and joint set properties (see 4.3.5.3 Rock Mechanical data) and the results are presented in Table 17 and Figure 63.

Scenario	Surface	RQD	Jn	Jr	Ja	Q´	A	B	C	N´	Area	Perimeter	HR
Positive	HWFW mid stope horizontal	70	2	4	1	140	0.92	0.2	5	129	375	80	4.7
Positive	HWFW mid stope vertical	70	2	4	1	140	1	0.2	5	140	375	80	4.7
Positive	End wall	70	2	4	1	140	0.56	0.2	5	78	250	70	3.6
Positive	Crown	70	2	4	1	140	0.16	0.2	5	22	150	50	3.0
Negative	HWFW mid stope horizontal	40	6	3	1	20	0.92	0.2	5	18	375	80	5
Negative	HWFW mid stope vertical	40	6	3	1	20	1	0.2	5	20	375	80	5
Negative	End wall	40	6	3	1	20	0.56	0.2	5	11	250	70	3.6
Negative	Crown	40	6	3	1	20	0.16	0.2	5	3	150	50	3.0

Table 17 - Modified Stability Graph stope size analysis.

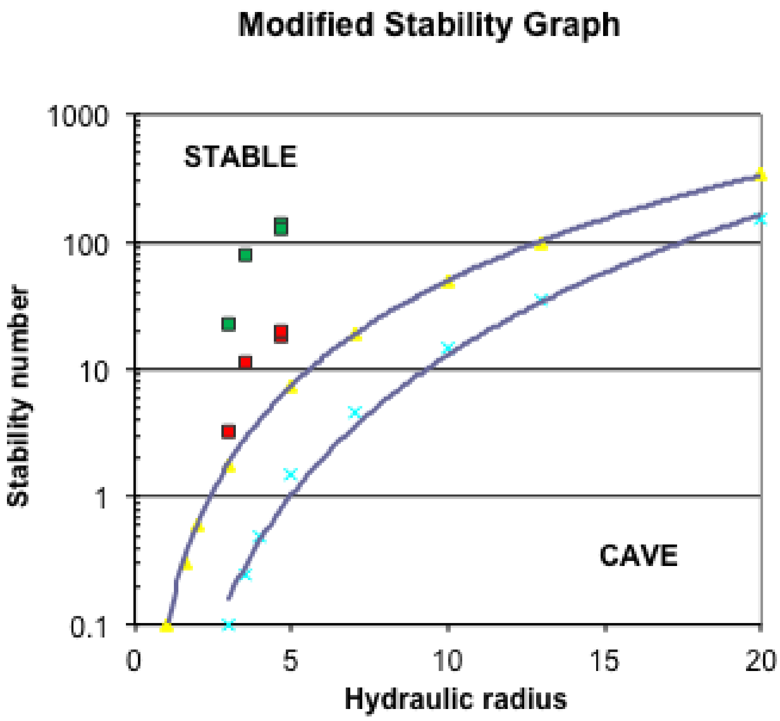


Figure 63 - Modified Stability Graph stope stability plot (Green = positive, Red = negative scenario).

It is concluded that both the positive and negative scenario stability analysis plot within the stable without support zone of the Modified Stability Graph. In this stage of the project the stope width and height as discussed are assumed reasonable and used in further design and economic evaluation. Due to the fact that the negative scenario still plots within the stable without support area of the stability graph it is well possible that the stope size turns out to be too conservative in a subsequent stage of project evaluation. However, the goal of this scoping study is to evaluate the possibility of underground mining. If mining turns out to be economic when using a 15 metre stope size, it certainly will be when using larger stope sizes. The possibility of using larger stopes has to be investigated once more detailed geotechnical data comes available.

4.3.8 MINERAL PROCESSING METHODS

Due to the fact that the Älgträsk project area is located in the Boliden area, all ore is transported by road to Boliden where a full processing facility with spare capacity is available. It is common practice within Boliden to transport raw ore from mines in the area to the centralized processing facility in the village of Boliden.

Metallurgical testwork has been carried out by Boliden to identify the best processing method for the Älgträsk gold ore. Gravimetric, flotation and cyanide leaching tests have been carried out on two sets of samples (Bolin, 2009). One set of samples was taken from mineralization with low arsenic whilst the second set of samples was taken from zones with high arsenic content. High As ore is considered to contain > 0.3% As in the raw material feed.

The following processing routes were selected for investigation by Boliden and SRK:

1. *Gravity and Flotation of both high As and Low As ore types*
2. *Gravity and Leaching of both High and Low As ore types*
3. *Gravity and Flotation of Low As ore and Gravity and Leaching for High As ore*
4. *Gravity and Flotation of High As ore and Gravity and Leaching of Low As ore.*

For the purpose of this scoping study it is assumed that the high As containing ore stream and the low As containing ore stream can be kept separate during mining and processing. This is considered to be an acceptable assumption due to the fact that As is concentrated in certain areas and it should therefore be possible to send the ore to the right process if grade control is carried out during extraction.

The individual concentrate products that result from each separate processing method are kept separate when transported to the smelter. However, if both high As ore and low As ore are concentrated by the same process (for example processing route 1), the products are combined and no separate refining of the end product takes place.

All 4 processing routes are included in the Technical Economic Model (TEM).

The process recoveries from gravity and flotation as well as leaching are shown in Table 18.

Gravity and Flotation				
High As ore	Recovery	Recovery	Recovery	Recovery
	Au	**Ag**	**Cu**	**As**
Gravity	40.0%	15.0%	0.8%	9.6%
Flotation	30.0%	30.0%	40.0%	10.7%
Total	**70.0%**	**45.0%**	**40.8%**	**20.3%**
Low As ore				
	Au	**Ag**	**Cu**	**As**
Gravity	40.0%	15.0%	0.8%	16.0%
Flotation	50.0%	50.0%	40.0%	5.0%
Total	**90.0%**	**65.0%**	**40.8%**	**21.0%**
Gravity and Leaching				
High As ore	Recovery	Recovery	Recovery	Recovery
	Au	**Ag**	**Cu**	**As**
Gravity	40.0%	15.0%	0.8%	9.6%
Leaching	45.0%	65.0%	0.0%	0.0%
Total	**85.0%**	**80.0%**	**0.8%**	**9.6%**
Low As ore				
	Au	**Ag**	**Cu**	**As**
Gravity	40.0%	15.0%	0.8%	16.0%
Leaching	53.0%	60.0%	0.0%	0.0%
Total	**93.0%**	**75.0%**	**0.8%**	**16.0%**

Table 18 - Process Recovery Factors for both Gravity & Flotation and Gravity & Leaching.

Test results from the concentration tests carried out by the Boliden Mineral Processing department indicate that the following gold grades can be assumed in the concentrate products:

Gravity Concentrate: 150 g/t

Flotation (Low As Feed): 150 g/t

Flotation (High As Feed): 67.5 g/t

4.3.9 TECHNICAL ECONOMIC MODEL (TEM)

A technical economic model (TEM) was constructed in Microsoft Excel in order to create a cash flow model for each to be evaluated mining scenario. Besides the mine and processing recovery factors, all costs related to the mining operation (including capital expenditures, operating costs and refining costs) have to be included in the model. This chapter will briefly discuss all costs included in the TEM and their source or reasoning behind the number. An example of the constructed TEM can be found in appendix B.1.

4.3.9.1 MINE CAPEX

Mine capital expenditures for the Älgträsk project are based on the Kankberg mine feasibility study of 2011 (Boliden Mineral AB, 2011) after discussions with senior management. A 15 percent contingency was factored in to account for potentially increased costs and/or extra expenditures once more detailed studies are completed. A summary of the mine capital expenditures is provided in Table 19.

Description	**Total Cost (MSEK)**
Power Distribution, Control and Radio System	40
Fresh Air Heating Plant	11
Ventilation Fans	3
Ventilation Shaft	4.6
Explosives and Detonator Storage facility	5
Mine Water Pumping System	4.5
Water Supply System	3
Water Treatment System	7
Change House and Mine Office	8
Unspecified Small Investments	4
Reclamation	0.3
Mobilization Cost for Contractor	2
Subtotal Mine CAPEX	92.4
15% Contingency	13.86
Total Mine CAPEX	**106.26**

Table 19 - Älgträsk Mine Capital Expenditures (after Kankberg mine feasibility study).

4.3.9.2 MINING OPEX

As the project is currently in a scoping study phase, no actual cost estimation has been carried out to date. A reasonable mining cost was estimated by the author to evaluate the Älgträsk Project. In order to come up with an estimate on mining costs several similar projects both within and outside the company were investigated. The results are presented in Table 20.

Project Name	Company	Method	Mining (SEK/t)	Backfill (SEK/t)	Total cost (SEK/t)	Production (kt/y)
Boliden Garpenberg - 2012	Boliden	SLOS - Pastefill	124	80	204	1484
Faboliden Gold Mine - 2012	Lappland Goldminers	SLOS - Pastefill	213	30	243	1400
Kittlia Mine – Agnico Eagle	Lappland - Finland	SLOS - Pastefill	Unknown	Unknown	226.55	255.5
Bjorkdal Gold Mine - 2012	Gold-Ore Resources Ltd.	SLOS - No Backfill	192.87	-	192.87	500

Table 20 - Mining and Backfill Costs.

From table 4 it can be observed that mining costs in sublevel longhole open stoping of similar type and size are in the range of 200 SEK/t. The Garpenberg operation shows a significantly lower mining cost at 124 SEK/t; this probably has to do with the fact that the Garpenberg orebody is a massive orebody, which reduces the average mining costs per tonne of ore. The operation that shows most similarity with the Älgträsk project in terms of orebody geometry and depth is the Bjorkdal Gold Mine that produces at a mining cost of 192.87 SEK/t (2011-2012 actuals). This is based on contractor mining at a rate of 500kt per year. Another very similar type operation is the Kittila Mine located in northern Finland. At a production rate of 255.5 Kt per annum the combined mining and backfill costs add up to 226.55 SEK/t.

As a conservative base case estimate a mining cost of 250 SEK/t (Mining & Backfilling) will be used for economic evaluation.

4.3.9.3 PROCESSING OPEX

Because the processing plant in Boliden will be used to process the Älgträsk ore, the operating costs are well understood. The numbers used in this study are the same as the ones used in the 2012 SRK Älgträsk Update Memo (Bradley, 2012).

Description	Unit	Total
Gravity or Flotation	SEK/t	98
Leaching	SEK/t	65.13

Table 21 - Processing costs Boliden central processing facility.

4.3.9.4 HAULAGE COSTS

Transportation from the Älgträsk mine to the Boliden processing facility and from the processing facility to the smelter will take place over existing roads. The total distance from mine to mill adds up to 42 km and the distance to the smelter is circa 50 km. Haulage costs are summarized in Table 22 and are taken from the 2012 SRK Älgträsk Update Memo (Bradley, 2012).

Description	Unit	Total
Mine to Mill Haulage	SEK/t	34.50
Mill to Smelter Haulage	SEK/t concentrate	33.00

Table 22 - Haulage Costs Mine to Mill and Mill to Smelter.

4.3.9.5 OTHER COSTS

As a contractor executes all Mining the cost per tonne is assumed to be an all-inclusive cost (e.g. Including equipment and personnel costs). Processing costs include all processing related costs and development costs are assumed to include all costs related to the development of a tunnel section. The one cost not included in any of the other costs is the general and administrative cost, which is essentially an expense for the Boliden technical office that is coordinating the mining operation. This cost is assumed to be the same as in the 2012 SRK study at this stage.

Description	Unit	Total
G & A Costs	SEK/t	13

4.3.9.6 SMELTER CHARGES

Treatment and refining of the produced concentrates takes place at the Ronnskar smelter. This is a Boliden owned smelter, which means that all treatment and refining charges are well documented. All costs are summarized in Table 23.

Gravity or Flotation Concentrate		
Description	**Unit**	**Total**
Concentrate Treatment	USD/ t concentrate	60.00
Payable Gold	%	96.50
Payable Silver	g/t deduction	30.00
Payable Copper	% deduction	1.00
Refining Charge - Gold	%	2.00
Refining Charge - Silver	%	5.00
Refining Charge - Copper	SEK/t Cu	0.00
Penalty Charge - As*	SEK/t concentrate	280.00
* applies if As>0.3% (gravity concentrate) As>0.2% (flotation concentrate)		
Leach Dore		
Description	**Unit**	**Total**
Treatment Charge >15% Gold	SEK/kg dore	225.05
Treatment Charge <15% Gold	SEK/kg dore	258.79
Payable Gold	%	99.70
Payable Silver	%	99.10

Table 23 - Treatment and Refining Charges Ronnskar

4.3.10 OPTIMIZATION APPROACH AND BASE CASE ASSUMPTIONS

The approach taken in the Älgträsk mine optimization is based on the strategic mine planning theory as described in section 2.4.1. The 6 subsequent steps that have to be undertaken are briefly introduced in this section and each individual step is discussed in more detail in individual sub-chapters.

1. A break-even grade calculation will provide the range in which the optimum cut-off grade is likely to be found.

2. The AMIRA Stope Optimizer will be used to design feasible stopes for each cut-off grade. Two stope optimization steps have to be executed
    a. A framework optimization will determine the optimum location of stopes as to maximize revenue.
    b. With the optimum location of stopes known, a second pass on the block model will provide more feasible stope designs that are aligned properly both horizontally and vertically (application of the so called 'stope smoothing').

3. Basic infrastructure (decline, level development, ore drives) will be designed manually for each set of stopes.

4. A scheduling package (Deswik.Scheduler) will be used for life of mine scheduling (long term planning) as to find a feasible mining sequence and production rate for the mine. The scheduling package will be used to evaluate:
    a. Mine sequencing
    b. Production rate
    c. Development rate

5. The resulting life of mine schedule will be used as an input to create a cash flow model (TEM) of the project.

6. Based on the cash flow model, the project NPV can be calculated.
7. Repeat step 2-6 for the full range of cut-off grades.

By repeating the optimization and design steps 2 – 6 it is possible to create a so-called Hill of Value graph. This graph presents all separately evaluated scenarios in a mountain or hill shaped graph in which the optimal strategy is easily found by searching for the 'top of the hill'. An example of a simple Hill of Value is shown in Figure 64 (Alford & Hall, 2009).

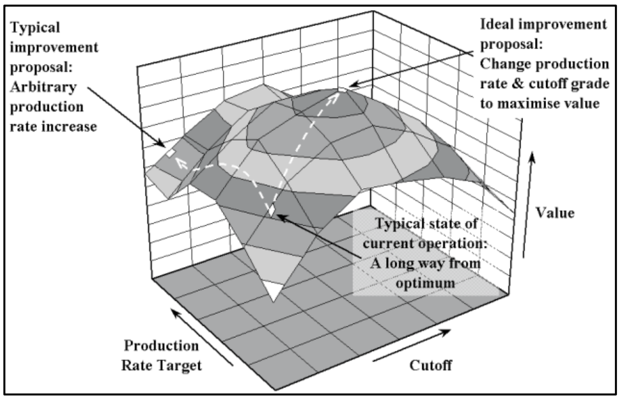


Figure 64 - Hill of Value example (Alford & Hall, 2009).

4.3.10.1 BASE CASE ASSUMPTIONS

1. Mining will progress from the upper to lower levels (descending). This will assure early production to offset capital and development costs.
2. A safety crown pillar of ca. 25m is left in place. This coincides with the upper (first) level of stopes. They are excluded from the mine plan at this stage as it remains to be seen whether or not they can be mined by a caving method.
3. All capital expenditures are done in the year prior to mining (Year -1).

4.3.11 BREAK-EVEN GRADE ESTIMATION

The break even grade is the minimum metal grade that a tonne of rock must contain in order to pay for its own mining, processing and refining. Only direct variable costs related to the mining, processing, and refining of the material are taken into account when calculating the break-even grade. A break-even grade calculation was prepared based on the cost factors described in the previous sections (Table 24). As not all costs can be calculated to full detail (eg. Transportation costs, Smelter charges per tonne of concentrate, As content, etc.), the calculated cut-off grade is an estimate of the range in which the real optimal cut-off grade is to be found.

Parameter	**Unit**	**Total**
Revenues		
Gold price (1200 USD/ oz)	SEK/g	289.39
Process Recovery	%	88.00
Payable Metal	%	98.10
Selling price		**249.82**
Refining Charge (2% of selling price)	SEK/g	5.00
Selling Royalty (0.2% of selling price)	SEK/g	0.50
Recovered Revenue	SEK/g	**244.33**
Costs		
Mining Costs	SEK/t	250
Overhead Costs	SEK/t	47.5
Processing Costs	SEK/t	105
Total Costs	SEK/t	**402.5**
Cut-off Grades		
Excluding Dilution		
Cut-off grade	**g/t**	**1.6**
Including Dilution		
Dilution	%	15
Dilution Au grade	g/t	0
Cutoff grade (incl. dilution)	**g/t**	**1.94**

Table 24 - Break-even grade estimation for Älgträsk Area of interest.

The estimated gold cut-off grade at a gold price of 1200 USD/oz (=289.39 SEK/g) is 1.6 g/t. With a dilution percentage of 15% at a grade of 0 g/t Au this means that the stope head-grade should be at least 1.94 g/t to cover mining, processing and refining costs. To cover the complete likely range of cut-off grades, stopes and infrastructure will be designed for 1.5 – 1.6 – 1.7 – 1.8 – 1.9 – 2.0 g/t Au. After scheduling of each design it should be possible to find the optimum cut-off grade for this specific deposit.

4.3.12 STOPE FRAMEWORK OPTIMIZATION

The first step in stope optimization using the optimizer is to find the best location for the stopes (the stope framework). The selection of a feasible stope size was discussed in section 4.3.7 and resulted in a stope width and sublevel interval of 15 and 25 metre respectively. These numbers are entered into the stope optimizer's optimization setup menu.

Once the sublevel interval and stope width are fixed (X- and Z-) the stope optimizer will evaluate the orebody in the Y direction in order to find the optimized stopes.

As the optimization technique is based on evaluating stope locations based on a regular grid it is not difficult to imagine that the location for a stope can be optimised by moving the optimization framework (grid) around. An example in which the framework origin is shifted upwards and to the left is visualized in Figure 65.

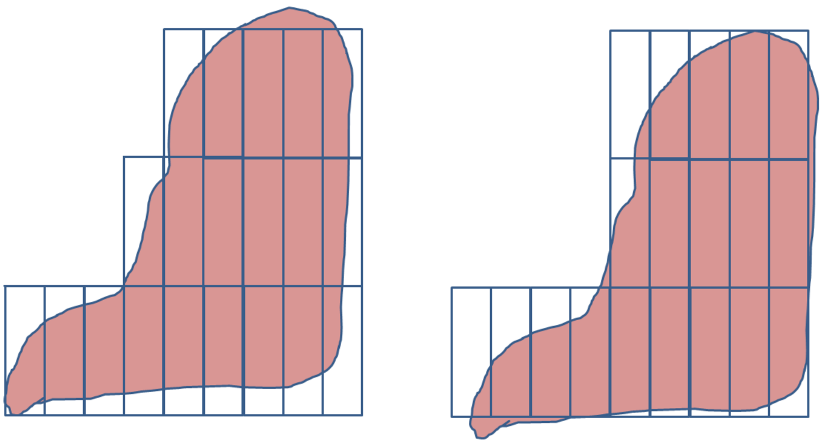


Figure 65 - Impression of the framework optimization step showing the framework origin (left) and (more) optimized displaced framework (right).

The framework optimization was carried out for all cut-off grades in the range 1.5 -2.0 g/t. The step distance in both the X- and Z-direction was set to 5 meters and at a stope width and height of 15 x 25 meters. This means that a total of 15 (3 x 5) framework origins were evaluated. The step distance of 5 meters was chosen to limit the computation time as the stope optimizer uses a brute force approach to find the optimal framework origin (the optimization process will be repeated 15 times in this case).

Example outputs of several cut-off grade evaluations in the range 1.5-2.0 are presented in Table 25. It was observed that the framework origin of evaluation case 9 (x-origin:5, z-origin:15) resulted in the highest total metal above the selected cut-off in all cases. Therefore this framework will be used to evaluate all cut-off grades in the range 1.5-2.0.

Cut-off grade	Optimum framework origin case #	# of stopes	Tonnage (t)	Average Au grade (g/t)
1.5	9	136	1,340,821	2.76
1.8	9	119	1,091,026	3.00
2.0	9	110	955,493	3.16

Table 25 - Example Framework Optimization Outputs (Tonnes and Grades).

An example framework optimization output is presented in Figure 66.

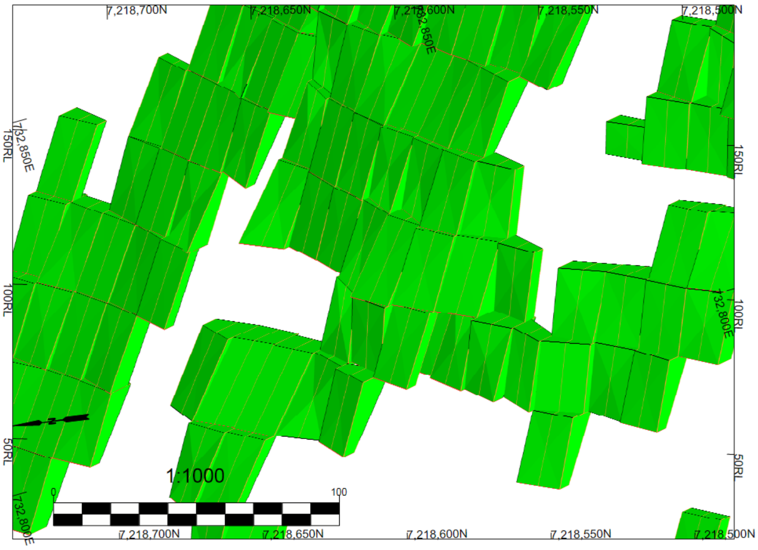


Figure 66 - Framework Optimization example (best scenario @ 1.8 g/t cut-off).

The stopes in this example are all optimized individually but do not represent a minable situation (the stopes should be lined up better in order to represent feasible mining conditions). The next step in the optimization process is to run a second optimization scenario within the optimized framework coordinates as found in the previous step. The only difference being this time the stope smoothing algorithm will be included in the process as to line up the stopes better and get a more realistic tonnage and grade indication. Stope smoothing cannot be combined with the framework optimization process due to the large overhead time involved in stope smoothing which would in that case have to be carried out for each evaluated framework. The large overhead times are caused by the fact that stope smoothing can involve upto 9 individual stopes when trying to smooth (the stope + all surrounding stopes) and it will not compromise to the boundary conditions as set by the user (e.g. Maximum internal waste percentage, cut-off grade, etc.).

4.3.13 STOPE OPTIMIZATION AND SMOOTHING

The same set of stopes as shown in Figure 66 is again presented in Figure 67. One can clearly observe that the individual stope shapes have changed in order to better line up with the surrounding stopes.

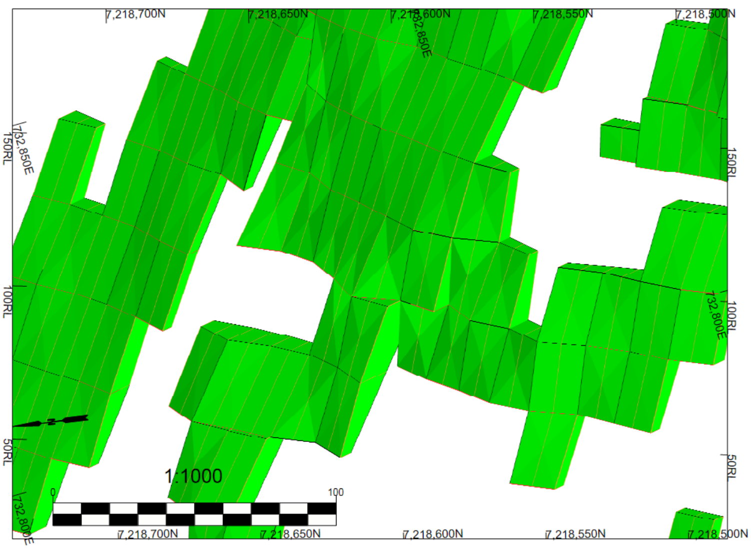


Figure 67 - Optimized Stopes after 'stope smoothing' (best scenario @ 1.8 g/t cut-off).

Although boundary conditions are not violated during the stope smoothing process, tonnages and grades change slightly when stopes increase or decrease in size. The resulting tonnages and grades are presented in Table 26. At a cut-off grade design of 1.8 g/t a small increase in stope tonnage and a decrease in average gold grade is observed. This is a to be expected result of the smoothing process and understandable when comparing the figures 51 and 52. No extreme differences are observed between the optimization results in Table 26 and Table 25 and therefore the smoothened stopes are assumed to be correct.

Cut-off grade	# of stopes	Tonnage (t)	Average Au grade (g/t)	Average Stope Length (m)
1,5	134	1.324.684	2,69	10
1,6	126	1.227.367	2,78	10
1,7	123	1.169.281	2,84	9
1,8	118	1.096.242	2,92	9
1,9	116	1.030.978	2,97	9
2,0	108	937.579	3,07	8

Table 26 - Tonnages and grades of the stopes after stope smoothing.

4.3.14 INFRASTRUCTURE DESIGN

In order to evaluate production rates for the mining project it is necessary to design basic infrastructure to connect and access the optimized stopes. Table 27 summarizes the design criteria for the Decline, Level development and Ore drives as used by Boliden.

Tunnel Type	Size (W x H)	Gradient	Comments
Decline	6 x 5 metre	1:7	Turn radius: 25m
Level development	6 x 5 metre	Max 1:7	
Ore drives	5 x 5.5 metre	0*	* During this stage

Table 27 - Standardized development design criteria as used by Boliden.

In order to reduce costs, a central decline is designed running through the central area between all 3 mining areas (Figure 68). For stability reasons this decline cannot be constructed within 25 meters distance from the ore drives which is assured by the level development. Due to the relatively small stope lengths (see Table 26) the logical direction of stoping is longitudinal which means all ore drives are constructed within the orebody itself, thereby reducing development costs significantly compared to transverse stoping.

No ventilation design was carried out for the level of this study and all costs related to ventilation are assumed to be included in the initial capital expenditures and infrastructure development costs.

The design of infrastructure is unlike the stope design, still a manual trial-and-error process and the presented design is therefore not guaranteed to be the optimal infrastructure design. It is beyond the scope of this project to design optimal infrastructure however, due to the fact that the infrastructure design undergoes very little changes over the range of selected cut-off grades (due to the tabular shape of the ore lenses) it is believed that the resulting mine plan, schedule and cash flow model are a good indication of the capabilities of the stope optimization software to find the optimal cut-off grade based mine plan.

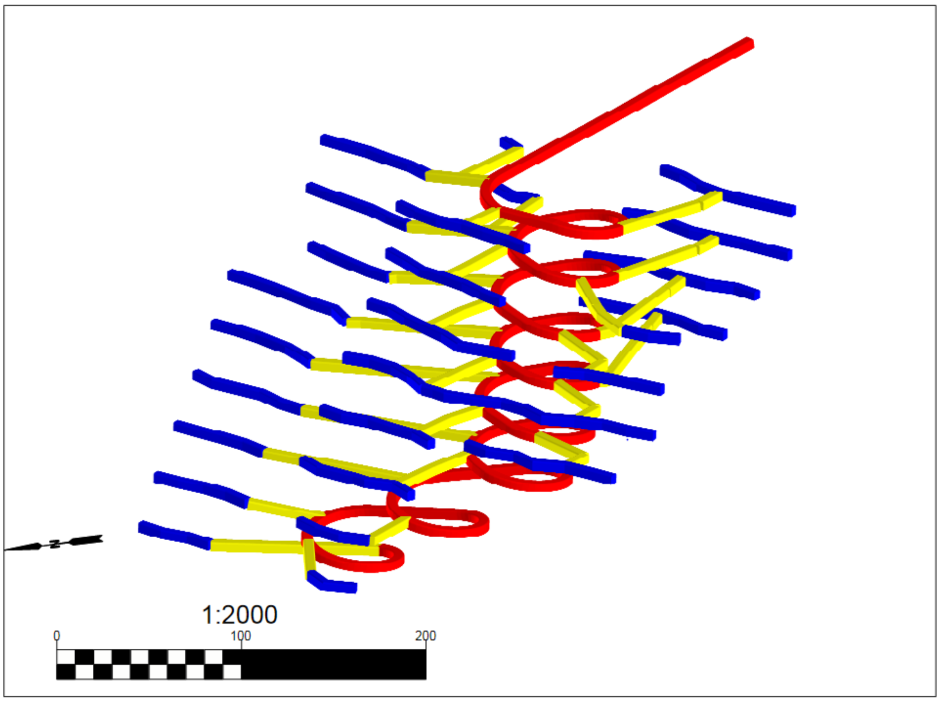


Figure 68 - Infrastructure design (Decline=red, Level development=yellow, Ore drives=blue).

4.3.15 LIFE OF MINE SCHEDULING

With the stopes and required infrastructure designed the next step is to import the stopes and tunnel lines into a scheduling package. Deswik.Scheduler is integrated in the Deswik.CAD package and is therefore the scheduling package of choice. The goal of using the scheduler is to create a feasible mining sequence and analyse production rates. The output from Deswik.Scheduler is a material flow schedule including tonnage and grade figures that can be fed into the TEM to obtain a project NPV.

The scheduling process can be divided into three separate steps, namely;

- Creating task solids
- Dependency Creation
- Task Rates, Resource Rates & Resource Levelling

4.3.15.1 CREATING TASK SOLIDS

Each 'activity' (e.g. decline development, production drilling) was supplied to the scheduling software package. This allows for the creation of dependencies (the interconnectivity between activities). An activity is part of a development or production task (e.g. a section of the decline or production from a stope). At the current stage of this project the following activities were supplied to the scheduling software:

Activity type	Color
Development	
Decline	Red
Level	Yellow
Ore	Blue
Production	
Drilling	-
Stoping	-
Backfilling	-

When all the tunnel centrelines and stope optimizer output shapes are assigned with an activity type, task solids are created automatically and subsequently interrogated against the Älgträsk block model to obtain tonnage and grade information for further scheduling. The resulting task solids for the 1.8 g/t cut-off mine design are presented in Figure 69.

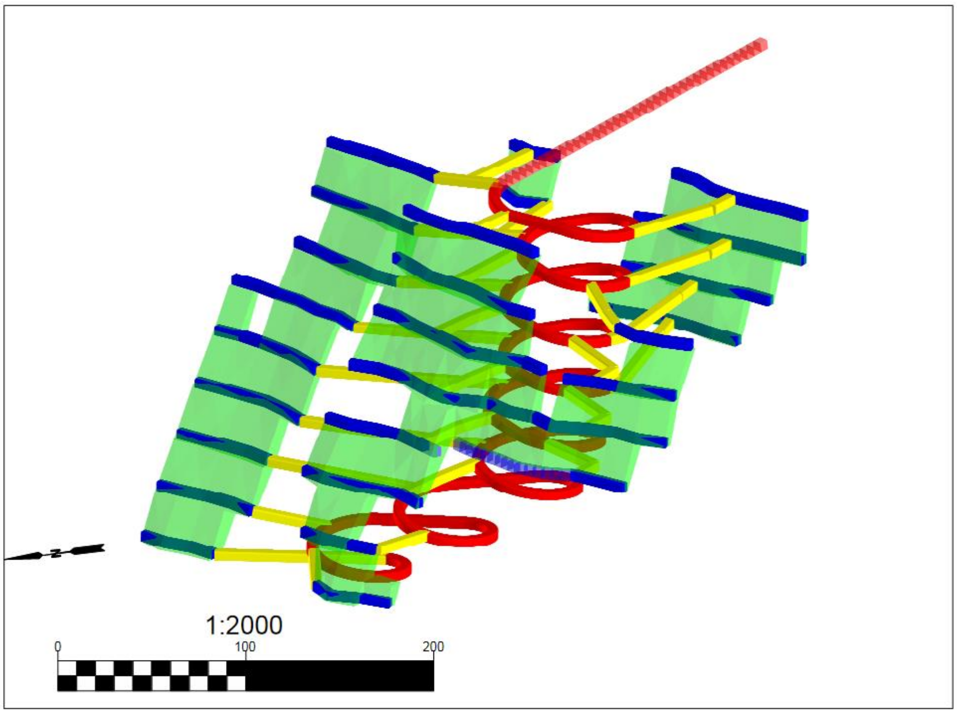


Figure 69 - 1.8 g/t mine design task solids.

4.3.15.2 DEPENDENCY CREATION

Dependencies are links between two task solids that prevent the next task from starting before the previous task is finished. In case of the development tasks this means that a level development task cannot start before the decline has reached the specific level. In the same way, an ore drive task cannot be started before the level development has reached the respective ore drive starting point. Finally, the mining of a stope cannot start before the ore drive for the specific mining area and level is finished. Besides the (more) obvious development dependencies, there also constraints between the stopes that are being mined. Both lateral and vertical constraints exist:

Laterally:

Once a stope is mined out, it is being backfilled and left to cure for 3 weeks (or 21 days). Production in the adjacent stopes can therefore not start before the backfill is fully cured.

Vertically:

The overall mining progression is downwards to offset development cost and capital expenditures.

In the same way as the lateral stope-dependency, mining of a stope on the level below a stope of which the backfill is not yet fully cured is not possible either. The vertical dependencies are further explained based on the visualization in Figure 70.

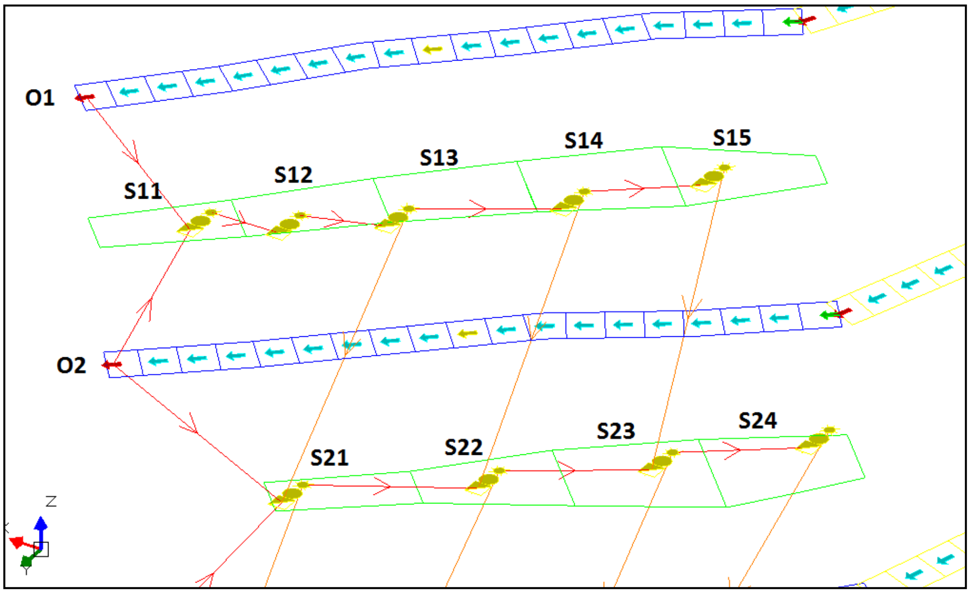


Figure 70 - Lateral & Vertical dependencies (arrows indicating progression direction)

Once the development of ore drives O1 and O2 is finished, the production of stopes on the S1 level can start. The mining of stopes is progressing from S11 towards S15. The production of each stope is visualized by three distinct icons that represent the Drilling, Stoping and Backfilling of the stope. The production of stope S12 can only start once the backfill in stope S11 has cured. The production of stope S21 (situated vertically under stope S12) can only start once the backfill in stope S11, S12 and S13 is fully cured. Because stope S13 can only be produced after stope S12 has cured and stope S12 can only be produced after stope S11 has cured, a vertical dependency is only necessary between stope S13 and S21.

The dependency creation between stopes (the production sequence) is mainly carried out manually and visually checked by animating the resulting production sequence. The end-result is a web of lateral and vertical dependencies that represent a feasible production sequence (see Figure 71).

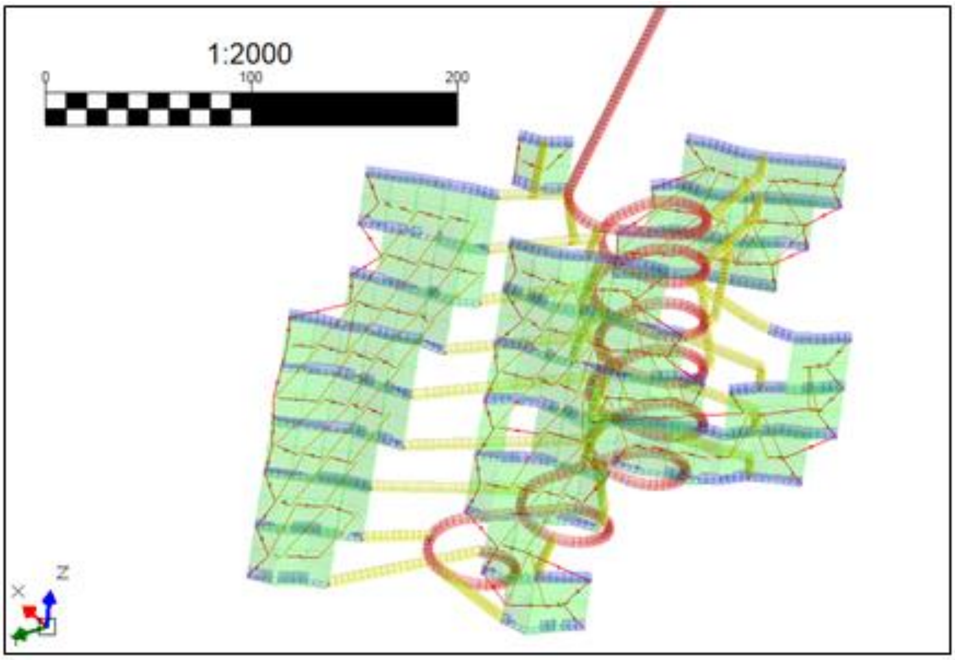


Figure 71 - Task solids and final production sequence impression (1.7 g/t mine design).

4.3.15.3 TASK RATES, RESOURCE RATES & RESOURCE LEVELLING

Now that all the task solids are defined and linked by the dependency rules, the respective task rate needs to be assigned to each solid and subsequently a resource to complete the task. The task rate and resource together determine the duration of a task and whilst considering all tasks together, a life-of-mine development and production schedule is obtained.

The task rates as used by Boliden are presented in Table 28.

Activity Type	Task type	Task rate / duration	Resource	Resource Rate	Resource Type
Development					
Decline	Fixed rate	40-60m/mo	Development Jumbo	400m/mo	Effort driven
Level	Fixed rate	40-60m/mo	Development Jumbo	400m/mo	Effort driven
Ore Drive	Fixed rate	40-60m/mo	Development Jumbo	400m/mo	Effort driven
Production					
Drilling	Fixed duration	1d	Production Drill Rig	300m/d	Effort driven
Stoping	Fixed rate	500t/d	LHD	500t/d	Required
Backfilling	Fixed duration	21d	n.a.	n.a.	n.a.

Table 28 - Task Rates & Task Durations.

The development task rate is 40 to 60 meters per face per month. However, the Development Jumbo has a capacity of 400m/month. Therefore the Jumbo can be assigned to multiple tunnel faces during a month. To allow the Jumbo to do this, it is assigned as an Effort driven resource, meaning that the resource rate compared to the task requirements will determine the number of resources required (Deswik Mining Consultants, 2013). For example, at a task rate of 40 meters per month, the resource can theoretically develop 10 separate tunnel faces when running at full capacity.

The stoping task rate is 500t/d which is equal to the resource rate. Because the costs used in the Technical Economical Model are contractor prices based on a mine production rate of circa 500kt/year, and it is unknown how many LHD's, trucks etc. is accounted for, the resource rate of the LHD is set to 500t/d based on a Required resource type. This means that if a stope is in production, an LHD is required regardless of the task rate (although the same in this case). This is used in the scheduling process to create scenarios for a production rate of 1000t/d (2 LHD's) and 1500t/d (3 LHD's).

The project scenarios that were investigated as part of this study are:

Cut-off grade (g/t)	Development Task Rate (m/mo)	Target Production Rate (tpd)
1.5/1.6/1.7/1.8/1.9/2.0	40/60	1000/1500

Table 29 – Investigated Development Task Rates and Mine Production Rates.

This results in a total of 24 different life of mine schedules and with 4 different processing routes, a total of 96 Cash Flow Models (Figure 72) of which the results will be discussed in the next section.

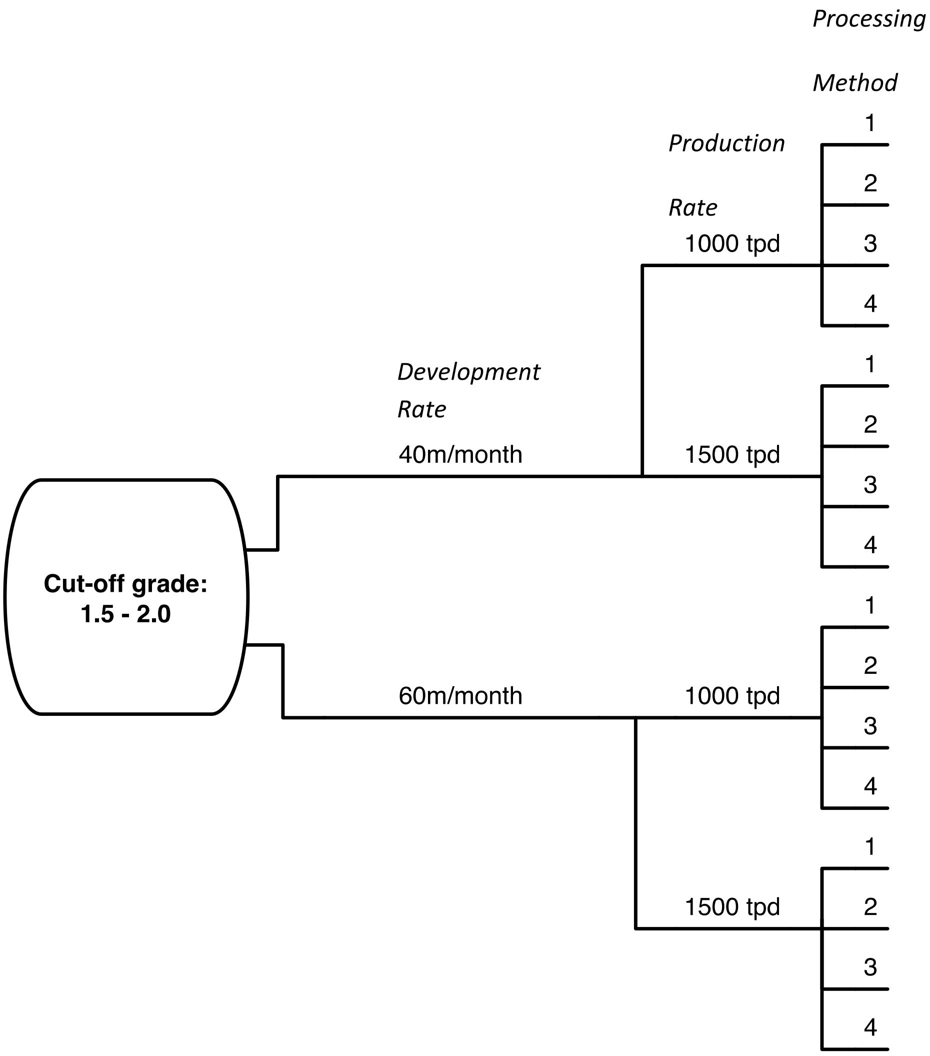


Figure 72 - Overview and build-up of project scenarios.

4.3.16 OPTIMIZATION RESULTS

By creating mine designs and multiple production schedules for each cut-off grade it is possible to find the optimal project scenario and analyze the influence of certain parameters (e.g. gold price, mining costs) to the project NPV over a full range of cut-off grades rather than just a single grade design. The first section of this chapter deals with the results obtained from the base case scenario whilst the second part of the chapter will look at sensitivity analysis of the main input parameters.

4.3.16.1 SCHEDULING RESULTS

Because all the mine designs were scheduled at the same 2 Production and Development rates (4 cases for each cut-off grade) it is possible to create a graph of the Life of Mine as a function of the cut-off grade. This graph is presented in Figure 73.

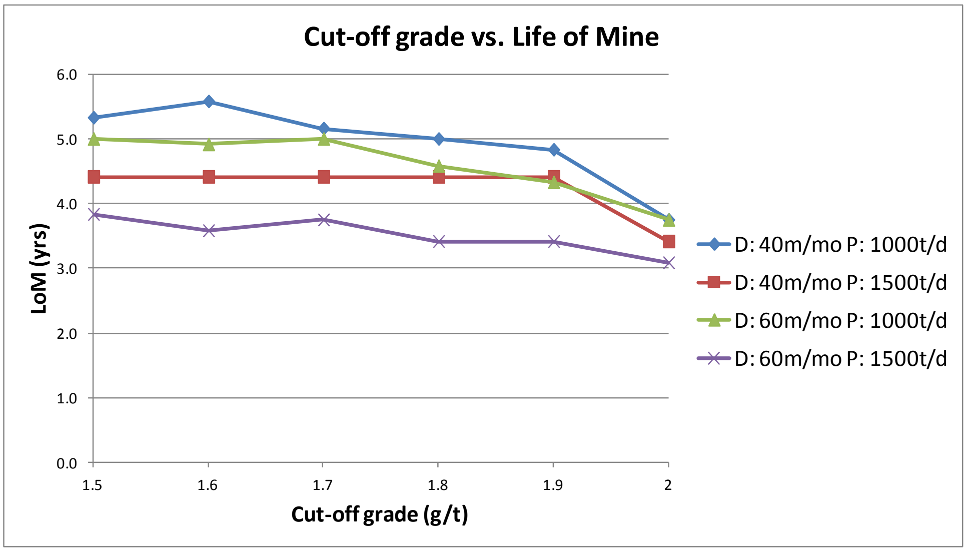


Figure 73 - Cut-off grade vs. Life of Mine.

What can be observed in the graph in Figure 73 is that at the high development rate of 60m/mo and a production rate of 1500t/d the life of mine is shortest and at the lowest development rate of 40m/mo and a production rate of 1000t/d the total life of mine is longest. What also can be observed (in combination with Figure 74) is that the life of mine at a development rate of 40m/mo and a production rate of 1500t/d is fully depending on the development rate over the cut-off range 1.5-1.9 indicating that the production capacity is not used properly. For the other (development/production) combinations the delay between final development and final production reduces with increased cut-off grades. This is to be expected because most of the required development stays the same over the full range of evaluated cut–off grades whilst the mineable tonnages decrease with increased cut-off grades (reduced stope lengths).

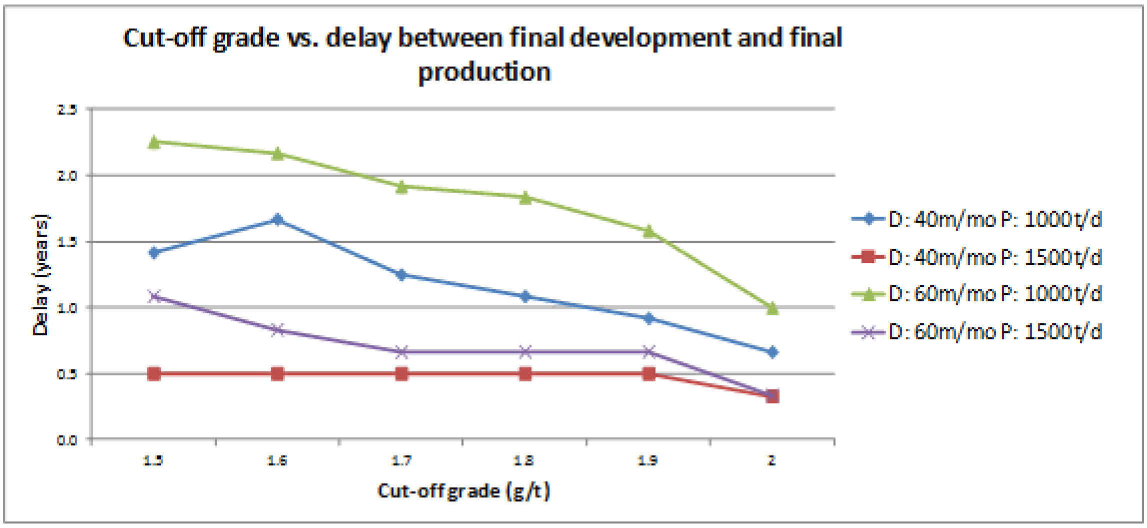


Figure 74 - Delay between final development and final production.

Important in any life of mine production schedule is that the material flow from the mine to the processing plant does not fluctuate too much. A constant ore flow means that the processing facility is used at its full capacity and the revenue stream does not fluctuate too much.

The cumulative RoM tonnage curves for each cut-off grade based mine design are collected in appendix B.2. When analyzing the graphs for linearity it is noted that although at a production rate of 1500t/d the curve starts of steeper, it quickly starts to curve meaning that there is no longer a steady production rate. The curves representing a production rate of 1000t/d show better linearity indicating that a steady production rate is sustained over a longer period of time.

Appendix B.3 summarizes the utilization of the production target capacity for each of the cut-off based mine designs. These graphs show that in general, target production capacity is only met (and/or exceeded) over a short period of time. This has to do with fact that many ore drives are constructed during the early life of the mine. Because the ore drives are situated within the ore body they contribute to the ore production. It is observed that at a target production rate of 1000 stoping tonnes per day, circa 80% of the capacity is sustained over a longer period of time. The target production rate of 1500stoping tonnes per day is more difficult to sustain over a longer period of time and after an early short utilitzation of 100% the production rate drops rapidly.

In general it can be concluded that the orebody is too small to sustain the target production rates by using this mining method. The required development to access the orebody is disproportional to the amount of mineable ore and due to the lateral and vertical dependencies there are not enough production faces (stopes) available to sustain a target near-constant material flow.

4.3.16.2 CASH FLOW ANALYSIS

A separate TEM was created for each individual mining scenario. A total of 96 scenarios including various cut-off grades, production schedules and processing methods were analysed within Microsoft Excel and linked to a control sheet for evaluation. This section will deal with the base case scenario results after which a classic sensitivity analysis is performed on the optimal solution.

4.3.16.2.1 BASE CASE SCENARIO OPTIMIZATION RESULTS

The base case scenario includes all costs and metal prices as shown in Table 30 and previously discussed in section 4.3.9.

Parameter	Unit	Totals
Exchange Rate	SEK:USD	7.5:1
Discount Rate	%	10
Gold Price	USD/oz	1200
Silver Price	USD/oz	20
Mining Cost	SEK/t	250
Processing Cost (Gravity/Flotation)	SEK/t	98
Processing Cost (Leaching)	SEK/t	163.13
Development Cost	SEK/m	25,000
Mine Capital Expenditure	MSEK	106.26

Table 30 – Älgträsk Base Case scenario value/costs assumptions.

These base case prices were inserted into the cash flow model for each scenario resulting in the Hill of Value graph presented in Figure 75.

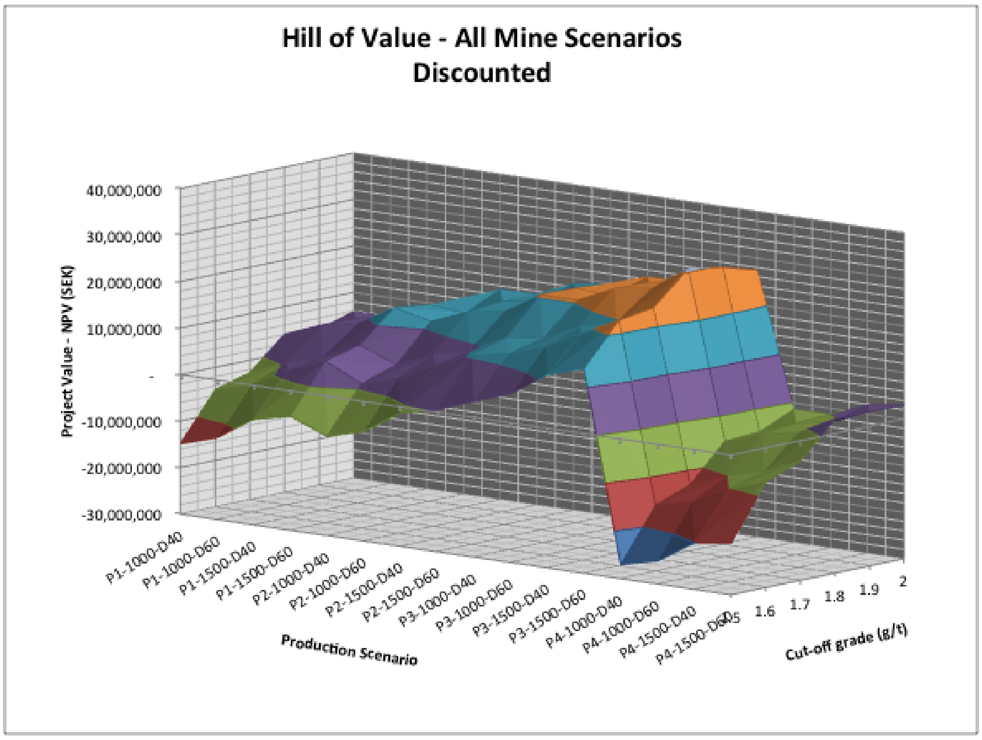


Figure 75 - Hill of Value – All mine scenarios at base case parameters (Mountain top is optimum scenario).

In the Hill of Value Graph it is observed that processing method P3 (Gravity and Flotation of Low As ore and Gravity and Leaching of High As ore) results in the highest project value on a discounted basis. The production scenarios for processing method 3 were further analysed at the target rate of 1000 and 1500 stoping tonnes per day in Figure 76 and Figure 77.

1000 t/d Case - Undiscounted

In the undiscounted upper graph of Figure 76 it can be clearly observed that there is a maximum profit to be made at a cut-off grade of 1.8 g/t in combination with a development rate of 40m per month on a non-discounted basis. This is slightly better than the profit at a development rate of 60m per month. The difference is explained by the fact that for the lower development rate the combination of stopes that are in production at the same time, result in a slightly better ore stream mixture of high and low arsenic ore which reduces processing costs and smelter charges resulting in an overall increased profit.

1000 t/d Case - Discounted

The optimal cut-off grade is also 1.8 g/t on a discounted basis in combination with a development rate of 60m per month. With a maximum project NPV@10% 24,2 MSEK (= 3,2 MUSD) and IRR=19%, it is clear that for the base case price assumptions this project is not economically viable.

The advantage of the ore mixture at a development rate of 40m per month on an undiscounted basis is overruled by the increased life of mine at this development rate (Figure 73). This clearly shows the time value of money concept in relation to the maximization of project NPV.

At a development rate of 40 meters per month there is no longer an optimum found within the range 1.5-2.0 g/t due to the discounting of future cash flows.

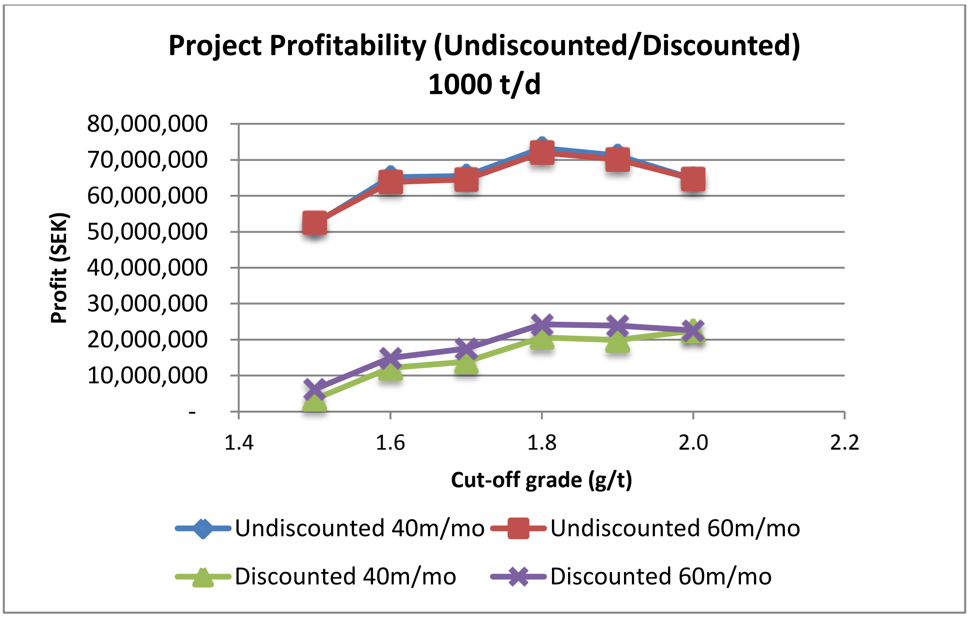


Figure 76 - Base Case Scenario Undiscounted & Discounted Cashflow vs. Cut-off grade - 1000tpd.

1500 t/d Case – Undiscounted

Again it is observed that the maximum profit is to be made at a cut-off of 1.8 g/t in combination with a development rate of 40m per month. The reason for it being better compared to the 60m per month development rate is again believed to be the mixture of the ore stream and resulting smelter charges.

1500 t/d – Discounted

At an increased production rate (and therefore decreased life of mine) we see that the project NPV slightly increases and that he optimum cut-off grade (the grade that maximizes the NPV) is now 1.8 g/t at a development rate of 60m per month. Again it is observed that at a development rate of 40meters per month, the optimal cut-off grade is not found. When comparing Figure 77 to Figure 73 it can be concluded that the life of mine at a development rate of 40m/month is mainly controlled by the development rate. This results in an extended life of mine and, with the time value of money in mind, a higher optimum cut-off grade.

With a maximum project NPV@10% of 31 MSEK (= 4,1 MUSD) at a cut-off grade of 1.8 g/t the project has an Internal Rate of Return of 23%. The profitability of the project increases only slightly compared to the lower target production rate as there is no large difference in the life of mine between the two options (due to the fact that the mineable reserves are very small). However, the slightly positive NPV indicates that the project could become feasible whenever circumstances improve (eg. a higher gold price or the discovery of additional ore reserves).

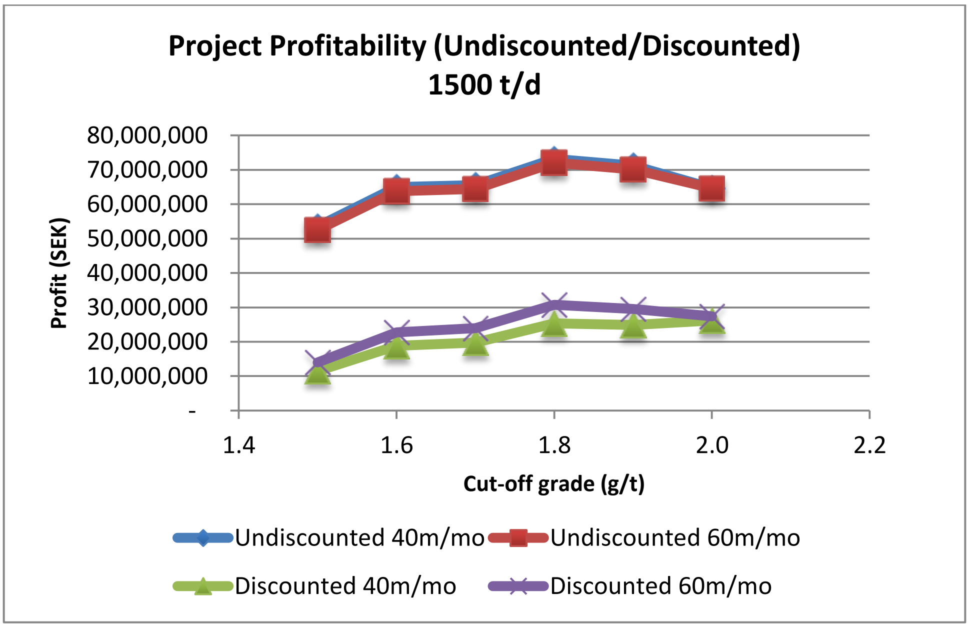


Figure 77 - Base Case Scenario Undiscounted & Discounted Cashflow vs. Cut-off grade - 1500tpd.

4.3.17 SENSITIVITY ANALYSIS

Once a mining and processing scenario is selected based on the cash flow analysis it is common practice to perform a sensitivity analysis on the base case scenario assumptions as to analyze the impact to the project NPV by changing an input parameter. The standard approach is to vary one of the inputs whilst keeping the other parameters at their base case value. The result is a graph that represents the influence on the project profitability when changing one of these parameters.

This sensitivity analysis approach was applied to the optimum scenario, which is in this case:

Cut-off Grade: 1.8 g/t

Production Rate: 1500tpd

Development Rate: 60m/mo

Processing Scenario: Gravity and Flotation of Low As ore and Gravity and Leaching of High As ore (Scenario 3).

The evaluated cases in the sensitivity analysis are presented Table 31 and the resulting sensitivity plot is presented in Figure 78.

Parameter	**Unit**	**-20% Totals**	**-10% Totals**	**Base Case**	**+10% Totals**	**+20% Totals**
Exchange Rate	SEK:USD	6	6.75	7.5	8.25	9
Gold Price	USD/oz	960	1080	1200	1320	1440
Silver Price	USD/oz	16	18	20	22	24
Mining Cost	SEK/t	200	225	250	275	300
Processing Cost (Gravity/Flotation)	SEK/t	78.4	88.2	98	107.8	117.6
Processing Cost (Leaching)	SEK/t	130.50	146.82	65.13	179.44	195.76
Development	SEK/m	20000.00	22500.00	25000	27500.00	30000.00
Mine Capital Expenditure	MSEK	85.01	95.63	106.26	116.89	127.51

Table 31 – Input Parameters to the classic sensitivity analysis.

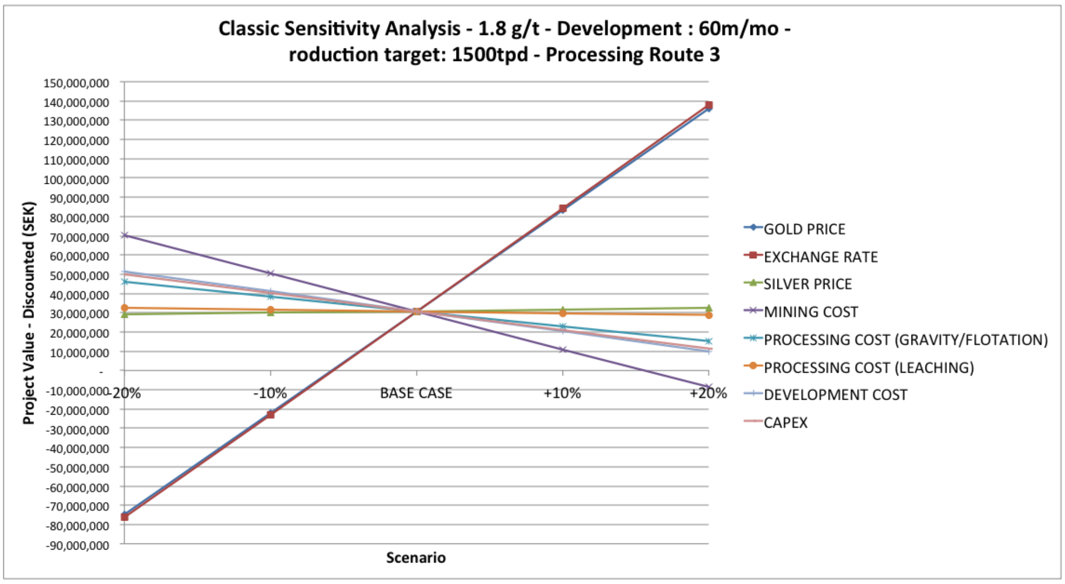


Figure 78 - Classic Sensitivity Analysis - Processing route 3.

Based on the sensitivity analysis the following can be concluded;

Gold Price

The project is very sensitive to the gold price. This has to do with the project, at current price assumptions, being a marginal project. At base case assumptions however, the project is just about breaking even meaning that whenever the gold price will increase this can be considered pure profit. Vice versa, whenever the gold price drops, the project will quickly become non-profitable since gold is the main mineral.

Exchange Rate

The same project sensitiviy is observed for the exchange rate. With mining taking place in Sweden, all costs are paid in Swedish Kronor (SEK) whilst gold is traded on the international market in US Dollars per troy ounce. Whenever the exchange rate goes up, one will receive more Swedish Kronor for each dollar earned whilst whenever the exchange rate goes down, the project NPV will rapidly drop below 0.

Silver Price

Although silver mineralization occurs regularly within the Älgträsk deposit it is the silver price compared to the gold price that, together with the too low-grade occurrence, that makes this mineral’s influence to the project NPV insignificant.

Mining Cost

The perhaps most interesting conclusion based on this classic sensitivity analysis is the fact that the mining cost is the biggest cost contributor to the project NPV. At the same moment, as discussed previously, this is also the least well-known cost factor and therefore it is noted that a more detailed study on mining cost should be executed.

Processing Costs

With only 17%* of the ore being of the High As type and being processed by Gravity and Leaching these processing costs contribute less to the project profitability compared to the Low As ore stream that is processed by Gravity and Flotation. Because 83%* of the ore is processed by this method, the influence of this processing cost is larger though still relatively small compared to the mining costs. It should also be noted that the processing plant is already up-and-running which means that processing costs can be considered well known and are unlikely to deviate much from the base case assumptions.

**Percentages apply specifically to the 1.8g/t mine design*

Development Costs

The base case assumption used as development cost is 25.000 SEK/metre tunnel. This is an average number is used by Boliden for both decline and level development as well as ore drives. With little geotechnical data available at this point in time, this number is assumed reasonable. Whenever more detailed geotechnical data becomes available, a more detailed tunnel and support design should be realized and costs reconsidered. The influence of development costs to the project profitability is considered to be limited. Only if very negative or positive rock conditions are encountered the project profitability is affected.

4.3.18 CONCLUSION

The Mine Optimization Approach introduced in section 2.4 was applied to one of Boliden's mineral resources and a total of 96 different scenarios were evaluated. A break-even calculation provided the first estimate of the cut-off grade and the AMIRA Stope Optimizer was used to design feasible stopes for a range of cut-off grades. Access development to the production areas was designed and the resulting mine designs were imported into Deswik.Scheduler to perform life of mine scheduling of the project at different development and target production rates. The resulting tonnage and grade schedules were imported into a Technical Economic Model in Microsoft Excel for cash flow analysis. The cash flow analysis based on 4 different processing methods was calculated and resulted in multiple Project NPV's. By combining the 96 different scenarios in a Hill-of-Value type chart, the optimal project strategy could be obtained.

The optimum strategy proved to be following scenario:

Cut-off Grade: 1.8 g/t

Production Rate: 1500tpd

Development Rate: 60m/mo

Processing Scenario: Gravity and Flotation of Low As ore and Gravity and Leaching of High As ore (Scenario 3).

It is concluded that although the process of cut-off grade optimization proved to be a time-consuming process, it did increase the project NPV by circa 35 % compared to the 1.6g/t break-even estimate. The use of the AMIRA Stope Optimizer significantly reduced the time required for stope design and proved to be a valuable tool in strategic mine planning as it allows the engineer to rapidly create mine designs.

The Algrtrask underground project proved to be a very marginal mining project and of high risk due to the required large investments compared to the potential profit and life of mine. The project is extremely volatile to the gold price and with a life of mine of less than 4 years from start to finish, the risk inherent in the fluctuation of the gold price over time cannot be minimized.

4.4 CASE 3: ASSESSING GRADE RISK IN THE ÄLGTRÄSK UNDERGROUND STUDY

DISCLAIMER:

The following study on grade risk is based on a different resource model than the one currently being used by Boliden Mineral AB. The applied approach to data preparation, data analysis, variogram modeling and resource model estimation/simulation is explained in depth to clarify fundamental differences. The risk assessment approach is applicable in any mining project.

4.4.1 INTRODUCTION

The case study in section 4.3 resulted in an optimum mining strategy that maximizes project NPV. It was also noted that the the project is very marginal and extremely sensitive to the gold price. This lead to the believe that the succes of the project will be largely based on the accuracy of the resource model (representing the spatial location of high grade zones within the orebody). This case study will asses the spatial grade uncertainty present in the mineral resource by means of stochastic simlation of equally true realizations of the mineral resource. The approach as proposed in section 2.5 of the report was used as a guideline in this study and each step will be subsequently explained.

4.4.2 EXPLORATORY DATA ANALYSIS

Table 32 shows an overview of all exploration drilling carried out at the Älgträsk project site to-date (Boliden Mineral AB, 2013).

Drilling method	Period	Hole nr	Total lenght (m)	Contractor
Core	1987-1990	1-116	15 363	Boliden
Core	1996-1997	117-138	5 889	Boliden
Core	2004	139-147	1 507	Prospekteringsteknik Norr
Core	2005-2007	148-268	22 397	Rate
RC	2006	5001-5010	838	SweDrill
Core	2008-2010	269-365	17 126	Rate
RC	2008	5011-5061	4 150	Borr VVS
Core	2010-2011	366-373	2 687	Styrud
Core	2012	374-387	2 638	Styrud
Core	2012-2013	388-403	2 936	Styrud

Table 32 - Drilling at Älgträsk to date (4-30-2013).

A dataset containing 2m composites based on the exploration drill hole database was provided by Boliden Mineral AB together with a resource estimate report. The Älgträsk drill holes were assayed for multiple components and are summarized in Table 33.

Assayed Grades	
AU_PPM	PB_PROC
AG_PPM	AS_PROC
CU_PROC	S_PROC
ZN_PROC	MO_PPM

Table 33 - Assayed Components in the exploration drill holes.

From Table 32 it is noted that two drilling methods were used in the exploration campaigns (Reverse Circulation (RC) and Diamond Core drilling (BH)) by several different operators. Before an estimation or simulation of the ore grades can be executed it is necessary to determine the required estimation and simulation algorithms. The selection of the required algorithm depends on the grade distributions in the Reverse Circulation and Borehole datasets. If the grade distributions of both datasets are the same, the two can be combined into one single dataset and Ordinary Kriging (OK) and Sequential Gaussian Simulation (SGS) methods can be used for grade interpolation/simulation into the block model. Whenever the two datasets show different grade distributions, other methods should be used for grade interpolation/simulation such as co-Kriging or Sequential Gaussian co-Simulation.

4.4.2.1 DATA CLEANING

Not all samples taken during drilling (both in BH and RC holes) were assayed for all elements in the laboratory for reasons unknown. To avoid an empty assay field in the drill hole database whenever a test was not conducted, a default value of ‘0’ was supplied by Boliden Geologists. Although this method results in a conservative estimate, some of these ‘0’ values were removed from the composite dataset by applying a selection criterion as they clearly introduced a measurement bias in the drill hole database. A zero value is accepted as a ‘measured zero grade’ for a sample whenever other elements in the same sample have been tested and show some grade. Whenever all elements show a ‘0’ grade value, it is assumed that no tests have been carried out on this sample (or at least they were not inserted into the database) and the sample is removed from the dataset to avoid the introduction of a measurement bias. The resulting cleaned composite dataset was used for further analysis.

4.4.2.2 DATASET DISTRIBUTIONAL SIMILARITY EVALUATION

The cleaned composite dataset was split into an RC and BH dataset and loaded into The Stanford Geostatistical Modeling Software package (SGeMS) for evaluation. SGeMS is a software package that can be used for data analysis, variogram modeling as well as estimation and simulation of block models.

Due to the limited availability of exploration data, the assay values of all ore lenses were combined according to the mineralization area they belong to. In the Älgträsk project area, two mineralization areas are defined based on orebody orientation (Nyhem and Liden, Figure 79) and both drill hole datasets were split accordingly. As this study focused on a small area within the Nyhem Mineralization, the Liden area is was not further investigated.

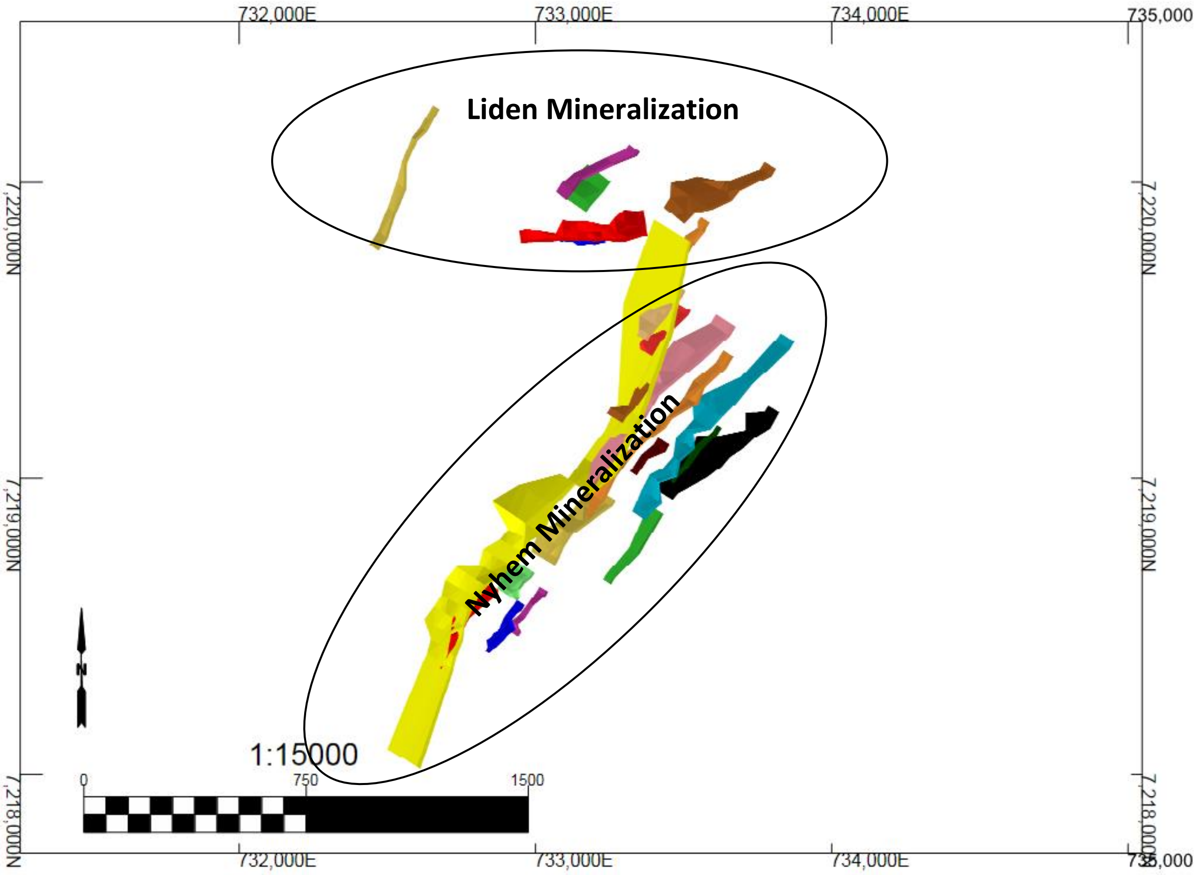


Figure 79 - The separate Ore Zones at Älgträsk (Boliden Mineral AB, 2013).

Figure 80 shows the histogram of both the RC and BH drill hole dataset covering the Nyhem mineralization. Both datasets show a positively skewed distribution which is a common grade distribution for mineral deposits (Wellmer, 1998). Visual comparison of the BH and RC dataset indicates similar distributional behavior especially in the lower grade range from 0 to 10 g/t Au. The maximum grade of 30 g/t in both datasets is caused by the introduction of a top cap in gold grade of 30 g/t.

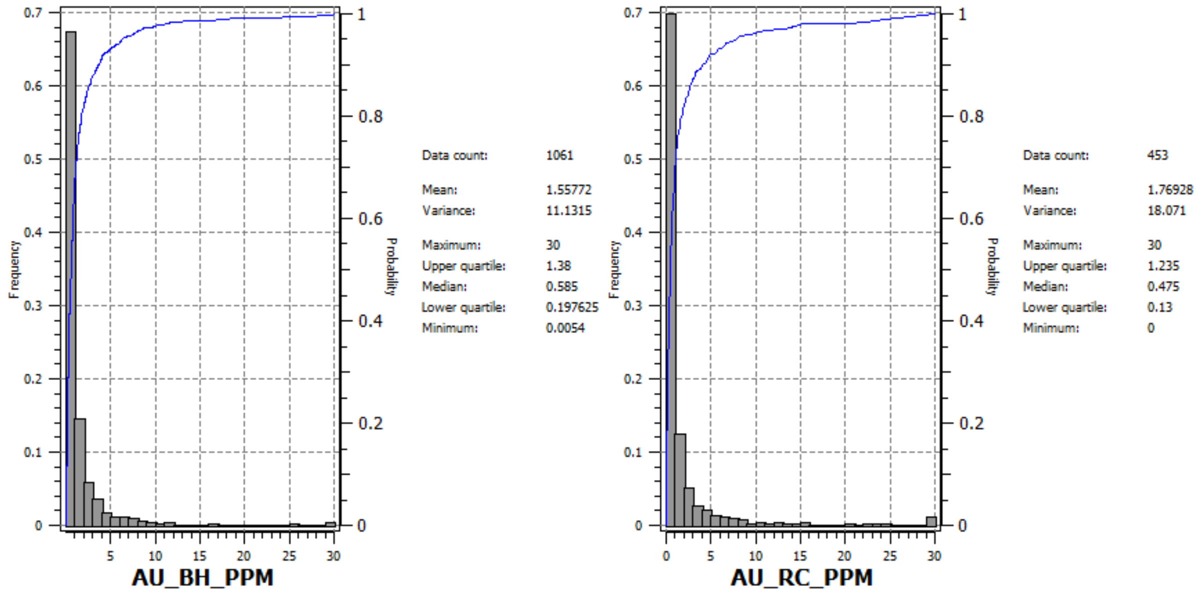


Figure 80 - Histograms for BH and RC datasets.

When both datasets are combined into a new histogram (Figure 81), only one distribution is visible (a positively skewed distribution with a single peak). This is considered a first indication that both the RC and BH datasets have the same grade distribution and the datasets can be combined.

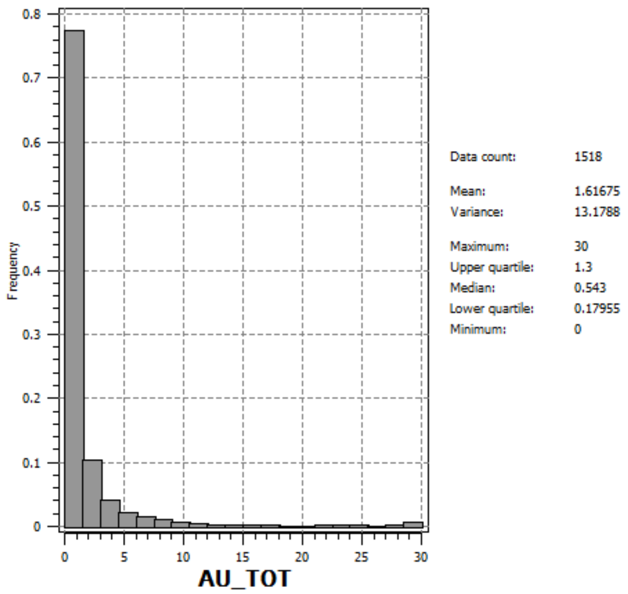


Figure 81 - Histogram of the combined RC and BH datasets.

To further analyze similarity, both datasets can be analyzed in a Quantile-Quantile plot (QQ-plot). The QQ-plot is a method to compare sample datasets to a theoretical distribution (eg. normal, logarithmic etc.) By plotting the quantiles of a sample dataset to the quantiles of a theoretical distribution the nature of the dataset can be verified. Whenever the distribution of the dataset is the same as the chosen theoretical distribution, the QQ-plot will show linearity. When the theoretical distribution is replaced by a second sample dataset, it is possible to find if the two datasets have the same distribution as in that case, the QQ-plot will also show linearity.

Figure 82 shows the Quantile-Quantile plot for the RC versus BH dataset. The left graph shows the skewed datasets whilst the right graph shows the QQ-plot of the normalized datasets. The left graph shows clear linear behavior in the lower grade range (0-10) which is in line with the observations in the histogram analysis. For the higher grade values the linearity is less and this is most likely caused by the fact that only a few high grade values were measured in the drill holes (the grade distribution is highly skewed and 95% of the grades are less than 10 g/t) and these values can be considered as outliers. The measured high gold grades are in that case considered nuggets. A second possible reason for the non linear behavior (the discrepancy between BH and RC data) is a result of the drilling and sampling method that is applied. When RC drill holes are sampled, part of the gold may get lost as fine particles (gold dust) which can only be avoided if these fines are collected whilst drilling and are added to the collected sample.

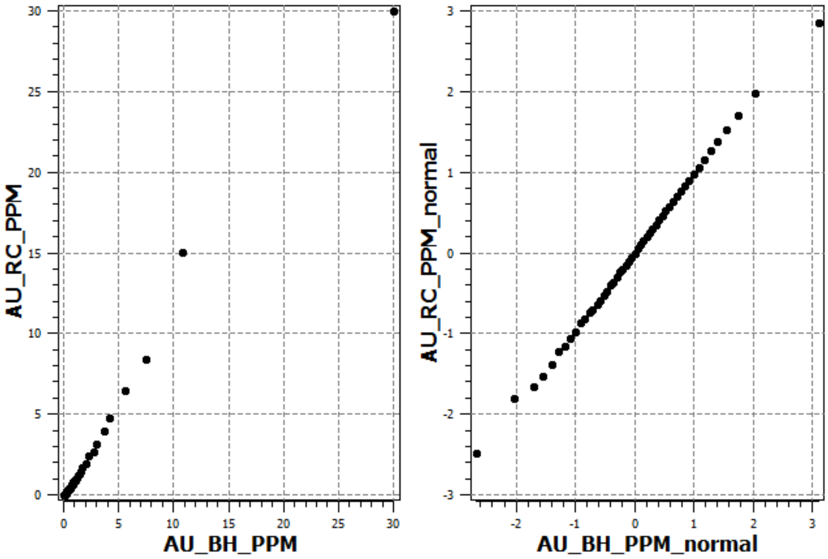


Figure 82 - QQ-plot skewed datasets (left) and normalized datasets (right).

In order to better analyze the two datasets for distributional similarity; it is common practice to transform skewed data into a normalized (Gaussian) space. By doing so, the influence of the outliers present in the dataset is reduced. SGeMS was used to normalize the datasets and the result of these transformations is shown in Figure 83. It is concluded that both datasets fit to a normal distribution with a mean of circa 0 and variance 1. The RC distributions is noisier due to the limited number of samples (453) compared to the BH dataset (1061 samples).

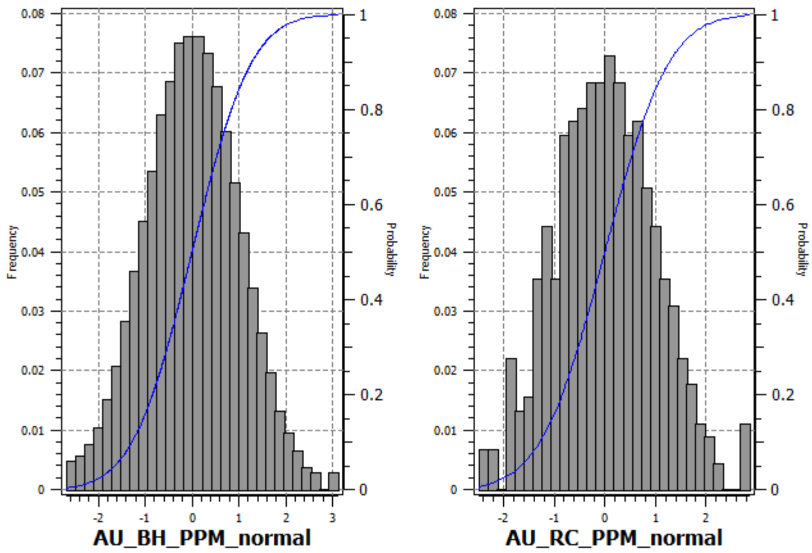


Figure 83 - Normalized Datasets BH (left) and RC (right).

The normalized datasets are again presented in a QQ-plot (Figure 82, right). The fact that circa 95% of the data (with a gold grade between 0 and 10 g/t) already shows linearity in its non-transformed skewed form is a good indicator of distributional similarity. The QQ-plot of the Gaussian transformed data strengthens the indication that the two datasets can be combined into a single set. As a final step in the distributional similarity evaluation, the down-hole variograms of both datasets are compared.

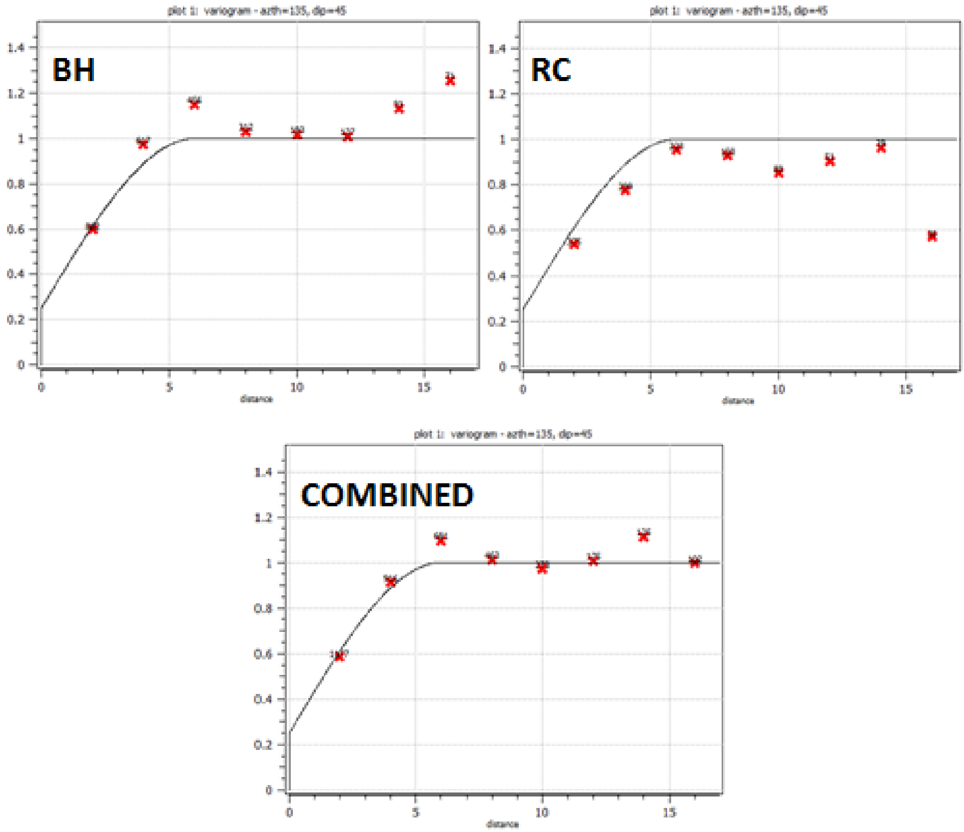


Figure 84 - Down Hole Variogram Model BH (top left), RC (top right) and Combined (bottom) dataset.

Figure 84 shows the down-hole variogram models of the normalized BH and RC datasets as well as a down-hole variogram of the normalized combined dataset. It is observed that the experimental BH and RC variogram fit to the same theoretical variogram model. Based on the down-hole variogram analysis it is concluded both datasets fit to a variogram model with a nugget of 0.25, a sill contribution of 0.75 and a range of 6m. When the same nugget, sill and range are supplied to the combined dataset, it is found that the experimental variogram can be fitted with the same model parameters.

Based on the Histogram Analysis, the QQ-plot analysis and the down-hole Variogram Analysis it is concluded that both datasets can be combined into one single dataset. This combined dataset will be used for directional variogram modeling, grade estimation and stochastic simulation. Because the datasets can be combined, Ordinary Kriging is the method of choice for grade estimation and Sequential Gaussian Simulation can be used for grade simulation.

4.4.3 VARIOGRAM MODELING

The next requirement for block model estimation and simulation is/are variogram model(s). The variogram model will describe the spatial relationship (variance) between points in a 3D space. From the down-hole variogram model it is noted that points are related to one another up to a range of 6 meters. As this may be true for samples down a drill hole, it is unlikely that the mineral deposit will show the same relationship in other directions (eg. along strike, down dip). This requires the application of directional variogram modeling to find the deposits' directional grade anisotropy. The results from the down-hole variogram model are however not completely meaningless. Due to the fact that the down hole variogram is based on the most sample points (grades are measured at intervals down the drill hole), the nugget effect (which is a measurement of the true nugget effect in neighboring samples as well as sampling errors) is considered to be most precise and is therefore used when creating directional variograms.

The standard approach to directional variogram modeling is to create variograms in many directions and to find the direction that shows the largest spatial relationship (the longest range). This would then be the principal direction of continuity and the semi-major and minor directions of continuity are taken perpendicularly to the major direction.

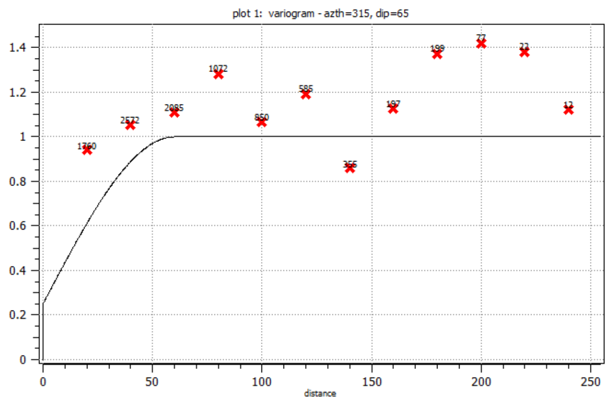
In this scoping study all exploration data for the ore lenses that have roughly the same dip and dip direction are combined into one dataset. For this reason the orientations for the directional variograms (Table 34) are also chosen based on geological orientation of these ore lenses.

Direction	Dip Direction/Dip	Description
Major	315/65	"Down Dip"
Semi-Major	225/00	"Along Strike"
Minor	135/25	"Across Strike"

Table 34 - Orientation of Directional Variograms.

The directional variograms as defined in Table 34 are presented in Figure 85. Variogram modeling has proven to be difficult for the major and semi-major direction due to the limited availability of closely spaced drill holes. It turned out to be impossible to reproduce the variogram model ranges of 63m (major) and 60m (semi-major) as found by the Boliden Resource Department (Boliden Mineral AB, 2013) in these directions.

Based on the results of directional variogram modeling it is concluded that too little data is available to determine preferred directional continuity. However, to compare the impact of directional variograms, two series of estimations and simulations were performed. The first series uses the directional variograms with ranges as defined in Figure 85. These variogram models are in line with the variogram ranges used by Boliden Mineral AB.

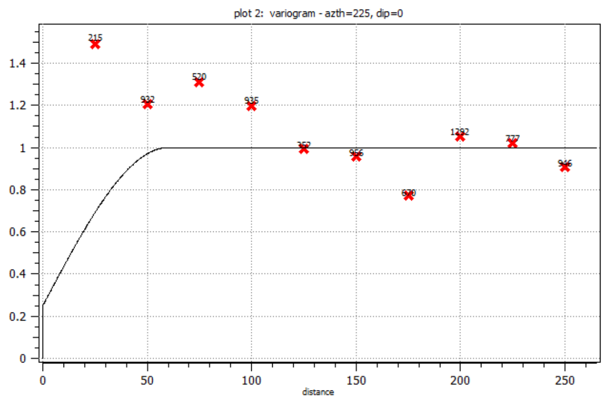


Orientation: 315/65

Nugget: 0.25

Sill Contribution: 0.75

Range: 60m

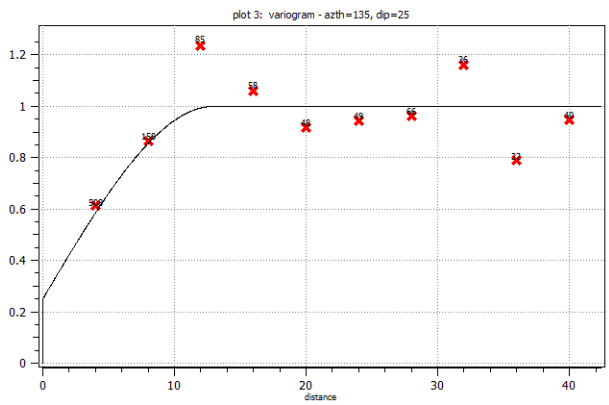


Orientation: 225/00

Nugget: 0.25

Sill Contribution: 0.75

Range: 60m



Orientation: 135/25

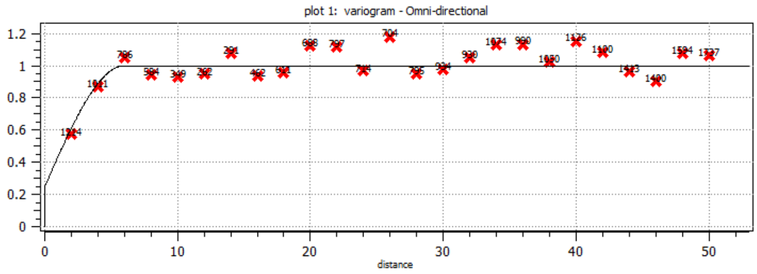
Nugget: 0.25

Sill Contribution: 0.75

Range: 13

Figure 85 - Directional Variogram Models in Major (top), Semi-major (middle) and Minor (bottom) direction of continuity.

For the second series an omni-directional variogram is proposed as the input model for grade estimation and simulation (Figure 86). Although the constructed directional variograms indicate the likeliness of a directional trend, it is impossible to quantify this trend with the available data and the use of an omni-directional variogram with a short range of 6m is deemed acceptable as a conservative estimate for estimation and simulation until more exploration data becomes available.



Orientation: Omni-directional

Nugget: 0.25

Sill Contribution: 0.75

Range: 6m

Figure 86 - Omni-directional variogram for grade estimation/simulation.

4.4.4 THE ÄLGTRÄSK AREA OF INTEREST

A full optimization process was carried out for the ore lenses on the south-western edge of the Nyhem mineralization (Figure 79, Figure 61). The goal of this study is to analyze the grade risk present in the optimized project strategy that resulted from the Älgträsk Underground Study presented in section 4.3. The geological wireframes representing the ore lenses that were not part of this earlier study were removed from the geological model and not included in further grade estimations/simulations.

4.4.5 BLOCK MODEL ESTIMATION

A different interpretation of the exploration drill hole data resulted in the combination of the RC and BH datasets into one dataset. Variograms were modeled according to this new dataset which resulted in the creation of 2 new estimated block models in addition to the existing Boliden model (The Boliden Estimation Method is provided in appendix C.1).

Directional Model:

An estimated model based on directional variograms with the same orientations and ranges as used by Boliden Mineral AB.

Omni-directional Model:

An estimated model based on 1 omni-directional variogram 'in all directions' with a range of 6 meters as this is the only variogram found by the author after extensive variogram modeling.

All Block Model Estimations were carried out with Datamine Studio 3 using the composite drill hole data and variogram models. Zonal control was used to ensure that blocks inside an ore lens are interpolated only with the grades measured in the respective lens. To limit the down-hole effect in grade estimation (as a result of most of the measured data can be found in a down-hole direction), the maximum number of samples coming from a single drill hole was set to '3' whilst the minimum number of samples used to interpolate a grade block was set to '4'. This assures a grade interpolation using at-least two drill holes.

The resulting estimated block models are presented in Figure 87 for visual reference (larger plots are presented in appendix C.2) and the according grade-tonnage curves can be found in Figure 88.

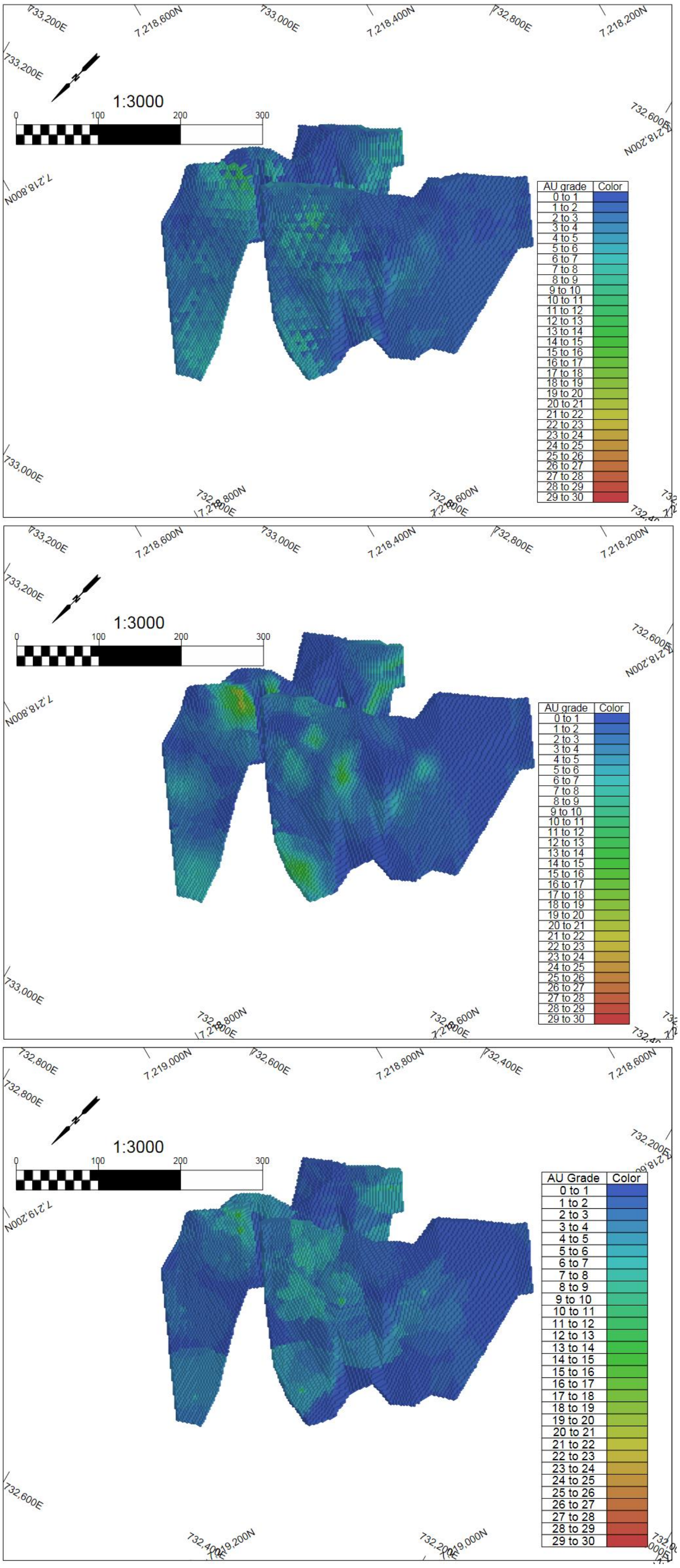


Figure 87 - Estimated Block Models by Boliden (top), directional variograms (middle), omni-directional variogram (bottom).

When visually comparing the block models it is observed that all three models show similar grade patterns with each model interpolating higher grade zones slightly different depending on the variogram model (directional or omni-directional) or other grade interpolating calculations (Boliden model). When comparing the same models based on their respective grade tonnage-curves it is observed that the average grade of all models is circa 2 g/t (@ 0 g/t cut-off). Depending on the estimation settings some variations exist in terms of tonnes of ore with a certain average grade for different cut-off grades. The larger range of grades in the Directional Variogram Model is a result of the directional spatial relationship as described by the range of the variograms used in interpolation. This behavior is not observed in the Boliden Model as the Resource Department at Boliden estimated two gold grades in the model based on the separate RC and BH datasets. The resulting RC and BH gold grades were converted into a final gold grade by the application of a weighting factor. This weighted average has a smaller grade range compared to the Directional Variogram Model which is based on the combined BH and RC dataset. The Omni-directional variogram shows a smaller window of gold grades as a result of the small variogram range (6m).

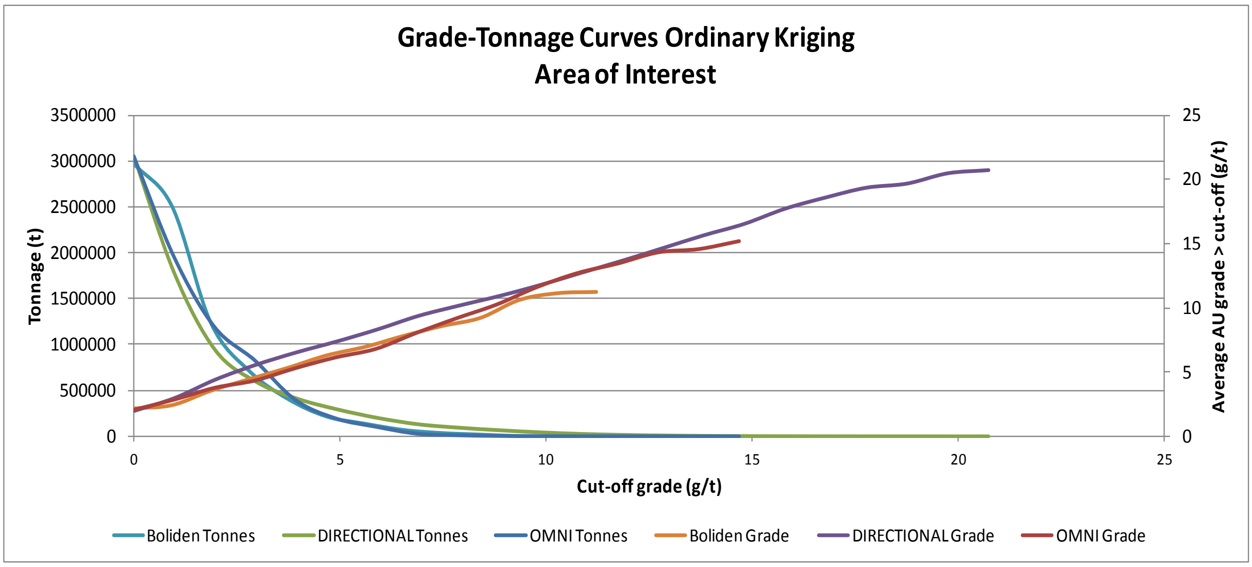


Figure 88 - Grade Tonnage Curve comparison between 3 estimation methods.

The individual ore lenses (lens 1, 2 and 3) were interrogated against all estimated block models (Table 35) and it was observed that on average all estimation models return similar total tonnage and grade values for the ore lenses. The average gold grade within the individual lenses differs slightly with each estimation method and this is accounted to the variogram models used (the spatial relationship used in interpolation) in combination with the available drill hole data.

	Ore Lens 1		*Ore Lens 2*		*Ore Lens 3*	
Ordinary Kriging Model Type	**Tonnage (t)**	**Average Grade (g/t)**	**Tonnage (t)**	**Average Grade (g/t)**	**Tonnage (t)**	**Average Grade (g/t)**
Boliden	1,865,660	1.85	577,905	1.86	631,931	2.24
Directional Variogram	1,869,072	1.76	579,332	1.88	632,584	2.05
Omni-directional Variogram	1,870,197	1.83	577,905	2.05	631,931	1.78

	All lenses combined	
Ordinary Kriging Model Type	**Tonnage (Mt)**	**Average Grade (g/t)**
Boliden	3.1	1.93
Directional Variogram	3.1	1.84
Omni-directional Variogram	3.1	1.86

Table 35 – Comparison of tonnages and average grades in the individual ore lenses.

4.4.6 STOPE OPTIMIZATION ON THE ESTIMATED BLOCK MODEL

The estimation method as used by the Resource Department at Boliden Mineral AB does not allow direct comparison with the block models resulting from stochastic simulation (due to the way that the RC and BH dataset are treated). It was therefore decided to use the directional variogram based model and the omni-directional variogram model as starting points in risk analysis.

The directional variogram model is more comparable to the Boliden model in terms of variogram ranges and location of the high and low grade areas within the orebody whilst the omni-directional variogram model was found by the author to be the only variogram model to describe grade relationships in the orebody based on the available drillhole data.

As a starting point for stope optimization using the new estimated models it was assumed that the optimum project strategy for the Älgträsk Area of Interest is still the optimum as found in the Älgträsk Underground Study, regardless of the different resource models created.

The optimum strategy proved to be following scenario:

Cut-off Grade: 1.8 g/t

Production Rate: 1500tpd

Development Rate: 60m/mo

Processing Scenario: Gravity and Flotation of Low As ore and Gravity and Leaching of High As ore (Scenario 3).

This means that only the optimized stopes at a cut-off grade of 1.8 g/t were further processed into a life-of-mine plan with resulting production schedule and Technical Economic Model for cash-flow analysis.

Figure 89 shows the stope optimizer stope design outputs based on the directional and omni-directional estimated models with the original Boliden model based optimization output shown for reference purposes. The location of the stopes in the optimization outputs are comparable to the block models shown in Figure 87 in terms of the location of optimized stopes and the higher grade zones as observed in the models.

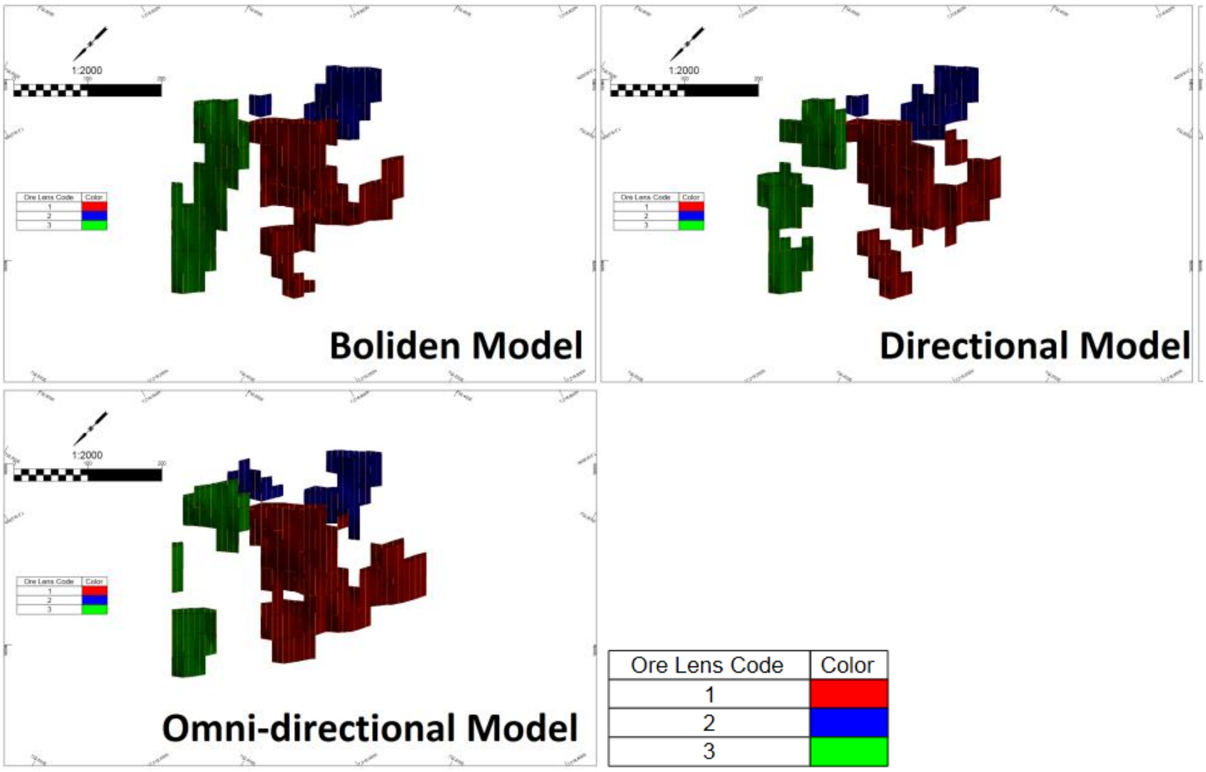


Figure 89 - Stope Optimizer outputs (@ 1.8 g/t cut-off) comparison between estimated models.

Table 36 summarizes the total stope tonnages and average grade for each Stope Optimizer output set of stopes. It is once again noted that the Directional model is very comparable to the Boliden model in terms of number and stope tonnages, however, the average grade is higher in the directional model as RC and BH drill hole data is treated equally in the estimation process and no weighting factor is applied.

Model	# of Stopes	Stope Tonnes (t)	Average AU Grade (g/t)
Boliden	118	1,096,242	2.91
Directional	123	1,045,255	3.66
Omni-directional	129	1,280,820	3.09

Table 36 - Optimized Stope Tonnages and Average Grade comparison between estimated models.

The Omni-directional model based optimization shows an increased tonnage compared to the Directional model at a lower average grade. Although optimized stope layout looks significantly different, the author believes this to be the most correct optimization based on the available drill hole data. This is clarified in Figure 90 which shows the Omni-directional model based optimized stopes and the intersecting drill holes (black dots). It is observed that areas where no drill hole data is available (examples noted by red circles) do not show any optimized stopes as is the case in the Boliden and Directional model. This indicates that the grade smoothing effect in the mineral resource model was somewhat reduced due to the use of a short range omni-directional variogram model. The use of this variogram assures that high gold grades are not interpolated in areas where there is no measured ’evidence’ for such high grades.

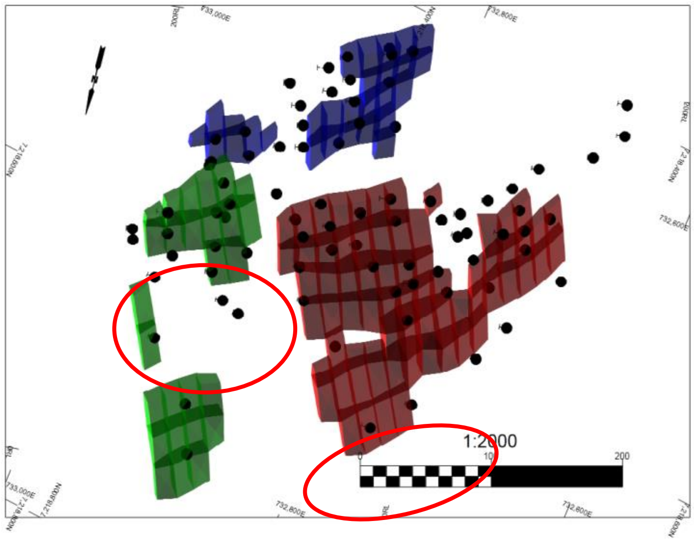


Figure 90 - Omni-Directional Model Stope Optimization with Intersecting Drill Holes.

Based on the comparison between the different block models and resulting stope optimizations it is concluded that the variogram ranges that are being used have a significant impact on the grade estimation in terms of tonnage and grade, but also on the final stope locations. The variogram modeling step is as such a very, if not most important step in the estimation process and care should be taken when creating variograms.

4.4.7 BLOCK MODEL CONDITIONAL SIMULATION

Because a single drill hole dataset is used in this study, Sequential Gaussian Simulation is the method of choice to create simulated block models. GSLIB (Stanford University, 2009) was used to perform the simulations. The **G**eostatistical **S**oftware **Lib**rary is a set of freely available software to perform sequential Gaussian simulations.

Again two sets of simulations were created based on the same Directional and Omni-directional variograms as used to create the estimated block models. The process of creating and validating the simulations is explained here for the directional model and a similar explanation for the omni-directional model is provided in appendix C.3.

4.4.7.1 SIMULATION BLOCK MODEL REQUIREMENTS

In order to correctly simulate block grades a regular block size has to be selected as the simulation software cannot deal with sub-blocking. Due to the limited thickness of the individual ore lenses a small block size of 2.5x2.5x2.5 metre was selected (the minimum sub-block size used in the Boliden block model). The small block size ensures a good fit to the geological wireframes and as simulations were limited to the Älgträsk Area of Interest, computation times remained acceptable.

Another important criterion for the selection of a small block size is that a small block size reduces the optimization error when using Stope Optimizer. Stope Optimizer uses only block model cells for optimization and therefore a better ‘geological fit’ of the block model cells to the geological boundaries ensures a more correct optimization of the stope boundaries and minimizes the stope tonnage and grade error.

4.4.7.2 SIMULATION APPROACH AND RESULTS

The ore lenses were simulated with the drill hole data intersecting the specific zone (in the same way as Ordinary Kriging estimation with zonal control is carried out) and the resulting models were later combined into one final model. By simulating the zones separately, grade contamination between closely spaced ore lenses is avoided. 25 simulations of the orebody were generated and some example simulations and the simulations ETYPE-model are shown in Figure 91 together with the Ordinary Kriging Estimated solution.

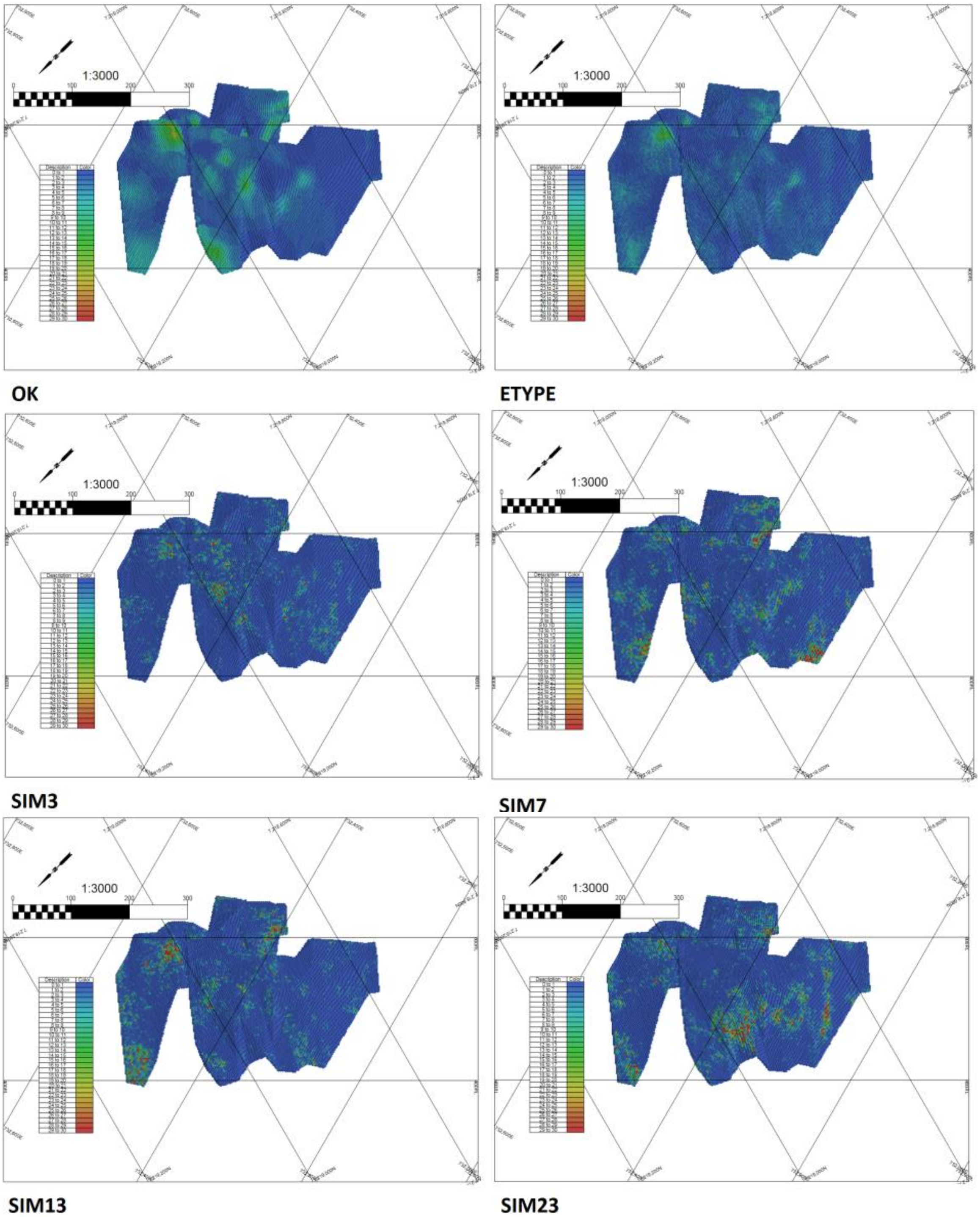


Figure 91 - Simulation examples and ETYPE (with OK block model for reference purposes).

4.4.7.3 SIMULATION VALIDATION

Before the simulated block models can be compared to the corresponding Ordinary Kriging Estimated block model and any conclusions can be drawn on grade risk the simulations have to be validated. The approach to validate Gaussian simulations follows 3 distinct steps and each step is explained in this section.

Grade Distribution Reproduction

The first assumption in sequential Gaussian simulation is that the grade distribution found in the drill holes is in fact the true grade distribution of the full deposit. As such the simulated block models should have the same grade distribution as the composite data.

The grade distributions of all 25 simulated block models are shown in Figure 92 together with the grade distribution of the composite data (green) and the grade distribution resulting from the Directional Ordinary Kriging Estimated block model. It is concluded that the simulated block models have approximately the same grade distribution as the composite data and therefore, on grade distribution only, the simulated block models are a valid interpretation of the grade distribution in the deposit.

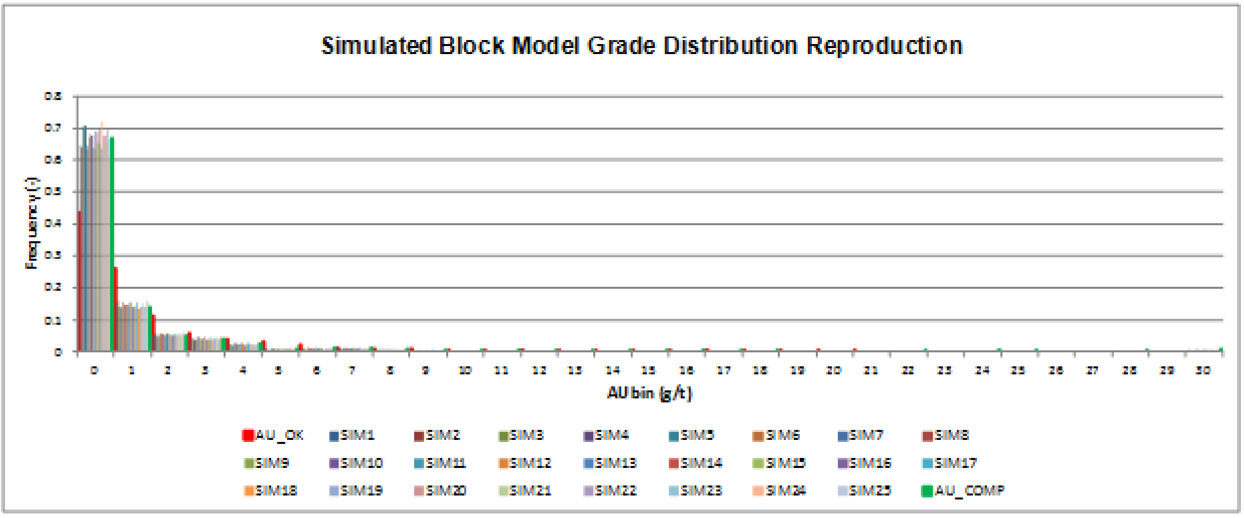


Figure 92 - Grade Distribution Analysis between Simulated and Composite data.

Variogram Reproduction

The subsequent step in the validation process is variogram reproduction. The simulated grades should have the same spatial relationship as modeled by the variograms and in order for the simulations to be valid it should be possible to reproduce the variograms as determined from the composite data. Experimental variograms were produced for all simulated block models and compared to the modeled variograms.

The variogram reproductions in the major, medium and minor direction of continuity are presented in Figure 93. When comparing the average experimental variogram of all simulations (red line, Figure 93) to the theoretical model (green line, Figure 93) as found during variogram modeling it is observed that the sill and range of the simulation based experimental variograms are the same as the theoretical model. Some deviation from the theoretical model is observed especially at short lag distances. The deviation is caused by the unfavorable dataset which consists of very little (closely spaced) samples in the 315/65 (down dip) and 225/00 (along strike) directions. As a result of the lack of data points the variogram modeling in these directions is poor and this causes the noise in the simulated experimental variograms. The experimental variograms in the 135/25 direction deviate less from the theoretical model as more data points are available in this direction due to a favorable drill hole orientation. The overall shape of the experimental variograms is in line with the theoretical variogram and due to the limited amount of available data in this project stage the variogram reproduction is deemed acceptable. It should be noted that whenever more sample data becomes available, the process of variogram modeling, block model estimation and simulation should be re-executed.

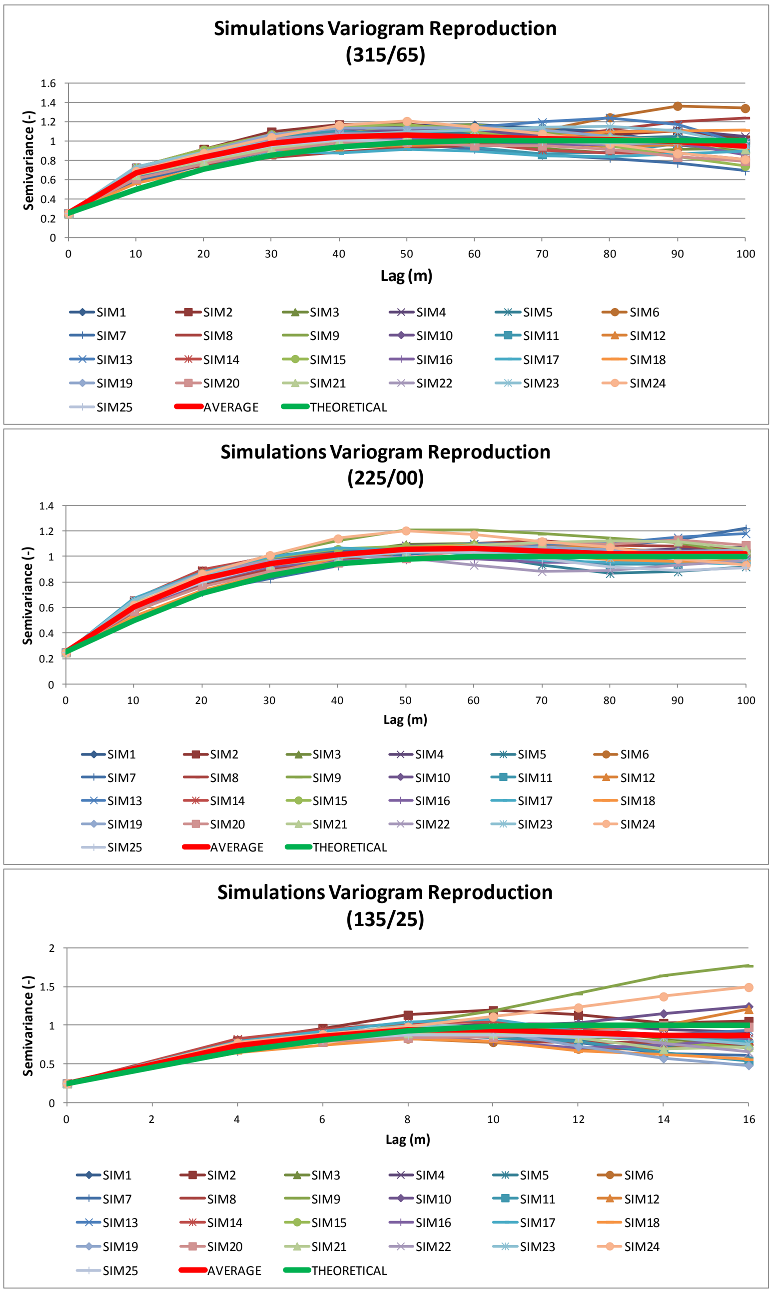


Figure 93 - Simulations variogram reproduction.

Measured Data Reproduction

The final check to validate the simulated block models is to visually check if measured grades are reproduced in all simulated block models as fixed values at their measured location. A visual check confirmed this as was to be expected because the measured data has been assigned to the grid nodes before simulations were executed.

All validation checks to the simulation results proved satisfactory and it is therefore concluded that the simulations can be compared to the Ordinary Kriging Estimated model and be used for risk assessment.

4.4.8 COMPARISON – ESTIMATION VS. SIMULATION

Figure 94a and Figure 95a show the grade-tonnage curves of the simulated block models compared to the estimated block model based on the same variogram model. Due to the linear estimation nature of the Kriging algorithm a smoothing effect appears in the grade distribution after estimation. The result of this smoothing effect is that the lower grade material is over-estimated whilst the higher grade material in the resource is under-estimated. This effect is clearly visible when analyzing the tonnage (black line) and grade (red line) curves. The high grades that were present in the composite data are not reproduced in the estimated block model whilst the lower grades are over-estimated (this effect is also observed when analyzing the grade distribution reproductions in Figure b). Depending on the variogram ranges used in estimation, the effect is weaker or stronger but the composite data grade distribution is never reproduced.

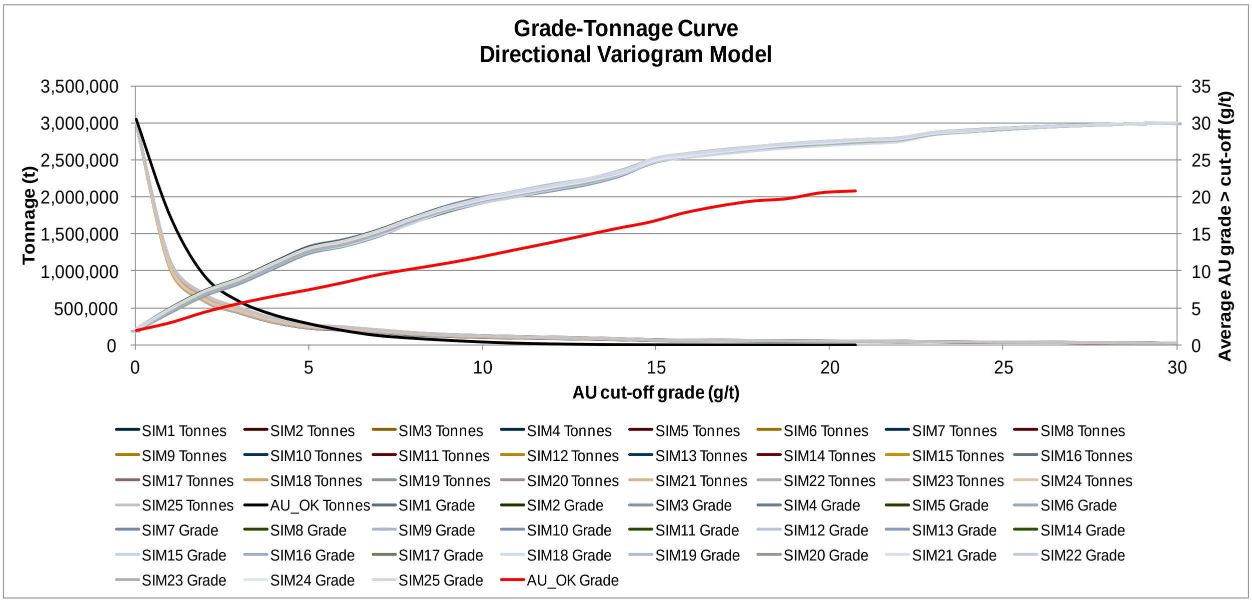


Figure 94a - Grade-Tonnage curve comparison Directional estimation vs. simulations.

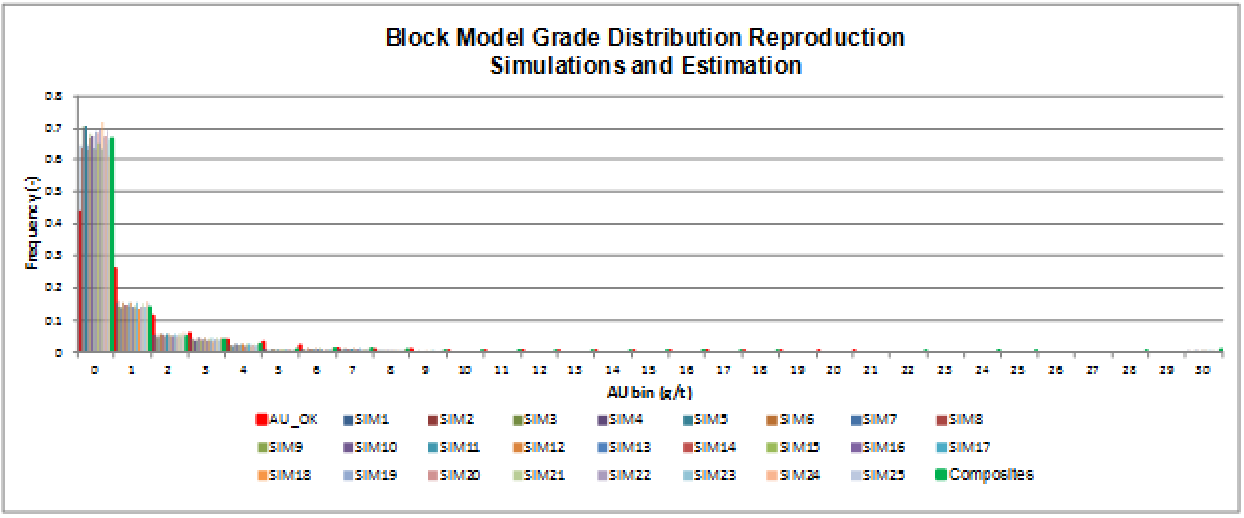


Figure 94b - Block Model Grade Distribution Reproduction (Directional Models).

The simulated models show a better reproduction of the composite grade distribution. This is one of the strong capabilities of Sequential Gaussian Simulations (SGS) as it respects and reproduces the 'measured grade distribution' which is known to exist within the orebody.

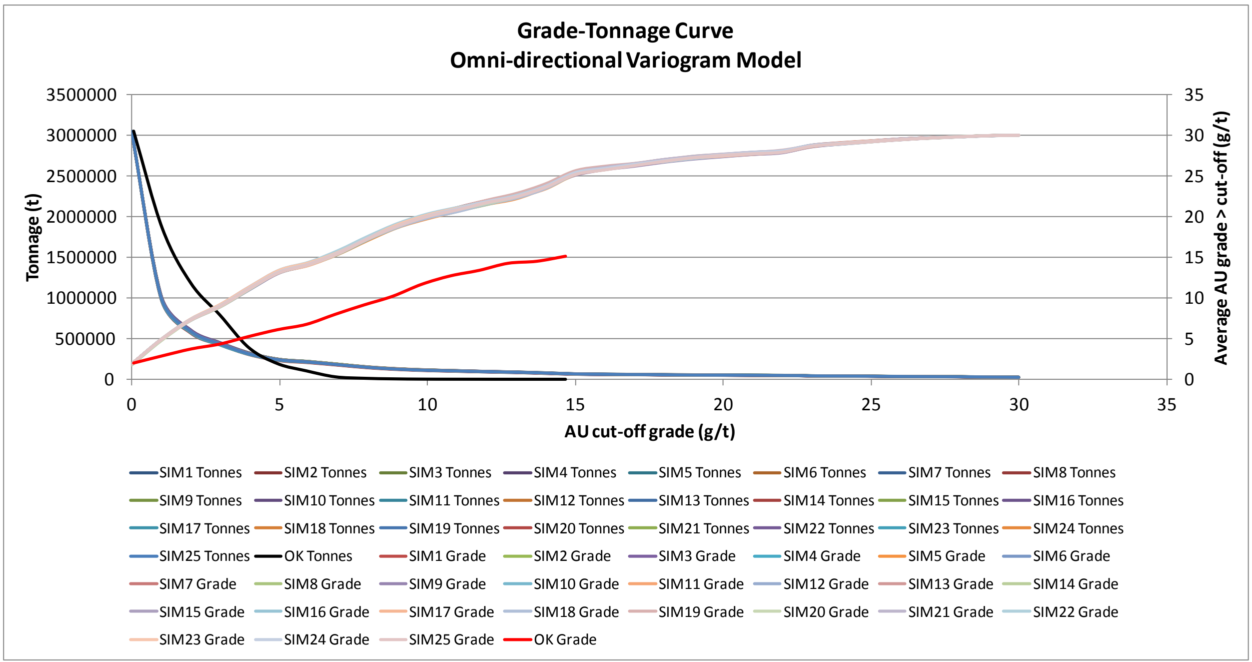


Figure 95a - Grade-Tonnage curve comparison Omni-directional estimation vs. simulations.

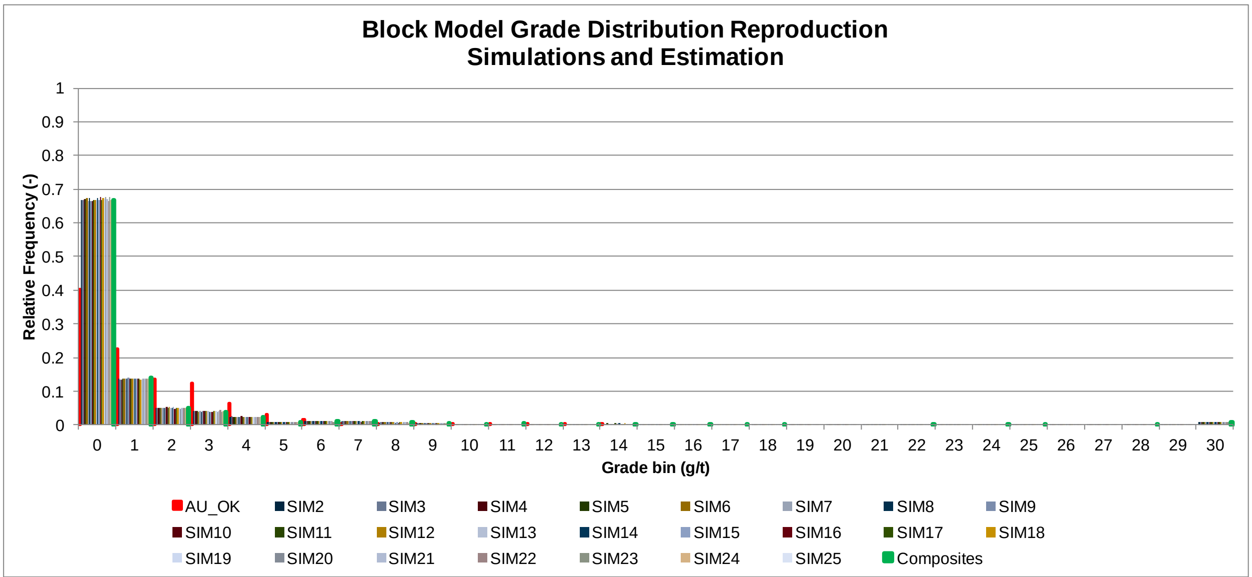


Figure 95b - Block Model Grade Distribution Reproduction (Omni-directional Models).

Another effect of the Kriging Estimation algorithm is the positive grade interpolation in areas surrounding a high measured grade in a drill hole. Grades are interpolated by applying weighting factors to surrounding measured grades. Due to the limited exploration drilling being carried out to date, the number of measured data points is low meaning that the few available points define the whole area. If the measured grades are relatively high, the area surrounding these points is likely to be over-estimated in terms of grade. In other words, measured high grade drill hole samples have got a too great influence in the estimation of the surrounding blocks whilst there may not be enough evidence to support grade continuity. Both estimated block models presented in Figure 96 show this behavior.

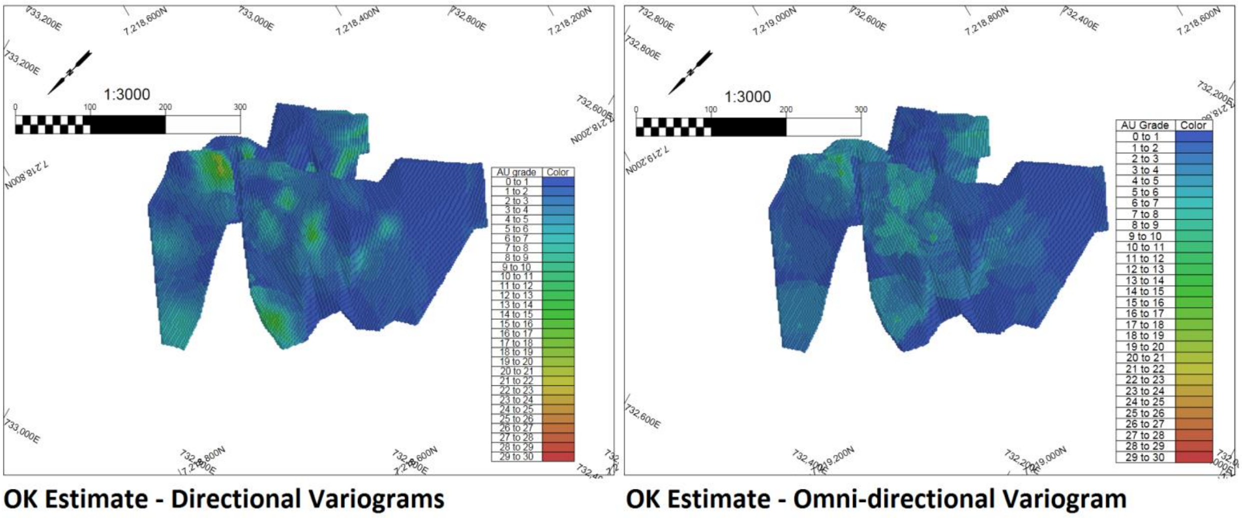


Figure 96 - Positive grade interpolation in estimated block models.

The incorrect reproduction of the grade distribution and the positive grade estimation form a potential hazard in mine optimization. A too optimistic interpretation can result in positive project decisions and project profitability expectations that can never be achieved.

The simulated block models aim to account for the spatial grade uncertainty by simulating ‘equally true’ realizations of the mineral resource. Once a set of simulations for an orebody is created the conditional variance for each block can be calculated which is a good indicator of variability in the resource estimate. The resulting conditional variance models are presented in Figure 97.

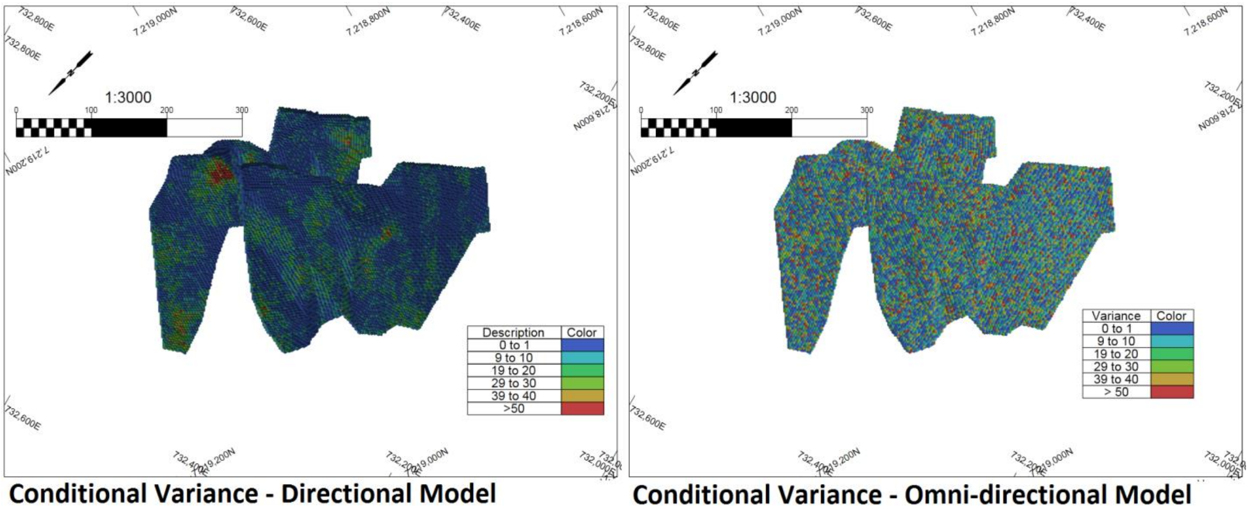


Figure 97 - Conditional Variance (Simulated block models).

When comparing the conditional variance of the directional model with the directional Kriging estimated model in Figure 96 it is observed that the high grade areas estimated by the Kriging algorithm are, based on 25 equally true simulations of the same model, also the areas with higher grade uncertainty (higher conditional variance). This indicates that a level of uncertainty will be present in the previously optimized stopes that need to be investigated.

The conditional variance of the Omni-directional model is showing a high level of ‘noise’ caused by high grade uncertainty within the simulated block models. The high uncertainty is a result of the single omni-directional variogram used to describe spatial relationships between grades. The range within grades are spatially correlated was interpreted to be 6 meters (as this was the only valid variogram produced by the author) whilst the drill hole spacing is up to 10 times larger. The high scatter in conditional variance is a result of random grade distribution reproduction outside the range of the variogram. Only within the direct vicinity of a borehole, conditional variances are low (<1) as is shown in Figure 98 indicating that there is a higher level of confidence in these grade blocks.

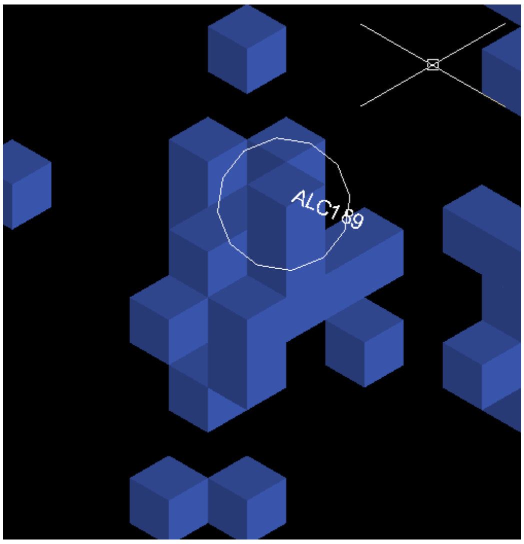


Figure 98 - Low conditional variances in the direct vicinity of borehole ALC189 (Omni-directional simulations).

Based on the noise level present in the conditional variance model, optimized stopes based on the omni-directional estimated block model are expected to show lower confidence levels compared to the directional estimated model based stope optimizations. This will be investigated in the next section of this report.

4.4.9 ASSIGNING CONFIDENCE LEVELS TO STOPE DESIGNS

This section explains the process of evaluating the uncertainty present in an optimized stope design that is based on an estimated block model only. The optimized stope designs are interrogated to simulated block models in order to assign a level of grade confidence to the 'optimized stopes'. First the approach to uncertainty assessment will be presented after which the results of the Directional and Omni-directional model will be discussed.

4.4.9.1 UNCERTAINTY ASSESSMENT APPROACH

To analyze the uncertainty present in optimized stopes based on the Ordinary Kriging Estimated block model a confidence level of achieving the estimated stope headgrade was assigned to each stope. To do this the optimized stopes (based on the estimated resource model) were interrogated against all 25 simulations of the same resource model. This resulted in an Ordinary Kriging estimated stope grade and 25 simulated grades for each stope. The 25 conditional simulated stope gold grades were evaluated against the Kriging based stope gold grade by the following equations to determine the stope confidence level:

$$C_i = \begin{cases} 1, & S_i \geq AU_{OK} \\ 0, & else \end{cases}$$

$$Confidence\ level = \frac{1}{n}\sum_{i=1}^{n} C_i$$

In which S_i is the simulated stope grade and C_i is an integer (0 or 1) assigned to a simulated stope grade to define if it is equal or higher to the Ordinary Kriging Estimated stope grade. The confidence level is the total number of simulated stope grades divided by the total number of simulations that meet the Ordinary Kriging Estimate and is expressed as a number between 0 and 1.

4.4.9.2 CONFIDENCE LEVELS IN ESTIMATION BASED OPTIMIZED STOPES

The confidence levels for the Directional- and Omni-directional Estimated model based stope designs were calculated according to the previously described method and are graphically presented in Figure 99 (larger plots can be found in appendix C.4).

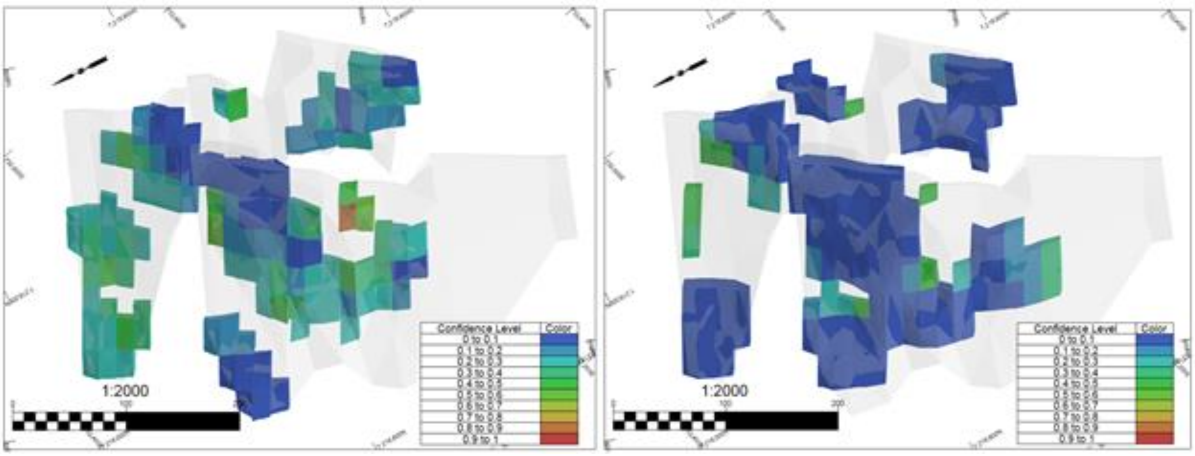


Figure 99 - Confidence Levels for the Directional (left) and Omni-directional (right) Kriging Estimated model based optimized Stope Designs at Älgträsk.

When analyzing the confidence levels of the estimation based stope designs it is found that in general the confidence levels are very low. The Directional Model based stope designs show confidence levels that are rarely over 50% and the majority of stopes show confidence levels between 0 and 30%. The stope-to-stope confidence levels together with the ordinary Kriging grades in comparison to the simulated grade quantiles are presented in Figure 100. It is observed that the ordinary Kriging estimated stope headgrades are almost always in the 3rd and 4th quantile of the simulated grades which means that the likeliness of a stope meeting the kriging estimated headgrade when taken into production is generally low. The positive stope headgrade estimate is explained by the positive smoothing effect present in Ordinary Kriging Estimation which results in smeared-out high grade areas surrounding a measured high grade. This effect is not present in sequential Gaussian simulation and hence the simulations are a better representation of the possible stope headgrades.

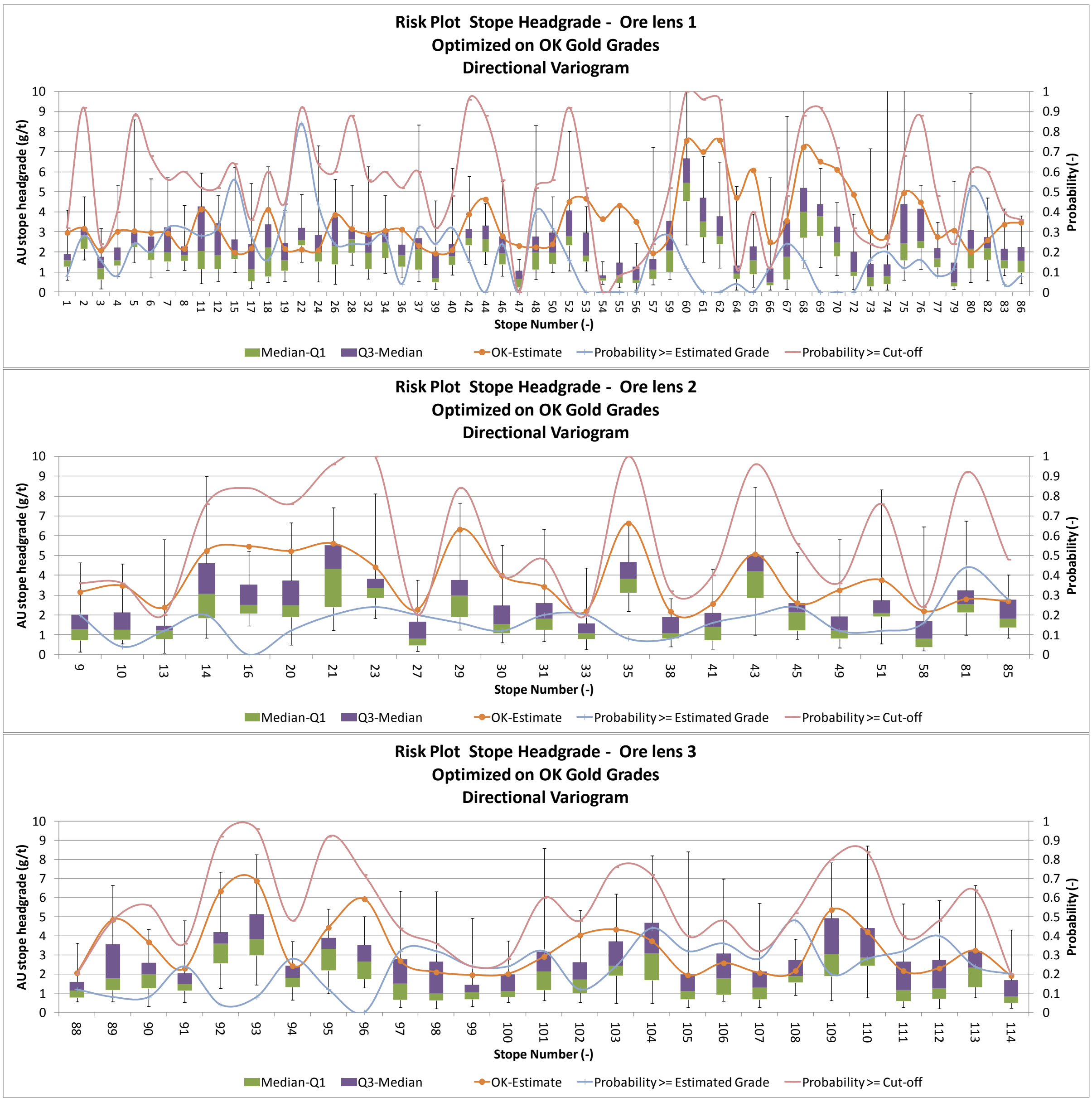


Figure 100 - Stope-to-Stope confidence level box plots (Directional Variogram Model).

The same analysis was performed on the omni-directional estimated model based stope optimizations and the stope-to-stope results are presented in Figure 100. Based on the conditional variance of the 25 simulated models the confidence level of stopes meeting the estimated head-grade was expected to be low. This is confirmed by the box-plots in Figure 100 in which it is observed that the Kriging estimated stope head grades are in general more positive than all simulated stope grades (the Kriging estimated grade is higher than the maximum grade observed in all 25 simulations).

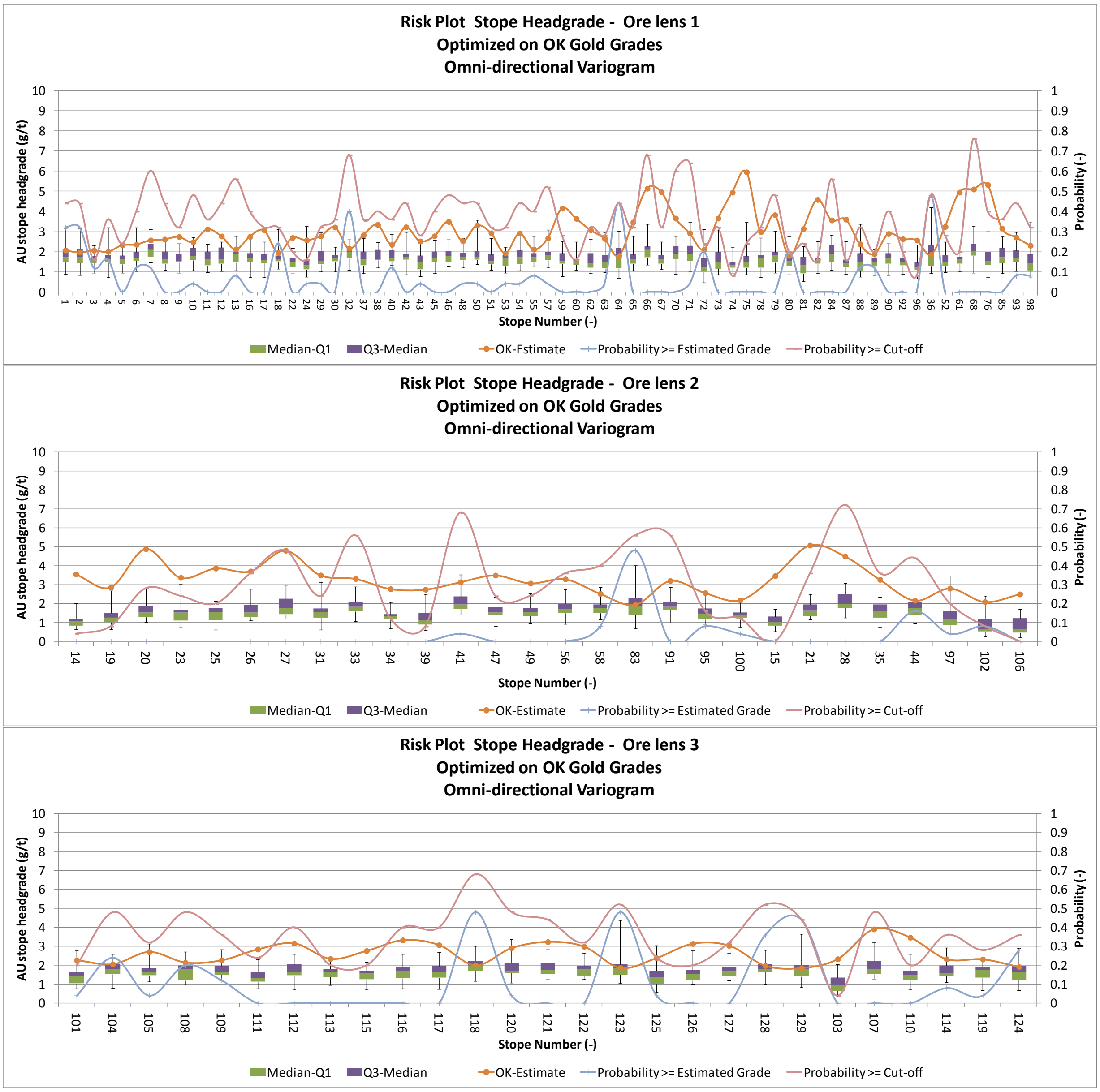


Figure 101 - Stope-to-Stope confidence level box plots (Omni-directional Variogram Model).

The difference between estimated and simulated grades can in this case be explained by the short variogram range used in estimation and simulation. Whilst the use a short variogram range does not prevent the occurance of the grade smoothing effect present in ordinary Kriging estimation, this effect is avoided in block model simulations. Due to the wide drill hole spacing compared to the variogram range, there is virtually no constraint to possible ‘equally true’ block model realizations (the possible grade simulations between measured grades are unconstraint as there is only a correlation between grades over a very short distance). This results in a wide range of possible grades for each cell in the block model with the median of the simulations converging to the average grade of the composite grade distribution.

It is concluded that an omni-directional variogram with a short range (in combination with a wide drill hole spacing) does not provide enough information to correctly estimate and/or simulate high and low grade areas within the orebody.

The omni-directional variogram was the only valid variogram found by the author after extensive variogram modeling but fails to describe the spatial relationship between grades to the extend where stope optimization is justified (the resolution is too low). The omni-directional variogram based resource model is only to be used as an, most likely conservative, estimate of the overall grade of the orebody and can be used to make decision on whether or not to proceed with exploration activities. Further exploration should provide the necessary data to determine spatial grade relationships (preferably directional if present) over longer ranges which will decrease the Kriging estimation error and will constrain the conditional simulation algorithm. Only then, high grade concentrations can be distinguished in the mineral resource.

As the omni-directional resource model proved to be non-suitable for stope optimization and related risk assessment it was decided to continue this study with the directional resource model to show the concept of including grade risk during the stope optimization process. It is assumed that the directional spatial relationships as determined by the Resource Department at Boliden do exist in the drill hole data although they were not personally reproduced by the author.

4.4.10 CASH FLOW ANALYSIS OF THE ESTIMATION BASED OPTIMIZATION

The Directional Estimated Model based optimized stopes as presented in Figure 99 (left) were connected with necessary infrastructure for life of mine scheduling and cash flow analysis. The resulting mine design is shown in Figure 102 and the according undiscounted yearly (cumulative) cash flows for the estimated and all simulated stope grades are presented in Figure 103.

The cash flow analysis emphasizes the project risk involved in the mining project when only the estimated model is used to develop mine plans. Based on the equally true simulations the small mining project proves to be of very high risk with only one simulation resulting in a positive cumulative cash flow on an undiscounted basis. Based on this cash flow analysis it is concluded that the project, optimized on the estimated block model, is not profitable.

There is too little information about the deposit resulting in a large smoothing effect when estimating grades in the mineral deposit. The optimization based on this smoothened model results in an over-estimation of the project profitability. The simulation model based cash flow results confirm that the project is unlikely to meet expectations whenever the decision would be made to start mining based on the current knowledge level.

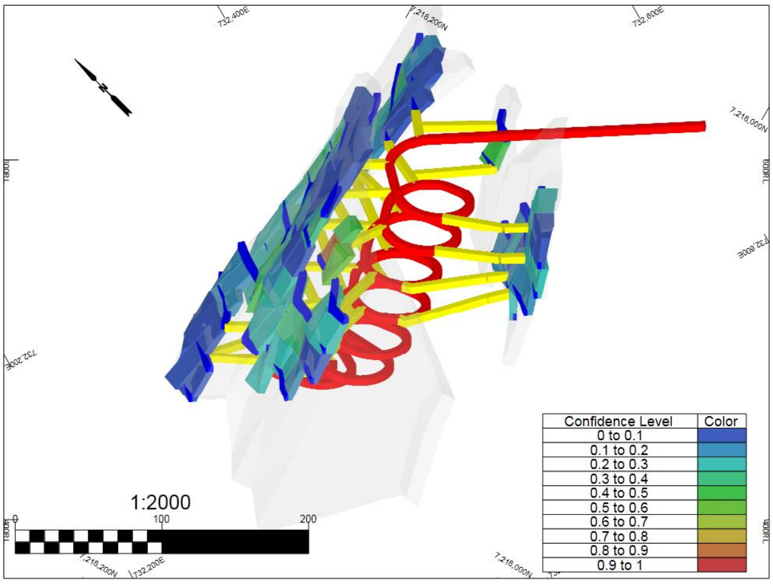


Figure 102 - Directional Estimated Model based stope designs and infrastructure (Au cut-off: 1.8 g/t).

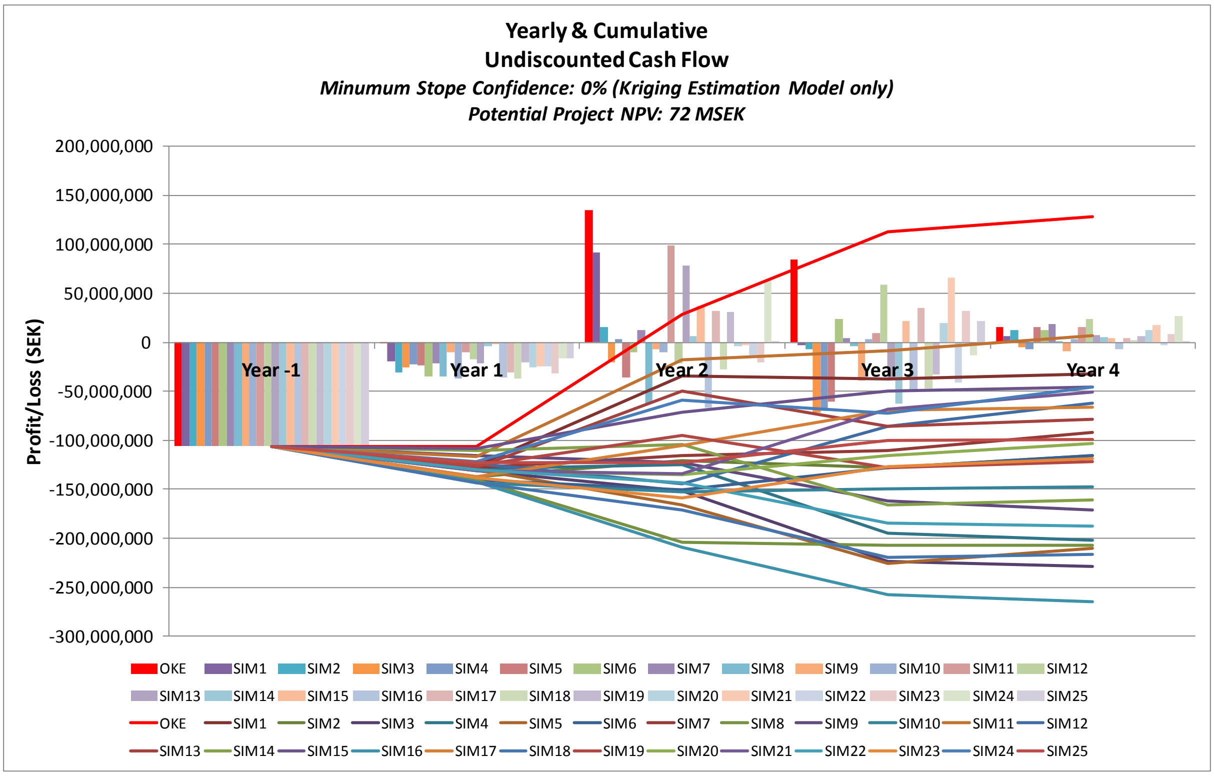


Figure 103 - Yearly (cumulative) Cash Flow based on the OK optimized stopes.

4.4.11 INCLUDING GRADE RISK IN STOPE OPTIMIZATION

In the previous section the stope optimization process was executed on the Ordinary Kriging Estimated block model and the resulting optimized stope shapes were evaluated against the simulated block models to assign a confidence level to each stope. As such the simulated block models were used only to back-analyze the confidence level of each stope meeting the estimated headgrade. This proved that the decision to optimize a mine design on the estimation model only is a high-risk decision, especially when resource confidence is low as a result of limited exploration.

It would be of great benefit if the estimated block model and the simulated block models could be used simultaneously when optimizing the stope shapes for a certain cut-off grade. By assigning a minimum confidence level to the final optimized stope prior to optimization, the risk of a stope head grade being below cut-off can be reduced.

The AMIRA Stope Optimizer can perform a stope optimization at a certain cut-off grade whilst considering the Kriging estimated model and simulated models simultaneously and as such optimize stopes at a certain minimum confidence level decided by the end-user. The final optimized stope will ‘at least’ be above the desired cut-off grade in a set percentage of the simulations. As an example; when optimizing stopes at a cut-off grade of 1.8 g/t and a minimum confidence level of 20% (based on 25 simulations) will result in stope shapes equal or above this cut-off grade in the estimated model as well as at least 5 of the 25 simulations. By subsequently increasing the minimum confidence level, the risk of under-performance of the optimized stopes decreases as more simulations have to meet the cut-off.

As a tradeoff for increased confidence, the number of stopes and stope tonnages are likely to reduce with increasing minimum confidence levels. The reduction of the stopes and tonnages is a quantification of the spatial grade uncertainty in the resource model. A confidence level of 100% can never be realized as a degree of uncertainty will always remain. This is a result of the spacing between drill holes and number of samples taken from drill holes. As such, a 100% confidence level would require an infinite number of drill holes which is unrealistic. The acceptable confidence level should be a trade-off between exploration costs and the cost of uncertainty (i.e. the range of possible project profit).

The AMIRA Stope Optimizer was used to re-optimize the underground stope designs at multiple minimum target confidence levels. The resulting optimized stope designs at 0%, 20%, 40%, 60% and 80% minimum target confidence are presented in Figure 104 (larger plots can be found in appendix C.5). It is observed that with increased confidence levels, the total number of stopes and consequently stope tonnage reduces rapidly (Figure 105). At a minimum confidence level of 60%, the stope configuration has reduced to an unmineable situation.

To show the possibilities of risk-based stope optimization and its impact on project profitability ranges the 40% confidence level was chosen for further analysis. Although this would still be considered a high risk optimization (stopes are accepted even if there is less than 50% chance for the stope to meet the cut-off grade) it is expected that the economic risk of the project has reduced to a certain extend compared to the optimized project based on the Kriging Estimated Model only (Figure 103).

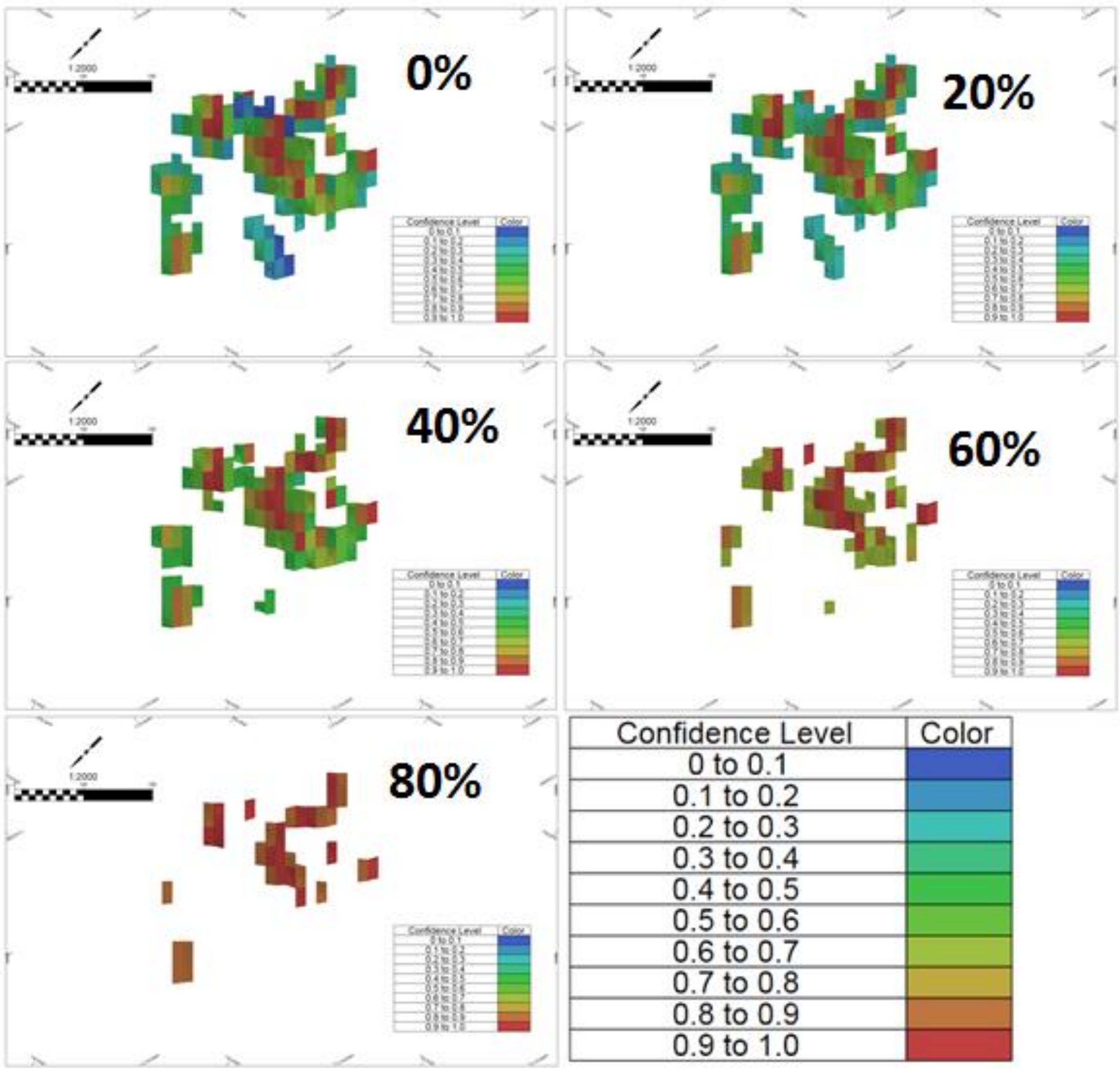


Figure 104 - Optimized Stope Designs at 0-80% minimum confidence level.

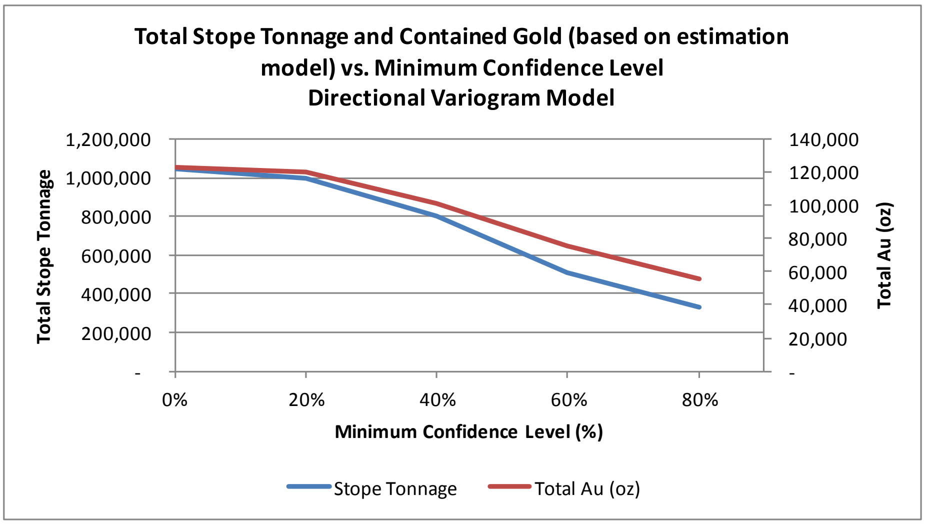


Figure 105 - Total Stope Tonnage and Average Grade with increased minimum confidence levels.

The optimized stopes at a minimum confidence level of 40% were connected with the required mine infrastructure in order to develop life-of-mine schedules and perform subsequent cash flow analysis. The resulting mine design is presented in Figure 106 and a summary of the project economics (cash flow analysis) is presented graphically in Figure 107.

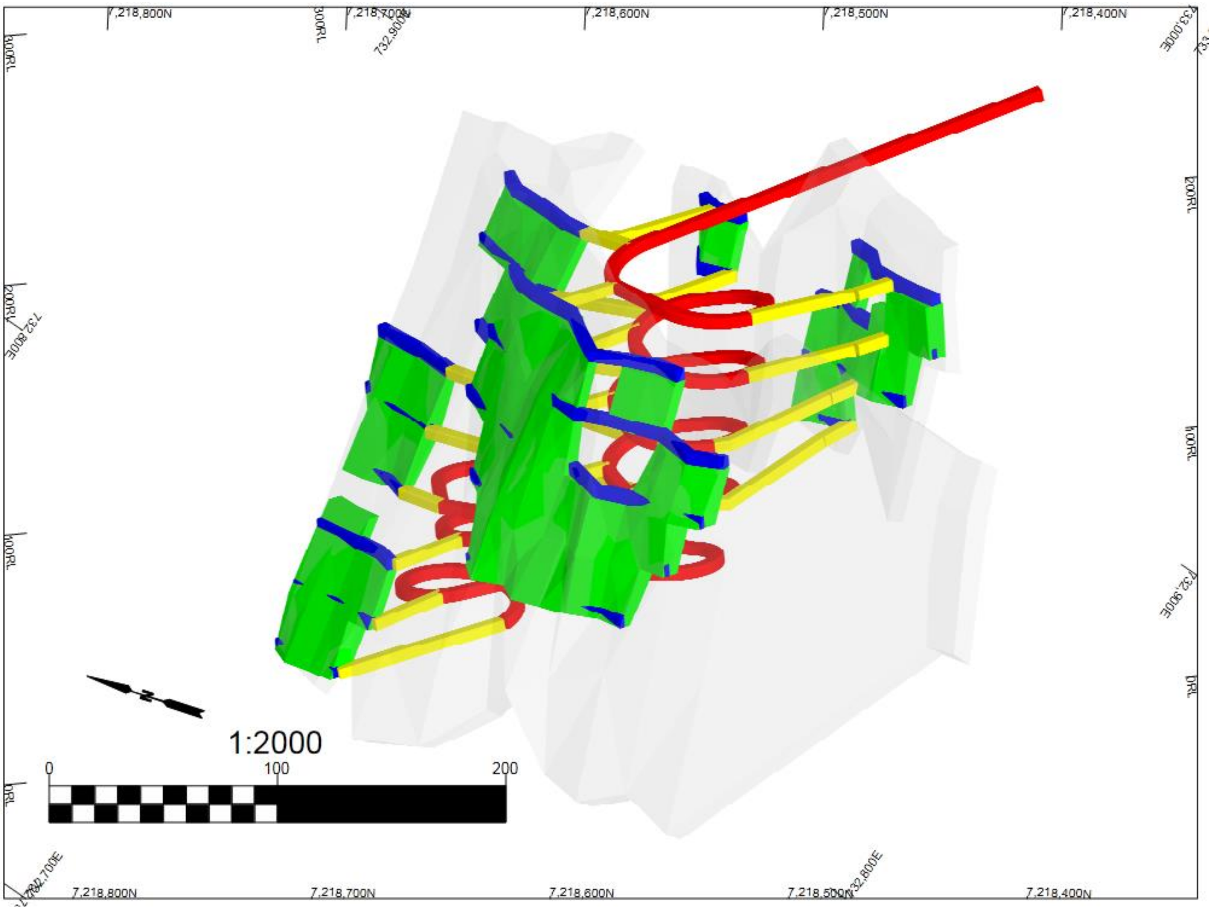


Figure 106 - Directional Model based stope designs and infrastructure (Au cut-off: 1.8 g/t @ 40% stope confidence).

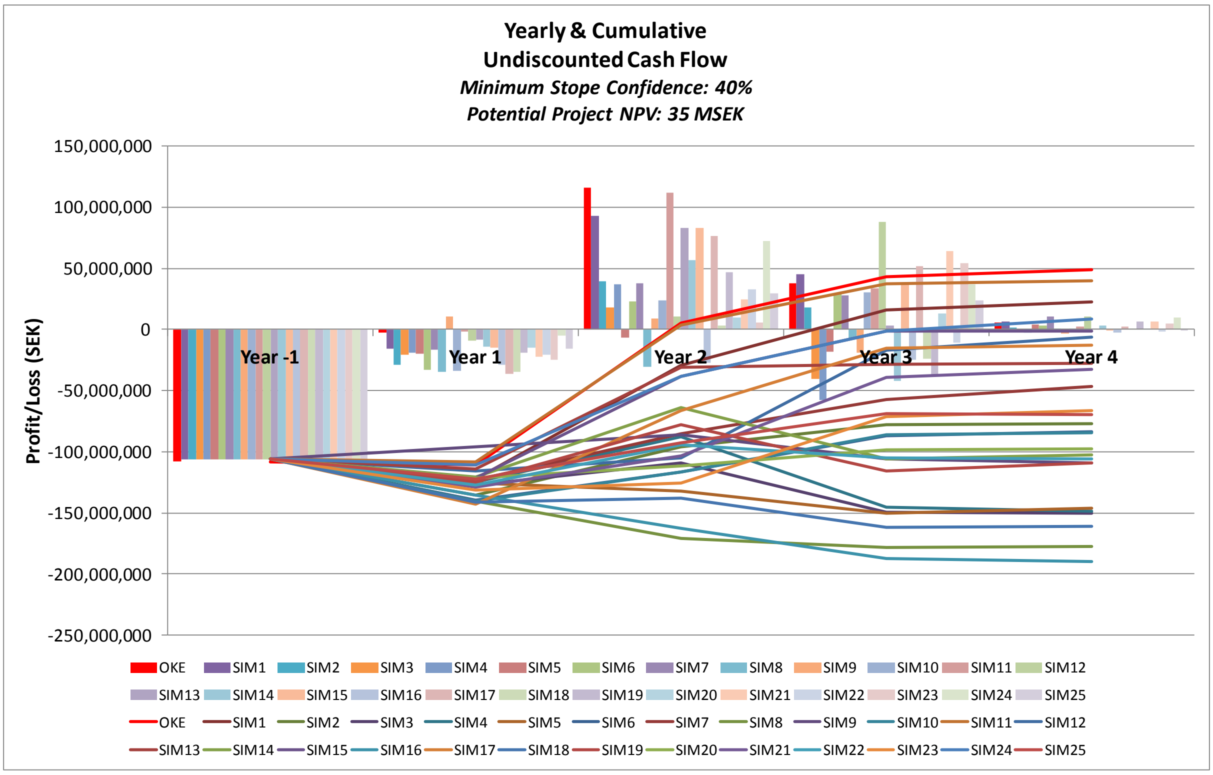


Figure 107 - Yearly (cumulative) Cash Flow based on Au cut-off: 1.8 g/t @ 40% stope confidence.

4.4.12 CONCLUSION

When the cash flow in Figure 107 is compared to the cash flow in Figure 103 the following is concluded:

Potential Project NPV

The Potential Project NPV (or the NPV based on the Estimated block model) has reduced by circa 50%. This is caused by the reduced number of stopes and hence reduced total stope tonnage and total producible gold content. If one would base project decisions on the estimated model only and stope confidence would not be considered, clearly the first strategy (resulting in a potential project NPV of 72MSEK compared to 35MSEK of the second strategy) would be preferred.

Risk Consideration

Together with the cash flow based on the estimated block model, the cash flows resulting from each of the simulations for the same project strategy are presented in Figure 103 and Figure 107. Two conclusions can be made on the observations of the graphs, namely;

The risk involved in the first project strategy (at a minimum confidence level of 0%, the estimated model only) is extremely high. Virtually all the cash flows resulting from 25 simulations of the same mineralization result in an undiscounted project loss (upto -265MSEK) with only one of the simulations resulting in an undiscounted profit of 7MSEK. Clearly this should result in the decision not to proceed with the project.

The second project strategy, for which a 40% minimum confidence level of meeting the stope cut-off grade of 1.8 g/t was considered, shows a slightly better result. By introducing a confidence level in the optimization process, stopes are optimized by simultaneous consideration of the 25 simulated models in addition to the estimation model . Stope shapes are adapted to satisfy the confidence conditions in all simulations and if this is not possible, are removed from the project.

The increased grade confidence is therefore expected to have a positive impact on project economics which is confirmed when analyzing the cash flow in Figure 107. Besides the positive project profitability based on the estimated block model, 3 out of the 25 simulation based cash flows also result in an undiscounted project profit. Based on 25 simulations this would mean that the project has a 12% chance of resulting in an undiscounted project profit which is an improvement compared to the first project strategy. Clearly this is still considered a high risk project (as was expected due to the low confidence level of 40%) and should still result in the decision not to proceed with the project. However, the strategy of implementing grade risk early in the mine optimization process by creating simulated block models proves to be valuable in the quantification of project risk.

4.4.13 MINERAL RESOURCE CONFIDENCE

The quantification of grade risk by creating a series of simulated block models in addition to the kriging estimated model proved to be valuable in the mine optimization process. The application of a new mine optimization strategy revealed that the optimal mine plan as found in a previous study was in fact based on a too optimistic grade interpolation of the deposit. Data analysis, variogram modeling and subsequent simulation revealed that insufficient exploration data is available to locate high grade zones within the deposit. Therefore some recommendations are proposed in this section to increase geological knowledge to a level that allows for mine optimization.

4.4.13.1 THE JORC CODE

Figure 108 shows the general relationship between exploration results, mineral resources and ore reserves according to the JORC code (Joint Ore Reserve Committee, 2012). The JORC code is the Australasian Code for reporting of exploration results, minerals resources and ore reserves and provides guidelines for the classification of a mineral deposit.

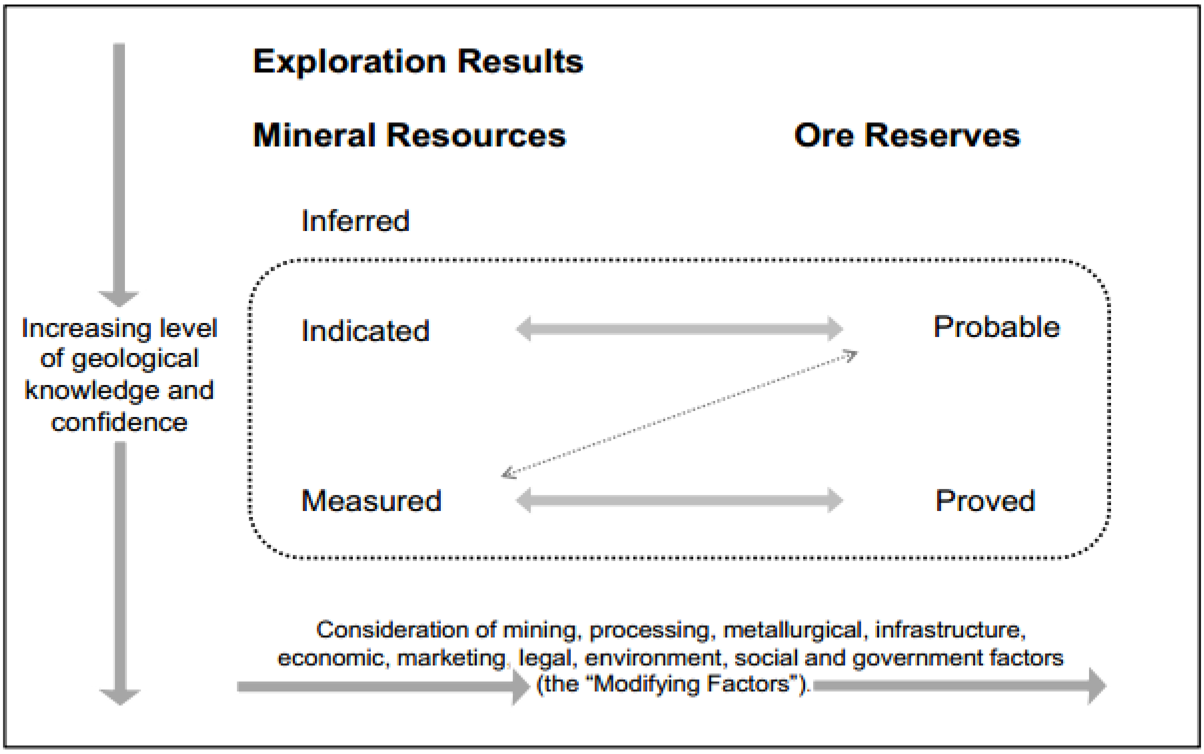


Figure 108 - General Relationship between Exploration Results, Mineral Resources and Ore Reserves (Joint Ore Reserve Committee, 2012).

It is important to note that exploration results can only be converted into inferred, indicated and measured resources depending on the level of geological knowledge and confidence whilst only indicated and measured resources can be converted into ore reserves by the application of the 'modifying factor'. The modifying factor is referred to as a detailed technical and economical study ((pre-) feasibility study). Inferred resources are considered to have a too low level of geological knowledge for a technical and economic study to be carried out in order to convert the resource into a reserve. Guidelines as to what is considered a sufficient level of geological knowledge are provided but not quantified (e.g by a maximum drill hole spacing, number of samples, etc.).

4.4.13.2 CURRENT ÄLGTRÄSK RESOURCE CLASSIFICATION

Based on the JORC guidelines to resource classification as provided in appendix C.6 the Älgträsk deposit can at best be considered an Inferred mineral resource.

The Älgträsk mineral resource quantity and grades are estimated based on limited geological evidence and sampling (a large drill hole spacing and limited sample database). Based on the nature of the deposit, geological continuity can reasonably be assumed (thin tabular deposit extending along strike at surface and picked up in drill holes at increasing depth). Grade continuity is implied (the assumption as used by Boliden Mineral AB on directional grade correlation) but could not be verified by the author (only an omni-directional variogram was produced).

The techniques used for exploration (Boreholes and RC holes) are expected to be carried out with care implying that the available data is correct and could, when additional exploration becomes available, be used to upgrade the inferred mineral resource into an Indicated mineral resource.

The JORC Code statement that inferred mineral resources cannot be converted in ore reserves by application of the modifying factor was also observed in the risk analysis of the Älgträsk deposit. The level of knowledge was concluded to be too low for an acceptable technical and economical study of the deposit.

4.4.13.3 IMPROVING THE ÄLGTRÄSK GEOLOGICAL KNOWLEDGE AND RESOURCE CLASSIFICATION

The following excerpt was taken from the JORC code (Joint Ore Reserve Committee, 2012) and describes the requirements for the classification of part of a mineral resource as Indicated.

> 22. **An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade (or quality), densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.**
>
> **Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes, and is sufficient to assume geological and grade (or quality) continuity between points of observation where data and samples are gathered.**
>
> **An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Ore Reserve.**
>
> *Mineralisation may be classified as an Indicated Mineral Resource when the nature, quality, amount and distribution of data are such as to allow confident interpretation of the geological framework and to assume continuity of mineralisation.*
>
> *Confidence in the estimate is sufficient to allow application of Modifying Factors within a technical and economic study as defined in Clauses 37 to 40.*

The classification of part of a mineral resource as Indicated is only possible when quantity, grade, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of the modifying factor (technical and economical studies) to evaluate the economic viability of the project. The geological evidence as mentioned should be derived from sufficient and reliable exploration data. The exploration data should be sufficient to assume geological and grade continuity between sampled locations.

Based on this classification guideline it is clear that increased geological knowledge is required to upgrade the Älgträsk Inferred Mineral Resource into an Indicated Mineral Resource. The following measures are proposed to upgrade the knowledge level to a level for which the mineral resource can be reported as an Indicated Mineral Resource that allows for a detailed technical and economical study:

Additional Exploration Drilling

Multiple Resource Estimates were carried out for the Älgträsk area of interest mineral deposit. Depending on the variogram model and estimation approach used, some variability between the ore lenses is observed whilst the total tonnage and grade for all lenses is considered similar for all three models with the Boliden estimate being slightly more positive.

The Directional Variogram Model and Omni-direction Variogram Model were assessed for spatial grade uncertainty by creating a series of equally true realizations of the spatial grade distribution within the orebody. Only an omni-directional variogram was reproduced by the author based on the available exploration drill hole data and the subsequent 'equally true' simulations showed a high variability in spatial grade interpretation. This emphasizes the limited knowledge of the deposit and hence its Inferred Mineral Resource status.

To research the impact of using directional variograms to model spatial grade correlation and quantify the spatial grade risk, it was assumed that the directional variograms are a valid description of spatial grade correlation. The subsequent 'equally true' simulations showed significantly less variability which emphasizes the necessity to model directional grade continuity for the deposit in order to upgrade the it to an Indicated Mineral Resource.

In order to validate the assumed variograms for directional grade correlation, exploration drilling should be intensified (more sample data for variogram modeling) and the drill hole spacing, which is in many cases over 50 metre (almost the assumed variogram range), should be reduced in order to correctly model the spatial grade correlation over the full variogram range.

Once the directional correlation is correctly modeled by means of variograms, stochastic simulations of the ore lenses (in combination with the estimated resource model) can be used to pinpoint areas with high grade variability that need further infill drilling to increase spatial grade confidence.

The process to increase geological knowledge as proposed above requires a significant amount of new exploration drilling. The question to be asked is whether the costs of additional exploration are justified by the existing knowledge of the deposit. The author believes that based on the available exploration data (although limited) there is no justification for such additional drilling yet due to the limited size of the delineated Area of Interest Deposit (3.1 Mt), the relatively low average grade and marginal potential project profitability (based on the positive Estimated resource model).

The author proposes that before any infill drilling is carried out, the potential of the deposit is investigated. A limited series of holes are proposed to define the extend to depth of the ore lenses as this could potentially result in a significant increase of the mineral resource size.

Estimated exploration drilling costs are presented in Table 37 and proposed drill holes are presented schematically in Figure 109. The target of the proposed drill holes is to identify the vertical extend of ore Lens 1 over the next 100m as this is the biggest ore lens within the area of interest. To do this it is proposed to drill two holes that potentially intersect Lens 1 at increased depth (these holes are seperated 50m vertically) . The hole lengths are extended to investigate the possible vertical extend of ore Lens 2. It is proposed that this series of holes is repeated along strike over the full length of the area of interest (450m) at intervals of 50 meter. This results in a total of 18 drill holes.

Exploration Method	Costs
Rock, RC or Diamond Sampling	200 SEK/sample
RC Drilling (max length 300m)	440 SEK/m
Diamond Drilling (max length 1000m)	900 SEK/m

Table 37 - Average Exploration Drilling Costs (After Rudenno, 2012), Costs converted from AUD to SEK at 1AUD=5.87SEK and rounded off to 10's of SEK.

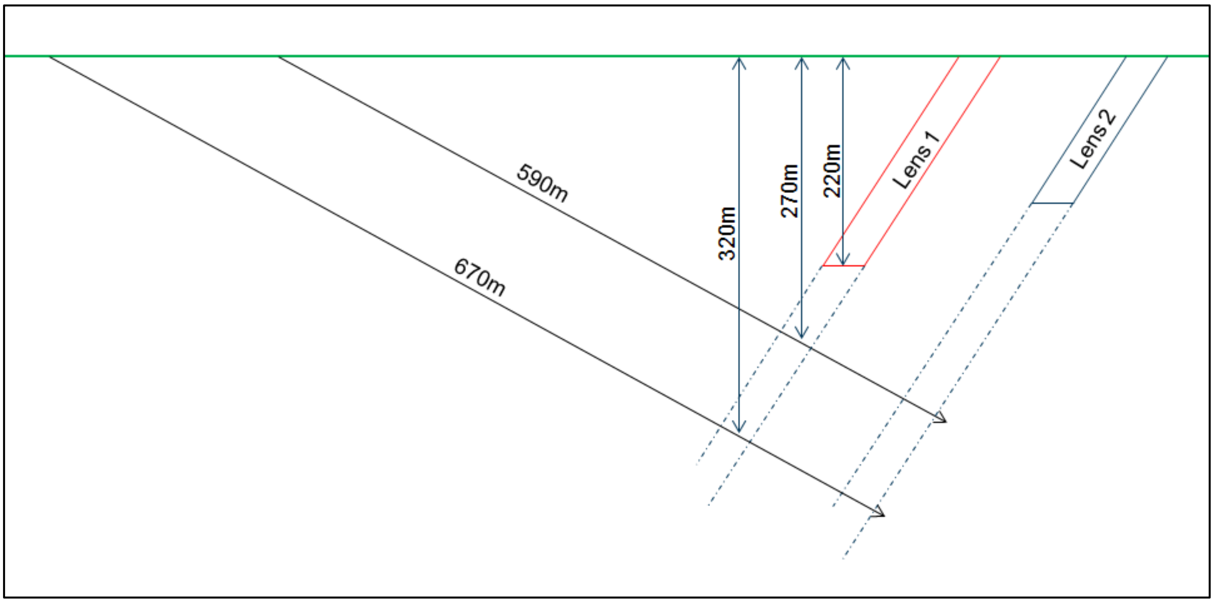


Figure 109 - Schematic representation of proposed drill holes.

Based on Figure 109 and the exploration drilling costs in Table 37 an estimate of the additional exploration costs was calculated. The resulting costs for a ’shallow’ (270m vertical depth) and ’deep’ (320m) drill hole are presented in Table 38.

Description	Hole length (m)	Drilling cost (SEK)	Sampling Cost (SEK)	Total Costs (SEK)
Shallow	590	531,000	4000	535,000
Deep	670	603,000	4000	607,000

Table 38 - Cost Breakdown for Shallow and Deep drill holes.

For 9 ’shallow’ and 9 ’deep’ drill holes the total exploration costs add up to 4.8MSEK and 5.5MSEK respectively. The exploration campaign should target the shallow drill holes first and if succesful, the deep holes should be drilled. Only if these exploration drill holes prove the orebody to be significantly larger than previously assumed and measured grades are satisfactory, the directional spatial correlation between measured data in drill holes should be investigated by reducing the drill hole spacing. The focus should in that case be the deeper part of the orebody where drill hole spacing is currently non-sufficient for resource estimation. It is also proposed to drill a series of holes inside and with the same dip as the orebody as this will provide many datapoints required for variogram modeling. The costs of these holes will depend on the results of the resource expansion drilling campaign.

5 CONCLUSIONS

The aim of this research project was:
'To introduce and evaluate an optimization approach for sublevel stope mine optimization using the AMIRA Stope Optimizer that includes grade-risk quantification and can be used by Boliden engineers for future mine planning and economic assesment.'

This aim was achieved by subsequent research of the following objectives:

1. Evaluate the AMIRA Stope Optimizer capabilities and limitations
2. Develop an underground mine optimization process in which the AMIRA stope optimization software is integrated and test the process on one of the company's mineral deposits.
3. Investigate the possibility to quantify grade-risk in the optimized stope design and evaluate its' impact to the profitability of a mining project.

The conclusions to the research project are presented here according to this same structure;

1. EVALUATE THE AMIRA STOPE OPTIMIZER CAPABILITIES AND LIMITATIONS

The AMIRA Stope Optimizer was evaluated to define its capabilities and limitations.

The following was concluded based on the evaluation of the AMIRA Stope Optimizer:

- The Stope Optimization algorithm functions best when the optimization constraints are kept as loose as possible. The optimization algorithm should be given 'as much freedom as possible' to find the optimal stope shape solution. In practice this means that the maximum geotechnically and practically allowable stope sizes and angles should be used as the optimization constraints

- When the optimized stopes are compared to manually designed stopes for the same sublevel the Stope Optimizer results are considered satisfactory. However, the optimized stopes are not an exact match because:

  - The stope optimization algorithm applies the same strict set of design rules to each stope whereas an engineer can apply slightly different constraints to each individual stope.
  - The stope optimization algorithm evaluates the smallest mining unit (a slice) against the cut-off grade and builds up the the stope by combining the best combination of slices. An engineer would design a full stope shape and as such evaluate the stope head grade to the cut-off grade. The stope optimization algorithm is therefore concluded to be more accurate.
  - The stope optimization algorithm cannot follow the true geological or grade boundaries. The optimization of the final stope shape is limited to the top and bottom corner points of the seed shape and for instance 'rounded' stope shapes can therefore not be designed. As such, optimized stope shapes should always be validated against the block model before the stope is taken into production.
  - The stope optimization algorithm optimizes stopes individually without taking the surrounding stopes into consideration as an engineer would do whilst designing stopes manually. This can result in a technically non-feasible set of stopes. The stope smoothing functionality proved to be a good tool to line-up adjacent stopes to make the overall design more feasible.

Although some limitations to the optimization algorithm were observed, overall the stope optimization software is able to find the economic zone within a mineral resource and design optimized stopes at a user defined cut-off to a level of detail where they can be used for strategic mine planning. The main advantage of the stope optimization software is the reduced time spent to design stopes at different cut-off grades. Where the manual design of stopes at a specific cut-off grade can take days up-to weeks, the AMIRA Stope Optimizer can deliver results (suitable for strategic planning) within a matter of hours or days.

2. DEVELOP AN UNDERGROUND MINE OPTIMIZATION PROCESS IN WHICH THE AMIRA STOPE OPTIMIZATION SOFTWARE IS INTEGRATED AND TEST THE PROCESS ON ONE OF THE COMPANY'S MINERAL DEPOSITS.

- *General Concept*

The goal in the optimization of mining projects is to maximize profit and is expressed in terms of the Net Present Value. The work of Lane (Lane, 1988) proved that there is a relationship between cut-off grade and the NPV in a way that there is a cut-off grade for which the Net Present Value of a mining project is maximizes.

Although Lane's mathematical theory is validated in theory (based on the grade-tonnage curve of a deposit only), its practical application is difficult as more factors have to be considered (e.g. the changing spatial location and concentration of metals in a deposit and resulting development and production factors) that cannot be quantified in mathematical equations.

- *The Mine Optimization Approach*

There is no mathematical solution to the underground mine optimization and the process of project optimization can therefore be considered an iterative process as presented in Figure 110.

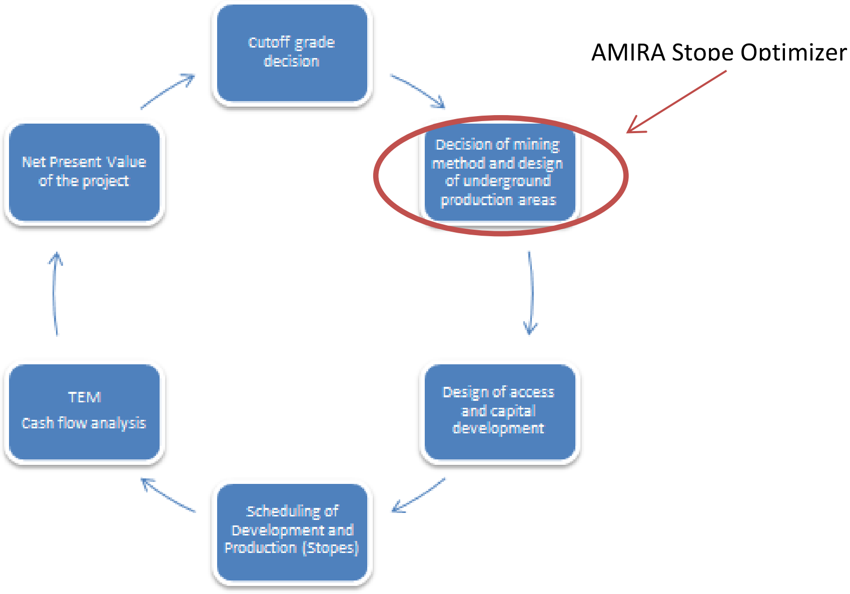


Figure 110 - The iterative underground mine optimization approach.

Due to the fact that all subsequent optimization steps as presented change with the selection of a (new) cut-off grade, the process of underground mine optimization can be a time consuming process. It is for this reason that many mines are operating at a cut-off grade equal to the break even grade. It is unlikely that the break-even grade is the right cut-off grade that maximizes NPV.

By implementing the AMIRA Stope Optimizer in the underground optimization process, the total time to complete one iteration is significantly reduced.

- ***Boliden Optimization Approach***

This proposed underground mine optimization process is considerably different from the mine optimization approach that is the current state-of-the-art practice at Boliden in that the Boliden approach does not allow for the evaluation of different cut-off grade scenarios. Instead, a cut-off grade (break-even estimate) is applied to raw drill hole data and the orebody delineation and grade estimation focuses only on the economic (above cut-off) part of the deposit. This limits the possibilities for mine optimization.

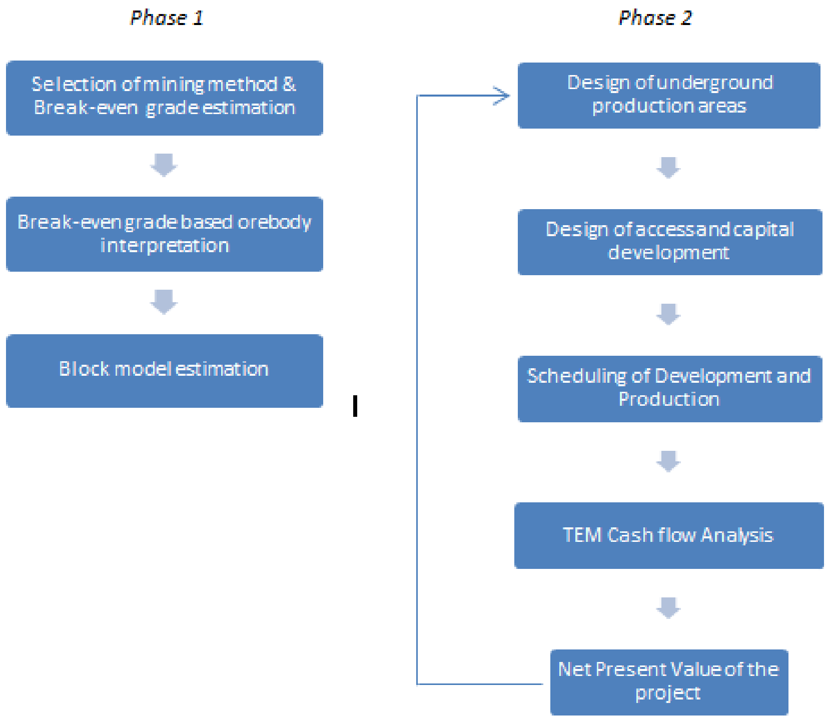


Figure 111 - The current Boliden approach to mine optimization.

By application of a cut-off in raw drill holes, valuable information about the deposit is lost. Whenever first assumptions change (e.g. change of mining method, metal prices, mining costs, processing costs etc.), the cut-off grade will change as well. If the cut-off grade drops, this means that the whole planning process has to be re-executed starting from the first step of *Phase 1*.

The current approach to mine optimization at Boliden does not allow for the succesful implementation of the AMIRA Stope Optimizer. The stope optimization software should be used to find the economic zone and design optimized stopes in an unconstrained (in terms of cut-off value in raw drill holes) resource model. By evaluating stope designs at a range of cut-off grades and performing subsequent mine design and economic evaluation it is possible to find the optimal mine design that will maximize project profitability.

- ***The Älgträsk Underground Study***

The AMIRA Stope Optimizer was successfully integrated into the mine optimization process and a case study was undertaken to validate the process on one of Boliden's mineral deposits (The Älgträsk Underground Study).

A break-even calculation provided the first estimate of the cut-off grade and the AMIRA Stope Optimizer was used to design feasible stopes for a range of cut-off grades. All subsequent steps in the iterative mine optimization process were carried out and the optimum strategy, resulting in a project NPV of 31MSEK, proved to be following scenario:

Cut-off Grade:	1.8 g/t
Production Rate:	1500tpd
Development Rate:	60m/mo
Processing Scenario:	Gravity and Flotation of Low As ore and Gravity and Leaching of High As ore (Scenario 3).

It was concluded that although the process of cut-off grade optimization proved to be a time-consuming process, it did increase the project NPV by circa 35 % compared to the 1.6g/t break-even estimate. The use of the AMIRA Stope Optimizer significantly reduced the time required for stope design and proved to be a valuable tool in strategic mine planning as it allows the engineer to rapidly create mine designs.

The Algrtrask underground project proved to be a very marginal mining project and of high risk due to the required large investments compared to the potential profit and life of mine. The project is extremely volatile to the gold price and with a life of mine of less than 4 years from start to finish, the risk inherent in the fluctuation of the gold price over time cannot be minimized.

3. INVESTIGATE THE POSSIBILITY TO QUANTIFY GRADE-RISK IN THE OPTIMIZED STOPE DESIGN AND EVALUATE ITS' IMPACT TO THE PROFITABILITY OF A MINING PROJECT.

With the optimization process introduced and validated on one of Boliden's mineral deposits it was decided to research a way to quantify the amount of risk involved in the defined optimum. Multiple types of risk were considered after which 'grade-risk' was selected for investigation as it is the risk of grade uncertainty in a mineral resource, that will affect all further optimization steps. A mineral resource with a high grade-risk, will result in a high risk of stopes not meeting the estimated grade and as a result of that, the project profitability is endangered.

Estimation vs. Simulation

Estimation algorithms such as Kriging type algorithms tend to smooth locally measured grades over the area surrounding the measured point. As a result of this effect, lower grades tend to be over-estimated whilst higher grades are generally under-estimated. Although the overall grade in the orebody may be interpolated correctly by the Kriging algorithm, the smoothing effect is a serious concern when trying to pinpoint high grade areas in an orebody and define areas to mine or leave in place.

Stochastic simulations give a better representation of the grade distribution within mineral deposits and can be used to quantify grade uncertainty. Stochastic simulations are not affected by the smoothing effect and both the grade distribution as well as the spatial relationships between grades as measured in the drill hole data are resproduced in the simulated resource models. By creating multiple 'equally true' realizations of the mineral resource model the grade-risk present in the mineral resource can be quantified by calculating the conditional variance (Figure 112). A higher conditional variance indicates a larger grade uncertainty (larger grade-risk).

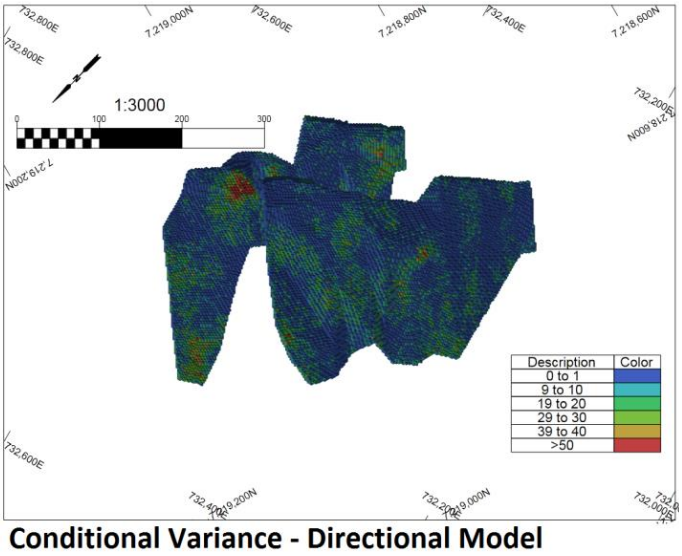


Figure 112 - Conditional variance (directional model) Algtrask underground study.

A modified version of the underground mine optimization approach was proposed (Figure 113) in which the AMIRA Stope Optimizer is used to optimize stopes whilst simultaneously considering the estimated as well as the simulated resource models. By requiring the stope to be above cut-off in the estimated as well as a percentage of the simulated models, the risk of stopes not meeting the minimum cut-off grade is reduced.

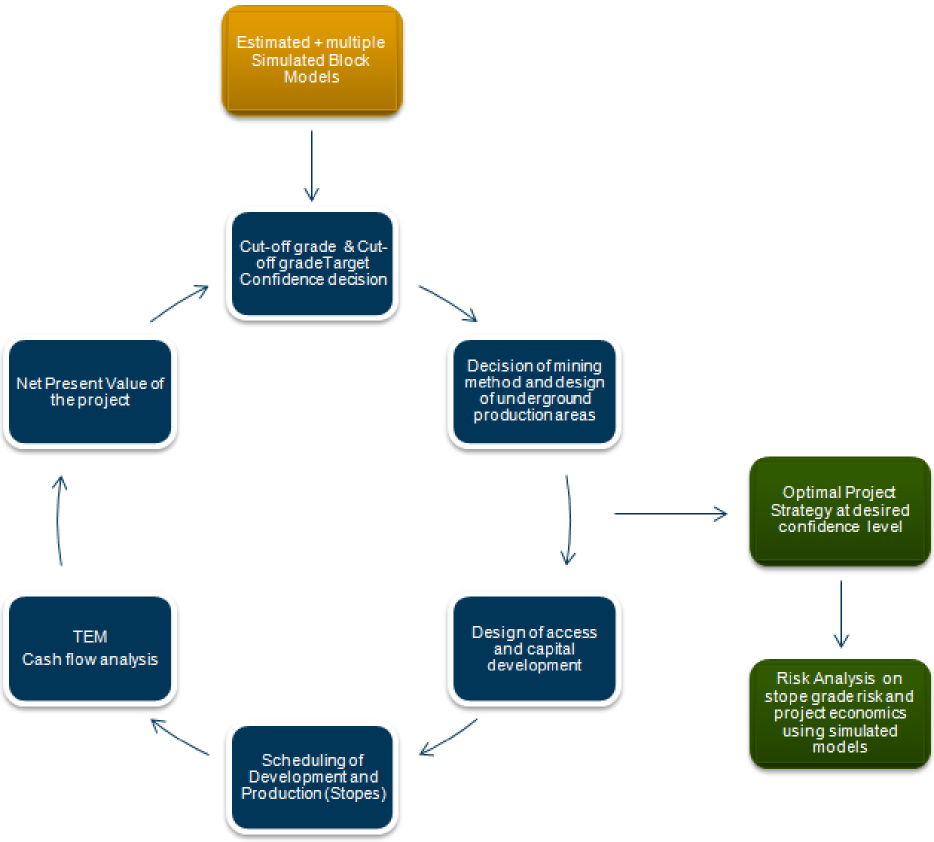


Figure 113 - The mine planning approach using resource estimation and simulation models simultaneously.

- ***The Älgträsk Underground Study***

The risk based mine optimization approach was compared to the general optimization approach by a case study evaluating the Boliden Älgträsk mineral deposit. The full process from drill hole dataset to grade estimation and simulation was undertaken to obtain estimated resource models as well as simulated resource models based on the same variogram models and drill hole data.

Two sets of estimations and simulations were made based on the variogram model(s) used.

- Directional variograms with ranges as used by Boliden
- Omni-directional variogram with a range of 6m (the only variogram observed by the author)

First the risk of optimizing a mine plan on the estimated resource model only was investigated. Stopes were optimized on the resource model only and compared to the simulated models to assign a confidence level of the stope meeting the desired grade. The majority of stopes showed confidence levels of 0 – 30% which indicates that there is a high grade risk which will probably result in poor production figures if it was decided to start mining.

The economic risk of the estimation based optimization was evaluated by designing infrastructer, scheduling of development and production and subsequent cash flow analysis. This resulted in the cash flow presented in Figure 114.

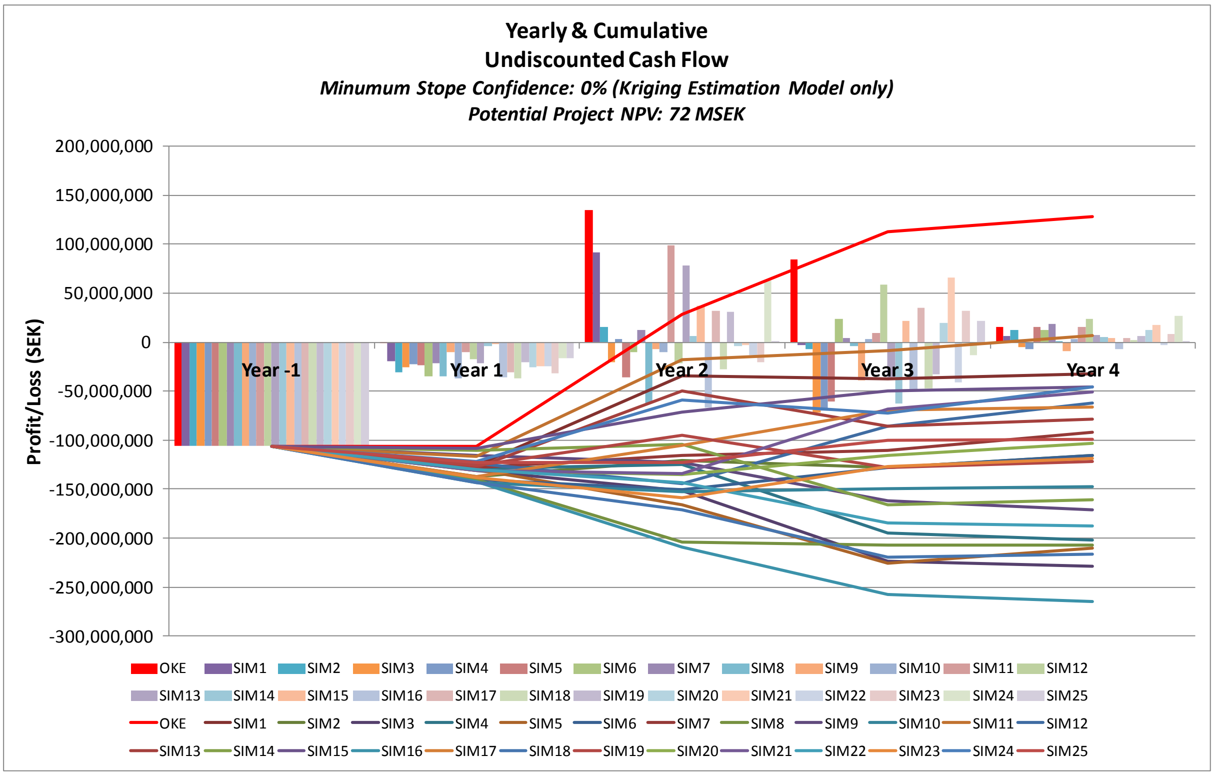


Figure 114 - Yearly (cumulative) Cash Flow based on the OK optimized stopes.

The cash flow analysis proved that the estimated resource model is likely to be too positive in terms of high grade areas within the deposit. The simulations indicated much lower stope grades resulting in stopes not meeting the expected cut-off grade and subsequent economic losses due to much lower smelter returns.

Secondly, the risk based mine optimization approach was evaluated in which the AMIRA Stope Optimizer is used to optimize stopes by simultaneosly considering the estimated and simulated block models. As a result of the consideration of multiple resource models, the number of stopes as well as the total stope tonnage reduced rapidly with increased minimal confidence constraints applied to the optimization process.

The case of 40% confidence (Figure 115) was evaluated as it was the maximum confidence level for which an acceptable number of stopes and tonnages was returned to design a mine, perform scheduling and evaluate the cash flow. It is observed that although based on the estimated model the potential NPV is reduced by circa 50%, the grade-risk is also somewhat reduced with 3 out of 25 simulations resulting in a positive cash flow.

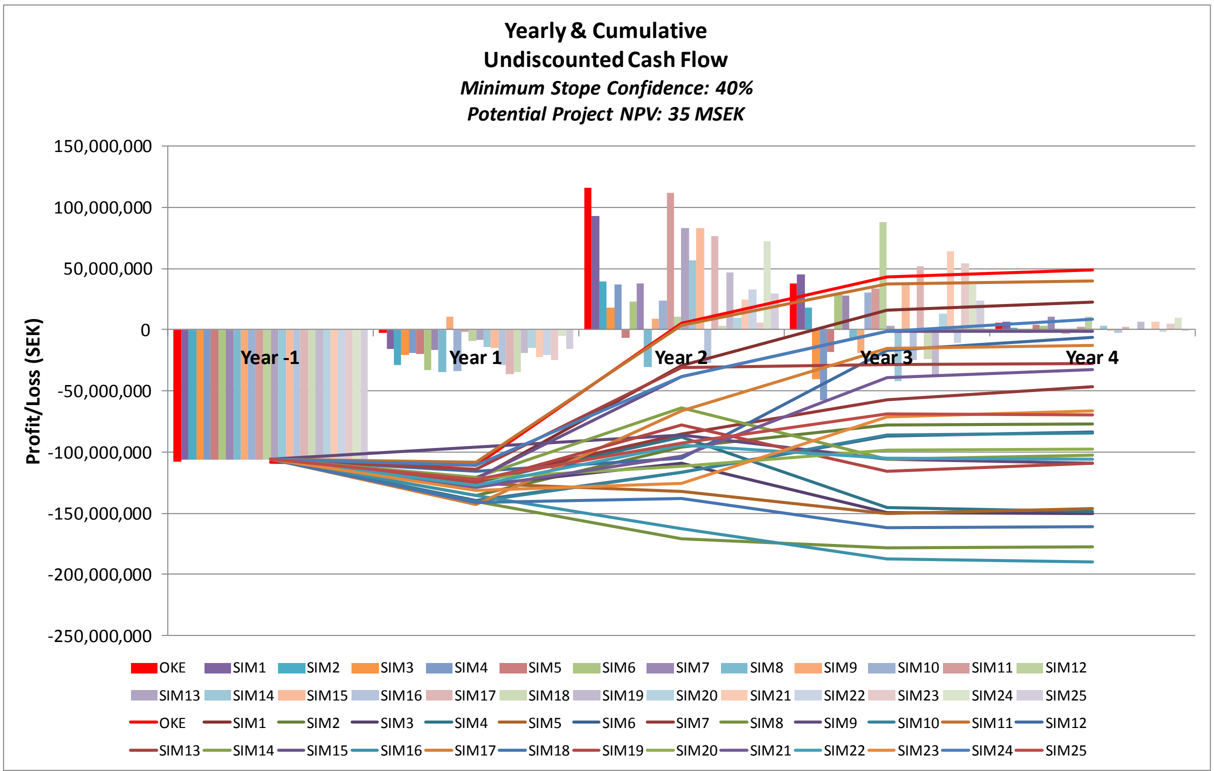


Figure 115 - Yearly (cumulative) Cash Flow based on Au cut-off: 1.8 g/t @ 40% stope confidence.

The risk based mine optimization approach was succesfully tested for the Älgträsk mineral deposit to reduce grade risk. It is concluded that due to the limited exploration drilling, the Kriging estimated resource model provides a too positive interpretation of the grade distribution in the orebody. This is accounted to the smoothing effect present in the estimation algorithm.

By creating a set of ’equally true’ simulations of the deposit it is possible to quantify this grade risk and asses the economic risk that is present in estimated resource model based optimized stope designs. When the estimated model and simulations are used simultaneously whilst optimizing stopes, it is possible to obtain optimized stopes at different levels of confidence. In case of the Älgträsk mineral deposit this resulted in a slightly reduced economic risk of the project strategy although the number of stopes and total stope tonnage was reduced as well.

6 RECOMMENDATIONS

Recommendations as derived from the research work performed are presented in this chapter. The recommendations are subdivided in recommendations for Boliden to succesfully implement the AMIRA Stope Optimizer and recommendations for further research in the field of underground mine optimization.

6.1 RECOMMENDATIONS – BOLIDEN

In order to succesfully apply the proposed optimization methods as proposed in this research project, some fundamental changes have to be made to the existing planning process. The recommended changes to the existing planning process can be subdivided into 3 main phases:

- Resource Modeling
- Cut-off Grade Based Mine Optimization
- Project Strategies Analysis and Risk Assessment

The recommendations to each step will subsequently be discussed:

6.1.1 RESOURCE MODELING

Recommendation: *Model orebodies on true lithological boundaries and create estimated and simulated resource models simultaneously to allow for the evaluation of multiple mining scenarios and quantify mineral resource model uncertainty.*

The most important proposed change in this phase is the delineation of the orebody based on lithology. By delineating the orebody on lithology rather than by applying a cut-off grade in the raw drill holes, the full mineral deposit is modeled and represented in the Datamine block model.

The second proposed change as opposed to current practice is that consideration should be given to start simulating block models besides estimation to quantify spatial grade uncertainty. The completed case studies as part of this research project have shown that the quantification of grade risk is a valuable source of information and can be used for multiple purposes.

1. The modeling of spatial grade uncertainty can help to identify locations for further exploration drilling.
2. Quantification of the risk of under-performance of the mine design based on an estimated resource model and subsequent quantification of economic risk.
3. Reduction of the risk of 'below expectations' performance of stopes by using the simulated models in conjunction with the estimated model in the AMIRA Stope Optimizer to optimize stopes at a certain minimum confidence level.

The output of this phases is a Datamine block model with an estimated grade attribute field as well as n *simulated grade attribute fields.*This model can subsequently be used in the *cut-off grade based mine optimization phase.*

The process steps to create the required resource model are presented in Figure 116. The loop (presented in red) denotes two stages in this phase where during which it should be decided whether or not more exploration drilling is required to increase resource knowledge.

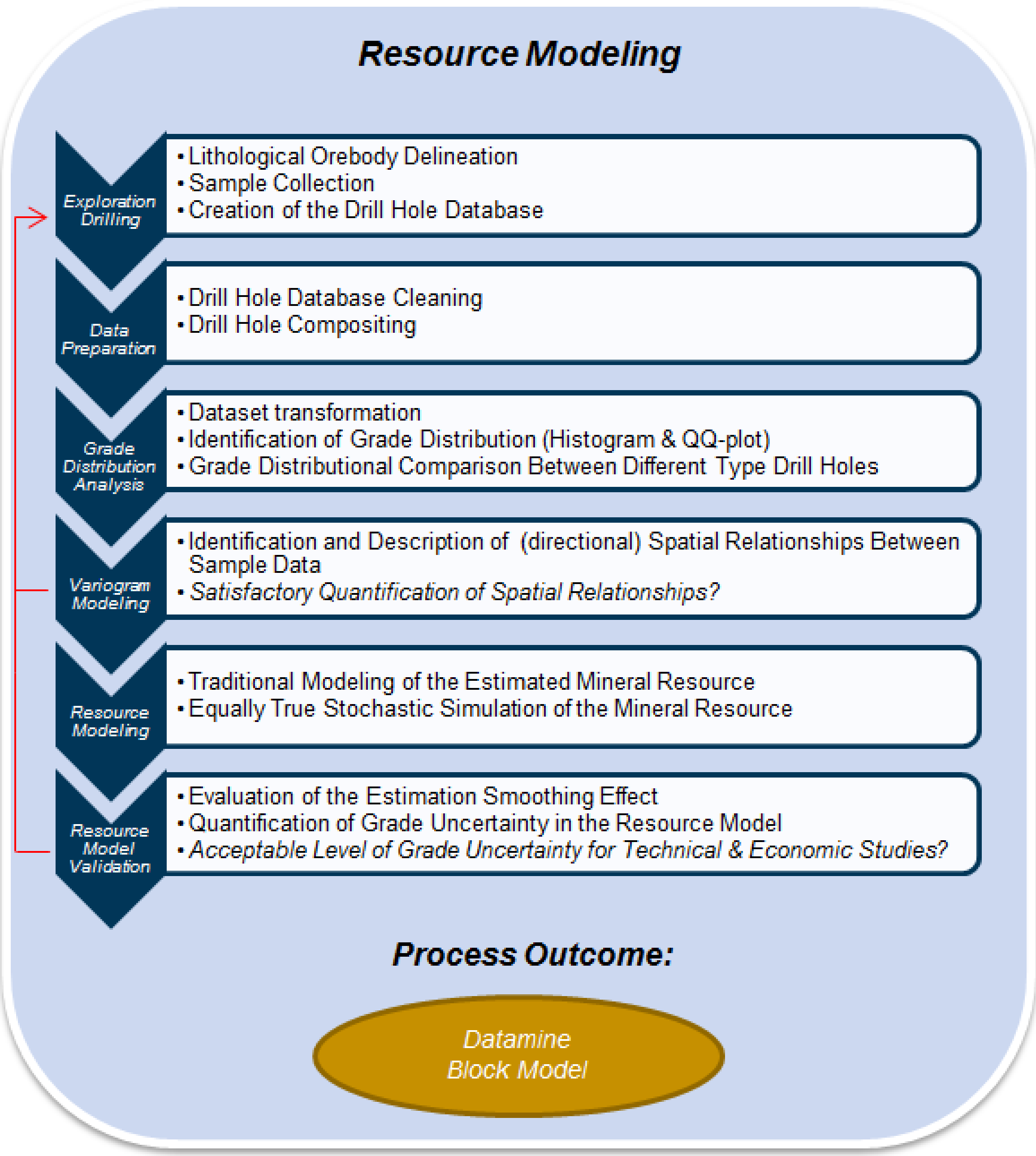


Figure 116 - Proposed Resource Modeling process.

6.1.2 CUT-OFF GRADE BASED MINE OPTIMIZATION

Recommendation: *Move the cut-off grade based optimization step from the Resource Geology Department to the Mine Planning Department and use the AMIRA Stope Optimizer to identify mineable areas within the resource model.*

It is important to notice that, because the resource model is now a representation of the complete orebody, not all material is automatically mineable as higher and lower grade areas are both present in the resource model.

The use of the AMIRA Stope Optimizer is proposed to be used to identify the economic zone within the resource model. As the resource model is no longer constrained by a cut-off grade, it is possible to investigate a series of cut-off grades and resulting stope designs. This is a valuable improvement as it is not longer required to re-interpret and re-create a new resource model before a new stope design can be optimized.

This allows for the evaluation of multiple what-if scenarios such as changing mining or processing costs, contractor costs, or changing metal prices. Theoretically, multiple scenario's could be created and adopted whenever first assumptions change.

By using the stope optimization software in this phase, little time is lost compared to the existing optimization approach as the time required by the optimization software to identify the economic zone and optimize stopes is easily offset by the time reduction in manual resource modeling and manual stope design. The focus of the engineer can be shifted more towards the scheduling and cash flow analysis resulting in the evaluation of more scenario's in the same period of time.

The cut-off grade based mine optimization phase is presented in Figure 117 which shows the iterative nature of the process. The process itself is based on the Datamine block model (the mineral resource model) that is obtained in phase 1. The output of the cut-off grade based mine optimization phase is a series of possible mine plans and related cash flow models.

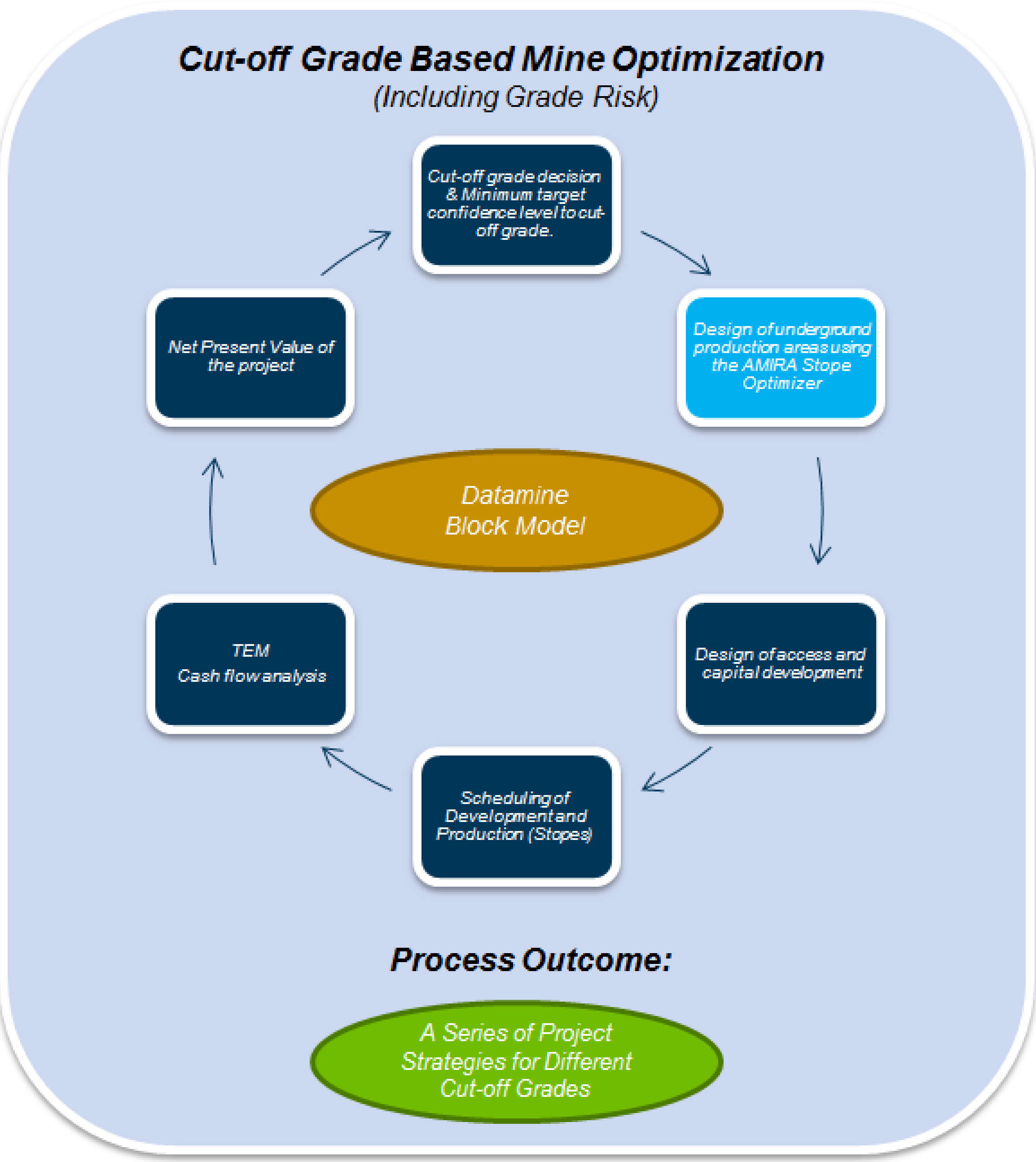


Figure 117 - Proposed Cut-off grade based mine optimization process.

6.1.3 PROJECT STRATEGIES ANALYSIS AND RISK ASSESSMENT

Recommendation: *Evaluate the obtained project strategies and grade&economic risk involved in each strategy to identify the best strategy or best next step.*

Although a level of uncertainty will always remain in the mineral resource and subsequent mine design, it is important to define the acceptable level of uncertainty. The process to decide on acceptable grade-risk is presented in Figure 118.

The confidence level of individual stopes is determined and can be used to decide whether a stope should be mined or not. For instance, a stope with a lower confidence level may be acceptable to mine if it is surrounded by stopes with a high confidence level whilst the decision not to mine a low confidence stope may be taken if it requires a large amount of additional infrastructure development to access the stope. Additionally, areas of high uncertainty in the resource model that were not recognized in the resource modeling phase (phase 1) are easily observed in when analyzing the stope confidence levels. Comparison of the stopes and the mineral resource block model is therefore essential during this stage and may result in the decision to increase the geological knowledge by means of further exploration (going back to phase 1).

In order to convert grade risk into economic risk it is proposed to analyze the cash flow model based on the estimated resource model with the cash flow models resulting from the equally true simulations of the mineral resource model. Instead of obtaining a single estimated project Net Present Value, a range of equally likely NPV's is obtained that can be used to decide to proceed with the project or to go back to phase 1 or 2. In the most extreme case when the project strategy turns out to be of very high risk and the potential profit is marginal, the decision may be made to discard the project completely.

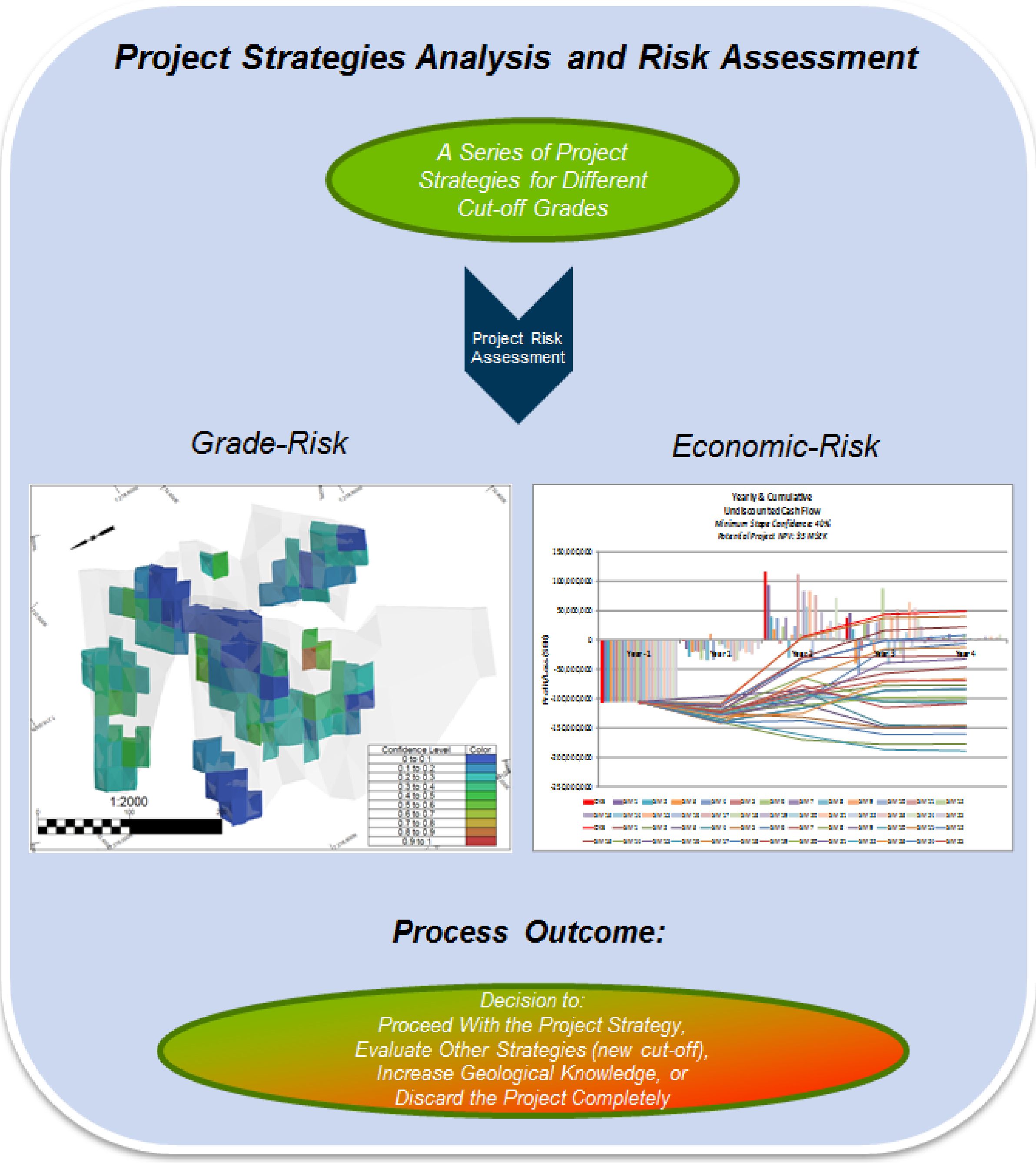


Figure 118 - Proposed project strategies analysis and risk assesment process.

6.2 RECOMMENDATIONS – FURTHER RESEARCH

- ***Perform a detailed cost study on mining OPEX***

In this research project all mining OPEX were based on company estimates or similar mining projects and were considered to be constant regardless of production rate. Because operating costs are usually depending on the amount of ore produced and the rate at which it is produced a detailed study should be executed to create cost curves as a function of production rate. This will allow for a better economic assesment of different project strategies and will be a valuable source of information for future studies.

- ***Complete the project risk assessment***

In this research project, only the grade uncertainty (grade-risk) was investigated and used to quantify economic risk. As there are many other risks that can have a negative impact on the economic viability of a project, it is proposed that further research identifies these risks and develops a method to quantify the risk and assess their economic impact on project profitability. If successful, this risk assesment can be incorporated into the final technical economic model. Research could focus on the risk of dynamic metal prices over the life of mine of the project as well as the uncertainty in OPEX and CAPEX.

- ***Identify the optimum Stope Sequence and Ore Blending strategy***

The limited size of the Älgträsk mineral deposit only allowed for the evaluation of a single production sequence. A reasonable assumption is that an optimized stoping sequence exists that has a positive impact on project profitability. In the same way there may be an optimum way to blend stopes and obtain an optimal ore flow in which the gold grade is constant over time and for instance, the grade of penalty elements is reduced to a minimum to minimize refining charges.

- ***Research the possibility of applying 'Dynamic Cut-off Grades' to different areas in a mineral resource.***

This project focuses on mine optimization at a fixed cut-off grade regardless of the stope location in the mineral resource. The cut-off grade was determined based on the metal price and basic cost of mining, processing, transportation and refining + an amount for infrastructure development. The same cut-off grade was applied to each stope. Further research could provide an answer to the question if a dynamic cut-off grade can be applied based on the spatial location of a stope in the mineral resource and the development required to reach this stope. An example research topic could be the identification of a cut-off grade based on depth or -similar to open pit mining- determining a cut-off grade for expansion of an existing mine where large portions of infrastructure are already in place.

- ***Mine Infrastructure Optimization***

This research project focused on the optimization of underground stopes in a vein type gold deposit. As such the shape of the orebody was reasonably confined regardless of cut-off grade. This made the adaption of the infrastructure design to each cut-off grade relatively easy. One can imagine that in case of a large orebody (thick, or irregularly shaped) , the economic zone within the mineral resource can change quite significantly in terms of location and shape depending on the chosen cut-off grade. An automated optimization process to design basic level development and declines would under such circumstances certainly be of added value to reduce the required time to complete one iteration of the optimization process.

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Personal Comments

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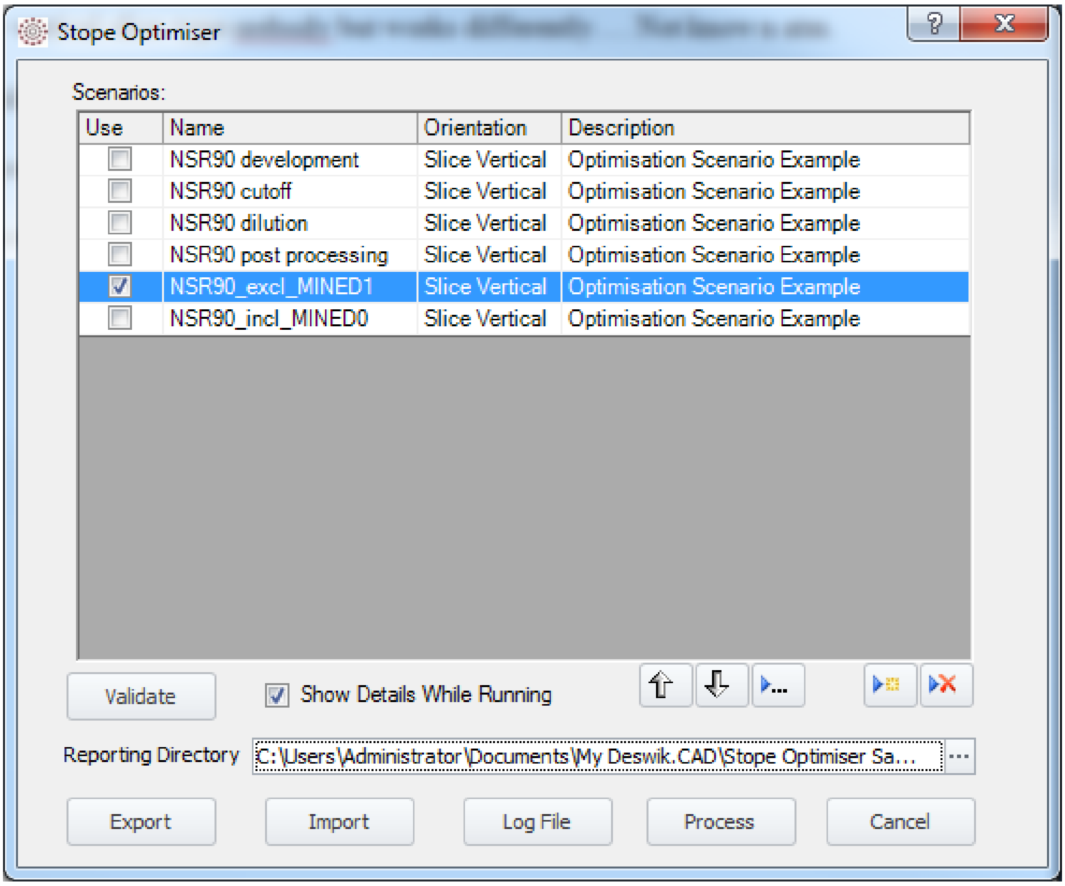
8 APPENDIX A – STOPE OPTIMIZER GUIDELINE

Stope optimizer – Scenarios



Slice vertical – slices the blockmodel vertically, for vertical orebody and find stopes on a horizontal axis. Used for sub vertical orebodies, narrow vein deposits or intrusive orebodies.

Slice horizontal – looks in a vertical direction at a horizontal orebody, for a room and pillar type mine. Not evaluated in this study.

Prism – type of slice vertical which has a particular prism shape that is projected along the optimization axis. Used in sublevel caving design.

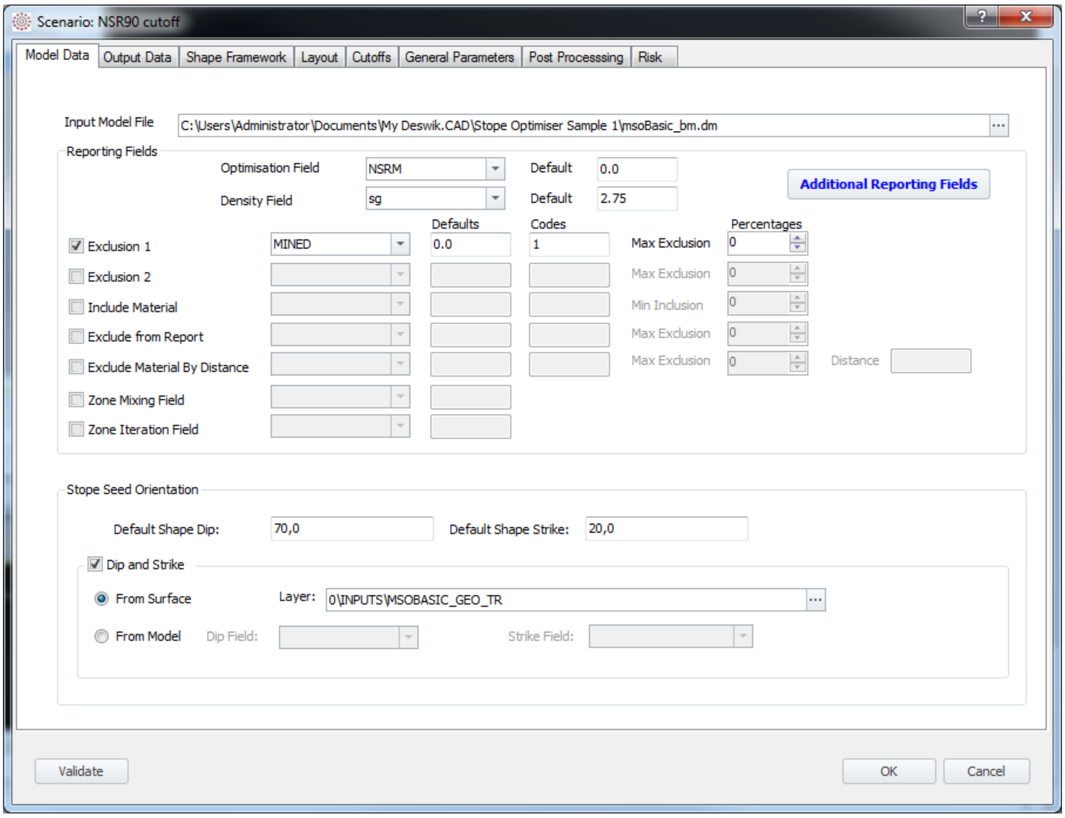
Validate – validates if the setup settings are correct.

Export – export optimization scenario data to xml file.

Log file – Opens the log file for the latest executed optimization scenario.

Process – Starts the optimization process for the selected scenarios.

Model Data



Input model file – The input block model for optimization should be selected. Only Datamine block models are supported at this moment.

Reporting Fields

Additional Reporting Fields – Other fields within the block model to be reported based on the final stope shapes. For example additional metal grades or any other attribute fields present in the block model. The fields are purely reporting fields and no optimization takes place for the attribute fields selected here.

Optimisation field – field in the blockmodel you want to optimize for (default attribute value has to be defined). In practice this would be the METAL GRADE or NSR.

Density field – In order for the optimizer to correctly calculate stope tonnages, a density attribute field has to be present in the block model.

Exclusion 1 and 2 – By using this option it is possible to exclude certain blocks from the optimization process. For example one can give all blocks in the block model an attribute field 'MINED' and give mined-out blocks a value of 1. By excluding blocks that have an attribute value of 1 for the MINED attribute, these blocks will not be included in the optimization process and hence no stopes will be designed containing these blocks. In a similar manner one can define maximum percentages of blocks with a certain attribute field to be included in the optimization process.

Include material – The opposite of Exclusion.

Exclude from report – Exclude from report is slightly different from the Exclusion or Include material fields. When using exclude from report and the MINED attribute example as introduced in the Exclusion description, the blocks that are already mined out will be used during the optimization process. However, grade and tonnages for these blocks are not accounted for in the stope tonnage and grade calculations as the blocks are already mined out and one wants to avoid counting metal content twice.

Exclude material by distance – A minimum distance from blocks with a certain attribute value can be applied. For instance a fault zone that is modeled into the block model with an attribute FIELD and value 1 for a fault zone and 0 for all other areas in the block model. We can now define a minimum distance between the designed stope shape and the fault zone. This distance is measured in the direction of optimization only.

Zone mixing field – This function can be used to restrict the occurrence of multiple code values for a field in a stope. If multiple ore lenses exist, and they have an attribute field that represents each zone (e.g. ZONE=1,2,3, etc) one can select this field as the zone mixing field and avoid the mixing of the ore lenses in a single stope. To use this functionality one has to supply the list of ore zones that cannot be 'mixed' in the *Mixing Zone Definition* (to be found in the General Parameters tab).

Zone iteration field – Not functioning correctly at moment of evaluation.

Stope Seed Orientation

! Dip and strike have to be specified as apparent dip and strike with respect to the stope framework origin (usually the block model origin).

Default shape dip – The default dip of the slices that will be created during the slice evaluation stage.

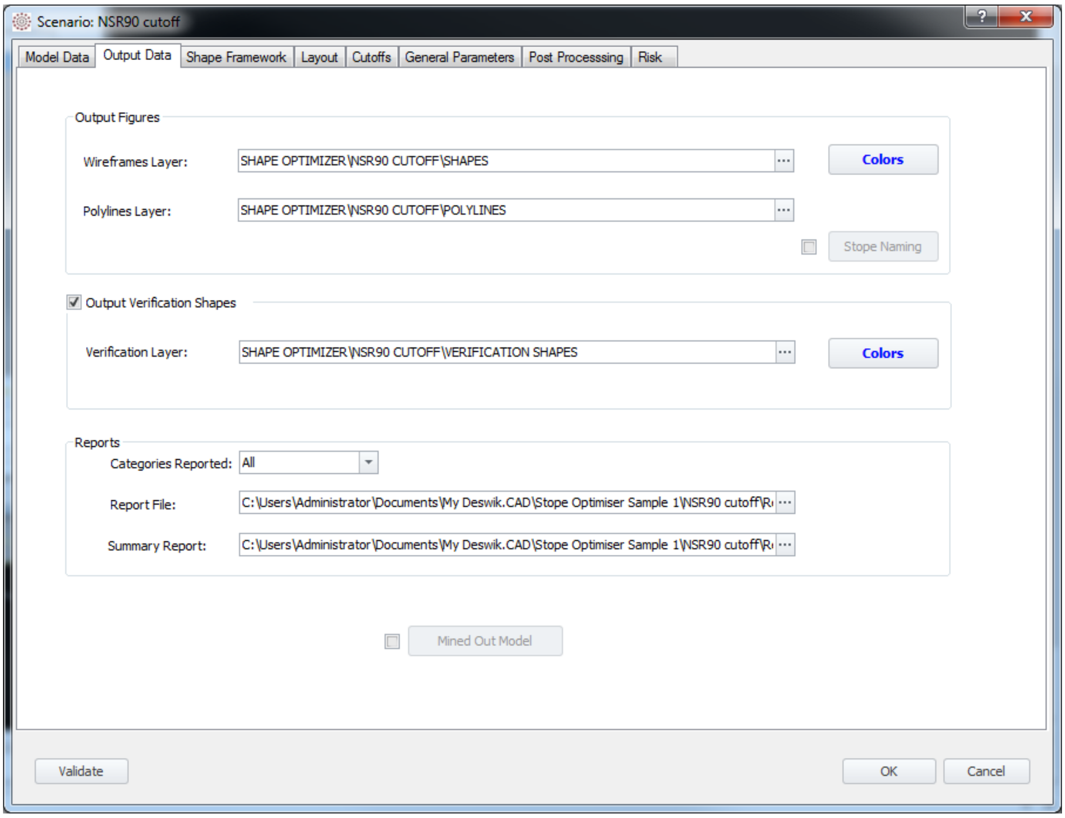
Default shape strike – The default strike of the slices that will be created during the slice evaluation stage.

Dip and strike

From surface -- select a surface wireframe that follows the dip and strike of the orebody. By using a wireframe, slices will be generated based on the local dip and strike of this wireframe during the slice evaluation stage.

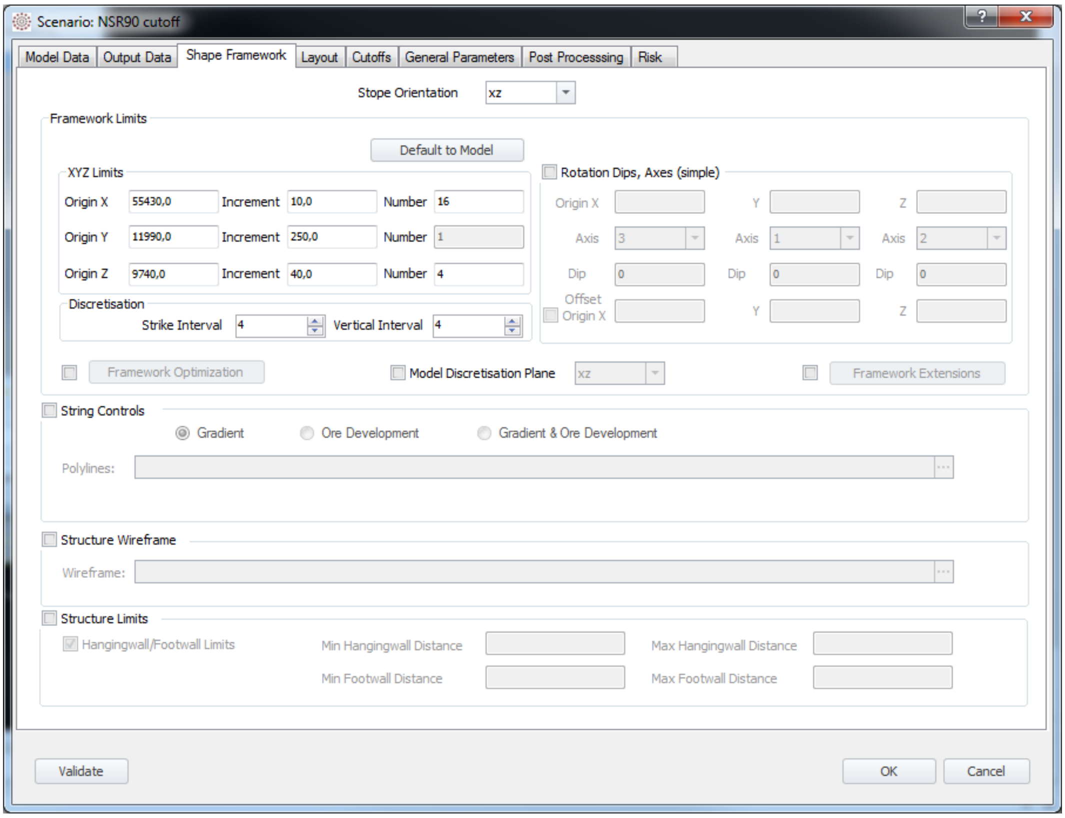
From model – reads the dip and strike from a field in the block model (not evaluated)

Output Data



Define the output figures and verification shapes that will be created during the optimization process as well as their colors and reporting file names and locations.

(Stope) **Shape Framework**



Framework limits

This is where you specify the stope shape framework. For sublevel stope mining use a slice vertical setup (selected in the scenarios menu) meaning that the stope orientation (or stope shape framework) is either in the XZ or YZ plane and is subsequently optimized in the Y or X direction respectively.

Framework	U axis	V axis	W axis	NX	NY	NZ	XINC	YINC	ZINC
Vertical XZ	X	Z	Y	N	1	M #	(XMAX-XMIN)/N	YMAX-YMIN	(ZMAX-ZMIN)/M

Ignored for gradient string and SectionXZ or SectionYZ cases, but NZ*ZINC must still define the Z extent

XYZ limits

Origin XYZ - This is where the framework origin is defined (the starting point for optimization). The origin is the place where the Optimizer will start placing stopes and is the coordinate of the left lower corner of the first stope. By clicking the *Default to Model* button, the block model origin will be set as the framework origin and the increments will be equal to the block model block size.

Increment X and Z- Increment defines the width and height of the to be designed stope.

Increment Y – Defines the length of the stope in the optimization direction. This should be a large enough number to completely cover the orebody thickness.

Number – Number defines the number of stopes to be designed in the horizontal and vertical direction.

Rotation Dips, Axes

This functionality can be activated whenever the strike of the orebody does not line up with the default strike of the optimization framework based on the block model origin.

As an example we have an orebody of which the strike is rotated 45 degrees with respect to the block model origin. To align the optimization framework with the strike of the orebody, the block model origin has to be rotated 45 degrees.

Origin XYZ - The origin of the *block model (the original framework origin).*

Axis – The axis around which the rotation has to take place to align the framework with the orebody.

Dip – The angle the framework has to be rotated around the specified axis.

In our example we want the rotation to be 45 degrees counter-clockwise around the Z (3) axis to line up the X-axis of the framework with the orebody.

Discretisation

Discretisation – This is the subdividing of the original blocks in the block model into smaller sub blocks. It is required for two purposes:

1. It is used during the stope shape annealing stage where the optimizer tries to maximize the stope profit by trying different stope shapes.
2. Maintaining an accurate volumetric estimate of the material mined out in stoping units and sub-units. Multiple sequential passes are made for stoping units, stoping sub-units and development. The ore available to subsequent passes is reduced by that taken in earlier passes. So for instance if half a block is used in one stope, we still want to have the other half of the block available for the optimization of the next stope.

Framework extensions – In this menu it is possible to force the optimizer into optimization at a pre-defined level spacing (Z-direction) or section spacing (X or Y direction). It gives the end-user the possibility to define more complex stope shape framework (level spacings or sections) rather than the default stope shape framework.

Model Discretisation Plane –The cells and subcells in the input block model are subdivided (or discretised) for the stope shape annealing process. The plane in which the cells are split is normally the the model plane that is most closely aligned with the stope orientation plane. In cases where the model or framework (or both) are rotated, another model discretisation plane may be selected.

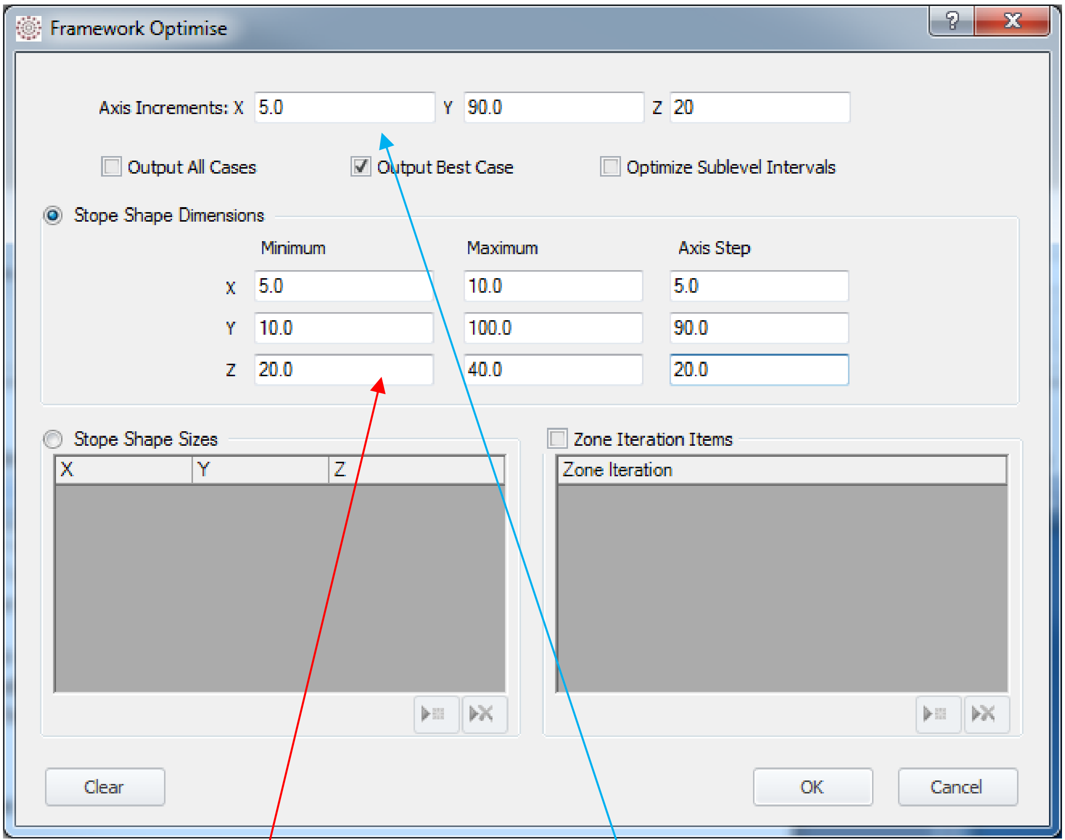
Framework Optimization

"During initial mine design the position of the origin for the stope shape framework, and the choice of stope dimension can both be considered variables in the analysis. If only a limited number of cases need to be evaluated, a scenario can be evaluated for each specific case. To enable all possible cases to be evaluated a framework optimisation procedure is supplied.

The stope size ***minimum, maximum and increment*** *are supplied for the axes specified by the stope orientation plane, and the* ***step size for the origin shift*** *along the same axes.*

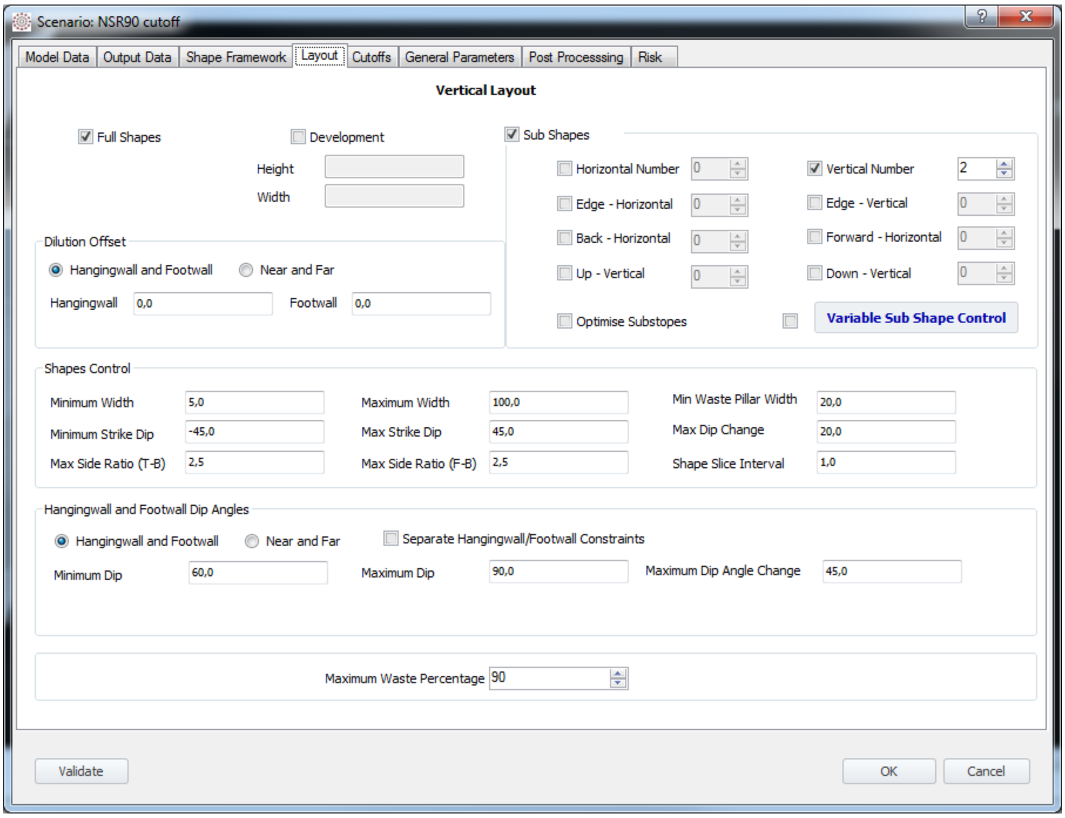
The initial framework specification defines the extent of the volume to be considered. To minimise the number of combinations to be considered the stope size increment should be a sub-multiple of the stope dimension, and the origin shift should be a sub-multiple of the stope sizes. For example the stope sizes might be 20-35 in increments of 5 and the origin step size is also 5. An initial scenario should be run to estimate the likely runtime for all combinations.

If a zone iteration field is defined in the block model and each zone is spatially separate then a framework will be optimised for each zone identified in a list of zones. This feature can be useful when there are a significant number of "pods" to be evaluated."



Case	Zone	Stope_X	Stope_Y	Stope_Z	Origin_X	Origin_Y	Origin_Z
1	0.0	5.00	250.00	20.00	55430.00	11990.00	9740.00
2	0.0	5.00	250.00	40.00	55430.00	11990.00	9740.00
3	0.0	5.00	250.00	40.00	55430.00	11990.00	9760.00
4	0.0	10.00	250.00	20.00	55430.00	11990.00	9740.00
5	0.0	10.00	250.00	20.00	55435.00	11990.00	9740.00
6	0.0	10.00	250.00	40.00	55430.00	11990.00	9740.00
7	0.0	10.00	250.00	40.00	55430.00	11990.00	9760.00
8	0.0	10.00	250.00	40.00	55435.00	11990.00	9740.00
9	0.0	10.00	250.00	40.00	55435.00	11990.00	9760.00

Layout



Full Shapes – When this box is selected you actually confirm that you want to create the full size stope shapes that you defined by means of the increment in the *Shape Framework* tab, *Framework Extensions* or the output of the *framework optimization*.

Development (Height & Width) – It looks for Development shapes that fit into a height and width and achieve your cut-off. They're not necessarily practical shapes but can be used when you design your development through an orebody to get some more ore out. Development shapes have the lowest optimization priority so they only appear in sections of the orebody where ore does not fit in full shapes or sub shapes.

Sub Shapes

Sub shapes can be defined in two ways:

- Fixed dimensions by specifying the number of sub-shapes in the X and Z direction (by ticking the respective boxes)

- Specifying a user defined list of shapes (use this option without ticking the full shapes box.)

By specifying the list shapes manually it is possible to define irregular shapes such as stope and drive shapes or primary and secondary stope shapes.

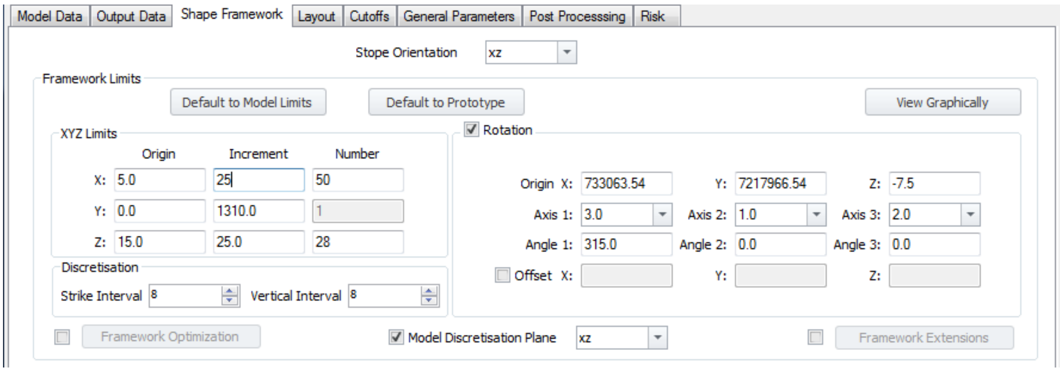
By default sub shapes are processed in the order in which they are supplied (by user or automatic list) and the first economic sub shape will be presented as the optimal stope design.

EXAMPLE

This example explains how to setup the stope optimizer to optimize a primary-secondary stope sequence with 10 and 15 metre stopes.

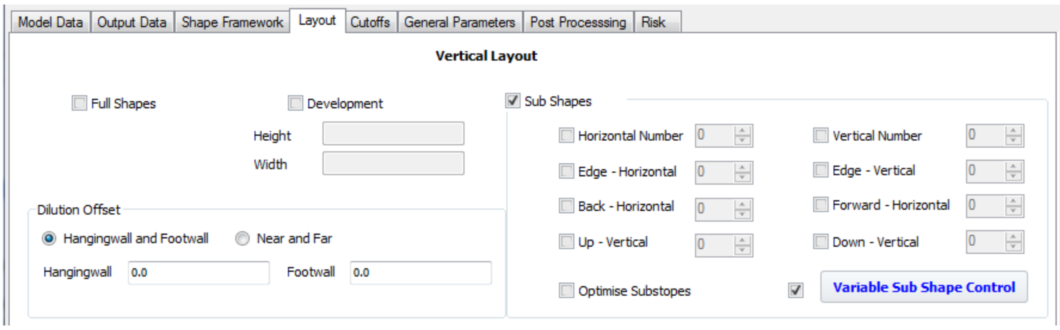
Step 1:

In the Shape Framework tab the XYZ limits are defined as previously explained. In this example we want to define a stoping sequence of 10 and 15 metre stopes but since we can only supply a single increment in the X direction we add up the stope widths (10+15=25) and supply 25m as the increment.



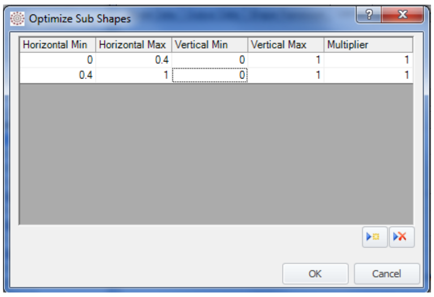
Step 2:

In the Layout tab we untick the 'Full Shapes' box and tick the 'Sub Shapes' box and 'Variable Sub Shape Control' box which will activate the respective button.



Step 3:

Click the Variable Sub Shape Control' button and the following window will pop-up:



The primary-secondary stope sequence is now supplied by means of a fraction between 0 and 1 of the full increment (the 25m supplied in the Shape Framework tab). The 10 metre stope is defined as 0 to 0.4 whilst the 15 metre stope is defined as 0.4 to 1.

In the same way it is possible to define a vertical split or any other stope shape combination (in combination with changing the settings in the Shape Framework tab).

Optimise Substopes – If this box is ticked, the optimal choice of sub shapes is evaluated to maximize value/grade/metal. So the optimizer will look at, and select the best combination of sub shapes. Optimizing sub shapes will only have an effect if the list of sub shapes supplied is overlapping.

The stope optimizer does not check the combination of full shapes and sub shapes simultaneously and give an output that optimizes grade/value/metals.. So if a solution exists with a full shape and a sub shape(s) (which in fact has a higher grade), the optimizer will still output the full shape because of the optimization priority (full shapes, subshapes, development).

Dilution Offset

Here you can specify the dilution as average meters into the rockface over the full hangingwall and footwall. It can assist the engineer to estimate the grade of the stope external dilution (overbreak).

Near and Far will keep one side of the orebody as Hangingwall and one side as Footwall along the whole orebody. Hangingwall and Footwall will apply the dilution offset based on which stope wall is hanging (This can change on a stope to stope basis).

Shapes Control

Under Shapes Control it is possible to define stope design constraints to guide the stope shape annealing stage.

Minimum Width – The minimum width of the final stope shape.

Maximum Width – The maximum width of the stope shape you are optimizing. You could choose a large number here if there is potential to do transverse stoping. In the *Post Processing* tab you can later sub-divide these large stope shapes into smaller shapes.

Min Waste Pillar Width – The minimum width of a waste pillar between two adjacent optimized stopes.

Minimum Strike Angle / Maximum Strike Angle – This is where the minimum and maximum deviation from the default shape strike is defined. When for instance the default shape strike is 90 degrees with respect to the framework origin (as supplied in the Model Data tab) and we allow the local stope strike to deviate from that 90 degrees by 30 degrees, then we enter -30 and 30 as the minimum and maximum strike angle.

Max Strike Change – This limits the difference between the strike angle of the front and back of the stope. To allow full freedom to the previous example, the maximum strike angle change has to be set to 60 degrees.

Max Side Ratio (Top-to-Bottom) – Side to length ratio. The top can be X times the bottom or the bottom can be X times the top.

Max Side Ratio (Front-to-Back) – Also a ratio but applies to the sidewalls of the stope.

Shape Slice Interval – This interval defines the slice thickness (the thickness of the smallest mining unit) that is used during the slice evaluation stage.

Hangingwall and Footwall Dip Angles

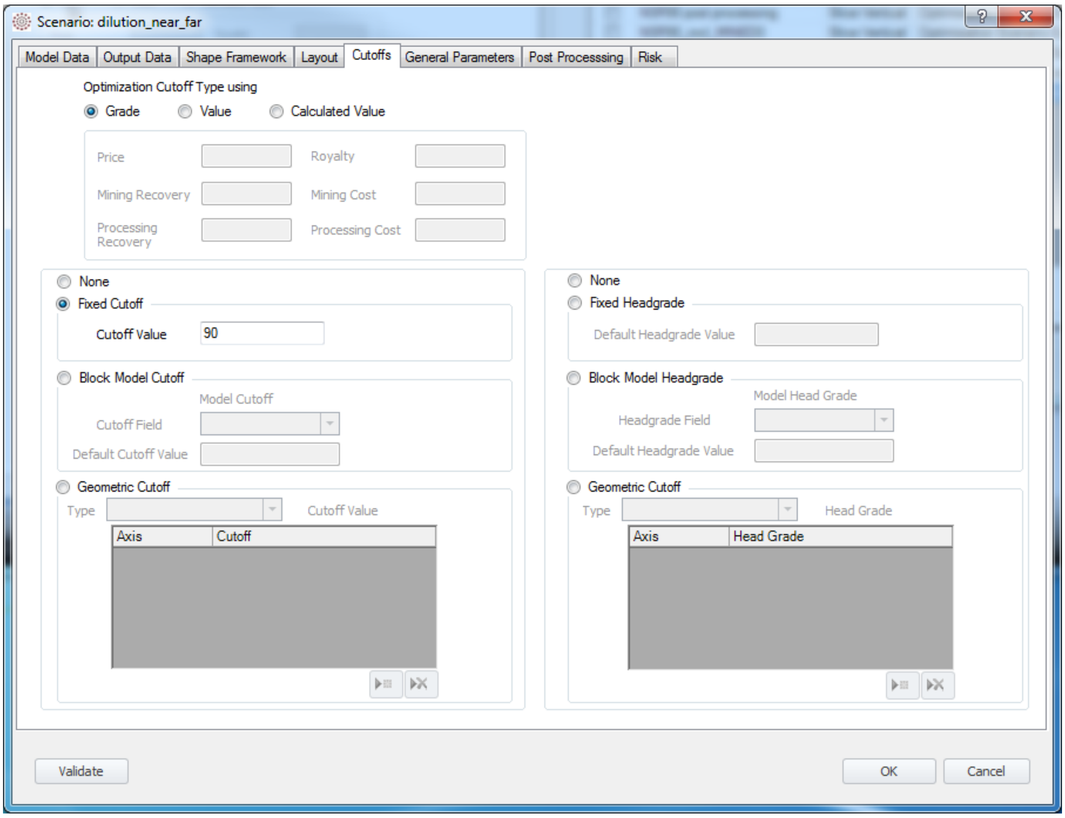
Minimum Dip – The minimum dip for the hanging wall and/or footwall of a final stope shape. Usually this angle > angle of repose in order for broken ore to flow to the stope drawpoint automatically.

Maximum Dip – The maximum dip for the hanging wall and/or footwall of a final stope shape.

Maximum Dip Angle Change – This is the maximum angle between the hanging wall and footwall of the final stope shape.

Maximum Waste Percentage – The maximum percentage of material below cut-off that you allow in a stope. This is also known as the maximum internal waste percentage.

Cutoffs



Several ways exist to define the cut-off for the optimization scenario:

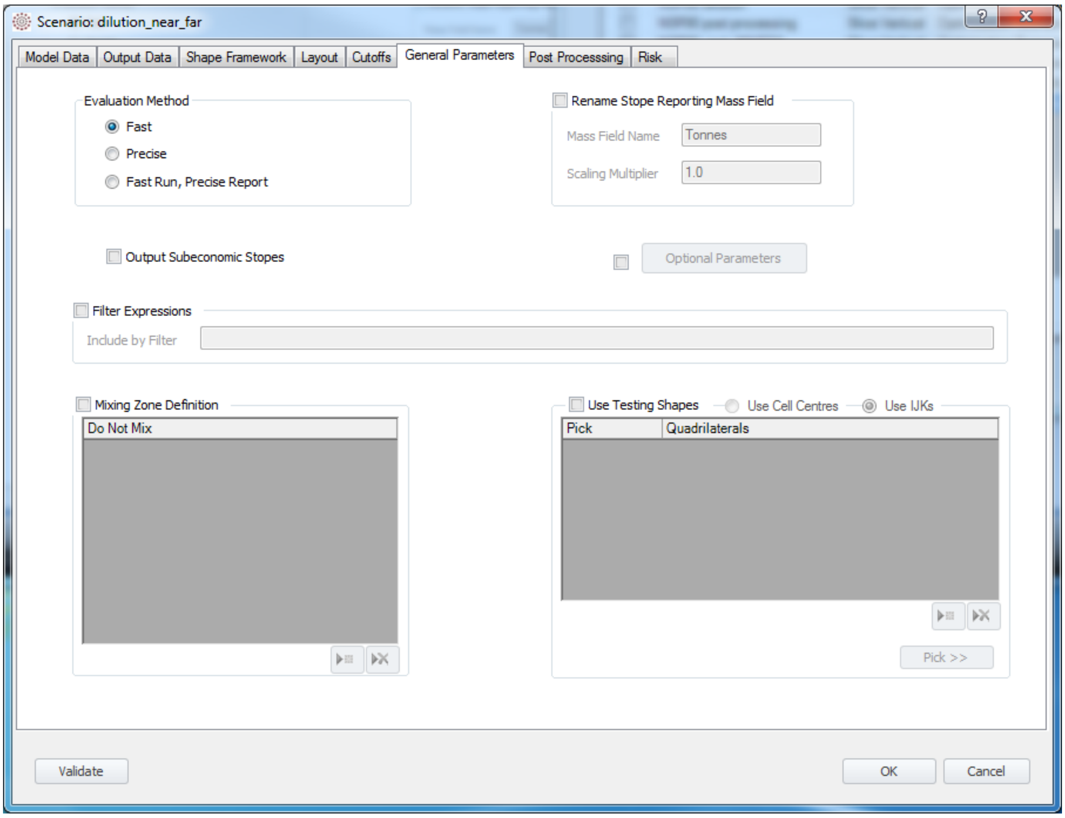
Calculated Value – Simple break-even cost inputs can be supplied after which the Stope Optimizer will calculate the resulting cut-off grade. In general it is not recommended to use this method to define the cut-off. These type calculations are easily performed in a spreadsheet package such as Microsoft Excel.

Grade – 3 possibilities for grade type optimization are available:

- Select a fixed cut-off throughout the whole orebody.
- Use a cut-off that is stored in an attribute field within the block model. A default value has to be applied in case the attribute field is empty.
- Use a geometric cutoff based on the shape of the orebody. You can input a range of values for geometric cutoffs. Applicable Geometry types are: Thickness, Area, Height and Mass

Value – The same possibilities as in Grade optimization although Value optimization allows for the use of negative values in the block model.

General Parameters



Evaluation Method

Fast – A new ray-casting type method to calculate stope grade and tonnage. It is believed to be more accurate compared to the partial cell investigation.

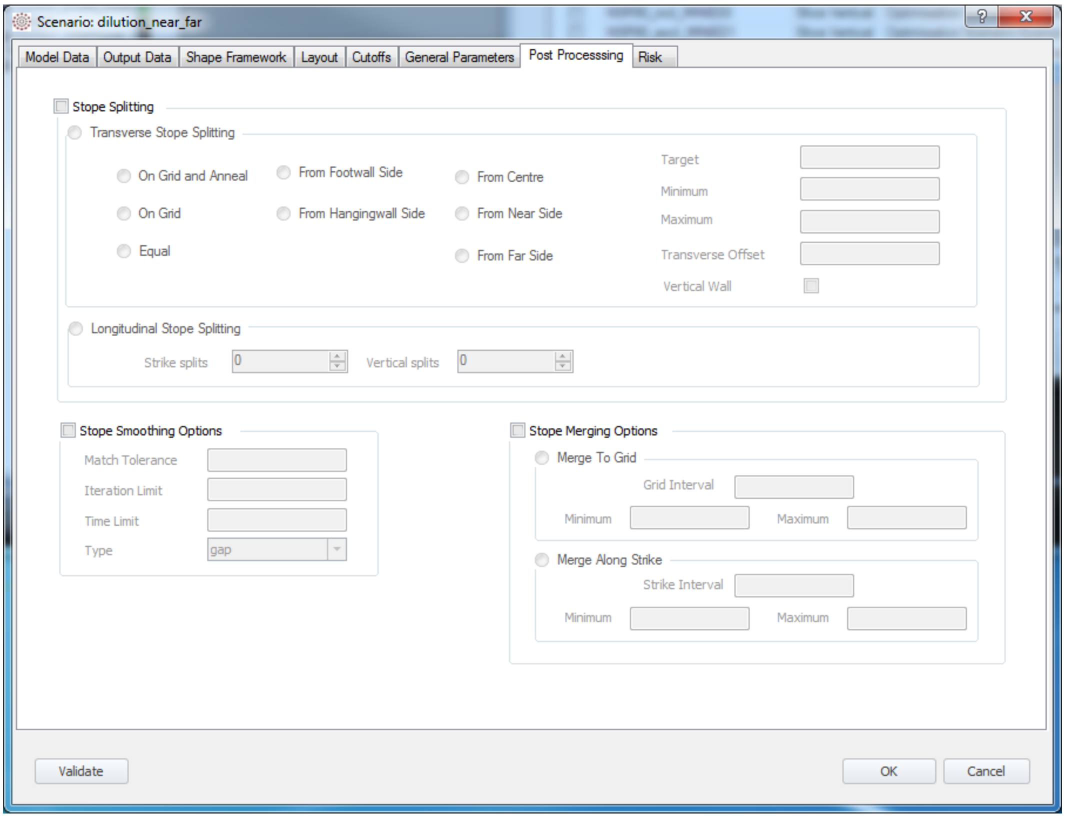
Precise – This a classical partial cell investigation to calculate stope grade and tonnage. It is used by most mining software packages today.

Fast Run, Precise Report – The Optimization process is based on the ray-casting type method that is believed to deliver more accurate optimization results. However, the final stope volumes, tonnages and grades are reported based on the partial cell investigation for further use in the Deswik.Scheduler.

Mixing Zone Definition

Do Not Mix – This list is used in conjunction with the Zone Mixing Field (Model Data tab).

Post Processing



Stope splitting – Here it is possible to for instance divide a 100m wide stope in sections of say, 10 meters for transverse stoping.

Stope Smoothing – stitches the stope edges together to create a more mineable shape

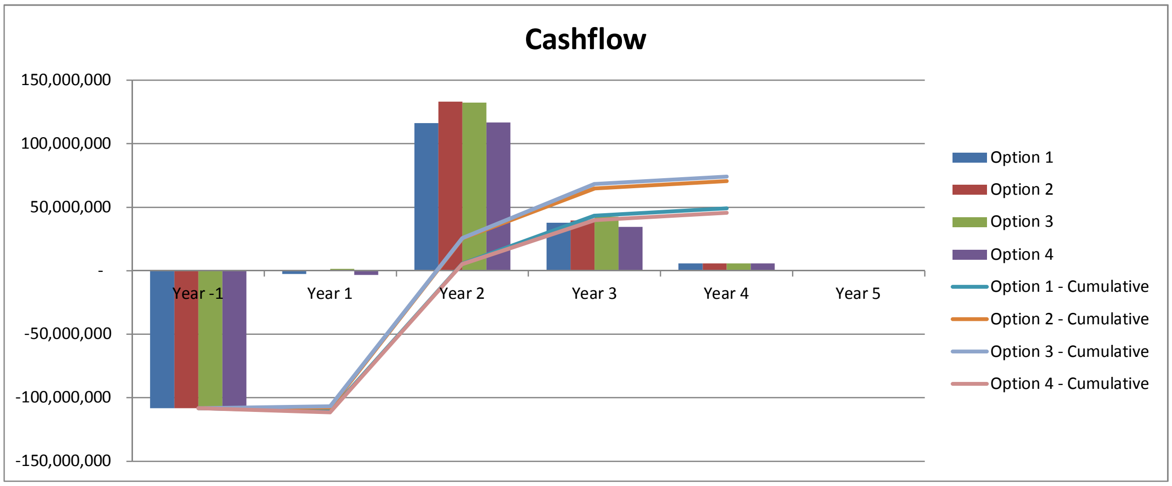
9 APPENDIX B – ÄLGTRÄSK CASE STUDY

1. *Example Technical Economic Model*
2. *ROM Cumulative tonnage curves*
3. *Target Production Capacity Utilization*

1. Example Technical Economic Model

Price Inputs	Exchange Rate	USD:SEK		1.00	7.5
		USD/oz	SEK/oz	SEK/g	
	Gold Price	1200	9000		289.39
	Silver Price	0	0		0.00

Other Inputs		
CAPEX (year -1)	SEK	- 108,260,000



Annual Cash Flow

Processing Method	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	NPV	IRR
Option 1	- 108,260,000	- 2,490,498	116,175,091	37,796,861	5,801,797	-	16,225,845	18%
Option 2	- 108,260,000	621,895	132,995,107	39,449,837	5,812,245	-	32,570,689	25%
Option 3	- 108,260,000	1,467,657	132,457,613	42,637,689	5,801,797	-	35,036,698	26%
Option 4	- 108,260,000	- 3,336,261	116,712,584	34,609,009	5,812,245	-	13,759,836	16%

Cumulative Cash Flow

Option 1	- 108,260,000	- 110,750,498	5,424,593	43,221,454	49,023,251
Option 2	- 108,260,000	- 107,638,105	25,357,001	64,806,839	70,619,084
Option 3	- 108,260,000	- 106,792,343	25,665,271	68,302,960	74,104,757
Option 4	- 108,260,000	- 111,596,261	5,116,324	39,725,333	45,537,578

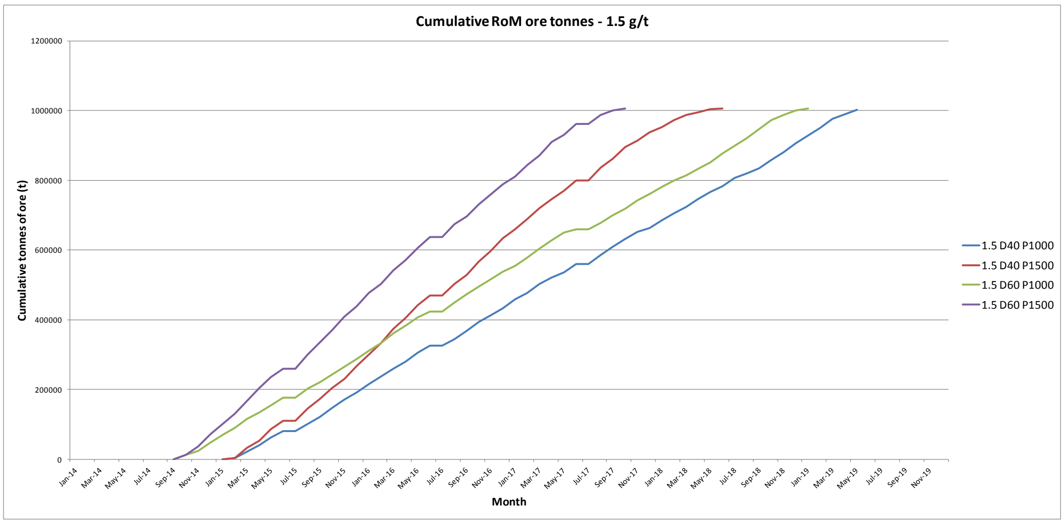
COG	1.8		
	Gold Price	289.39	SEK/g
	Silver	0.00	SEK/g
	Copper	26	SEK/lb

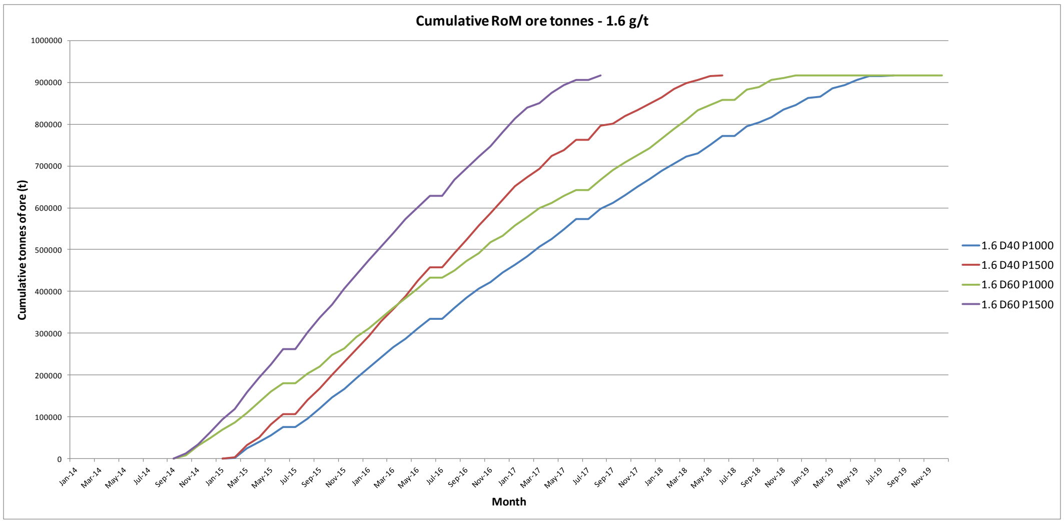
					Jan-14	Feb-14	Mar-14	Apr-14	May-14	Jun-14	Jul-14	Aug-14	Sep-14	Oct-14	Nov-14	Dec-14	Jan-15
	Tonnes & Grades																
	Period Tonnes	0	Tonnes		0	0	0	0	12315.51674	6604.820869	0	6736.335108	3774.533029	25050.62164	24564.56694	35818.47873	44150.0021
	Period grade AU (g/t)	0	AU_OK		0	0	0	0	5.326520754	1.441395768	0	2.961645266	6.502605882	2.941766113	3.243325933	3.152421037	4.136921802
	Period gold production		kg		0	0	0	0	65.59885549	9.520160848	0	19.95063498	24.54430068	73.69306985	79.67089698	112.9149258	182.6451062
	Tonnes Low As - ORE&LOS	0	Tonnes Low As		0	0	0	0	7018.932551	2205.775887	0	3498.950635	2861.373998	17726.08551	16212.04055	21133.51321	25230.18674
	AS_PROC Low As - ORE&LOS	0	AS_PROC Low As		0	0	0	0	0.140596776	0.120676512	0	0.088684916	0.137270763	0.122310887	0.130266511	0.109670104	0.136841799
	AU_OK Low As - ORE&LOS	0	AU_OK Low As		0	0	0	0	5.698552737	1.7869276	0	2.250717196	6.272982992	3.32942025	3.491299035	3.529558387	4.362656531
	0	0		0	0	0	0	0	0	0	0	0	0	0	0	0	0
	0	0		0	0	0	0	0	0	0	0	0	0	0	0	0	0
	Tonnes High As - ORE&LOS	0	Tonnes High As		0	0	0	0	2606.651248	2908.309066	0	2789.322265	568.6481973	4158.630985	7009.325008	10559.9861	13390.13784
	AS_PROC High As - ORE&LOS	0	AS_PROC High As		0	0	0	0	0.468037233	1.041177788	0	0.733552434	0.365685713	0.353494669	0.63180489	0.902129613	0.923216912
	AU_OK High As - ORE&LOS	0	AU_OK High As		0	0	0	0	9.821451263	1.91815894	0	4.329183032	11.59759282	3.528921385	3.291303434	3.629072681	5.419994035
	0	0		0	0	0	0	0	0	0	0	0	0	0	0	0	0
	0	0		0	0	0	0	0	0	0	0	0	0	0	0	0	0
	Stoping Tonnes (incl. dilution 15%)		Diluted Tonnes		0	0	0	0	0	0	0	0	0	12736.08508	16988.0993	35177.5311	39990.72378
	Stoping Tonnes - LOS	0	Stoping Tonnes		0	0	0	0	0	0	0	0	0	11074.85659	14772.26026	30589.15748	34774.54242
	Stoping Tonnes - ORE&LOS	0	Stoping Tonnes		0	0	0	0	12315.51674	6604.820869	0	6736.335108	3774.533029	25050.62164	24564.56694	35818.47873	44150.0021
	Tonnes Production ore drives		Tonnes		0	0	0	0	12315.51674	6604.820869	0	6736.335108	3774.533029	13975.76505	9792.306674	5229.321253	9375.45968
	Development																
0	Meters - DEVELOPMENT	0	Meters		61.10882957	55.19507187	61.10882957	101.2815943	295.8363673	188.1132704	0	189.8813823	254.6682064	288.9115976	225.1167519	245.531851	291.2596986
	Recoveries																
	"Gravity / Flotation"																
Low As		Recovery															
Gravitation	Au (g)	40%			0	0	0	0	15999	1577	0	3150	7180	23607	22640	29837	44028
	Ag (g)	15%			0	0	0	0	0	0	0	0	0	0	0	0	0
	Cu (t)	0.80%			0	0	0	0	0.000	0.000	0	0.000	0.000	0.000	0.000	0.000	0.000
	As (t)	16.00%			0	0	0	0	1.579	0.426	0	0.496	0.628	3.469	3.379	3.708	5.524
Flotation	Au (g)	50%			0	0	0	0	19999	1971	0	3938	8975	29509	28301	37296	55035
	Ag (g)	50%			0	0	0	0	0	0	0	0	0	0	0	0	0
	Cu (t)	40.00%			0	0	0	0	0	0	0	0	0	0	0	0	0
	As (t)	5.00%			0	0	0	0	0	0	0	0	0	1	1	1	2
High As																	
Gravitation	Au (g)	40%			0	0	0	0	10240	2231	0	4830	2638	5870	9228	15329	29030
	Ag (g)	15%			0	0	0	0	0	0	0	0	0	0	0	0	0
	Cu (t)	0.80%			0	0	0	0	0	0	0	0	0	0	0	0	0
	As (t)	9.60%			0	0	0	0	1	3	0	2	0	1	4	9	12
Flotation	Au (g)	30.00%			0	0	0	0	7680	1674	0	3623	1978	4403	6921	11497	21772
	Ag (g)	30.00%			0	0	0	0	7680	1674	0	3623	1978	4403	6921	11497	21772
	Cu (t)	40.00%			0	0	0	0	0	0	0	0	0	0	0	0	0
	As (t)	10.70%			0	0	0	0	1	324	0	219	22	157	474	1019	1323
	Leaching																
Low As																	
Leaching	Au (g)	53%			0	0	0	0	21199	2089	0	4174	9513	31279	29999	39534	58337
	Ag (g)	60%			0	0	0	0	0	0	0	0	0	0	0	0	0
	Cu (t)	0.00%			0.000	0.000	0.000	0.000	0.000	0.000	0	0.000	0.000	0.000	0.000	0.000	0.000
	As (t)	0.00%			0.000	0.000	0.000	0.000	0.000	0.000	0	0.000	0.000	0.000	0.000	0.000	0.000
High As																	
	Au (g)	45%			0	0	0	0	11520	2510	0	5434	2968	6604	10381	17245	32659
	Ag (g)	65%			0	0	0	0	0	0	0	0	0	0	0	0	0
	Cu (t)	0.00%			0	0	0	0	0.000	0.000	0	0.000	0.000	0.000	0.000	0.000	0.000
	As (t)	0.00%			0	0	0	0	0.000	0.000	0	0.000	0.000	0.000	0.000	0.000	0.000
	Concentrate Grades																
	Flotation / Gravity																
Low As GRAVITY																	
	Au_conc (g/t)	150															
	Tonnes of Concentrate				0.0	0.0	0.0	0.0	106.7	10.5	0	21.0	47.9	157.4	150.9	198.9	293.5
	Concentrate grades																
	Au (g/t)				0.0	0.0	0.0	0.0	150.0	150.0	0	150.0	150.0	150.0	150.0	150.0	150.0
	Ag (g/t)				0.0	0.0	0.0	0.0	0.0	0.0	0	0.0	0.0	0.0	0.0	0.0	0.0
	Cu %				0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
	As %				0.00%	0.00%	0.00%	0.00%	1.48%	4.05%	0	2.36%	1.31%	2.20%	2.24%	1.86%	1.88%
Low As FLOTATION																	
	Au_conc (g/t)	150															
	Tonnes of Concentrate				0.0	0.0	0.0	0.0	133.3	13.1	0	26.3	59.8	196.7	188.7	248.6	366.9
	Concentrate grades																
	Au (g/t)				0.0	0.0	0.0	0.0	150.0	150.0	0	150.0	150.0	150.0	150.0	150.0	150.0
	Ag (g/t)				0.0	0.0	0.0	0.0	0.0	0.0	0	0.0	0.0	0.0	0.0	0.0	0.0
	Cu %				0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
	As %				0.00%	0.00%	0.00%	0.00%	0.37%	1.01%	0	0.59%	0.33%	0.55%	0.56%	0.47%	0.47%
High As GRAVITY																	
	Au_conc (g/t)	150															
	Tonnes of Concentrate				0.0	0.0	0.0	0.0	68.3	14.9	0	32.2	17.6	39.1	61.5	102.2	193.5
	Concentrate grades																
	Au (g/t)				0.0	0.0	0.0	0.0	150.0	150.0	0	150.0	150.0	150.0	150.0	150.0	150.0
	Ag (g/t)				0.0	0.0	0.0	0.0	0.0	0.0	0	0.0	0.0	0.0	0.0	0.0	0.0
	Cu %				0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
	As %				0.00%	0.00%	0.00%	0.00%	1.72%	19.54%	0	6.10%	1.14%	3.61%	6.91%	8.95%	6.13%
High As FLOTATION																	
	Au_conc (g/t)	67.5															
	Tonnes of Concentrate				0.0	0.0	0.0	0.0	113.8	24.8	0	53.7	29.3	65.2	102.5	170.3	322.6
	Concentrate grades																
	Au (g/t)				0.0	0.0	0.0	0.0	67.5	67.5	0	67.5	67.5	67.5	67.5	67.5	67.5
	Ag (g/t)				0.0	0.0	0.0	0.0	67.5	67.5	0	67.5	67.5	67.5	67.5	67.5	67.5
	Cu %				0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
	As %				0.00%	0.00%	0.00%	0.00%	1.15%	1306.79%	0	407.94%	75.91%	241.16%	462.15%	598.47%	410.08%
	Gravity / Leaching																
Low As																	
	kg of Dore				0.0	0.0	0.0	0.0	21.2	2.1	0	4.2	9.5	31.3	30.0	39.5	58.3
	Dore % metals																
	Au (%)				0%	0%	0%	0%	100%	100%	0	100%	100%	100%	100%	100%	100%
	Ag (%)				0%	0%	0%	0%	0%	0%	0	0%	0%	0%	0%	0%	0%
	Cu %				-	-	-	-	-	-	0	-	-	-	-	-	-
	As %				-	-	-	-	-	-	0	-	-	-	-	-	-
High As																	
	kg of Dore				0.0	0.0	0.0	0.0	11.5	2.5	0	5.4	3.0	6.6	10.4	17.2	32.7
	Dore % metals																
	Au (%)				0%	0%	0%	0%	100%	100%	0	100%	100%	100%	100%	100%	100%
	Ag (%)				0%	0%	0%	0%	0%	0%	0	0%	0%	0%	0%	0%	0%
	Cu %				-	-	-	-	-	-	0	-	-	-	-	-	-
	As %				-	-	-	-	-	-	0	-	-	-	-	-	-
	Recovered Revenue																
Gravity / Flotation																	
	Concentrate treatment	420	SEK/t														
	Payable Gold	96.50%															
	Payable Silver	30	(g/t deduction)														
	Payable Copper	0%															
	Refining Charge Au	2%	of selling price														
	Refining Charge Ag	5%	of selling price														
	Refining Charge Cu	-	SEK/t CU														
	As Penalty	280	SEK/t Concentrate if As>0.2%														
	FLOTATION	**Low As**	Earnings		-	-	-	-	5,584,896	550,361	0	1,099,609	2,506,272	8,240,639	7,903,222	10,415,293	15,369,188
			Treatment Charges		-	-	-	-	205,026	20,204	0	40,368	92,007	302,520	290,134	382,354	564,215
			Recovered Revenue		**-**	**-**	**-**	**-**	**5,379,870**	**530,157**	0	**1,059,241**	**2,414,265**	**7,938,119**	**7,613,088**	**10,032,940**	**14,804,973**
	FLOTATION	**High As**	Earnings		-	-	-	-	2,144,812	467,365	0	1,011,662	552,513	1,229,484	1,932,746	3,210,626	6,080,154
			Treatment Charges		-	-	-	-	122,544	26,703	0	57,801	31,568	70,247	110,428	183,439	347,390
			Recovered Revenue		**-**	**-**	**-**	**-**	**2,022,268**	**440,662**	0	**953,860**	**520,945**	**1,159,238**	**1,822,318**	**3,027,186**	**5,732,763**
	GRAVITY	**Low As**	Earnings		-	-	-	-	4,467,917	440,289	0	879,687	2,005,017	6,592,511	6,322,577	8,332,235	12,295,350
			Treatment Charges		-	-	-	-	164,021	16,163	0	32,294	73,606	242,016	232,107	305,883	451,372
			Recovered Revenue		**-**	**-**	**-**	**-**	**4,303,896**	**424,126**	0	**847,393**	**1,931,412**	**6,350,495**	**6,090,471**	**8,026,352**	**11,843,978**
	GRAVITY	**High As**	Earnings		-	-	-	-	2,859,750	623,153	0	1,348,882	736,684	1,639,313	2,576,995	4,280,834	8,106,871
			Treatment Charges		-	-	-	-	104,984	22,876	0	49,519	27,044	60,180	94,604	157,153	297,610
			Recovered Revenue		**-**	**-**	**-**	**-**	**2,754,766**	**600,276**	0	**1,299,364**	**709,639**	**1,579,132**	**2,482,391**	**4,123,682**	**7,809,262**
Leaching																	
	Payable Gold	99.70%															
	Payable Silver	99.10%															
	Treatment Charge if Au>15%	225.05	Sek/kg dore														
	Treatment Charge if Au<15%	258.79	Sek/kg dore														
		Low As	Earnings		-	-	-	-	6,116,300	602,728	0	1,204,237	2,744,744	9,024,738	8,655,215	11,406,311	16,831,570
			Treatment Charges		-	-	-	-	4,771	470	0	939	2,141	7,039	6,751	8,897	13,129
			Recovered Revenue		**-**	**-**	**-**	**-**	**6,111,529**	**602,258**	0	**1,203,297**	**2,742,603**	**9,017,699**	**8,648,464**	**11,397,414**	**16,818,441**
		High As	Earnings		-	-	-	-	3,323,903	724,294	0	1,567,814	856,251	1,905,382	2,995,256	4,975,638	9,422,663
			Treatment Charges		-	-	-	-	2,593	565	0	1,223	668	1,486	2,336	3,881	7,350
			Recovered Revenue		**-**	**-**	**-**	**-**	**3,321,311**	**723,729**	0	**1,566,591**	**855,584**	**1,903,896**	**2,992,919**	**4,971,757**	**9,415,313**

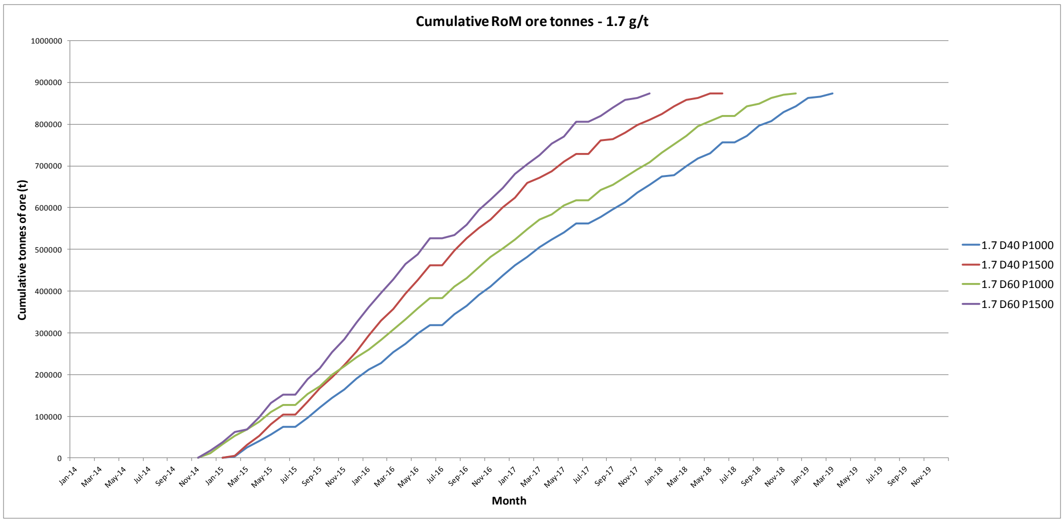
				Jan-14	Feb-14	Mar-14	Apr-14	May-14	Jun-14	Jul-14	Aug-14	Sep-14	Oct-14	Nov-14	Dec-14	Jan-15
TONNAGES	Stope Tonnes	t		0	0	0	0	0	0	0	0	0	11075	14772	30589	34775
	Ore Drive Tonnes	t		0	0	0	0	12316	6605	0	6736	3775	13976	9792	5229	9375
	Total Tonnes incl. dilution	**t**		**0**	**0**	**0**	**0**	**12316**	**6605**	**0**	**6736**	**3775**	**26712**	**26780**	**40407**	**49366**
203460	Total High As	t	0.31	0	0	0	0	2607	2908	0	2789	569	4159	7009	10560	13390
442993	Total Low As	t	0.69	0	0	0	0	7019	2206	0	3499	2861	17726	16212	21134	25230
	Development	m		61	55	61	101	296	188	0	190	255	289	225	246	291
	fraction High As	-		0.00	0.00	0.00	0.00	0.27	0.57	0.00	0.44	0.17	0.19	0.30	0.33	0.35
	Fraction Low As	-		0.00	0.00	0.00	0.00	0.73	0.43	0.00	0.56	0.83	0.81	0.70	0.67	0.65
RECOVERED REVENUES																
Option 1	*Gravity and flotation of High and Low As ore*															
	High As ore	SEK		-	-	-	-	4,777,034	1,040,938	-	2,253,224	1,230,584	2,738,370	4,304,710	7,150,868	13,542,025
	Low As ore	SEK		-	-	-	-	9,683,766	954,283	-	1,906,634	4,345,676	14,288,614	13,703,559	18,059,291	26,648,951
				-	**-**	**-**	**-**	**14,460,800**	**1,995,221**	**-**	**4,159,858**	**5,576,260**	**17,026,983**	**18,008,268**	**25,210,159**	**40,190,976**
				-	-	-	-	220	210	-	209	227	231	226	223	220
Option 2	*Gravity and Leaching of High and Low As ore*															
	High As ore	SEK		-	-	-	-	6,076,077	1,324,006	-	2,865,954	1,565,223	3,483,028	5,475,311	9,095,439	17,224,575
	Low As ore	SEK		-	-	-	-	10,415,425	1,026,384	-	2,050,690	4,674,015	15,368,194	14,738,935	19,423,766	28,662,420
				-	**-**	**-**	**-**	**16,491,502**	**2,350,389**	**-**	**4,916,645**	**6,239,238**	**18,851,222**	**20,214,245**	**28,519,205**	**45,886,994**
Option 3	*Gravity and Flotation Low As ore, Gravity and Leaching High As ore*															
	High As ore	SEK		-	-	-	-	6,076,077	1,324,006	-	2,865,954	1,565,223	3,483,028	5,475,311	9,095,439	17,224,575
	Low As ore	SEK		-	-	-	-	9,683,766	954,283	-	1,906,634	4,345,676	14,288,614	13,703,559	18,059,291	26,648,951
				-	**-**	**-**	**-**	**15,759,842**	**2,278,288**	**-**	**4,772,589**	**5,910,899**	**17,771,642**	**19,178,869**	**27,154,730**	**43,873,526**
Option 4	*Gravity and Flotation High As ore, Gravity and Leaching Low As ore*															
	High As ore	SEK		-	-	-	-	4,777,034	1,040,938	-	2,253,224	1,230,584	2,738,370	4,304,710	7,150,868	13,542,025
	Low As ore	SEK		-	-	-	-	10,415,425	1,026,384	-	2,050,690	4,674,015	15,368,194	14,738,935	19,423,766	28,662,420
				-	**-**	**-**	**-**	**15,192,459**	**2,067,322**	**-**	**4,303,915**	**5,904,599**	**18,106,563**	**19,043,644**	**26,574,634**	**42,204,445**
COSTS																
Development	25000	SEK/m		1,527,721	1,379,877	1,527,721	2,532,040	7,395,909	4,702,832	-	4,747,035	6,366,705	7,222,790	5,627,919	6,138,296	7,281,492
Mining	321.5	SEK/t ore		-	-	-	-	1,434,758	769,462	-	784,783	439,733	5,722,828	6,602,478	11,918,792	13,949,259
Processing Gravity/flotation	98	SEK/t ore	Option1	-	-	-	-	1,206,921	647,272	-	660,161	369,904	2,617,761	2,624,480	3,959,872	4,837,886
Processing Leaching	163.13	SEK/t ore	Option2	-	-	-	-	2,009,030	1,077,444	-	1,098,898	615,740	4,357,504	4,368,688	6,591,570	8,053,106
			Option3	-	-	-	-	1,424,136	891,905	-	854,774	410,660	2,948,355	3,150,966	4,836,729	5,952,642
			Option4	-	-	-	-	1,791,815	832,812	-	904,285	574,984	4,026,910	3,842,202	5,714,712	6,938,350
Total Costs																
			Option 1	**1,527,721**	**1,379,877**	**1,527,721**	**2,532,040**	**10,037,588**	**6,119,566**	**-**	**6,191,978**	**7,176,342**	**15,563,379**	**14,854,876**	**22,016,960**	**26,068,637**
			Option 2	**1,527,721**	**1,379,877**	**1,527,721**	**2,532,040**	**10,839,697**	**6,549,738**	**-**	**6,630,716**	**7,422,178**	**17,303,122**	**16,599,084**	**24,648,658**	**29,283,857**
			Option 3	**1,527,721**	**1,379,877**	**1,527,721**	**2,532,040**	**10,254,802**	**6,364,199**	**-**	**6,386,592**	**7,217,098**	**15,893,973**	**15,381,362**	**22,893,818**	**27,183,393**
			Option 4	**1,527,721**	**1,379,877**	**1,527,721**	**2,532,040**	**10,622,482**	**6,305,105**	**-**	**6,436,103**	**7,381,422**	**16,972,528**	**16,072,598**	**23,771,800**	**28,169,101**
PROFIT			**Option 1**	-1,527,721	-1,379,877	-1,527,721	-2,532,040	4,423,212	-4,124,345	-	-2,032,120	-1,600,082	1,463,604	3,153,392	3,193,199	14,122,339
			Option 2	-1,527,721	-1,379,877	-1,527,721	-2,532,040	5,651,805	-4,199,349	-	-1,714,071	-1,182,940	1,548,100	3,615,161	3,870,546	16,603,138
			Option 3	-1,527,721	-1,379,877	-1,527,721	-2,532,040	5,505,040	-4,085,911	-	-1,614,003	-1,306,199	1,877,669	3,797,507	4,260,912	16,690,133
			Option 4	-1,527,721	-1,379,877	-1,527,721	-2,532,040	4,569,977	-4,237,783	-	-2,132,188	-1,476,823	1,134,035	2,971,046	2,802,834	14,035,344

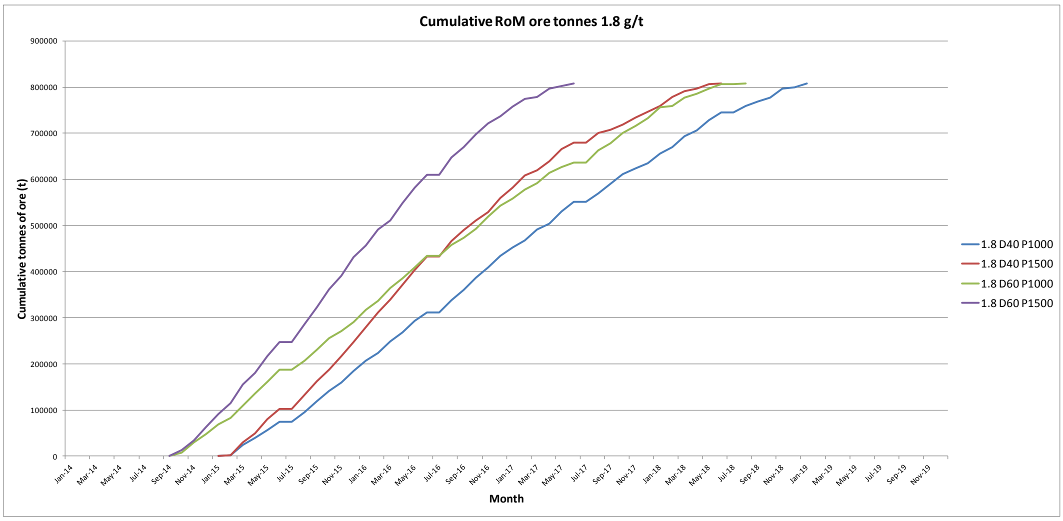
	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	NPV	IRR
Option 1	-108,260,000	-2,490,498	116,175,091	37,796,861	5,801,797		16,225,845	18%
Option 2	-108,260,000	621,895	132,995,107	39,449,837	5,812,245		32,570,689	25%
Option 3	-108,260,000	1,467,657	132,457,613	42,637,689	5,801,797		35,036,698	26%
Option 4	-108,260,000	-3,336,261	116,712,584	34,609,009	5,812,245		13,759,836	16%

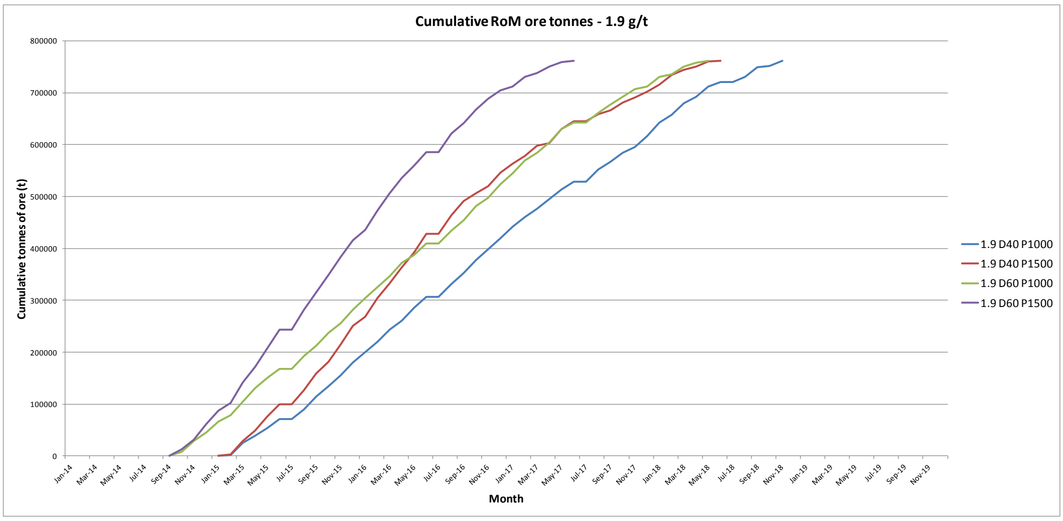
2. ROM Cumulative tonnage curves

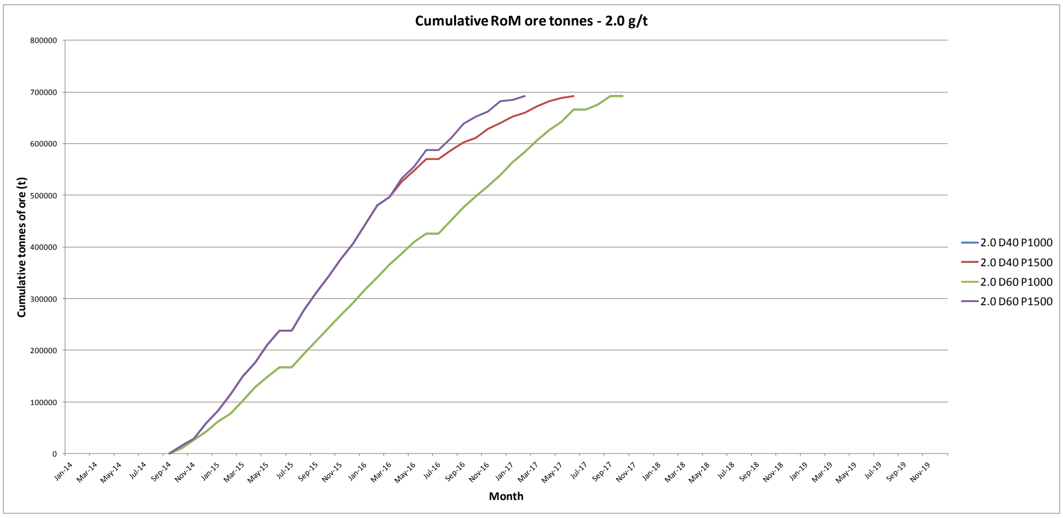




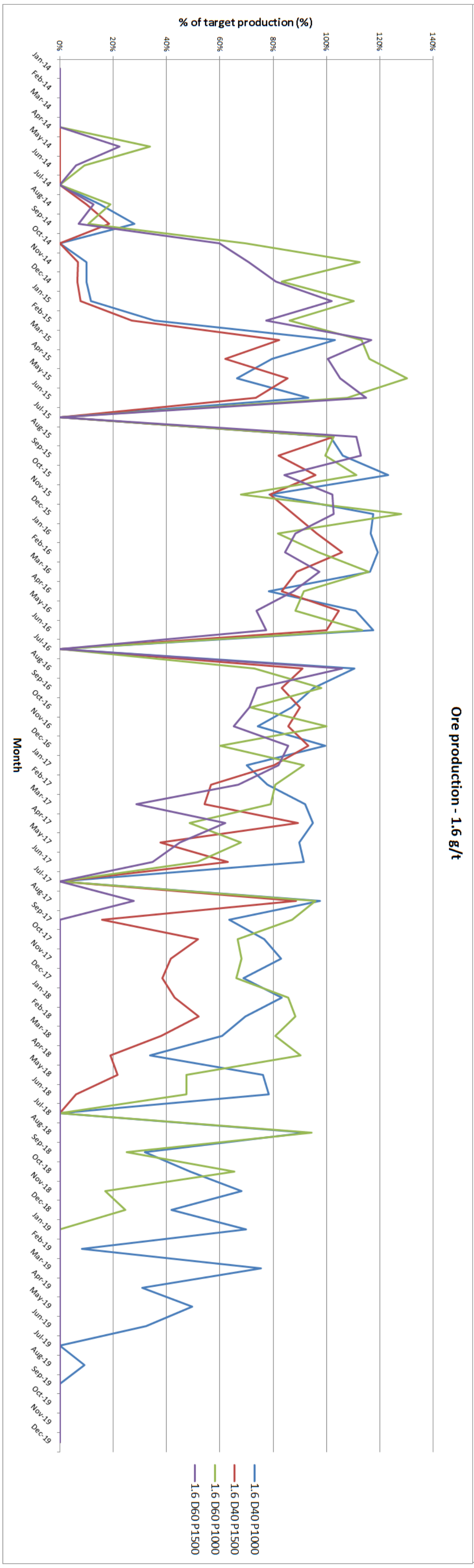


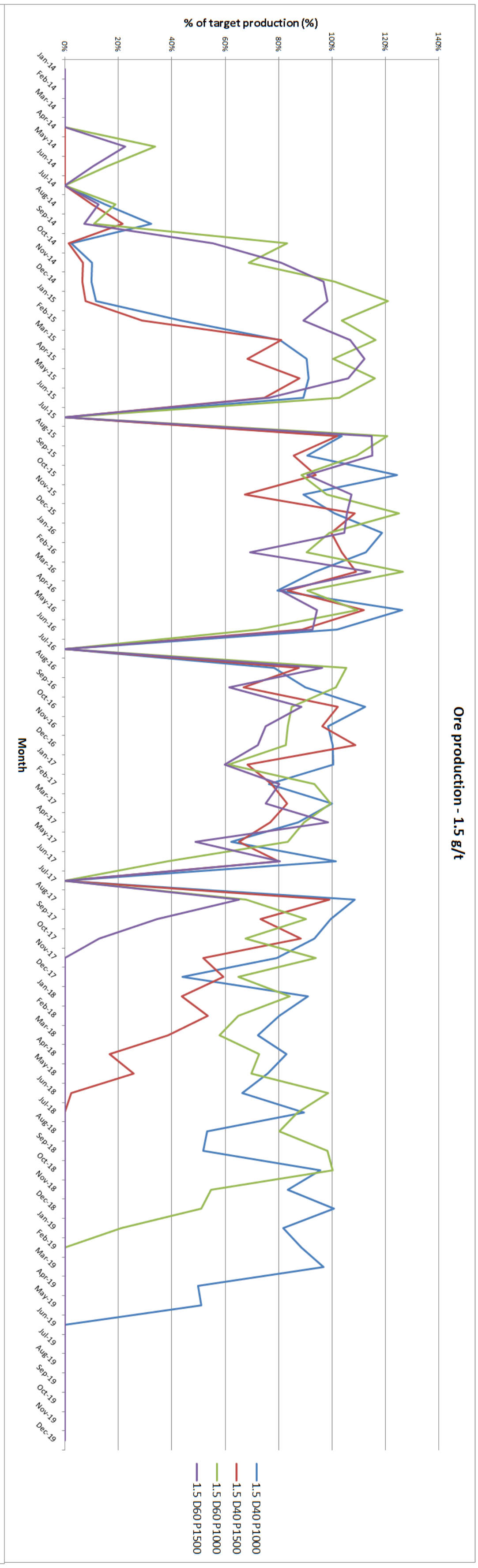

Cumulative RoM ore tonnes 1.8 g/t
Cumulative tonnes of ore (t)
Month
1.8 D40 P1000
1.8 D40 P1500
1.8 D60 P1000
1.8 D60 P1500

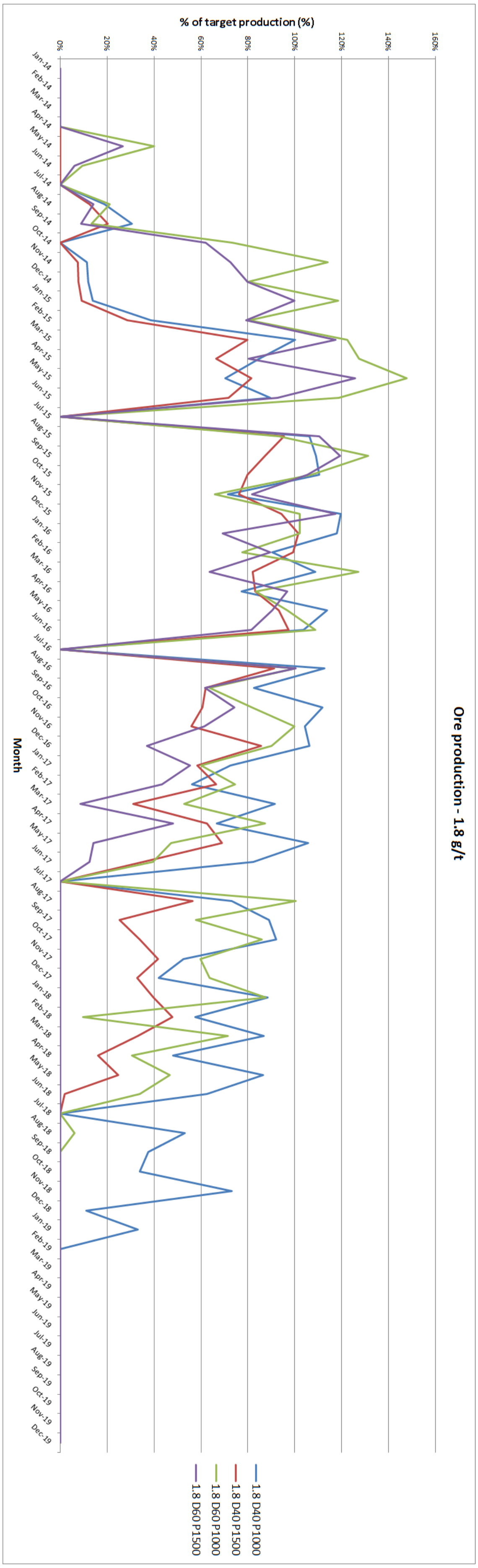

Cumulative RoM ore tonnes - 1.9 g/t
Cumulative tonnes of ore (t)
Month
1.9 D40 P1000
1.9 D40 P1500
1.9 D60 P1000
1.9 D60 P1500

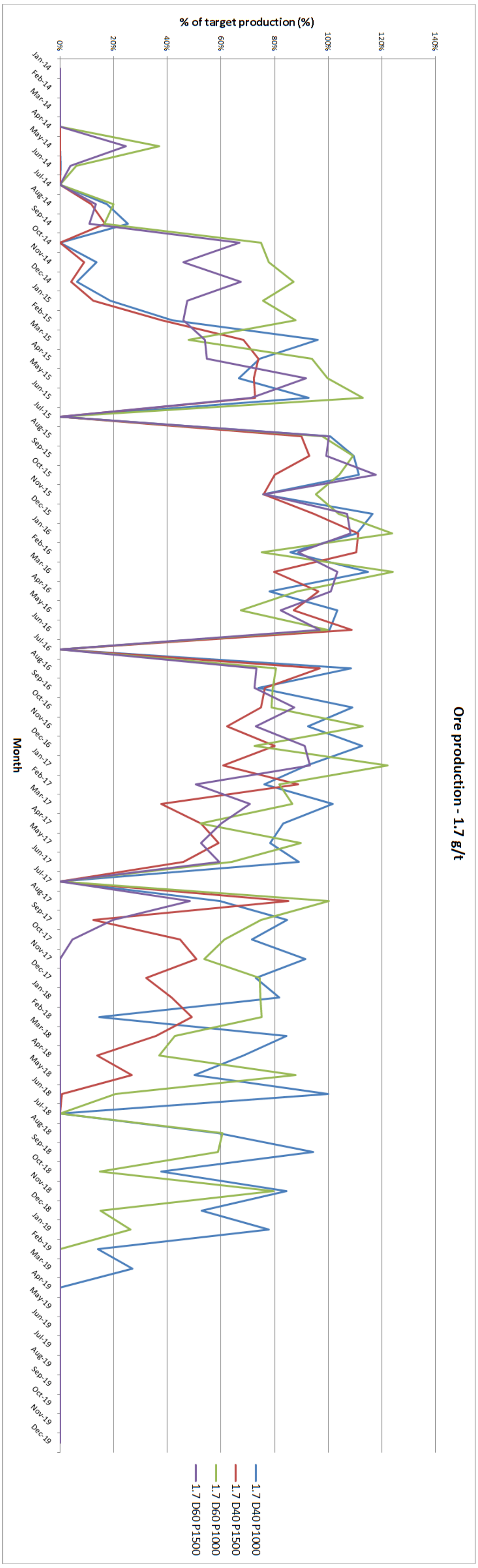

Cumulative RoM ore tonnes - 2.0 g/t
Cumulative tonnes of ore (t)
Month
2.0 D40 P1000
2.0 D40 P1500
2.0 D60 P1000
2.0 D60 P1500

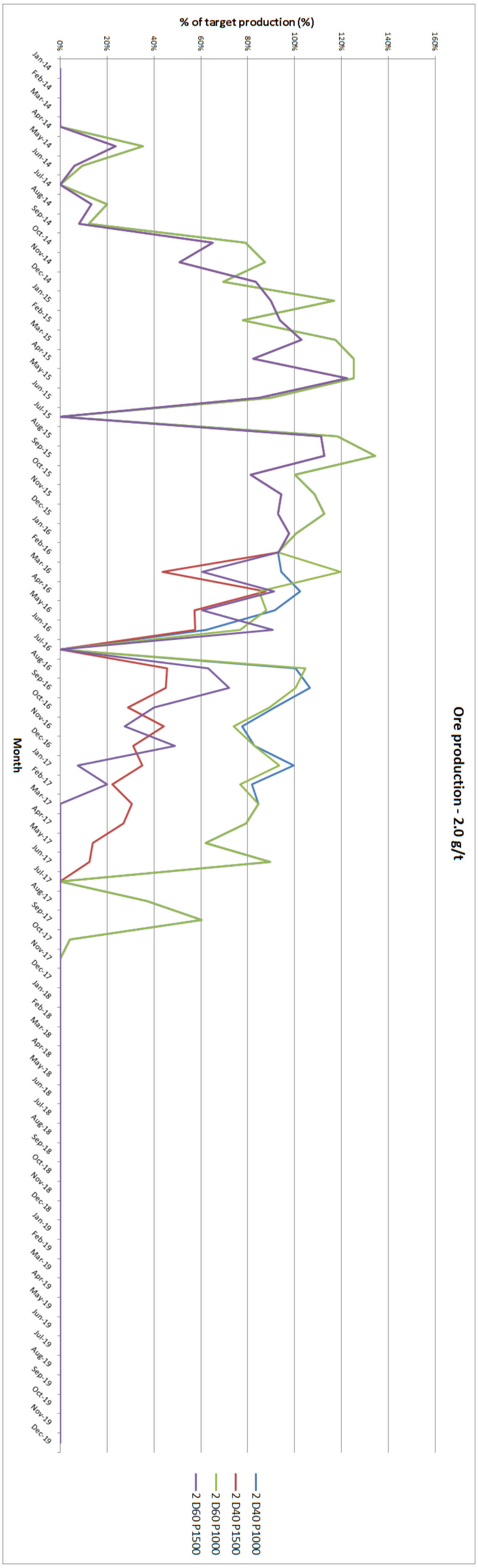
3. Target Production Capacity Utilization

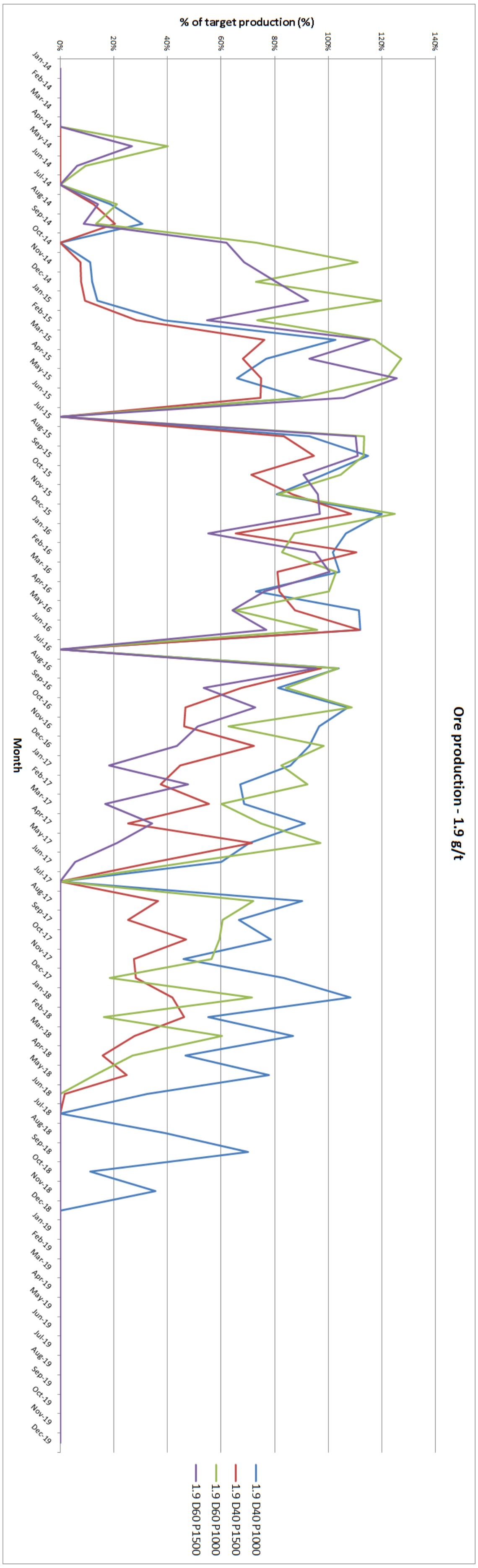





Ore production - 1.8 g/t
% of target production (%)
0%
20%
40%
60%
80%
100%
120%
140%
160%
Month
1.8 D40 P1000
1.8 D40 P1500
1.8 D60 P1000
1.8 D60 P1500


Ore production - 1.7 g/t
% of target production (%)
0%
20%
40%
60%
80%
100%
120%
140%
Month
1.7 D40 P1000
1.7 D40 P1500
1.7 D60 P1000
1.7 D60 P1500


Ore production - 2.0 g/t
% of target production (%)
Month
2 D40 P1000
2 D40 P1500
2 D60 P1000
2 D60 P1500


Ore production - 1.9 g/t
% of target production (%)
Month
1.9 D40 P1000
1.9 D40 P1500
1.9 D60 P1000
1.9 D60 P1500

10 APPENDIX C – GRADE RISK CASE STUDY

1. *Älgträsk resource estimation method (Boliden type)*
2. *Estimated block models*
3. *Creation and Validation of the Omni-directional variogram simulations*
4. *Confidence levels in estimated model based stope designs*
5. *Optimized stope designs at minimum target confidence levels (directional model)*
6. *JORC Guidelines to resource classification*

1. Älgträsk resource estimation method (Boliden type)

Variogram parameters

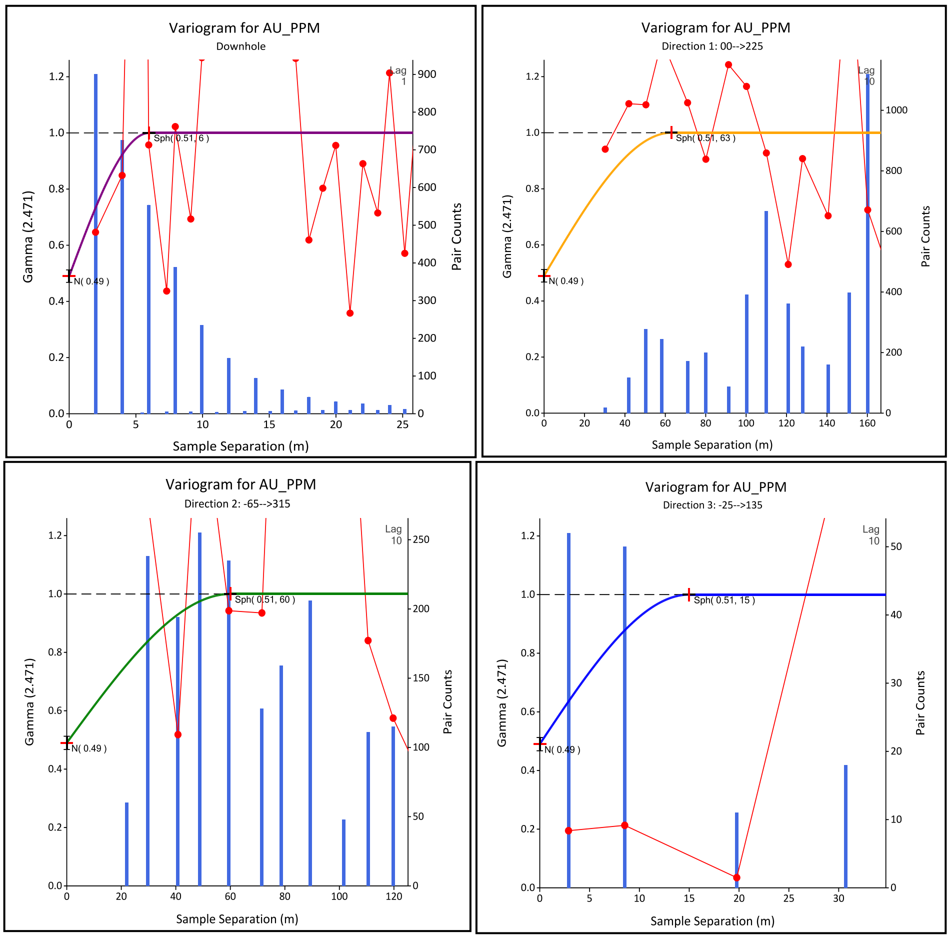
Snowden SuperVisor has been used to evaluate the estimation parameters, for exact estimation parameters see document MRE ALC 2013.xls DMS#603821.

The different domains within Nyhem had to be added together due to the too small sample number if kept separate and since the orientation of the Nyhem mineralizations is fairly similar this was decided to be acceptable. The different orientations of the mineralizations in the Liden area resulted in a too small sample population to be able to create good variograms.

Due to the nugget nature of gold the variograms were noisy but a variogram was created for the dataset called Au komp N bh (gold composites from drillhole data within the Nyhem mineralized area). Log and Normal Score transformation of the variograms were tested but the results did not improve.

The variograms created for Au were used for all the mineralized zones but together with different search ellipses.

Downhole	Omnidirectional
Direction 1	Direction of maximum continuity (along strike and plunge)
Direction 2	Direction of intermediate continuity
Direction 3	Direction of minimum continuity (across strike)



Grade estimation parameters

Search Volumes

For the estimation an anisotropic search ellipse was used. The parameters for the search ellipse are stored in the estparsv.dm file. Four different search volumes were used with different rotations around x, z, x considering the orientation of the different mineralization's/domains:

Domain	Angle 1	Angle 2	Angle 3
1-8	135	115	0
9	180	115	0
10	150	115	0
11	110	115	0

The same axial lengths were used for all search ellipses:

X: 60 m, Y: 60 m, Z: 15 m

Samples for each search volume:

Number of samples for the primary search volume: Min: 4, Max: 12

Number of samples for the second search volume (exp. factor 2): Min: 4, Max: 12

Number of samples for the third search volume (exp. factor 4): Min: 4, Max: 12

Due to the occasionally small amount of samples available for domain 11 the third search volume for domain 11 used the following criteria:

Number of samples for the third search volume (exp. factor 4): Min: 1, Max: 12

Estimation method

Au was estimated by using 'Ordinary Kriging'. Grades for Ag, Cu, Zn, Pb, Mo and S were estimated by using inverse distance.

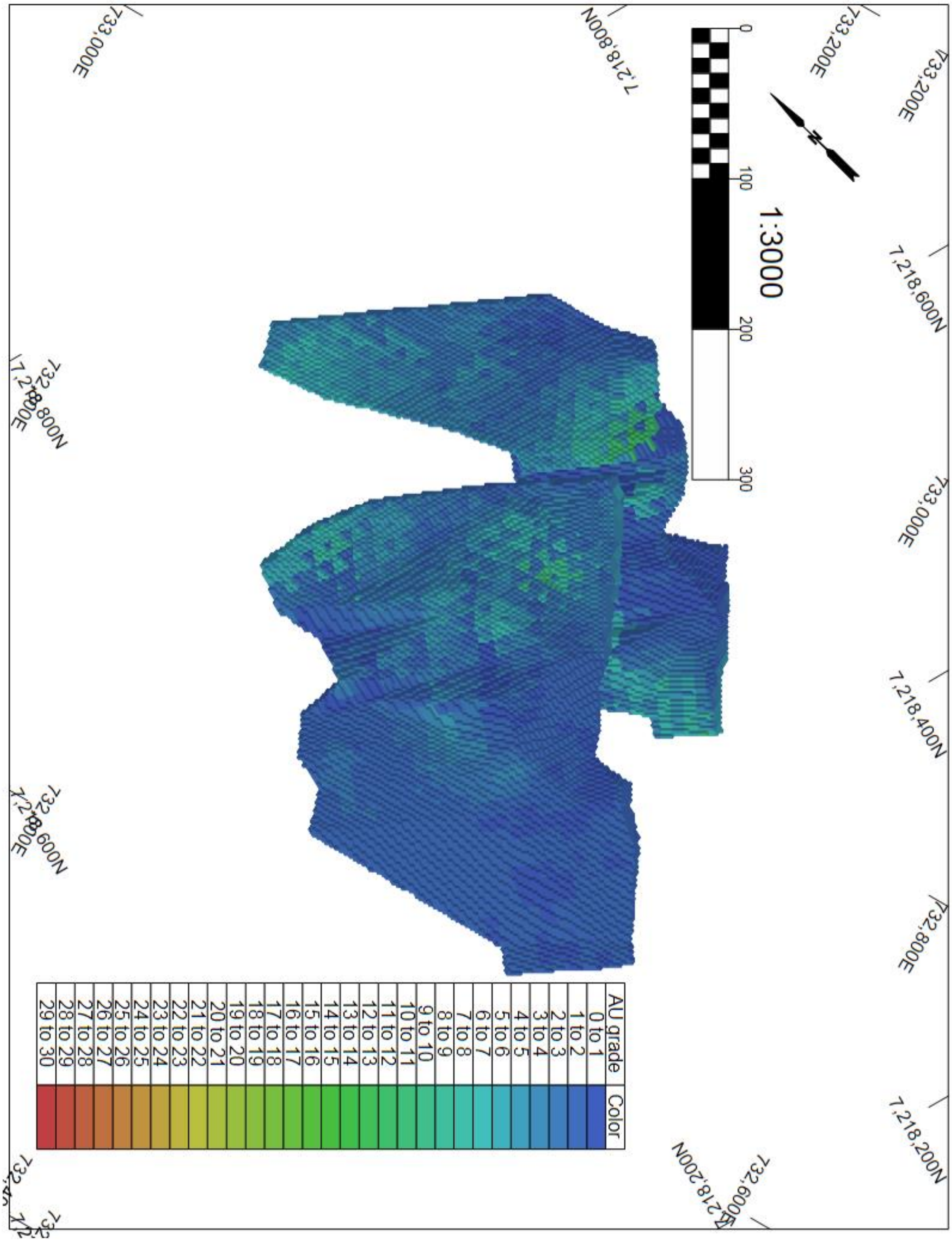
Since the Älgträsk area has been drilled both using core drilling and RC drilling and all of the data is used in this model the different drill hole methods were kept separate during the estimation and added together after the estimation by using weighting according to the variance.

A = (VARAU_RC/(VARAU_BH+VARAU_RC)

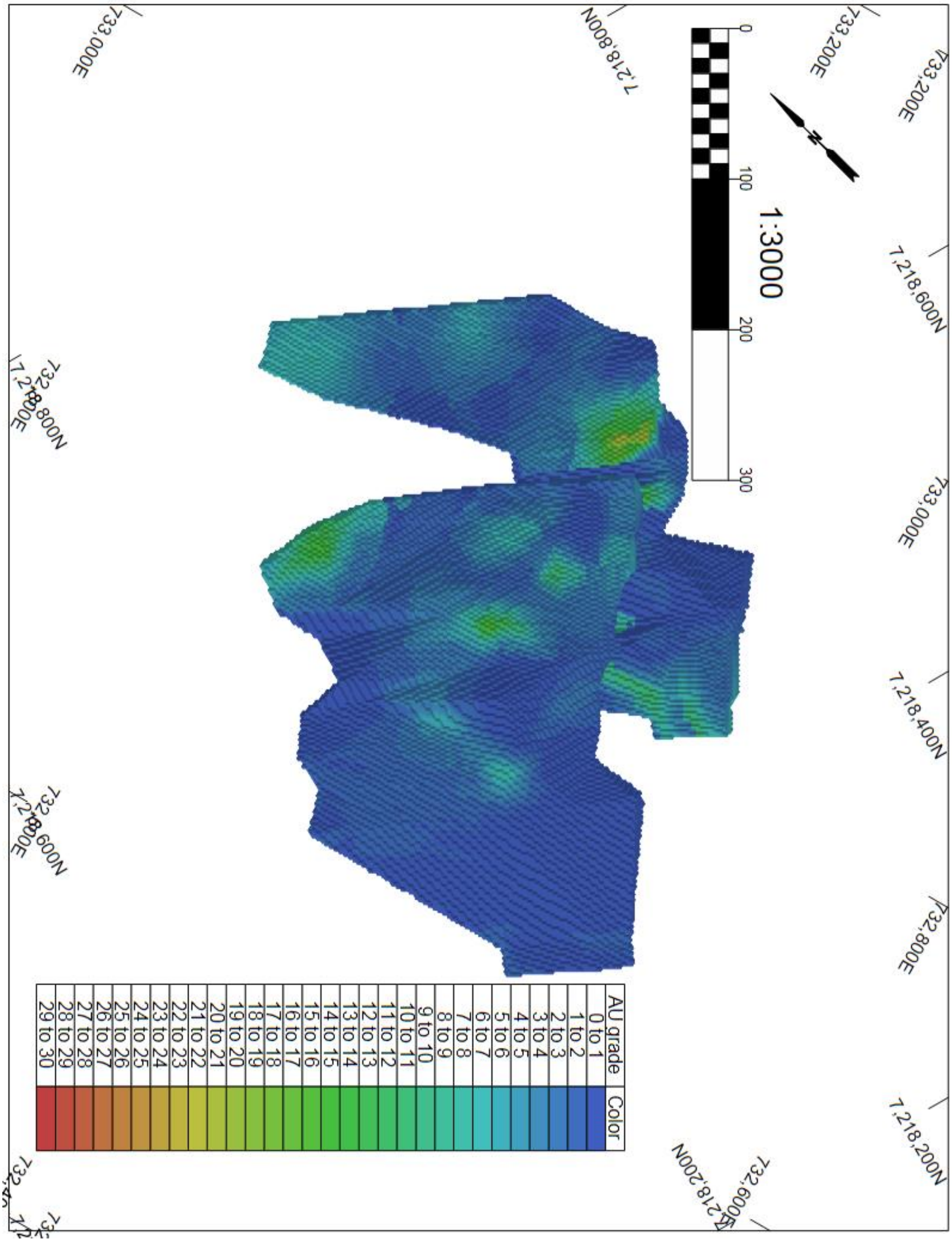
*AU_PPM = A*AU_BH + (1-A)*AU_RC*

2. Estimated block models

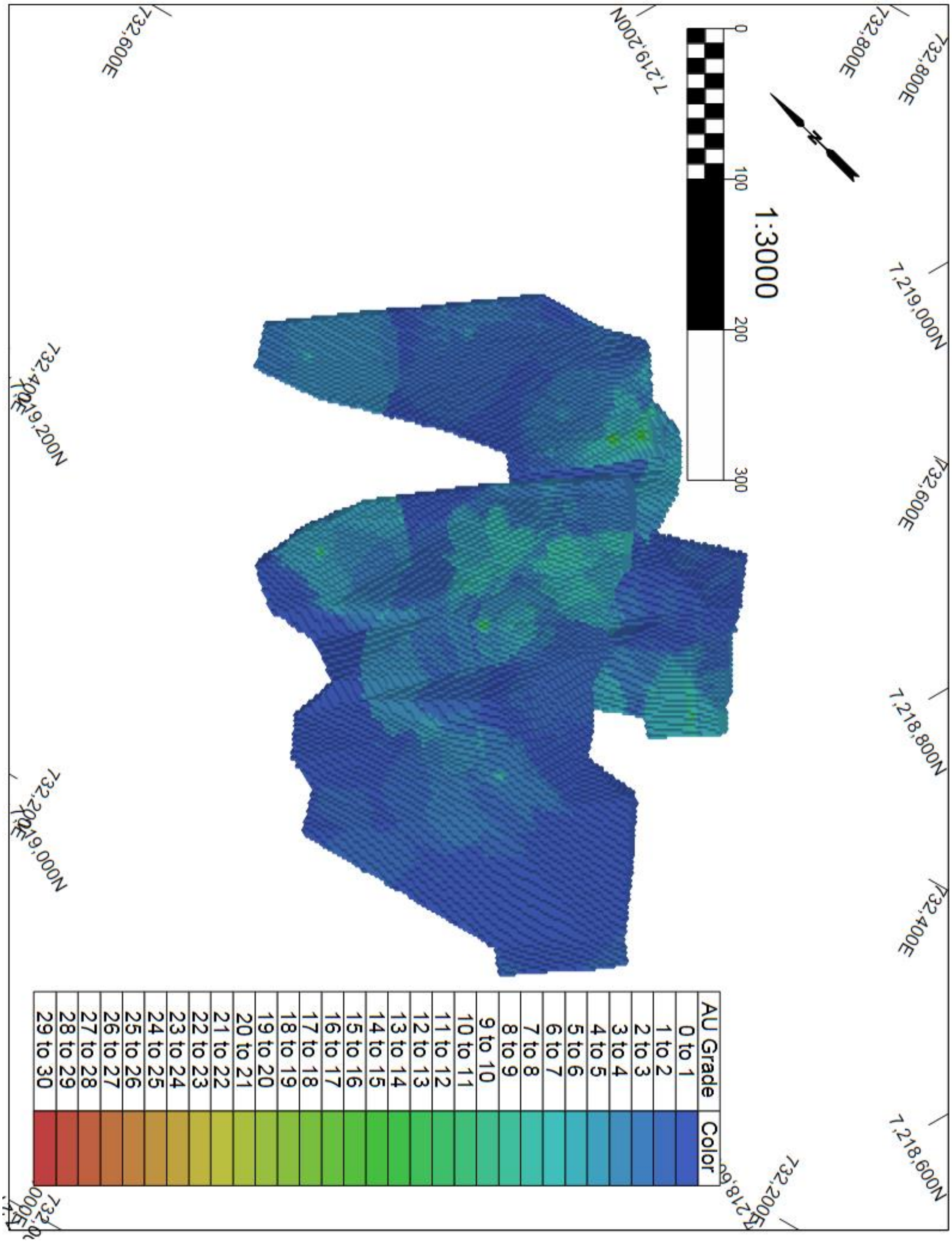
Boliden Model



Directional variogram model



Omni-directional variogram model



3. Creation and Validation of the Omni-directional variogram simulations

Because a single drill hole dataset is used in this study, Sequential Gaussian Simulation is the method of choice to create simulated block models. GSLIB (Stanford University, 2009) was used to perform the simulations. The Geostatistical software Library is a set of freely available software to perform sequential Gaussian simulations.

Again two sets of simulations were created based on the same Directional and Omni-directional variograms as used to create the estimated block models. The process of creating and validating the simulations is explained here for the omni-directional model and follows the same approach as the directional variogram simulations.

SIMULATION BLOCK MODEL REQUIREMENTS

In order to correctly simulate block grades a regular block size has to be selected as the simulation software cannot deal with sub-blocking. Due to the limited thickness of the individual ore lenses a small block size of 2.5x2.5x2.5 metre was selected (the minimum sub-block size used in the Boliden block model). The small block size ensures a good fit to the geological wireframes and as simulations were limited to the Algtrask Area of Interest, computation times remained acceptable.

Another important criterion for the selection of a small block size is that a small block size reduces the optimization error when using Stope Optimizer. Stope Optimizer uses only block model cells for optimization and therefore a better ‘geological fit’ of the block model cells to the geological boundaries ensures a more correct optimization of the stope boundaries and minimizes the stope headgrade estimation error.

SIMULATION APPROACH

The ore lenses were simulated with the according drill hole data for the specific zone (in the same way as Ordinary Kriging estimation with zonal control is carried out) and the resulting models were later combined into one final model. By simulating the zones separately, grade contamination between closely spaced ore lenses is avoided. 25 simulations of the orebody were generated and some example simulations and the simulations ETYPE-model are shown in Figure 119 together with the Ordinary Kriging Estimated solution.

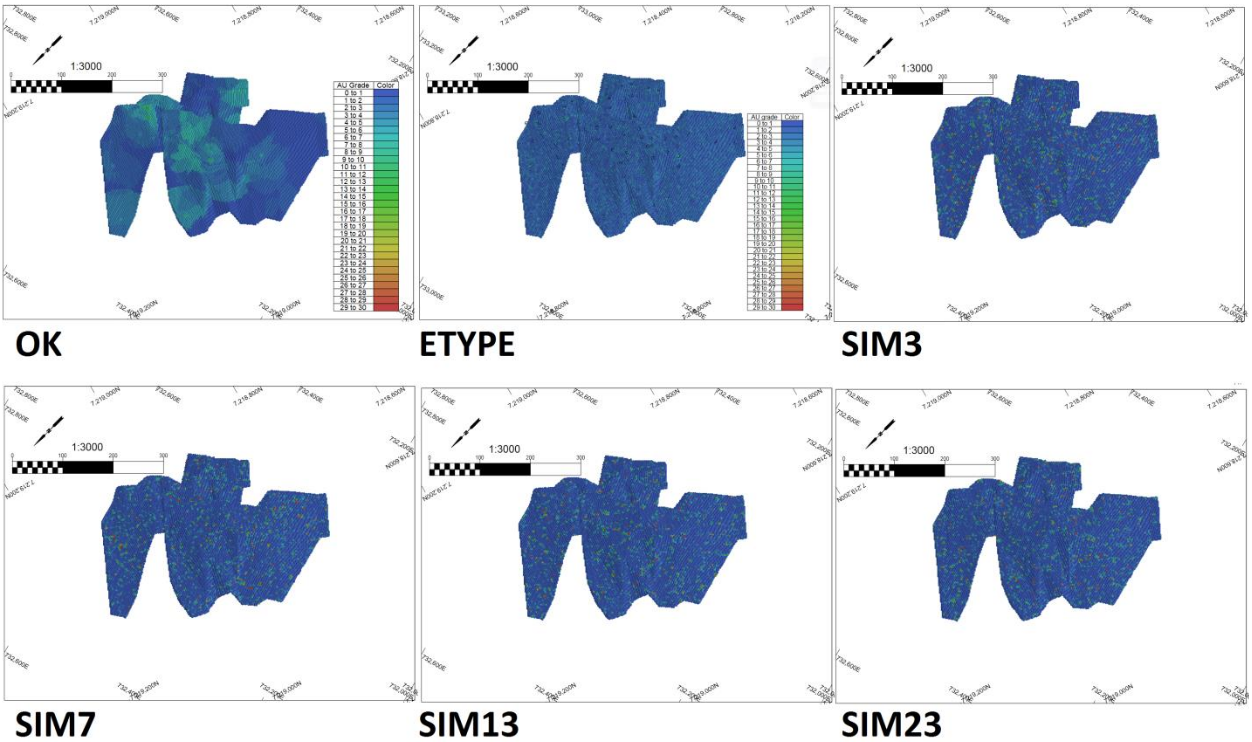


Figure 119 - Simulation examples and ETYPE (with OK block model for reference purposes).

SIMULATION VALIDATION

Before the simulated block models can be compared to the corresponding Ordinary Kriging Estimated block model and any conclusions can be drawn on grade risk the simulations have to be validated. The approach to validate Gaussian simulations follows 3 distinct steps and each step is explained in this section.

Grade Distribution Reproduction

The first assumption in sequential Gaussian simulation is that the grade distribution found in the drill holes is in fact the true grade distribution of the whole deposit. As such the simulated block models should have the same grade distribution as the composite data.

The grade distributions of all 25 simulated block models are shown in Figure 119 together with the grade distribution of the composite data (green) and the grade distribution resulting from the Omni-directional Ordinary Kriging Estimated block model. It is concluded that the simulated block models have approximately the same grade distribution as the composite data and therefore, on grade distribution only, the simulated block models are a valid interpretation of the grade distribution in the deposit.

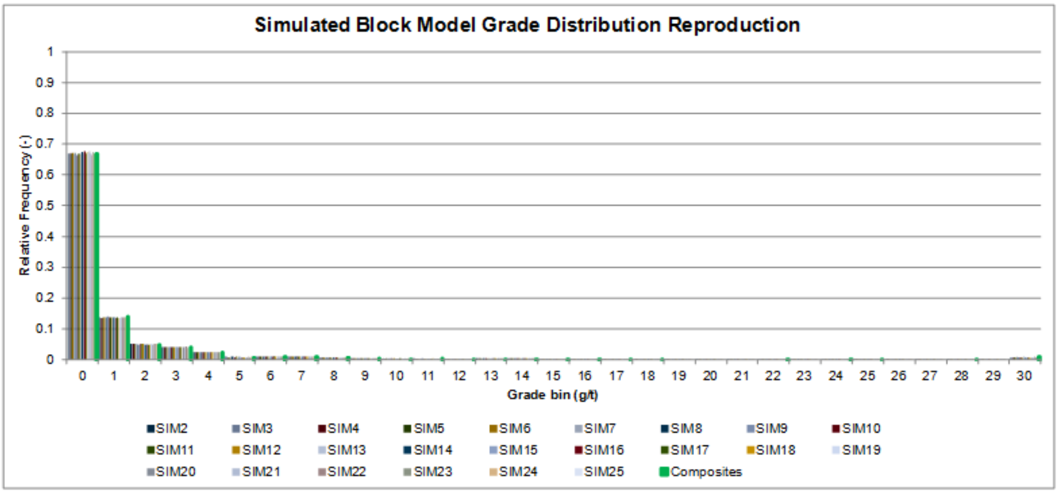


Figure 120 - Grade Distribution Analysis between Simulated and Composite data.

Variogram Reproduction

The subsequent step in the validation process is variogram reproduction. The simulated grades should have the same spatial relationship as modeled by the variograms and in order for the simulations to be valid it should be possible to reproduce the variograms as determined from the composite data. Experimental variograms were produced for all simulated block models and compared to the modeled variograms.

The variogram reproductions in the major, semi-major and minor direction of continuity are presented in Figure 121. When comparing the average experimental variogram of all simulations (red line, Figure 121) to the theoretical model (green line, Figure 121) as found during variogram modeling it is observed that the sill and range of the simulation based experimental variograms are the same as the theoretical model (6m). Some deviation from the theoretical model is observed but the overall shape of the experimental variograms is in line with the theoretical variogram and the variogram reproduction is deemed acceptable. However, whenever more exploration data becomes available, directional variogram modeling should again be tested in order to find the directional spatial relationships within the ore zones.

Measured Data Reproduction

The final check to validate the simulated block models is to visually check if measured grades are reproduced in all simulated block models as fixed values at their measured location. A visual check confirmed this as was to be expected because the measured data has been assigned to the grid nodes before simulations were executed.

All validation checks to the simulation results proved satisfactory and it is therefore concluded that the simulations can be compared to the respective Ordinary Kriging Estimated model and be used for risk assessment.

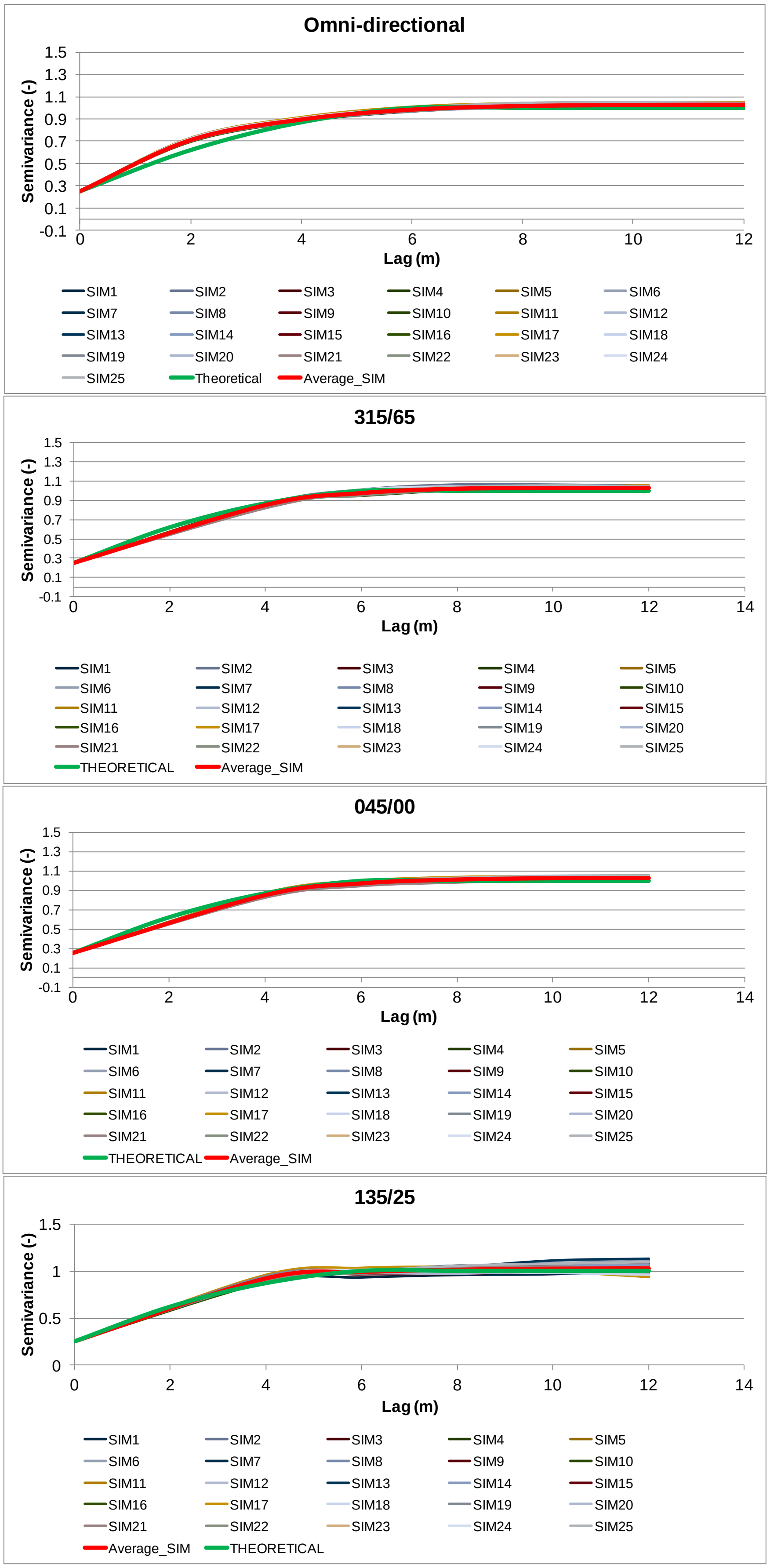
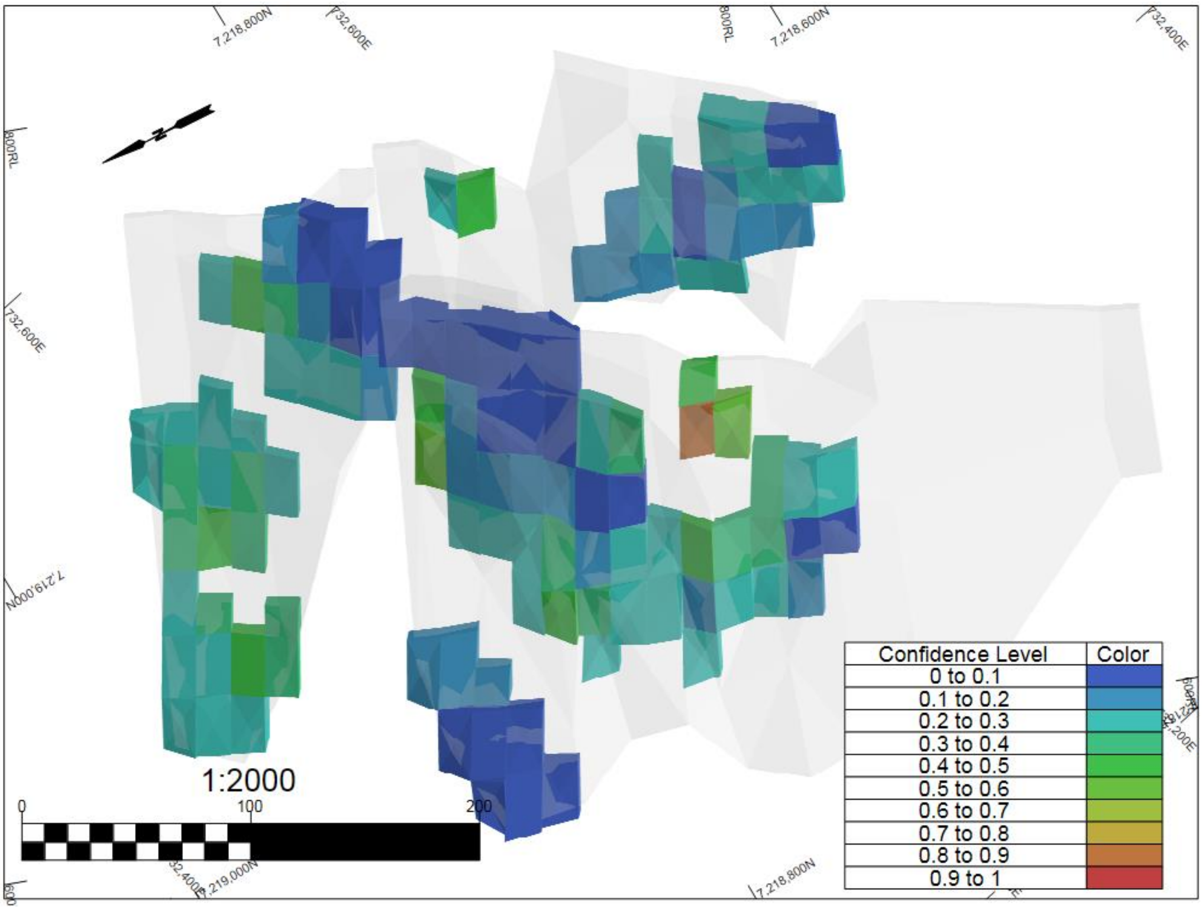


Figure 121 - Simulations Variogram Reproduction.

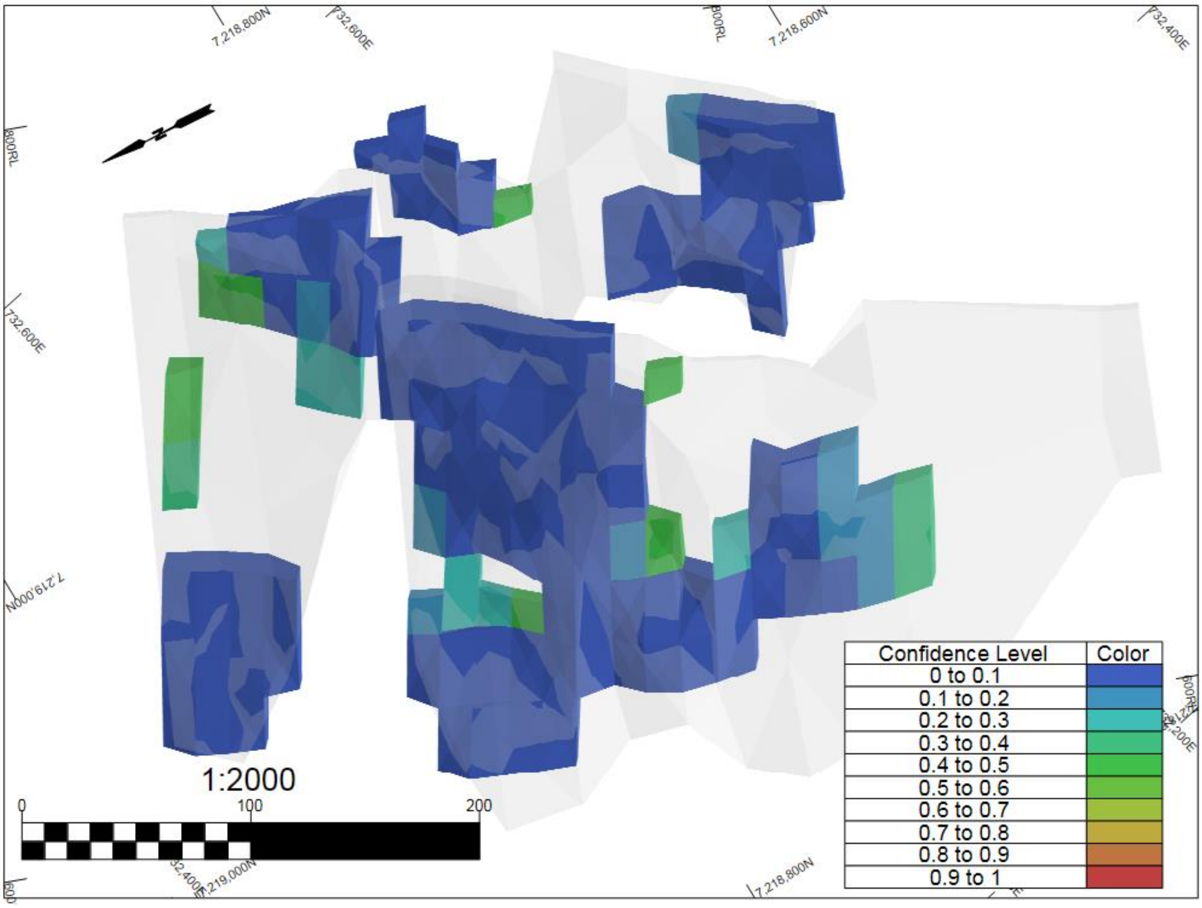
4. Confidence levels in estimated model based stope designs.

These plots show the probability that the estimation based optimized stope designs will contain the estimated head grade.

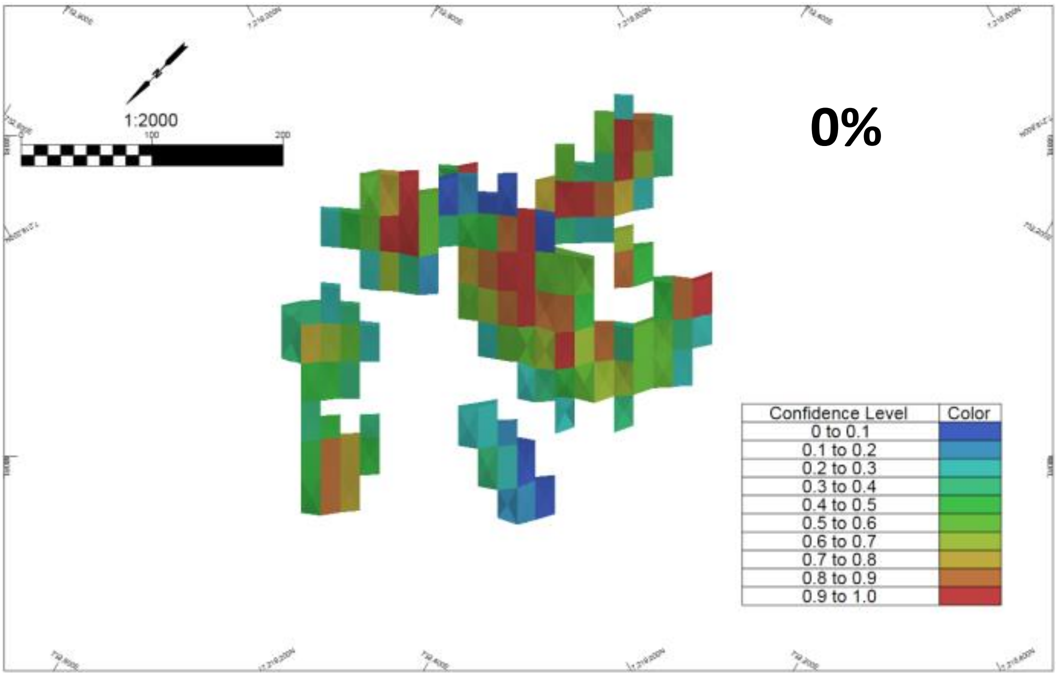
Directional Model

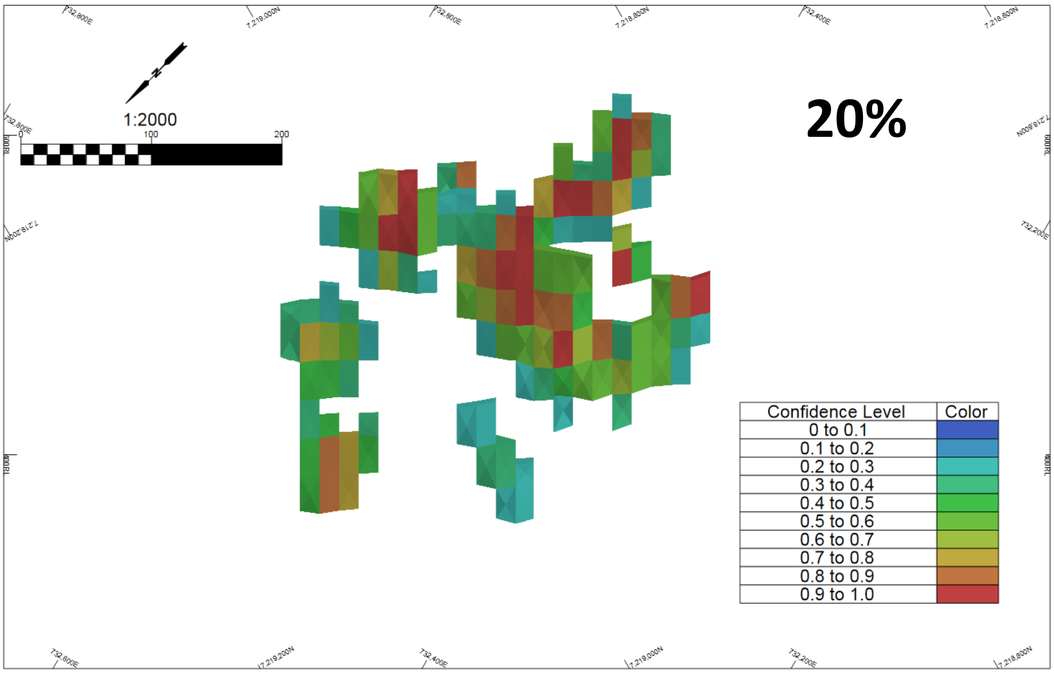


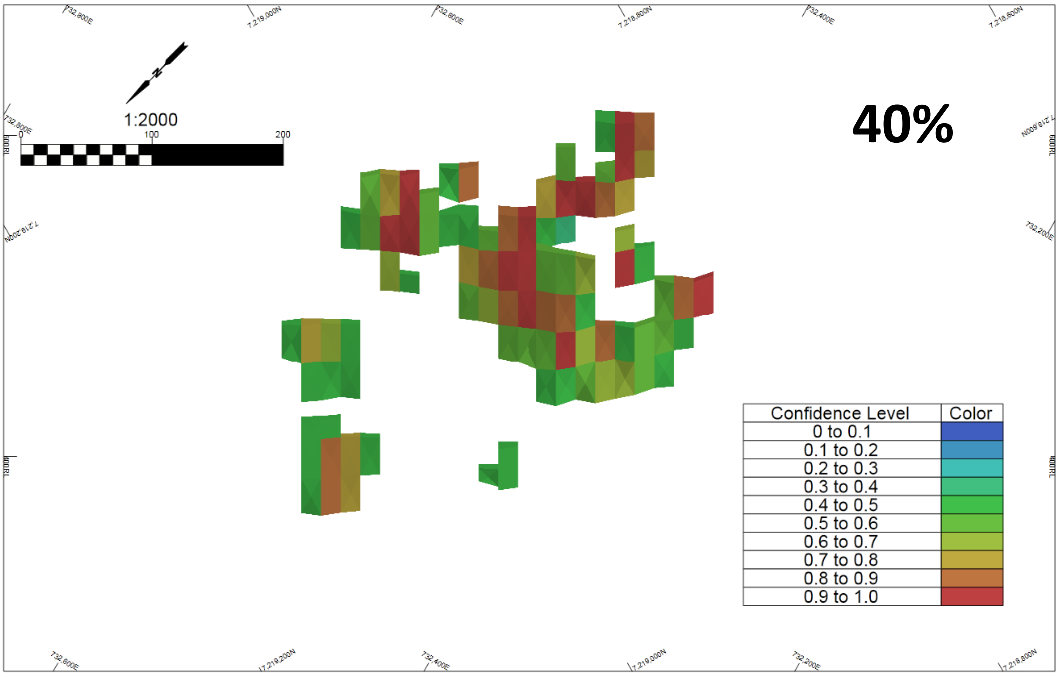
Omni-directional model

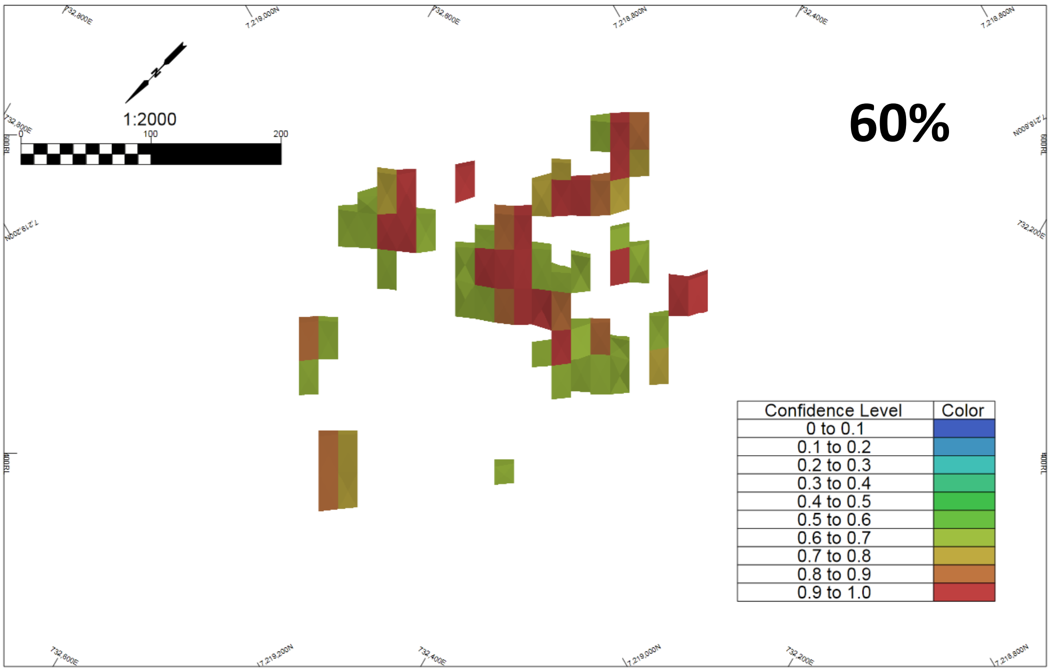


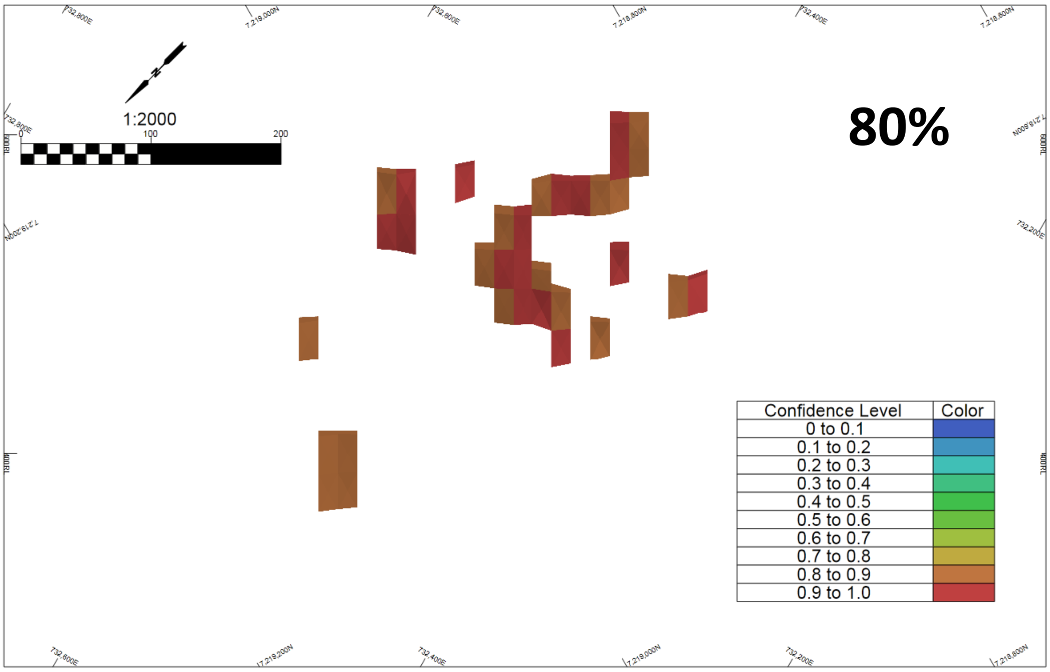
5. Optimized stope designs at minimum target confidence levels (directional model)






1:2000
0
100
200
40%
Confidence Level
Color
0 to 0.1
0.1 to 0.2
0.2 to 0.3
0.3 to 0.4
0.4 to 0.5
0.5 to 0.6
0.6 to 0.7
0.7 to 0.8
0.8 to 0.9
0.9 to 1.0


1:2000
0
100
200
60%
Confidence Level
Color
0 to 0.1
0.1 to 0.2
0.2 to 0.3
0.3 to 0.4
0.4 to 0.5
0.5 to 0.6
0.6 to 0.7
0.7 to 0.8
0.8 to 0.9
0.9 to 1.0


1:2000
0
100
200
80%
Confidence Level
Color
0 to 0.1
0.1 to 0.2
0.2 to 0.3
0.3 to 0.4
0.4 to 0.5
0.5 to 0.6
0.6 to 0.7
0.7 to 0.8
0.8 to 0.9
0.9 to 1.0

6. JORC Guidelines to resource classification

Reporting of Mineral Resources

20. **A 'Mineral Resource' is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade (or quality), and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade (or quality), continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling. Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories.**

    All reports of Mineral Resources must satisfy the requirement that there are reasonable prospects for eventual economic extraction (ie more likely than not), regardless of the classification of the resource.

    Portions of a deposit that do not have reasonable prospects for eventual economic extraction must not be included in a Mineral Resource. The basis for the reasonable prospects assumption is always a material matter, and must be explicitly disclosed and discussed by the Competent Person within the Public Report using the criteria listed in Table 1 for guidance. The reasonable prospects disclosure must also include a discussion of the technical and economic support for the cut-off assumptions applied.

    Where untested practices are applied in the determination of reasonable prospects, the use of the proposed practices for reporting of the Mineral Resource must be justified by the Competent Person in the Public Report.

    Geological evidence and knowledge required for the estimation of Mineral Resources must include sampling data of a type, and at spacings, appropriate to the geological, chemical, physical, and mineralogical complexity of the mineral occurrence, for all classifications of Inferred, Indicated and Measured Mineral Resources. A Mineral Resource cannot be estimated in the absence of sampling information.

21. **An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade (or quality) are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade (or quality) continuity. It is based on exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.**

    An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to an Ore Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

    The Inferred category is intended to cover situations where a mineral concentration or occurrence has been identified and limited measurements and sampling completed, but where the data are insufficient to allow the geological and grade continuity to be confidently interpreted. While it would be reasonable to expect that the majority of Inferred Mineral Resources would upgrade to Indicated Mineral Resources with continued exploration, due to the uncertainty of Inferred Mineral Resources, it should not be assumed that such upgrading will always occur.

    Confidence in the estimate of Inferred Mineral Resources is not sufficient to allow the results of the application of technical and economic parameters to be used for detailed planning in Pre-Feasibility (Clause 39) or Feasibility (Clause 40) Studies. For this reason, there is no direct link from an Inferred Mineral Resource to any category of Ore Reserves (see Figure 1).

22. **An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade (or quality), densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.**

    Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes, and is sufficient to assume geological and grade (or quality) continuity between points of observation where data and samples are gathered.

    An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Ore Reserve.

    Mineralisation may be classified as an Indicated Mineral Resource when the nature, quality, amount and distribution of data are such as to allow confident interpretation of the geological framework and to assume continuity of mineralisation.

    Confidence in the estimate is sufficient to allow application of Modifying Factors within a technical and economic study as defined in Clauses 37 to 40.

23. **A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade (or quality), densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.**

Geological evidence is derived from detailed and reliable exploration, sampling and testing gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes, and is sufficient to confirm geological and grade (or quality) continuity between points of observation where data and samples are gathered.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proved Ore Reserve or under certain circumstances to a Probable Ore Reserve.

Mineralisation may be classified as a Measured Mineral Resource when the nature, quality, amount and distribution of data are such as to leave no reasonable doubt, in the opinion of the Competent Person determining the Mineral Resource, that the tonnage and grade of the mineralisation can be estimated to within close limits, and that any variation from the estimate would be unlikely to significantly affect potential economic viability.

This category requires a high level of confidence in, and understanding of, the geological properties and controls of the mineral deposit.

Confidence in the estimate is sufficient to allow application of Modifying Factors within a technical and economic study as defined in Clauses 37 to 40.