Aalto University
School of Engineering

Dirk Joost Korte

Cut-off Grade Based Stope Optimisation to Maximise Value at the Garpenberg Mine

Master's thesis
School of Engineering

The Hague, 29.9.2019
Supervisor: Prof. Mikael Rinne
Advisors: Prof. Mikael Rinne
Hilmi Pehriz

Aalto University
MASTER'S THESIS 2019

Cut-off Grade Based Stope Optimisation

To Maximise Value at the Garpenberg Mine

Dirk Joost Korte

Thesis submitted in partial fulfillment of the requirements for the degree of Master of Science in Technology.
The Hague, September 29, 2019

Supervisors: Prof. Mikael Rinne (Aalto University)
Prof. Dr. Mike Buxton (Delft University of Technology)
Univ.-Prof. Dr. Bernd Lottermoser (RWTH)

Advisors: Prof. Mikael Rinne
Hilmi Pehriz (Boliden Minerals AB)

Aalto University
School of Engineering
European Mining Course

Aalto University

Aalto University, P.O. Box 11000, FI-00076 Aalto www.aalto.fi

Abstract

Author
Dirk Joost Korte

Title
Cut-off Grade Based Stope Optimisation to Maximise Value at the Garpenberg Mine

School School of Engineering

Master's programme European Mining, Minerals and Environmental Programme (EMMEP)

Major European Mining Course (EMC) **Code** ENG3077

Supervisor Prof. Mikael Rinne

Advisors Prof. Mikael Rinne and Hilmi Pehriz

Level Master's thesis **Date** September 29, 2019 **Pages** 77/2 **Language** English

Abstract

This research was aimed at determining the influence parameters have on the cut-off grade of sublevel stopes in the Garpenberg underground mine, and combing these to maximise value at the mine. Garpenberg is a multi-deposit poly-metallic mine owned by Boliden AB. Cost calculations were performed on data from 2018 to derive costs for each deposit and activity. Ore transport and rock bolting were two parameters which were suitable to be used for variable cost implementation. These costs were assigned to the different orebodies and cut-off grades were adjusted to reach a break-even level. Stopes were created using the Stope Optimizer software in Deswik, using the variable cut-offs and the technical constraints previously used for Garpenberg's Life of Mine Plan. Cut-offs were elevated or depressed to optimise NPV for each deposit separately and the optimal combination yielded an NPV increase of 530 Mkr, a 4% increase, compared to using static cut-off grades. More value can be obtained by combining cut-off strategies in Garpenberg as whole, in contrast to separate optimisation. Scheduling of newly created stoping areas within the Lappberget deposit proved to be beneficial for value. Finally, a financial analysis into double stopes of 50 m high was performed. The elimination of production levels provides the main cost reduction. However, lower selectivity and expected dilution will limit the financial gain from this method.

Keywords Cut-off, Sublevel Stoping, Garpenberg, Optimisation, Variable cut-off Scheduling,

url

Contents

List of Figures

List of Tables

Abbreviations

bkr billion Swedish Kronor

CAF Cut-and-Fill

CMS Cavity Monitoring System

CoG Cut-off Grade

HoV Hill of Value

HR Hydraulic Radius

kkr Thousand Swedish Kronor

Kt Kilo ton (Thousand Ton)

LHD Load Haul Dump

LoM Life of Mine

LOMP Life of Mine Plan

MIP Mixed Integer Programming

Mkr Million Swedish Kronor

MVV Mining Variable Value

NPV Net Present Value

NSR Net Smelter Return

RMR Rock Mass Rating

SEK Swedish Krona

SO Stope Optimizer

1. Introduction

Mine planning is an important part of the process in the operation of a mine, and it includes all plans which are made, and all decisions taken to allow for the most optimal extraction of the valuable. Mine planning includes broad multi-year plans which describe the goals of the mining project and include elements such as the general life of mine (LoM), up to detailed plans describing the allocation of equipment and crews within the mine, and all stages in between these plans (Hall 2014). Mine planning for open pit mining or underground mining require the same steps, but include different methods.

Underground mines require planning and construction of declines, shafts, development drifts, stopes and more. An underground mine will often extract the valuable material by constructing stopes. Stopes are openings in the ground from which the valuable can be extracted (King & Newman 2018). The manner in which these stopes are planned and designed will greatly affect the value of a mining operation. To be able to create more value from an existing deposit, a mining operation will aim at optimising the design of the mine and their stopes. A stope can have varying sizes and shapes. Its location and the scheduling of its extraction are two other important factors which will influence the mining operation. All these factors depend on different parameters, such as cut-off grade (CoG), the value of the ore, the geotechnical situation and the infrastructure of the mine. These parameters are different for every mine, and it is important to know the effect each parameter has on the stope design. This thesis aims to determine the influences several parameters have on the cut-off grade, and hence on the stope designs of the Garpenberg mine. This knowledge will be used to maximise the value of the mine.

This thesis is written at Boliden Minerals AB in Sweden. Boliden was founded in 1924 after finding a rich gold deposit nearby the village which is nowadays called

Figure 1.1. Location of Boliden's mines, smelters and offices. From: Boliden Group (2019)

Boliden. It operates nine open-pit and underground mines in Finland, Ireland and Sweden and processes raw materials into metals such as zinc, copper, gold, silver and more in five smelters in Finland, Norway and Sweden, as can be seen in figure 1.1.

1.1 Garpenberg

The Garpenberg mine is located in the southern part of Sweden, around 180 km North-West from Stockholm, figure 1.1 shows the Garpenberg mine and all other assets of Boliden. Earliest mining took place in the Iron Ages but Boliden acquired the mine in 1957. Ever since, the mine has expanded and is currently producing around 2.6 Mt of ore per year and production is ramping up until 3 Mt of ore per year. The mine is rich in zinc, lead and silver, which account for 90% of the revenue, while copper and gold make up the remaining 10% of revenue (Högnäs 2018).

The Garpenberg mining area contains several different ore lenses (which will be called deposits or orebodies throughout the report), which are illustrated in figure 1.3. This figure also indicates the orebodies which have been mined out in the past, like the Gransjön open pit in the North-East, and the Kanalmalmen deposit in the South-West. Currently, the ore is solely being extracted as an underground mine.

The newly discovered extensions of the orebodies are also shown in the figure. These deposits range between proven reserves and inferred resources, depending on their level of knowledge and confidence. The bottom end of the Lappberget deposit has not yet been reached, which will likely increase the current mineral reserve estimations of 76.1 Mt even further, allowing for a longer life of mine. (Högnäs 2018).

90% of the ore production is obtained by sublevel stoping, the rest is obtained by cut-and-fill (CaF) and tunnelling. Ore can be transported to tips by truck which will drop the ore into one of the two underground crushers, or it is removed from the working face first and stockpiled, after which it is transported to crushers later. The crushers are located at the 700 and 1087 level. From here, ore is crushed and hoisted up using a shaft. The Garpenberg mine has one ore shaft and one personnel shaft, as well as several ventilation and media shafts. Waste generated during development is transported using trucks which transport it either to the surface, or to secondary stopes to be used as backfill. All waste and ore transport by truck is operated by contractors.
When the ore reaches the surface it is placed on a stockpile and sent to grinding and milling stages, after which it is screened and flotation is used to produce zinc, lead and copper concentrate. The latter two include silver and gold as well. The concentrates are dewatered and the zinc and lead concentrates are transported by truck to the harbour and sent to one of the Boliden smelters. Copper concentrate is transported by rail to the Rönnskär smelter. (Högnäs 2018)

1.1.1 Geology

Garpenberg is located in a large sulphide deposit located in a heavily mineralised Paleoproterozoic igneous area. The host rock is mainly limestone which was formed in a shallow marine environment and altered to dolomite and skarns. It was folded, sheared and faulted into large syn- and antiform structures. Steeply dipping ore lenses occur at the contact zones between the altered limestones and the underlying siltstone. Mineralisation developed by penetration of metal-bearing fluid through faults created during volcanic activity, into the reactive limestones and created massive sulphide bodies (Högnäs 2018). A geological map is shown in figure 1.2 and indicates the main types or rock which are found in the mining area.

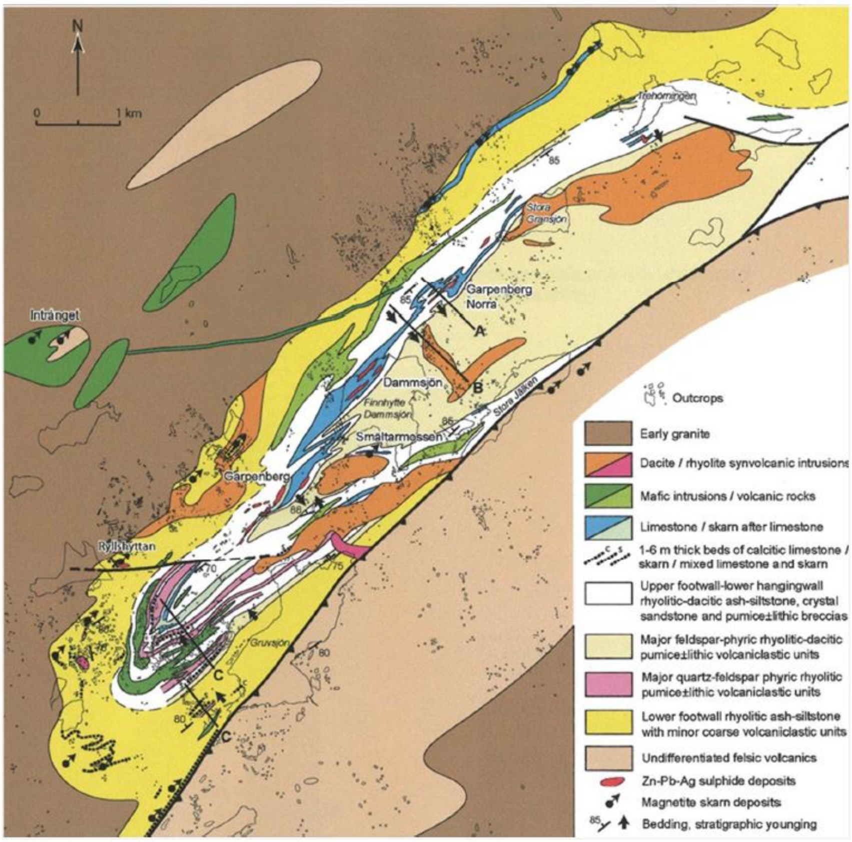


Figure 1.2. Geological map of the Garpenberg area. From: Högnäs (2018)

1.2 Mining Methods

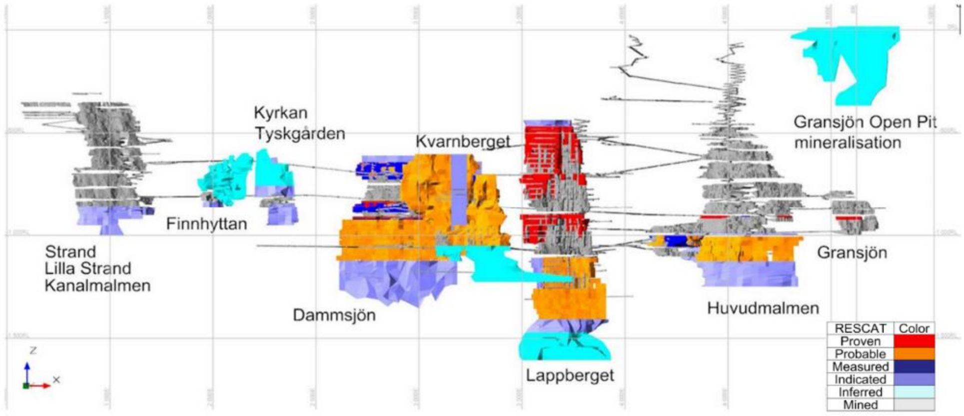
The Garpenberg Mine utilises two main mining methods: sublevel stoping and cut-and-fill. Some deviations from these methods also occur, and these will be described in this section.

1.2.1 Sublevel Stoping

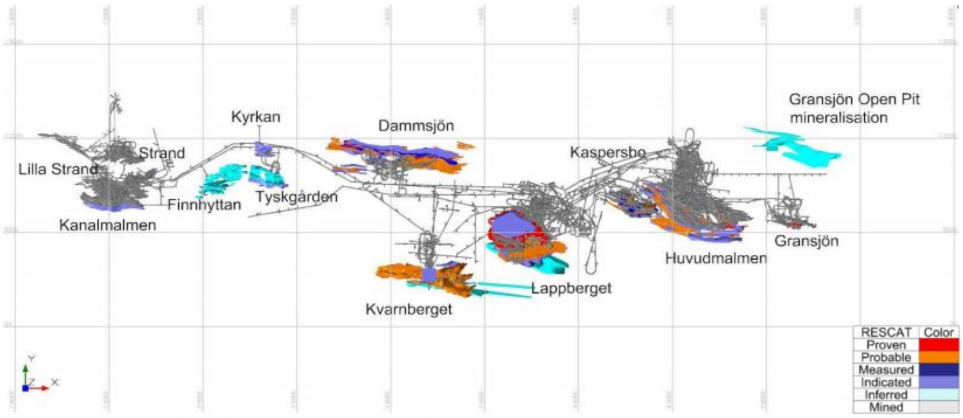
Sublevel stoping, or sublevel open stoping, is a widely used mining method throughout the world. It is used to extract ore from massive or tabular orebodies which can have have any dip, but preferably steeply dipping. Two different methods of accessing the ore exist: transverse stoping and longitudinal stoping.

Longitudinal Stoping

Longitudinal stoping is mainly used for more narrow orebodies, which are usually less than 15 m wide (Villaescusa 2014). In this setup, the stopes are created parallel to the strike of the orebody. The extraction sequence of the stopes is usually done in a retreating manner. Figure 1.4 b) shows the planning stage of a longitudinal stoping area at the Garpenberg mine. The stopes are shown in green, while the development is coloured blue. Boliden utilises another type of



(a) Side view



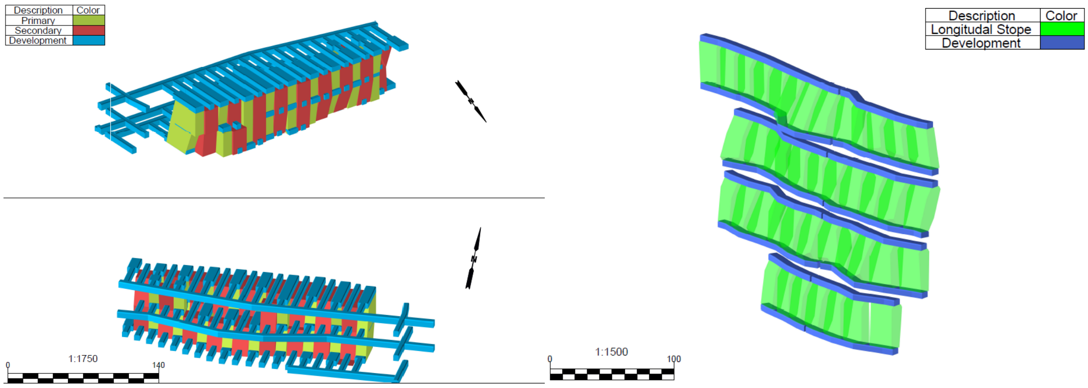
(b) Top View

Figure 1.3. Side and top view of the structures and all known deposits in the Garpenberg area at the end of 2018. Taken from: Högnäs (2018)

longitudinal stoping, called Rill mining. In this method, the stopes are extracted in a retreating manner, but backfilling is done simultaneously, yet delayed. This method is better known as Avoca mining. The extracted stopes are backfilled continuously in the same direction as the extraction takes place. By using this method, less opening slots needs to be created, meaning higher production rates could be achieved (Tommila 2014).

Transversal Stoping

Transverse stoping is used for thicker orebodies. The stopes are oriented perpendicular to the strike of the orebody. The primary stopes are extracted first, while the secondary stopes remain in place as pillar stopes (Villaescusa 2014). The primary stopes are often backfilled using cemented backfill or paste, after which the secondary stopes can be extracted. These are then in turn also backfilled, however, it is common to backfill these using unconsolidated rock. Figure 1.4 a) shows transversal stopes on the 850 level at the Garpenberg mine. Green indicates primary stopes, the secondary stopes are displayed in red. The development is shown in blue.



(a) Transversal Stoping **(b)** Longitudinal Stoping

Figure 1.4. Transversal and longitudinal stoping

Sublevel stoping can be a less suitable method for certain areas in the orebody. To solve this problem, Boliden uses another method, called *opping* mining. This method extends the possibilities of sublevel stoping, by accessing smaller high value areas of the orebody. A regular drift is created from which the rock above is blasted using uphole drilling. When this ore is extracted, the drift is used for regular sublevel stoping. Figure 1.6 shows an illustration of opping mining where a drilling jumbo is drilling uphole to access extra ore. The area with valuable ore is too small for the creation of a stope, or the geotechnical situation does not allow the creation of a stope, but this makes selective extraction of this ore possible. This method is used in lower Lappberget, as is shown in figure 1.5. The top drift of the upper stope is extended upwards by using opping mining. This allows the extraction of extra ore.

Double Stoping

The longitudinal and transversal stopes used in the Garpenberg mine are predominantly 25 m high. An exception of this are the double stopes, which are 50 m high. These double stopes are two regular transversal stopes stacked on top of each other, which are blasted at once. The aim of this method is to reduce the costs of mining a stope, by eliminating a production drift. Two 25 m high stopes require three production drifts, while a double stopes only requires two, which will result in lower drifting costs. A schematic overview of the double stoping method is visible in 1.7. The Garpenberg mine has previously used the double stoping method, however, in this case the 50 m stope was blasted in two blasts using three production levels. The double stope was merely used to allow for sufficient ore to be hauled during the summer recess. However, the mine is evaluating the implementation of double stoping to replace the 25 m stopes.

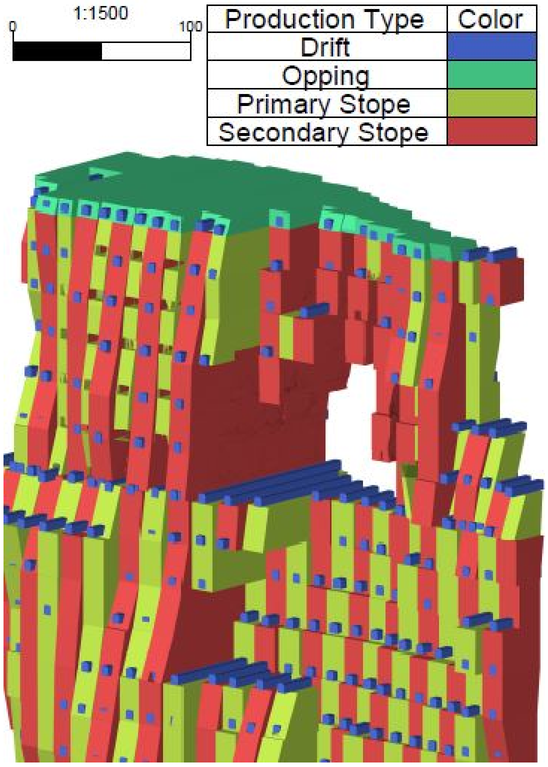


Figure 1.5. Opping in Lower Lappberget. Top drifts will be extended using opping to access extra ore. Opping is shown in green.

Figure 1.6. Illustration of opping Mining. From: Akerstrom (2010)

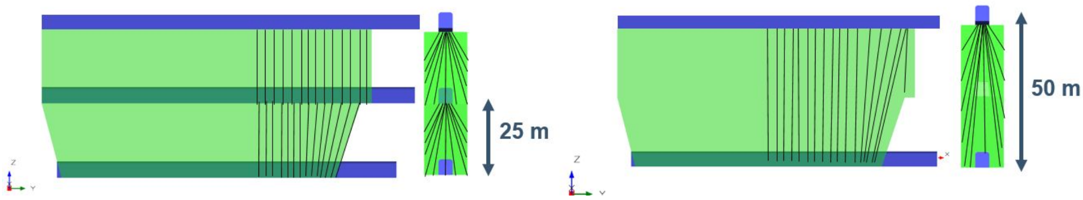


Figure 1.7. Schematic side and front view regular stopes of 25m height on the left, and double stopes of 50m on the right. Development is shown in blue and stopes are shown in green. Black lines represent drill holes.

1.2.2 Cut-and-Fill

Boliden has been using cut-and-fill as their mining method for a long time, and this mining method can be found back in many of the mines including the Garpenberg mine. CaF is a selective and small-scale mining method suitable for steeply dipping orebodies (Hamrin 1980). The method is suitable for irregular orebodies and allows for very selective mining. Figure 1.8 a) and b) show a CaF area on the 750 level in the Dammsjön area of the Garpenberg Mine. However, CaF is considered to be less productive than sublevel stoping and therefore CaF will be replaced by sublevel stoping. The narrow areas of the orebodies will be mined using longitudinal stoping, whilst transversal stoping will replace CaF in the wider areas of the orebody. The transition from CaF to sublevel stoping will lead to an increase in productivity and therefore a reduction in costs.

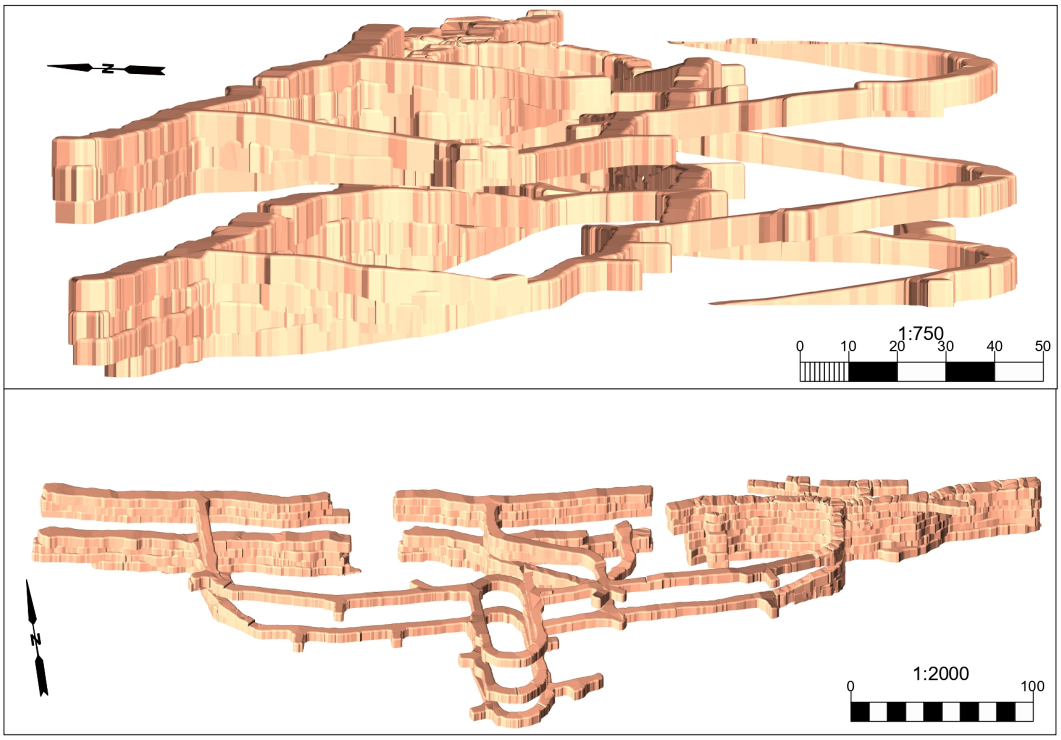


Figure 1.8. Cut-and-fill tunnels in the Garpenberg mine. Fig a) shows the consecutive tunnels which are built on top of the previous. fig b) displays the overview of multiple CaF areas in 750 level at the Dammsjön area

1.3 Cut-off Grade

A very general definition of a cut-off grade is a number which indicates the point between two alternative courses of action (Hall 2014). One course of action would be to distinguish the rock as waste, the other as ore. A more economical definition is the minimum grade or value of a rock which is needed to make it feasible to mine. This definition makes it possible to label rocks as either ore, or waste.

The cut-off grade can have different units, depending on the type of operation. Common units are g/t, % and net smelter return (NSR), which will be discussed in more detail in section 1.5.

1.3.1 Prior Research

The concept of cut-off grades has existed ever since mining occurred, it is however not known to be documented until the 16th century. This might be due to the fact that the concept of a cut-off grade is straightforward. If the costs of mining a certain amount of rocks are higher that its value, it will likely not take place. Georgius Agricola wrote, in 1556, about the need to separate material according to its properties before processing it to ensure a better economic performance of the mining operation (Rendu 2014).

The first well-documented cut-off grade theory was written several centuries later, by Mortimer in 1950. Mortimer's findings can be summarised as follows (Hall 2003):

I. The lowest grade of rock must pay for itself

II. The average grade of rock must provide a certain minimum profit per tonne milled

Mortimer uses break-even calculations to derive the required cut-off grades for the statements. Mortimer's statement I means that every tonne mined, needs to yield enough value to compensate for the costs of mining this tonne. Statement II means that the average grade of the mined tonnes (head grade), needs to be high enough to yield a certain amount of value, which will create a certain level of profit. However, this does not imply the operation is also profitable as a whole (Hall 2014). It is said that by combining statement I and II, Mortimer accounted for the geology in the cut-off derivations. (Hall 2003)

In 1964, Lane published the technical paper *'Choosing the Optimal Cut-off Grade'* (Lane 1964). Unlike Mortimer's definition, Lane also included capacity constraints for different parts of the extraction process, which were divided into mining, treatment and marketing (Hall 2014). Each of the three parts of the extraction process will have two cut-off grades: limiting cut-off and balancing cut-off grades. These six cut-off grades are then used to find the optimal cut-off grade according to Lane.

Apart from the capacity constraints, Lane viewed the goals of a mining operation differently. Whilst Mortimer used the two statements mentioned earlier, Lane viewed maximising cashflow as it's ultimate goal. By maximising cumulative cashflows and by discounting these, the Net Present Value (NPV), can be obtained. Making the NPV a key indicator to evaluate an operations performance.

Several more recent publications regarding cut-off derivations and the optimisation of the cut-off have been made ever since the work of Lane. The following paragraphs will briefly mention or describe these findings.

In 1992, Dagdelen proved that declining cut-off grades over time maximised the NPV, rather than using heuristic techniques (Dagdelen 1992). One year later he modified Lane's theory by adding an analytical method to find the balancing cut-off grade (Dagdelen 1993, Githiria 2016). These publications are based on open-pit mining, however, their base is applicable to underground cut-off grade derivation as well.

Several authors have researched the effect of varying cut-off grades throughout a deposit. It was found feasible having a varying cut-off grade in different areas of the deposit. This could be due to area dependant mining costs for example (Elkington 2009, Hall & Stewart 2007, Horsley 2005). The authors described how a cut-off grade can vary with location, even when all parameters remain constant. This could affect the LoM, but will have a positive effect on the mine's NPV. Wheeler & Rodrigues (2002) stated that development areas which are required for the extraction of stopes, could have a lower cut-off grade, as these areas need to be mined regardless of its value. These researches are strongly linked to this thesis, as the author aims to identify different cut-off grades for the deposits at the Garpenberg mine.

Hills et al. (2014) have researched varying cut-off grades in more detail and describe the introduction of a new measure to compare value in a multi-deposit underground mine: the Mining Variable Value (MVV). The authors have extended the use of NSR to include costs related to processing ore for different deposits, such as reagents used for flotation and power required for crushing and milling. A variable mining cost was also included into the MVV by implementing a variable haulage cost for ore from stope to mill. By using the MVV as the optimisation field in a stope optimiser, new stopes were created. Different production levels and cut-off grades were then analyses using a Hill of Value (HoV) to increase the operation's value.

Apart from a spatially varying cut-off grade, research is also aimed at varying cut-off grades over time. Lane was the first to include the time value of money into the cut-off grade calculations. Cash flows which are obtained from mining a certain block in the far future are much smaller than cash flows obtained when this block is mined earlier. Therefore, it could be beneficial to shorten the LoM. A result of this theory is that if the production capacity is constant, the cut-off grade will be raised to achieve this shortening of the LoM. (Horsley 2005)

Hall (2014) uses a HoV to illustrate the optimisation potential using a combination of cut-off grades and production capacities to improve NPV (Hall 2003, 2014, Hall & Stewart 2007). Figure 1.9 shows the HoV with value on the vertical axis, cut-off grade and production rate on the horizontal axes. The colours are random and indicate isolines of value. It shows that an increase in value is a function of both an increase in cut-off and an increase in production rates, and that achieving an optimal solution often requires an adjustment of both. The process of optimising the value of a mine is an iterative process which starts with a certain cut-off grade. Once this is known, different production scenarios can be analysed, after which the cut-off can be adjusted again. These scenarios which are created can be compared based on their cash flows which will be used to calculate the NPV. However, the optimisation process will have to repeated throughout the LoM as various parameters change over time, such as mining costs, metal prices or geotechnical situations. (Dagdelen 1992, Elkington 2009, Hall 2014)

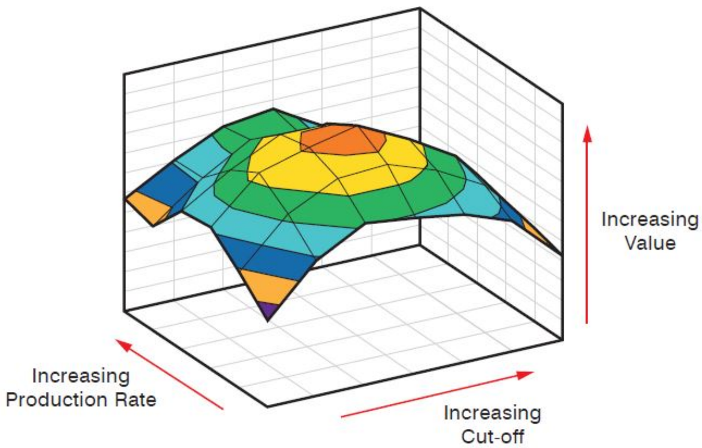


Figure 1.9. Hill of Value. Colours are random and indicate value isolines. Taken from: Hall (2014)

1.4 Rock Mechanics in Sublevel Stoping

This section will describe the methods used in Garpenberg to calculate the sizes of drifts and stopes which allow safe extraction of the ore. These used methods are well known for stope and drift designs and the data used is from the Garpenberg mine

The updated critical span curve is used to calculate the maximum span of drifts in which the operation is safe to operate. The graph is shown in figure 1.10. This empirical graph is based on 292 observation and uses the Rock Mass Rating (RMR) to deduce the maximum span for openings (Ouchi et al. 2008). The graph shows three zones: unstable, stable and potential unstable opening. The RMR is calculated using data from the area in which the drifts will be constructed, and the maximum span is deduced from the RMR. However, the RMR does not include stress or the effect of large scale structures which create wedges. Meaning reinforcement is required, even in the stable zone for competent rock. (Villaescusa 2014)

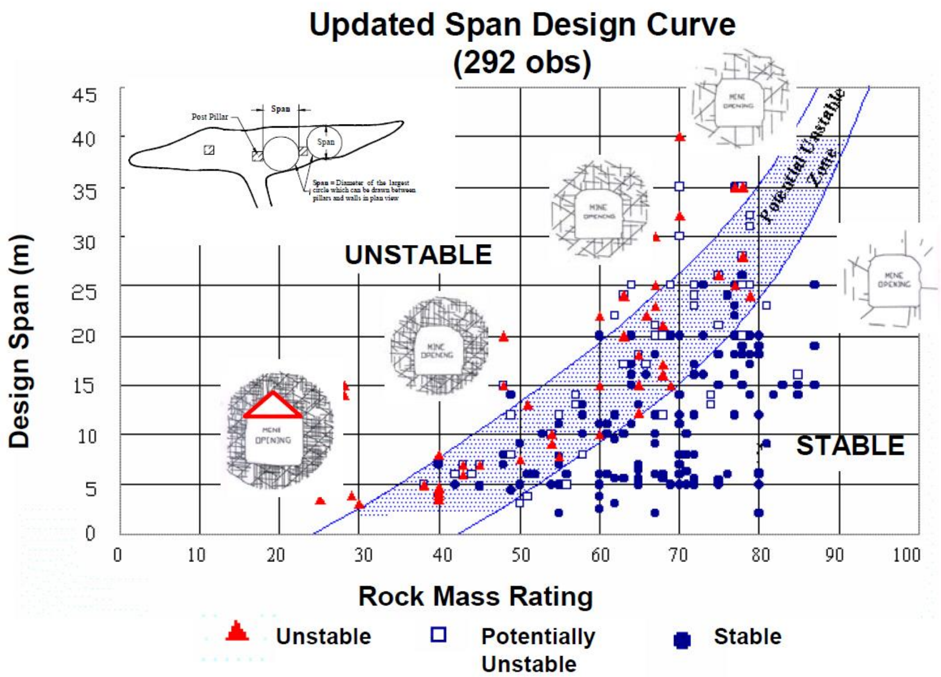


Figure 1.10. Updated Critical Span Curve used for the Dammsjön orebody. From: Fjellström et al. (2016)

EXAMPLE 1

The critical span curve is used for the Dammsjön area as follows:
Several drill cores were collected and the RMR's were calculated with average values ranging from 38 to 72. The drifts in Garpenberg are usually not wider than 10 m, and this horizontal line can be drawn in the graph as a

> start. The graph shows the majority of the area rock will be in the stable or potential unstable zone. Therefore the drifts can be constructed to be 10 m wide. However, the opening will be unstable for the poorest rock conditions. For these areas, a maximum span of 5-6 m is advised. (Fjellström et al. 2016)

Another method is used to asses the stability of the stopes. The updated stability graph method is more complex than the critical span curve method and includes stress in the rock. It defines three zones, stable, cave and transitional and uses the stability number and hydraulic radius (HR) of a stope to assess the stability. Similar to the critical span curve method, this graph is based on empirical data from various stoping operations. (Villaescusa 2014)

The hydraulic radius is calculated by dividing the area of a rectangle by its circumference. The stability number N' is calculated as follows:

$$N' = Q \times A \times B \times C \tag{1.1}$$

With:

N': Stability number [-]
Q: Tunnelling Quality Index [-]
A: Stress factor [-]
B: Rock defect orientation factor [-]
C: Design surface orientation factor [-]

Q is obtained by using the following formula:

$$Q = \frac{RQD}{Jn} \times \frac{Jr}{Ja} \tag{1.2}$$

With:

RQD: Rock Quality Designation [-]
Jn: Joint set number [-]
Jr: Joint roughness number [-]
Ja: joint alteration number [-]

The inclusion of stress in the stability assessment causes the method to be dependant on orientation. The stope wall which is oriented parallel to the principal stress orientation, will have a higher N' value than the stope wall which is oriented perpendicular to the principal stress orientation. Figure 1.11 illustrates orientation of the stresses to the transversal stopes in Dammsjön. It also indicates how length, width and height of stopes are annotated throughout this thesis.

The following example from the Dammsjön orebody will demonstrate the modified

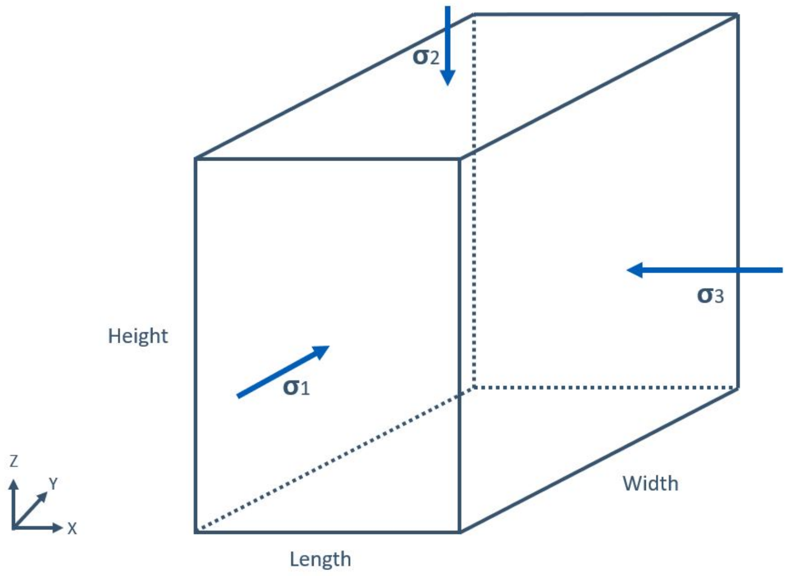


Figure 1.11. Stress orientations for transversal stopes in Dammsjön. Arrows indicate stress orientations. Principal stress orientation σ_1 is oriented perpendicular to the short side of the stope. σ_3 is oriented perpendicular to the long side of the stope. Vector sizes or stopes are not drawn to scale.

stability graph method for transversal stopes which are oriented parallel to the principal stress orientation, as is illustrates in figure 1.11.

EXAMPLE 2

The calculation of N' is done by using formula 1.1 and 1.2.

$$Q = \frac{80}{3} \times \frac{1}{2} = 13{,}3 \tag{1.3}$$

$$N' = 13{,}3 \times 0{,}1 \times 1 \times 8 = 10{,}7 \tag{1.4}$$

This N' is calculated for the long side of transversal stopes. In this example this wall is oriented perpendicular to σ_3, as is illustrated in figure 1.11.

Formula 1.5 shows the same calculation for the short side of transversal stopes, which are oriented perpendicular to the principal stress orientation σ_1. The significantly lower value for the short side of the stope is due to the principal stresses which are oriented perpendicular to this wall.

$$N' = 13{,}3 \times 0{,}1 \times 0{,}3 \times 8 = 3{,}2 \tag{1.5}$$

The two stability numbers can be inserted in the modified stability graph to asses the stability and maximum sizes of the stopes. Figure 1.12 shows two blue dotted lines for the stability numbers which have been calculated in formulas 1.4 and 1.5. The HR can be used to deduce the maximum stope sizes. The height of each stope is set at 25 m, due to the maximum effective drilling length. In order to remain in the transition zone, the long side of the stope has a maximum HR of 9. This is shown by the vertical dotted line in the same figure. A maximum stope width can be derived from this number, which is 64 meters.

Analogues to the calculations for the long side of the stopes, the maximum HR for the short side of the stope is 6, which limits the stopes' length to 23 m.

However, these are the maximum limits of stability in the transitional zone for the highest rock quality. Therefore, to ensure a safer operation the Garpenberg mine uses smaller stope sizes of 40 meters wide and 15 meters in length. These stope dimensions are indicated in figure 1.12 as the blue and red squares. The red square indicated the stope dimensions for the long side of the stope, while the blue square indicated the stope dimensions for the short side of the stope.

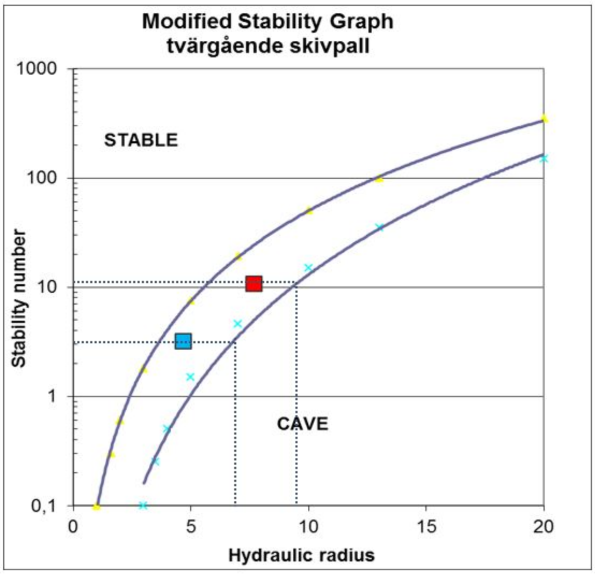


Figure 1.12. Modified Stability graph for Dammsjön orebody. Dotted lines: stability numbers. Red square: stope's side perpendicular to principal stress orientation. Blue square: parallel to principal stress orientation. From: Fjellström et al. (2016)

Double Stoping

The maximum stope dimensions are not solely dictated by geotechnical factors, but also practical, such as the maximum drilling length of longhole drilling. However, with new advances in drilling techniques, the possibility arises of creating higher stopes, or double stopes, as is described in section 1.2.1. However, the geotechnical stability of these stopes will need to be assessed again, as the HR will increase for the double stopes, causing a shift in the stability graph. Using the data provided in the example above, the new HR for the 50 m high stopes will be: 11,1 for the long side, and 5,7 for the short side of the stope. When looking at figure 1.12, this will shift the squares to the right, toward the caving zone. The blue square will remain in the transition zone, whilst the red square will be located in the caving zone. However, as the double stopes are proposed to be constructed in a location with the principal stress orientation perpendicular to the short side of the stopes, as is shown in figure 1.11, this will still be safe to operate. Furthermore, the Q value of the rock in Lappberget is generally higher than in Dammsjön, which will in turn lead to an increased N' value. This will shift the squares up in figure 1.12. However, more research into the geotechnical consequences of the double stopes is planned, which will be used to assess the feasibility of double stoping in Lappberget.

Moreover, unplanned dilution occurring from blasting and production will need to be assessed. Henning (2007) described stope height as a factor influencing unplanned dilution. Yao et al. (1999) concluded unplanned dilution was greatly influenced by stope heights greater than 40 meters, while Perron (1999) concluded that large 60 m high stopes in the Langlois mine ought to be divided using an extra sublevel, to reduce unplanned dilution. The double stope will likely generate more unplanned dilution than the current stopes of 25 m high. Furthermore, drilling deviation is also stated as a source of unplanned dilution by Villaescusa (2014). As the drill hole deviation will increase for longer lengths, this will also contribute to more dilution for the stopes. However, dilution depends on many more factors such as rock conditions and stope wall geometry, and research should be conducted into dilution in the proposed double stopes.

1.5 Net Smelter Return

Garpenberg implemented NSR to define the value of a mining block. This section will describe what is included in the NSR and how it is calculated.

It is essential for mining operations to know the value of the mining blocks. The value of a mining block can be defined in g/t, but deriving the value of polymetallic deposits is more complex. No single grade can be used, but the joint value of the metals should be considered. Furthermore, detrimental elements can be included as negative values in the NSR. The NSR is defined as the value the mining operation obtains from selling concentrate to a smelter. The value of a mining block can be derived from the value of the concentrate. The following factors are included into the NSR (Wellmer, Dalheimer & Wagener 2008):

- Transport Costs (concentrate)
- Treatment Losses
- Treatment Charges
- Refining Charges
- Metal Prices
- Penalties

The formula to calculate the NSR is shown below:

$$\text{NSR} = (g_c - \text{Loss}_t) \times P_{metal} - (C_f + T/C + R/C + Pen) \tag{1.6}$$

With:

NSR = Net Smelter Return [Kr/t]	C_f = Cost of freight [Kr/t concentrate]
g_c= Metal grades in concentrate [% or g/t]	T/C = Treatment Charges [Kr/t concentrate]
Loss_t = Treatment Losses [% or g/t]	R/C = Refining Charges [Kr/t metal]
P_{metal} = Price of metals [Kr/t]	Pen = Penalties [Kr/%][1]

The value of ore can be calculated based on the NSR of concentrate. However, recovery of the processing plant is a function of head grade. Boliden uses forecasting models to predict the recovery of metals over time (Mc Elroy 2016). This is translated into factors adjusting the recovery based on expected feed grade from the Life of Mine Plan (LOMP). By including the recovery based on the feed grade, a more reliable NSR calculation is obtained (Hall & Poniewierski 2016).

1.6 Stope Optimisation

Stope optimisation consists of several aspects which are all aimed at maximising the value obtained from a stoping operations, while being subject to constraints, such as physical, geotechnical and operational constraints. The most important aspect of stope optimisation is stope boundary optimisation, which will aim to optimise the location and the sizes of stopes, this is also called stope layout optimisation. Other aspects such as dilution optimisation, grade risk assessment, access design and scheduling will not be mentioned in this section, as it is less relevant

[1] Unit can be g/t, % exceeding the limit, or a fixed value if it occurs.

for this research. (Hou, Xu, Dowd & Li 2019)

Defining the location and boundaries of a stope is a complex problem to solve. When mining a certain block in an open pit, it is simple to see which blocks need to be mined before accessing the desired block. Figure 1.13 illustrates a 2D example of this problem. To be able to mine block X in figure 1.13 (left), all eight overlying blocks need to be mined (assuming a slope angle of 45°). This is not the case in underground mining. To mine block X in figure 1.13 (right) with a minimum stope size of four blocks and maximum of six, many options exist, one example could be (Y-1,Z-1) to (Y,Z-2). However, having a larger stope is also possible. It is clear the location and the boundaries (size) of the stope is a complex problem, even without block value or mining costs.

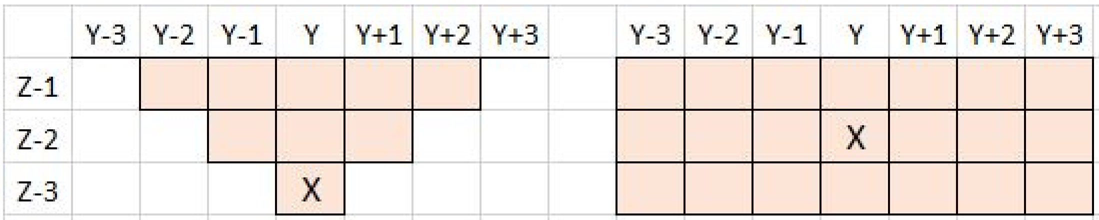


Figure 1.13. Graphical illustration of the difference between open pit (left) and underground mining (right). Based on: Ataee-pour (2005)

An underground stoping operation will try to maximise the value of their stopes, while obeying the constraints which occur in the mine. Throughout history, several methods and algorithms have been researched to optimise the stope layout, and by doing so, increase the value. Before stope optimisation algorithms were available, the location and size of stopes was decided on by a mining engineer. They used the block model to decide which areas would yield the most value, and the stope would be designed. The maximum stope dimensions are usually dictated by operational and geotechnical constraints, as explained in section 1.4, and other technical constraints can be implemented, such as drift accesses and maximum drilling length. Figure 1.14 shows a cross section of the Lappberget deposit. The colours indicate the NSR of the areas of the block model, the black lines indicate the boundaries. The stopes are drawn on a plane and solids are created after, as can be seen in figure 1.4 a). However, one can understand this is far from optimal. Even though some mines still use manual stope designs, several algorithms have improved this process significantly throughout the years. The following section will describe some relevant researches on this subject.

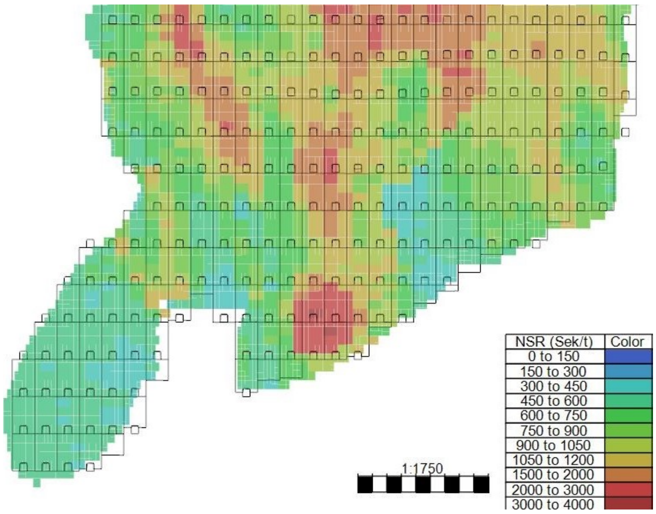


Figure 1.14. Section of the Lappberget deposit. The colours indicate the NSR value, the black lines are stope contours.

1.6.1 Prior Research

According to Hou et al. (2019), stope layout optimisation is the search through all possible combinations of blocks, to derive the best combinations of stopes to maximise the value of the operation. The first researcher to try to optimise this problem suitable for sublevel stoping was Cheimanoff in 1989, who came up with the Octree division method. This method uses geological data, such as borehole data to build a geological model. The model subdivided the model into mineable volumes after which the Octree division algorithm removed blocks which did not meet the set constraints, which mean they are unmineable. (Bai 2013, Sandanayake 2014)

Ovanic & Young (1999) used the branch-and-bound method to solve a mixed integer programming (MIP) problem regarding the start and finish location of stopes. The method aims to maximise the cumulative value of blocks in a row. If this value is maximised, the 'end' of that stope is defined. However, it only identifies the maximum stope size in one dimension. (Ataee-pour 2005, Erdogan & Yavuz 2017)

The floating stope method was developed by Alford in 1995 and its method is well described by its name. A stope with the minimum size is moved (or floated) around a block of the block model in three orthogonal directions. The grades for each possible stope are evaluated. Two boundaries (or envelopes) are obtained from this. Figure 1.15 illustrates the inner and outer envelopes of the floating stope method in 2D. In this illustration, the block size in the block model is 2 x 2, the minimum stope size is 8 x 8, and the maximum stope size is 24 x 16 (horizontal and vertical dimensions respectively). The outer envelope is the union of all possible stope shapes, meaning all stopes which have an average grade above cut-off. The inner envelope is the union of all stopes above cut-off which obtain the best value for each block. The outer envelope contains more blocks and has a larger tonnage than the inner envelope, while the latter has a higher average grade. The mining engineer will design the final stope positions based on the inner and outer envelope. The size of the final stope will depend on the goals of the mine. This could be to maximise tonnages produced, maximise value or combination of these two. The main disadvantage of the floating stope method is stopes can overlap and contain the same mining block and that it does not provide an optimal solution, but only an indicated of an optimal range. (Alford, Brazil & Lee 2007, Ataee-pour 2005, Sandanayake 2014)

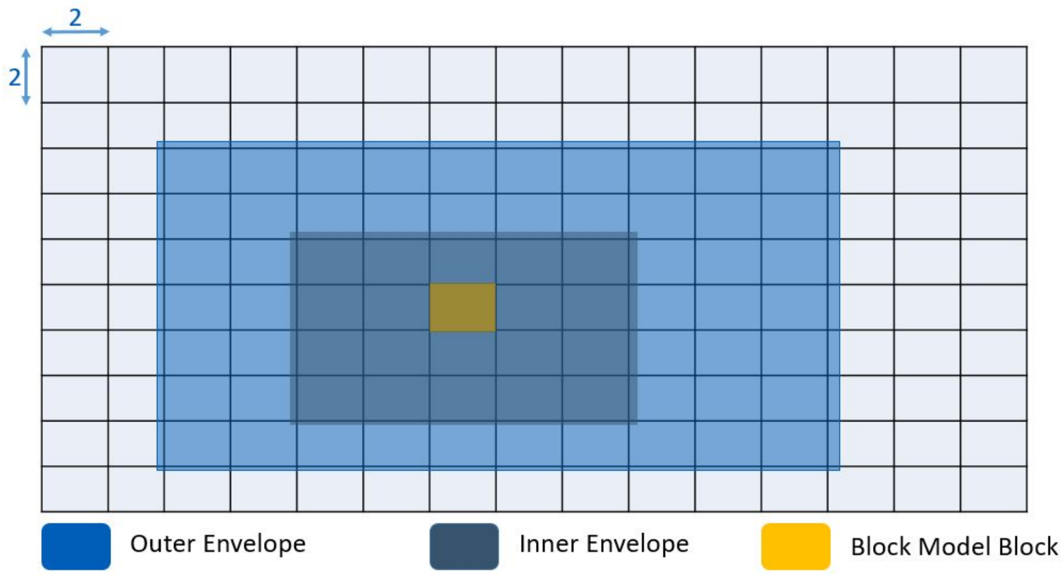


Figure 1.15. Schematics of floating stope agorithm. Block model block shown in yellow. Inner envelope is shown in dark blue. Outer envelope in light blue. Based on: Ataee-pour (2005)

Improvements on the floating stope method were put forward by Cawrse (2001) by developing the Multiple Pass Floating Stope Process (MPFSP). This method made it possible to include more parameters into the stope optimisation such as waste inclusions and head grade. Having more parameters set in the algorithm helped to produce envelopes which were more optimal. However, final stope designs are still to be produced by an engineer, which does not guarantee an optimal solution. (Erdogan & Yavuz 2017, Sandanayake 2014)

Ataee-Pour (2000) developed the Maximum value Neighbourhood (MVN) algorithm. This method starts by selecting blocks which need to be mined due to the minimum size of stopes. It then identifies the value of all blocks which are

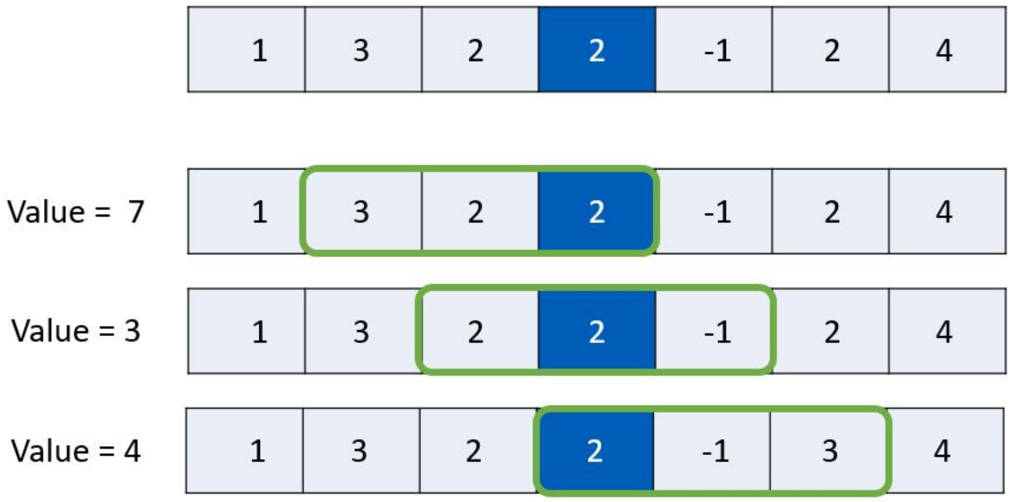


Figure 1.16. Illustration of the Maximum Neighbourhood Value Method. Initial block in dark blue. Green rectangles indicate possible neighbourhoods with their value indicated on the left. Based on Ataee-pour (2005)

adjacent (in the neighbourhood) and either includes them, or rejects them, based on their value. 1.16 illustrates the MVN method for a stope with minimum size of 3. The search for neighbourhood blocks starts around the dark blue block, and yields three options. A value is calculated from each option and the most valuable one is chosen. This method has the disadvantage that it selects a stope with the maximum value, which could lead to another high value block being left as a single block, meaning it cannot be mined anymore. This problem is visible in 1.16 by the block with value four. The algorithm searches the best combination of neighbouring blocks which could be included to meet the minimum stope size constraint. However, the block with value four can not be included and could possible not be mined. (Nhleko, Tholana & Neingo 2018, Sandanayake 2014)

Alford & Hall (2009) developed a fast stope optimisation tool. The tool creates a regular grid of the mineralisation first, and creates slices of this grid. These slices are evaluated on value and are aggregated into stopes if their value will increase the stope value. Stope shape annealing is performed by a hill-climb algorithm to adjust the shapes of the stopes after. Some stopes will not meet the accessibility constraints and a second optimisation is used to group all stopes which do meet the constraints only if they are grouped together. The optimal extraction levels and stope heights can also be identified. The ability to allow quick optimisation processes makes it capable to run it for several cut-off grades. Which can be compared afterwards to pick the optimal cut-off. This method was developed further to be used in the Stope Optimizer software which is used during this thesis. (Alford & Hall 2009)

Manchuk & Deutsch (2008) have researched the use of simulated annealing on stope boundary and scheduling optimisation. Figure 1.17 shows the process of annealing stopes. The corner points of a stope solid are moved randomly within a set range, and each newly formed stope is interrogated to the block model to obtain its value. The random relocation of stope points continues until the maximum value is reached. Unlike the method described by Alford & Hall (2009) which uses a hill-climb algorithm, this method uses the more complex simulated annealing optimisation algorithm which ensures the optimisation algorithm does not get stranded in a local maximum. This process however is computationally demanding and slow. (Bai 2013, Kumral & Villalba Matamoros 2017)

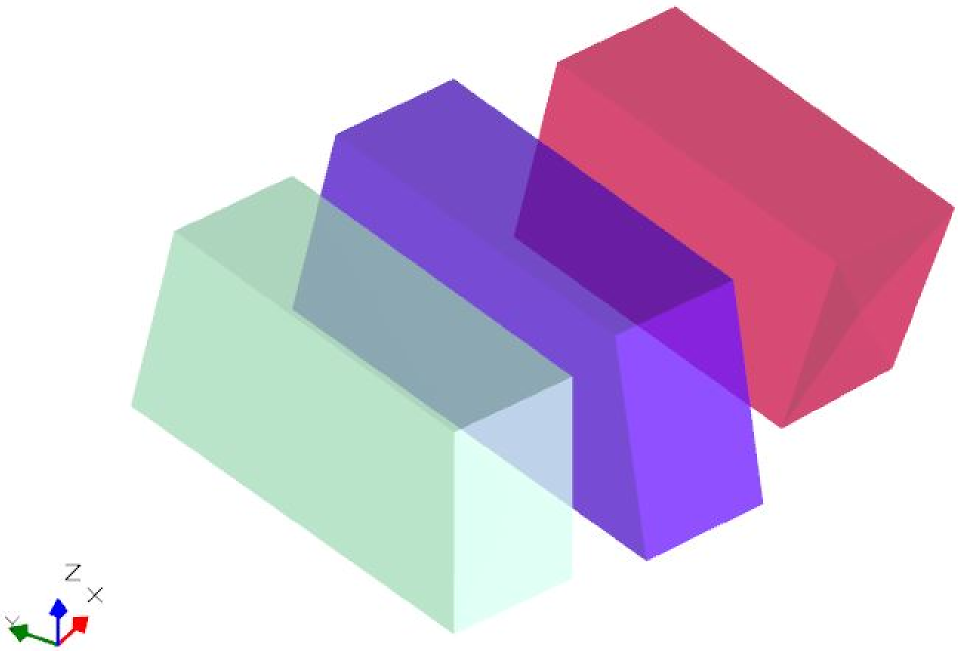


Figure 1.17. Example of three arbitrary adjusted stope shapes. Based on: Bootsma (2013)

Topal & Sens (2010) developed a heuristic stope boundary optimisation method. It creates a regular grid of the orebody and determines stope boundaries on this grid using costs and metal prices. The stopes are then visualised and the user can determine several selection strategies to improve the design. Overlapping stopes are removed based on the user's preferences. If the user sets metal content as its main goal, the overlapping stope with a higher metal content will be preserved, while the stope with the lower value will be removed. (Erdogan & Yavuz 2017, Sandanayake 2014)

Sandanayake (2014) published a research to improve the stope boundary optimisation problem. Similar to Topal & Sens (2010) and Alford & Hall (2009), the algorithm creates a regular grid of the orebody. It defines the stopes sizes and floats this stope throughout the orebody to identify all option. It discards overlapping stopes and compares the remaining options based on cash flows. Hence, costs and benefits are included in this algorithm, as well as geotechnical and physical constraints. However, as the algorithm evaluates all possible solutions, it is computationally demanding, which makes it less suitable for larger data sets. (Erdogan & Yavuz 2017, Nhleko, Tholana & Neingo 2018, Sandanayake 2014).

Bai (2013) developed a heuristic algorithm specifically for sublevel stoping operations. The algorithm is based on network flow method and vertical raises which are required before extraction can start in a stope. According to Nhleko et al. (2018), the main steps of the algorithm are: definition of a cylindrical coordinate system around a raise, creation of a link between footwall and hangingwall blocks, creation of a graph based on the vertical arc for the constraints, and selection of blocks to include based on the maximum distances from the vertical raise. Stope value is maximised by identifying the best location of the vertical raise, and its length. (Bai 2013, Sandanayake 2014)

1.6.2 Stope Optimizer

The *Stope Optimizer* (SO) software is used in this thesis to design stopes and to compare output for several scenarios. It was developed by Alford Mining Systems. This section will describe the processes behind the SO and explain how the software operates.

The optimisation objective of the SO is to optimise the shape and location of stopes to maximise the total recovered resource above cut-off. The SO operates using the algorithm as developed by Alford & Hall (2009). The process can be described using the following four stages:

1. Slice Evaluation

2. Seed Generation

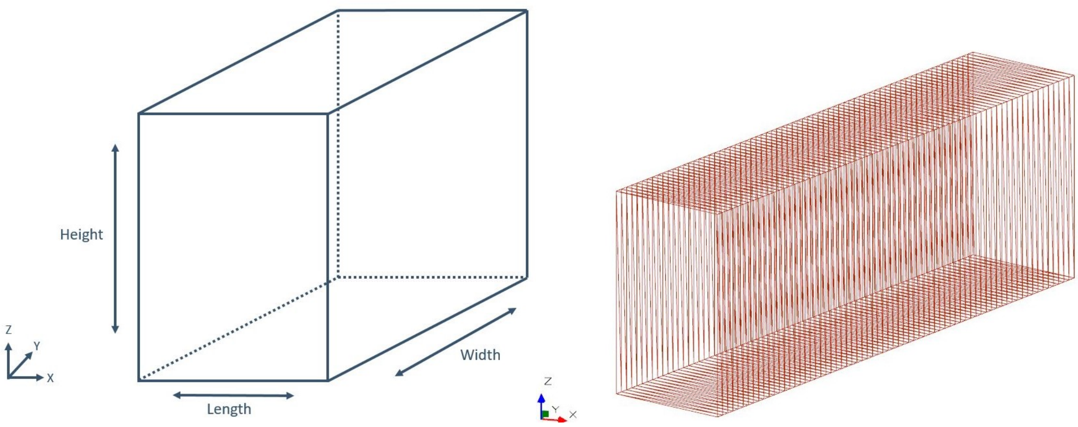
3. Stope Shape Annealing

4. Diluted Stope Shapes

To be able to run the optimisation, the SO requires the following input.

1. Geological Block Model
2. Cut-off grade
3. Technical Constraints

During stage 1, the mineralisation is sampled into a regular grid. This grid will contain the NSR values of the block model, the dimensions and locations. After, the SO creates slices within the regular grid. The slices are rectangles with dimensions (height and length) which are set by the user, and will be placed in a 2D grid, similar to what is illustrated in figure 1.14. The SO will select the location of the slices to include the most of the blockmodel as possible. Figure 1.18 a) illustrates the annotation of height, length and width of the stopes used by the SO, similar to what is shown in figure 1.11. The SO will continue the slice generating process into the orebody, generating new slices along the width of the stope at the same X and Z coordinates. Stope slices generated by the SO are visible in figure 1.18 b). This figure clearly shows the slices which are, in this example, created every 1 m.



(a) Schematic overview of stope layout **(b)** Stope slices. Based on Bootsma (2013)

Figure 1.18. Schematic stope Layout and Stope slices generated by SO.

During the second stage, the SO will combine the slices created in stage 1 into 'seed shapes' of stopes. The seed shapes are created by the aggregation of wire frames of the slices. Every slice which increases the stope's value will be included, while obeying the constraints which are set by the user, such as maximum stope width and pillar size.

The seed shapes of stopes generated in the second stage will go through a stope annealing process in stage 3. A hill-climbing algorithm is used to adjust the corners, and hence the shapes, of the stopes to seek a shape which has higher value, this process is similar as illustrated in figure 1.17. A hill-climbing algorithm will continue adjusting parameters as long as the solution obtained has a higher value. The stope shape annealing process will be performed while respecting the constraints implemented by the user. The result of the three-stage stope optimisation process will be a set of stopes with the highest value possible.

A results of the annealing process is possible extra dilution. The fourth stage will assess the dilution of each stope and discard them if the dilution has caused the grade of the stope to drop below the cut-off set by the user. The results after the fourth stage of the stope optimisation process are diluted stopes, shown in the CAD software. Each stope has been interrogated to the block model and will include information such as tonnage, volume, grade, dimensions, dilution etc.

The stopes can then be used by an engineer to plan the infrastructure around the stopes. However, there is no guarantee the created stopes are optimal, or technically feasible, even though the stope specific technical constraints have been obeyed. Factors which could have an impact on the feasibility of the stopes would be stress fields or infrastructural issues leading to accessibility problems. (Bootsma 2013, Bootsma et al. 2014, Mokos et al. 2014)

1.7 Problem Statement

Stope design can have a large influence on the value which a mining operation obtains from its resources. These designs are strongly influenced by the cut-off grade which is used. Currently, the Garpenberg mine produces ore from several different orebodies, Each orebody has different characteristics, which lead to varying mining costs per orebody, but the mining costs are considered equal. Therefore, the cut-off grade is also identical for each orebody.

To be able to apply the correct cut-off grade for each orebody and mining area, the mining costs should be well known. At the Garpenberg mine, it is not clear how a change in the amount of activity X required in deposit Y will influence the mining cost of that deposit. These activity-based costs will be called parameters throughout the thesis.

The influence these parameters have on the costs of a deposit or a mining area needs to be known to be able to calculate the break-even cut-off grade for each area. After which the cut-off grades can be optimised, leading to an optimised stope design, which will increase the value of the mining operation.

1.8 Research Objective

The aim of this thesis is to determine the influence activity-based parameters have on mining costs, and hence on the break-even cut-off grade of sublevel stopes in the Garpenberg underground mine. This knowledge will be used to optimise the cut-off grades and stope designs to maximise value.

This objective will be achieved by answering the following research questions:

1. What influence do the parameters have on the stope design?
2. How does scheduling of stope extraction influence the value of the mine?
3. What is the best combination of cut-off grades and scheduling to be used to maximise value?
4. Is double stoping a feasible method to improve the efficiency of sublevel stoping?

The fourth research question is less related to the cut-off grade based stope optimisation, but has close ties to the cost calculations.

1.9 Scope

This thesis aims to maximise the value of the Garpenberg mine by determining the influence of parameters on the cut-off grade. The following table (1.1) will state the subjects which have been included in this research, and which are left out of scope.

Table 1.1. Subject in and out of scope in this research

In Scope	Out of Scope
Five deposits of Garpenberg Mine	Other mines
Break-even cut-offs	Lane style optimised cut-offs
Deswik Stope Optimizer	Other stope optimisation software
Algorithm used in Stope Optimizer	Any other algorithm
Sublevel Stoping	Cut-and-Fill
Current infrastructure and mine plans	Designing new infrastructure
General double stoping benefits	Detailed production costs

1.10 Report Outline

1. **Methodology**
   This chapter will explain the approach of this thesis, as well as detailed information on all steps undertaken. Several steps are further explained using examples. The order which is described in this chapter will be continued throughout the results and discussion chapters.

2. **Results**
   The results which are found will be showed in this chapter. Tables and figures will help illustrate the findings of this research.

3. **Discussion**
   The discussion will deal with the analysis and interpretation of the results. The limitation of this research will be described and ideas for improving or continuing this research will be stated.

4. **Conclusions and Recommendations**
   Conclusions which can be drawn from this research will be put forward, and answers to the research questions will be given. This will be combined with recommendations for the Garpenberg mine.

2. Methodology

This chapter will describe the methods which have been used to achieve the objectives stated earlier. It will put forward the approach which has been taken in this thesis, after which it will provide more detailed information about the following steps:

- Cost calculations
- Cut-off calculations
- Parameter Identification
- Parameter Influence
- Using the Stope Optimizer
- Optimising the cut-offs
- Scheduling of extraction
- Double stoping

2.1 Approach

To be able to assess the influence of parameters on the cut-off grade and stope design, it was necessary to identify the costs for each deposit first. In this thesis, five orebodies were considered: Lappberget, Kvarnberget, Kaspersbo, Huvudmalmen and Dammsjön (see figure 1.3). Cost calculations aimed at distributing mining cost for each deposit, were used to derive static cut-off grades.

The parameters which were influencing the mining costs were identified and their influence on the cut-off grade was calculated. This information was used to implement variable costing to the deposits.

With the variable mining cost known, the cut-off grades could be adjusted such that a break-even situation was achieved. The *Stope Optimizer* embedded in *Deswik.CAD* was used to design stopes using the new variable cut-off grades.

The variable cut-off grades were optimised by evaluating possible elevated or depressed cut-off grades. All new cut-off grades were used to design new stopes, and their output was compared. As the new variable costing and cut-off grades had a large influence on the value of the stopes, scheduling was used to optimise the value.

Lastly, the financial feasibility of double stoping was analysed.

2.2 Cost Calculation

The total costs of Garpenberg of 2018 were used for this cost calculation. This meant calculating mining costs reversely, back to where they were incurred. Costs of Garpenberg are reported as mining, processing or surrounding costs (offices, roads, security etc). Each account was then distributed further, such as tunnelling, maintenance, equipment and more.
The second component used for cost calculations was activity data which included all activities and production details for 2018. The most valuable file was the logged activity file, which contained information on location, activity, duration, performance and more, of each type of machinery used in the mine. Further information used was surveyor data, data from a cavity monitoring system (CMS), feed intake into the processing plant, and excel sheets with specific information about haulage routes, production rates and more.

This data was used to distribute the direct costs of the Garpenberg mine, to determine the mining cost which was associated with a deposit. This distribution was based on ratios which were derived from the data explained above. An example is shown in table 2.1. This method was applied to bolting, drilling, charging, hauling, shotcreting and backfilling. Other direct costs not associated with the above-mentioned six processes were distributed based on produced tonnages or drifted meters.

EXAMPLE 3

Table 2.1. Rock bolts used and the share of each deposit.

Deposit	Rock bolts [-]	Ratio [%]
Kvarnberget	36178	31
Lappberget	58973	49
Huvudmalmen	101	0
Dammsjön	21382	20
Kaspersbo	0	0
Total	116634	100

Indirect costs such as maintenance of machinery and associated personnel costs were distributed based on activity duration of machinery for each deposit. Other indirect costs not associated with a deposit, such as general infrastructure in the mine, or shaft maintenance were distributed based on the production data. The same method was applied to the costs of the processing plant and the surrounding of the mine.

The distribution of costs per deposit also provided the opportunity to calculate the costs for a meter of development. By sorting the data accordingly, the costs for each type of development for each deposit could be calculated.

2.3 Cut-off Grade Calculation

With the mining costs distributed over the five deposits, the cut-off grade of each deposit could be calculated. Section 1.3.1 above mentioned the cut-off derivation methods. This section will describe the formula's used to calculate cut-off grades.

Break-even cut-off grade is the most straightforward method to calculate a cut-off grade. It includes the cost of mining a ton of rock, and is divided by the recovery multiplied by the product price, see formula 2.1. As mentioned in section 1.3.1, the break-even cut-off grade can be calculated in many ways. This all depends on which costs are included in the numerator.

$$\text{CoG (unit/t)}^{1} = \frac{\text{Cost (SEK/t)}}{\text{Recovery (\%)} \times \text{Product Price (SEK/t)}} \tag{2.1}$$

Currently, Boliden is using another method to calculate the cut-off grade of their mines. The costs of the complete mine are split into mining, processing and surrounding costs. These costs are divided by the produced ore and milled material respectively, which yields a break-even cut-off grade. The following formulas are used to calculate cut-off grades of the deposits:

$$\text{CoG}_{\text{mining}} = \frac{\text{Mining costs (SEK)}}{\text{Ore production (t)}} \tag{2.2}$$

$$\text{CoG}_{\text{Processing}} = \frac{\text{Processing costs (SEK)}}{\text{Milled Material (t)}} \tag{2.3}$$

$$\text{CoG}_{\text{Surrounding}} = \frac{\text{Surrounding costs (SEK)}}{\text{Milled Material (t)}} \tag{2.4}$$

$$\text{Overall CoG} = \text{CoG}_{\text{Min}} + \text{CoG}_{\text{Proc}} + \text{CoG}_{\text{Sur}} \tag{2.5}$$

All cut-off grades which will be mentioned from this point onward will be calculated using the above formulas, and are break-even cut-off grades. Several costs are omitted from the cut-off grade calculations: the depreciation and the sustaining capital expenses. More about this in chapter 4.

2.4 Parameter Identification

This part of the thesis dealt with identifying parameter which would influence mining costs. Four parameters were found to be suitable for this analysis: reinforcement, drill and blast, metal price and haulage distance. A change in a parameters is translated into a change in costs, which will lead to a change in cut-off.

Reinforcement was selected as a parameter for several reasons. Firstly, the spatial variety of reinforcement requirements throughout the mine and within a deposit. Secondly the large costs which are associated with reinforcements. The following costs are included in this parameter:

[1] %, g/t or other

- Rock bolts
- Rock bolting personnel
- Maintenance of rock bolting equipment

A change in reinforcement needed was based on geotechnical information. The rock bolting pattern in Garpenberg was adjusted based on the stress fields in the mining areas. Three types of bolting patterns were used: 1 x 1, 1,2 x 1,2 and 1,5 x 1,5 meter spacing. The bolting patterns used in 2018 for the deposits were analysed and by combining this with the amount of bolts used in that year, a cost factor was found. An example:

EXAMPLE 4

Table 2.2. Meters of drift bolted in Lappberget with certain rock bolting pattern

Pattern	**Meters bolted** [m]	**Ratio** [%]
1 x 1	311	6
1,2 x 1,2	895	16
1,5 x 1,5	4390	78
Total	5596	100

When comparing the amount of bolts used per m^2 compared to 1,5 x 1,5 (cheapest pattern), the following factors are found:

Table 2.3. Rock bolting Factors

Pattern	**Factor** [-]
1 x 1	2,25
1,2 x 1,2	1,56
1,5 x 1,5	1

By multiplying the ratios from table 2.2 with the corresponding ratios from table 2.3, Lappberget obtained a rock bolting factor of 1,16 in 2018.

Drill and blast was selected as these processes are the most time-consuming and could lead to larger differences in costs. Only longhole drilling and charging were considered. The following costs are included:

- Explosives
- Drill and blast personnel
- Maintenance of drill and blast equipment

An increase in drilled meters is considered to have an equal increase in the amount of explosives used. This was based on the amount of meters drilled per ton of ore in 2018.

Metal price influence was selected as a parameter to be analysed. This parameter is not locally varying, but is dictated by the metal market. Formula 1.6 displays how the NSR is calculated and the variable P_{metal} indicates metal prices. To calculate the NSR of a block, each metal grade in that block is multiplied by the earnings factor of that metal. The earnings factor already includes all costs for processing, average penalties for that LOMP and recovery. The recovery factor is adjusted using empirical results and estimates from the LOMP.

The earnings factor is multiplied by metal grade and metal price, to obtain the NSR of that metal. This process is repeated for all five metals. All five values are summed to obtain the NSR of one block.

Finally, haulage distances were selected as a parameter to assess its effect on mining costs. Haulage distance is defined as the distance an ore truck needs to drive from the area it is loaded by the LHD to an ore pass or crusher. Waste transport was not included in this parameter. Waste was mainly used for backfilling secondary stopes throughout the mine, which made it difficult to trace back the haulage distances.

The Garpenberg mine derived an empirical formula from historic transport data from contractors. Formula 2.6 describes the costs per ton of transported ore for a certain distance X. Elevation differences are not included, but are assumed to be included in distance. The formula is derived from historic transport data from several orebodies in the Garpenberg mine, and is hence an average cost per distance.

$$\text{Cost/t} = 16{,}568 + 0{,}01369 \times X \tag{2.6}$$

Equation 2.6 was used in combination with haulage distances and tonnages to calculate the cost of transporting ore from an area to an ore pass or crusher. This cost was used to adjust the cut-off grade, similar to other parameters. An example calculation can be seen below:

EXAMPLE 5

Primary stope Lapp 1257_1232 room 5 was excavated in 2018 and the closest ore destination was the crusher at level 1087. The distance to be covered was 840m. Using this in the formula above yields a cost of around 28 kr/ ton. 52 Kt of material had to be transported, therefore the costs of ore transport for this stope were 1 459 kkr. Formula 2.7 displays the derivation of this value.

$$(16{,}568 + 0{,}01369 \times 840) \times 52\,000 = 1\,459 \text{ kkr} \tag{2.7}$$

2.5 Parameter Influence

The four parameters which have been described in the previous section, were used to adjust the mining costs and cut-off grades. This section will briefly describe this process.

As described in section 2.2, mining costs of each deposit were known, as well as the costs of each of processes such as rock bolting, transport, personnel etc. Table 3.1 from the next chapter shows a clear overview of costs within each deposit. These costs were considered the average and base case for the calculation, which could be adjusted to reflect the actual costs. Using example 5 from section 2.4, the process will be explained below:

EXAMPLE 6

Costs for rock bolting in Lappberget in 2018 were 36 319 kkr. These costs were obtained by installing rock bolts in 5 596 meters of drifts, with various bolting patterns. The bolting pattern combinations caused Lappberget to have a bolting factor of 1,16, which is the base case of all of Lappberget. To find the influence of a parameter, the required situation in an area is used to compare this to the base case. For example the stopes on the right side of lower Lappberget in figure 2.1, where green and red indicate primary and secondary stopes respectively, which require a 1,5 x 1,5 bolting pattern. The yellow and blue colours indicate primary and secondary stopes which require 1 x 1 and 1,2 x 1,2 bolting patterns respectively. When looking at table 2.3, it is clear these stopes require more reinforcement than the average of Lappberget.

Hence the rock bolting costs for these areas are increased. Table 2.4 shows the new reinforcement costs for all of Lappberget in 2018, the newly implemented costs for primary and secondary stopes, respectively.

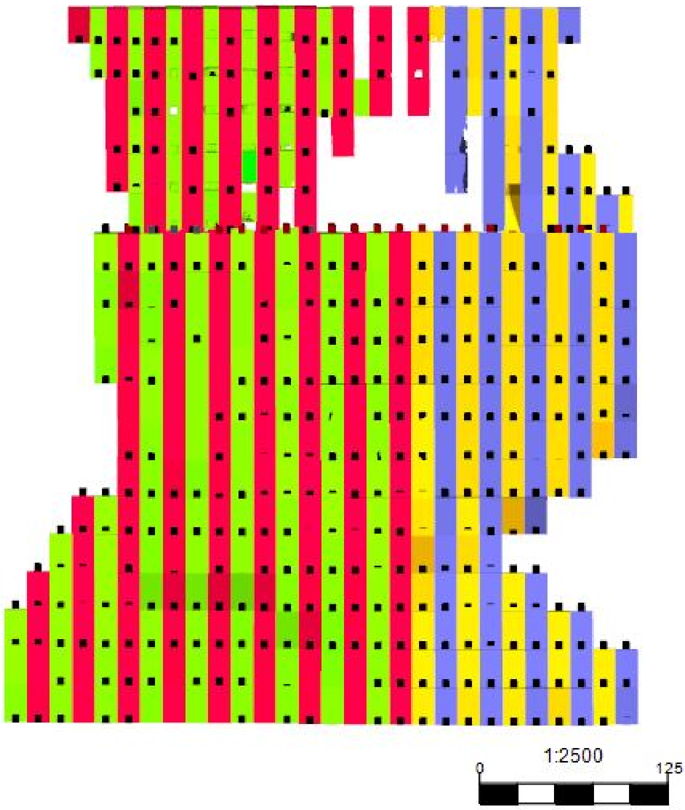


Figure 2.1. Lower Lappberget stopes. Colours indicate primary/secondary stopes (green/red) and primary/secondary stopes for higher stressed zones (yellow/blue)

Table 2.4. Rock bolting parameter difference in selected areas in Lappberget

Area	**Factor** [-]	**Cost** [kkr]
Current base	1,16	36 319
New Primary Stopes	2,25	70 460
New Secondary Stopes	1,56	48 992

The newly calculated mining cost for reinforcement will be used in formula 2.3 to acquire a new break even cut-off grade for that area.
The example above describes the process for rock bolting. Drill and blast is used in a similar manner. Ore transport is calculated differently however.

To implement the variable ore transport costing, formula 2.6 was used in reverse to obtain the average distance. As total cost of ore transport for each deposit was known, the average base case distance for each deposit could be calculated. When the average distance was known, varying haulage costs could be calculated. Every stope has a different haulage distance, but as the measurement of distances from stope to drop-off point was manual labour, stopes were grouped in areas. Each level would be divided into four to five areas, from which the distances were noted, and the costs were calculated. The following example illustrates distances and costs for mining areas.

EXAMPLE 7

Table 2.5 shows the haulage distances from stope to crusher and associated costs for some selected mining levels of Dammsjön. This process was done for all levels in four areas in Dammsjön. As the connecting drifts were located in various positions at each level, the haulage distances vary significantly throughout levels. To adjust the mining cost of an area, the average haulage distance for the whole Dammsjön area: 1904 m, was used to increase or decrease transport costs percentage-wise. Stopes in Area B at level 1172 would have a mining cost of: 445,7 kr/t, with the transport costs accounting for 43 kr/t. Due to a decrease in distance 12%, the transport costs now account for 39,6 kr/t, and total mining costs become 442,3 kr/t.

Table 2.5. Haulage distances for selected levels in Dammsjön and their associated costs

Level	Distance A [m]	Cost A [kr/t]	Distance B [m]	Cost B [kr/t]
1172	1739	41,1	1681	39,6
1147	2169	46,2	2173	46,3
1122	1951	43,2	1908	42,6

2.6 Stope Optimizer

The parameters defined in the sections above are used to adjust mining costs, which will be used for the variable costing. The break-even cut-off grade will be calculated and this value can be used to create new stopes. The following example clarifies why new stopes needed to be created:

EXAMPLE 8

Primary stope 1532_1507 Room 27 in Lower Lappberget has a tonnage of 21,9 Kt and an NSR of 451,6 kr/t. Based on the new variable mining cost, the extraction of this stope will cost 501,1 kr/t. This doesn't require any calculation to show this stope will cost more than it will yield.

This problem can be described in a more general sense:

$$\text{Profit} = \text{Tonnes} \times (\text{NSR - Mining Cost}) \tag{2.8}$$

With profit in sek and mining cost and NSR in kr/t.

If mining costs increase, profit will decrease. When the cut-off grade is increased however, the NSR will increase, and the profit will depend on the tonnage. Even though the tonnage will decrease due to the higher cut-off grade, the profit could potentially be higher than having a lower cut-off. This theory was used to maximise value. This section will describe the steps taken to create new stopes and how the variable cut-off grades were implemented.

The stope optimisation software *Stope Optimizer V3.1* which is embedded in *Deswik.CAD* version 2018 3.873, was used to create new stopes.

New stopes were created for all five deposits using different scenario's, such as: constant cut-offs for all deposits, elevated constant cut-offs, variable cut-offs within deposits, as well as varying technical constraints. The SO provided output in the form of polylines and solids of created stopes in Deswik.CAD which also included attributes such as tonnage, grades, NSR, cut-off, width, length etc. Apart from the Deswik.CAD output, excel files with data about each stope was created. The most valuable data are the dimensions, tonnages, volumes and NSR of a stope, and comparison of these made it possible to analyse the different scenarios.

New stopes were created to closely align current stope designs which are in place in Garpenberg (current will be used throughout this thesis to indicate the stopes and schedules which were designed by Boliden engineers and which make up the LOMP of Garpenberg). Areas with planned or existing drifts were used for the programme to create stopes. This meant that some areas which were deemed profitable by the stope optimiser, but were not planned to be mined, were not included in the scenario analysis. The majority of the settings which had been used to create the current stopes were preserved. This was done to ensure the new findings would be suitable for comparison to the current situation. Table 2.6 displays the most important technical constraints which have been used.

The general steps which are taken while using the SO are as follows:

1. Select blockmodel
2. Define region of optimisation
3. Set technical constraints
4. Set cut-off

Each deposit has a separate block model which uses metal prices of 2018. Block models are updated yearly with new NSR values and removing mined-out areas.

The region of optimisation was a box-shaped figure in which the programme would create and optimise stopes. The shape needed to be a multiple of the minimal dimensions visible in table 2.6. The areas within deposits, which were mentioned in section 2.5, meant the optimisation region needed to be aligned with these areas. An example of this was the Lappberget deposit, which was divided horizontally in five areas of equal size.

The technical constraints were mainly preserved from the current stope design, however, some changes were made. As the distinction between primary and secondary stopes could not be made within the SO, the creation of *substopes* of 10 m length was allowed. This was done to allow the creation of smaller stopes. The minimum width for transversal stopes was larger than 9 meters, but this dimension was kept in place to allow the creation of longitudinal stopes. The SO creates stopes in one direction only, as is explained in section 1.6.2. Meaning a combination of longitudinal and transversal stopes is only possible if the width of the transversal stopes is sufficiently small.

Table 2.6. Stope Optimizer Settings

Parameter	Value	Unit
Height	25	m
Length	15	m
Minimum width	9	m
Maximum width [1]	300	m
Pillar width [2]	8	m
Side ratio top-bottom	2	-
Side ratio front-back	2	-
Maximum dilution	30	%
Minimum dip	75	°
Maximum dip	105	°
Maximum change	20	°
Slice interval	1	m

Cut-off grades could be set in the SO as a constant number, or as a varying number. The varying cut-off could however only vary by stope dimensions, which was unsuitable for this research as the stopes have fixed dimensions, or by elevation (depth). This explains the horizontal division of orebodies. Each orebody has laterally and vertically varying characteristics, which lead to a varying cut-off grade. Each deposit could be divided horizontally, while grouping stopes together, and assigning one mining cost for that horizontal area. The vertical varying cut-off grade (with depth) could now be included into the SO. Figure 2.2 shows output of the SO for lower Lappberget and the optimisation region is shown as a purple box. The area within the purple box is one of five defined regions in lower Lappberget and the cut-off data was inserted in a way which is similar to that in table 2.5: each level within this box was assigned a cut-off grade.

2.7 Result Analysis

After generation of the stopes by the SO, the results could be analysed. The goal of this thesis was to find the influence certain parameters have on the stopes, and the main indicators for this were value, either profit or NPV, tonnages and dimensions. This section will explain how the created stopes were used to answer

[1] Can be divided manually in smaller stopes
[2] Pillar between stopes in Y direction, as shown in figure 1.18a

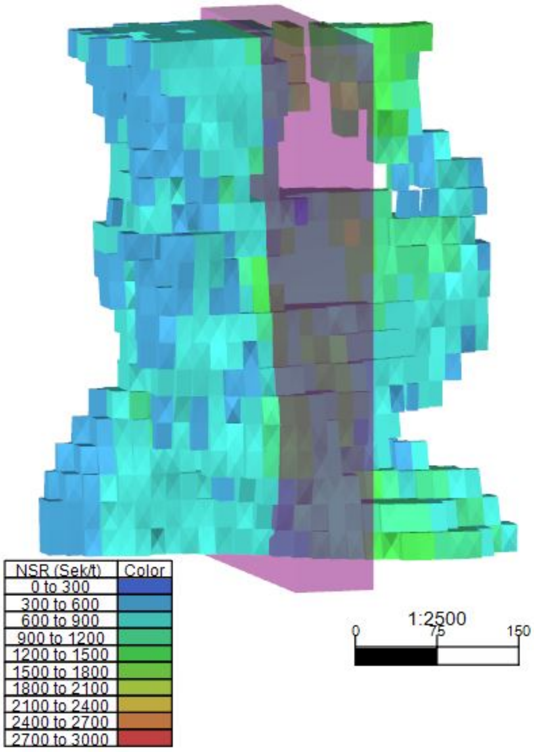


Figure 2.2. Region of optimisation for lower Lappberget. The colours of the stopes indicate NSR, the purple box indicates the region of optimisation

the research questions in this thesis.

Firstly, the generated stopes were analysed in Deswik.CAD and unmineable stopes, already mined-out stopes or overlapping stopes were removed manually. This was done to ensure similarity between newly produced stopes and current stopes, and to remove faulty stopes from the process.

Stopes which were generated in separate areas (as shown in figure 2.2), were merged to form complete deposits. Tonnages and dimensions of all stopes were analysed using pivot tables in Excel and Deswik. The variable mining costs were added to the new stopes as attributes and including tonnages and NSR in formula 2.8 yielded the profit of each stope. These steps could also be performed on the current stopes of the Garpenberg mine. By including the variable mining cost, and using the same profit and NPV calculations, a clear comparison is made between the current stopes and stopes created using the new method.

Each stope of the current planning also contained attributes regarding the scheduling, with starting date and ending date. This information was used to implement the same scheduling as the current stopes. With the addition of extraction scheduling data, the NPV of each stope could be calculated using the following formula:

$$\text{NPV} = \frac{\text{Profit}}{(1+0{,}1)^{\text{Start-2017}}} \tag{2.9}$$

With the NPV and profit in SEK and start being the year in which extraction of the stope starts. 0,1 is due to the discount factor of 10% which is used.

This method was used to obtain the profit and NPV of each stope of the deposits. However, to include the cost of development into the NPV calculations, spreadsheet analysis was required. All planning and scheduling of extraction and development for Garpenberg is done in *Deswik.Sched* version 2018.3.1011.0, and essential information regarding ore/waste production and development meters could be extracted. Information such as yearly ore production and capital and operational development meters per year were extracted from the scheduler and added to the economic models in the spreadsheets.

2.8 Optimal Cut-off

Optimisation of cut-off grades was performed after implementation of variable costing and variable cut-offs. Variable cut-off grades were elevated or depressed and new scenario's were created to compare the results. The cut-off grades were changed +-5%, +-10% and +-20% and tonnages, profits and NPV's were analysed. An increase in cut-off leads to lower tonnages in the stopes, but possibly increases the profitability of the stopes. This will be described using grade-tonnage curves in the results chapter.

2.9 Scheduling

The following example will demonstrate the reason for scheduling to be included in this thesis. The remainder of the section will describe the steps taken to obtain results.

EXAMPLE 9

Secondary stope 1407_1382 room 0 in lower Lappberget has a tonnage of 23,8 Kt and an NSR of 745 kr/t. The average mining cost for Lappberget is 455 kr/t. By using formula 2.8, this stope will yield a profit of 6,9 Mkr.

With variable costing included in the same stope, the mining cost is 490 kr/t, providing a profit of 6 Mkr, which is around 1 Mkr less than the the original stope.

Even though this example only highlights the differences between implementing mining cost within the current stopes, the same problem occurs in newly designed stopes. New stopes will already account for the higher mining costs by having a higher cut-off grade. However, the profit of a certain stope could still be different from the current stope. The extraction schedule of the stopes is not aligned to the profit distribution of the deposit anymore, and stopes which have been profitable in the older designs, may yield less or more profit in the current designs. This problems highlights the necessity of scheduling the stope extraction.

The ideal scenario of extraction stopes would be to mine the most profitable stopes first, and the least profitable stopes last. As formula 2.9 shows the profit is discounted each year and postponing the extraction to a further moment in time, will yield less value. Implementing variable costing will shift the profit distribution throughout the orebody. As is shown in example 9.

However, this simple scenario cannot be be implemented in practice for several reasons: the production capacity, mill constraints, physical constraints and geotechnical constraints.

The Garpenberg mine has the capacity of producing 3 Mt of ore per year. Meaning the amount of stopes which can be extracted annually is limited.

The processing plant requires a constant input of ore, and cannot handle varying ore grades. Making it impossible to mine all higher grade stopes earlier.

Regarding the physical constrains, a certain stoping area cannot be mined before the development has reached this level. Or a stope cannot be mined if the stope below is not backfilled yet.

Finally, the geotechnical constraints dictate the order of extraction for sublevel stopes. As explained in section 1.4, the mine operates a triangular retreat shape for stope extraction in order to prevent stresses building up in yet to be mined stopes. This limits the choice of stopes to be extracted.

Scheduling of stope extraction is done without any optimisation software, causing the process to become time-consuming, hence only lower Lappberget was scheduled. The first step taken to identify possibilities of value increase was to compute the difference between NPV and profit of each stope. The deposit was divided into areas, similar to the optimisation regions, but also split vertically, consisting of 6 stoping levels. These sectors were used to adjust all starting years of stopes in that sector by a few years. This was done to maintain the differences in scheduling from stope to stope.

Several scenarios were created using this method. Lastly, the stopes were adjusted individually to ensure the triangular retreat shape was kept in place, as well as the yearly production targets.

2.10 Double stoping

The research regarding the feasibility of the double stopes is uncomplicated. It uses the economic models created earlier for Lappberget, and new data from created double stopes is added to this economic model. The development meters from the economic model were also adjusted to represent the required development meters for double stopes. The new NPV's were analysed in the economic models.

3. Results

This chapter will mention the results which have been achieved by following the processes as mentioned in section 2.1. It will contain the following results:

- Cost calculation
- Cut-off grade calculations
- Parameter Influence
- Optimal cut-off
- Scheduling
- Double Stoping

3.1 Cost Calculation

Cost calculations were a prerequisite for any further calculations made. Not all details leading up to the result will be displayed, only important results will be shown. Despite the aim of analysing the costs of the five orebodies, only three have been analysed as insufficient mining activities have taken place in Huvudmalmen and Kaspersbo. Therefore, all further calculations are done for three remaining orebodies: Lappberget, Kvarnberget and Dammsjön. Section 3.3 will describe how cut-offs for the two remaining deposits were calculated.

Table 3.1 displays mining costs distributed into different elements of the operation for three orebodies. Sustaining capital costs consist predominately of development tunnels which are permanent, further explanation can be found in chapter 4. Costs of development are incurred through the various accounts, and sustaining capital is defined as a benefit to remove these costs from the total costs. Depreciation costs are not included in this cost calculation.

The table clearly shows Lappberget has the highest costs, and Kvarnberget and Dammsjön are significantly less costly. Personnel costs are the largest contributor to costs for each of the orebodies, which is similar in most mining operations (Poxleitner 2016). Secondly, it is clear that Dammsjön is not using any backfilling, which is due to the mining method used, CaF, which uses unconsolidated backfill. Another notable remark are sustaining capital costs of Kvarnberget and Lappberget. These are similar, yet their production numbers are not. This is due to the amount of new permanent development required in Kvarnberget.

A similar cost distribution can be seen in table 3.2, with Lappberget accounting for almost 80 % of costs incurred. This table includes processing and surrounding costs as well, however, sustaining capital is, analogously to table 3.1 not included. This is also the case for depreciation costs, which are not included in this break-even cut-off calculation.

Table 3.1. Mining cost in SEK

Account	**Kvarnberget** [kkr]	**Lappberget** [kkr]	**Dammsjön** [kkr]	**General** [kkr]
Rockbolting	- 7 839	- 12 661	- 4 849	-
Shotcrete	- 9 554	- 18 165	- 4 219	-
Drilling	- 6 224	- 11 821	- 1 763	-
Explosives	- 8 471	- 24 009	- 2 201	-
Ore Transport	- 13 641	- 69 450	- 6 287	-
Waste Transport	- 11 994	- 19 112	- 1 387	-
Diesel	- 3 867	- 16 876	- 1 111	-
Energy	- 2 399	- 6 352	- 945	-
Maintenance	- 24 255	- 81 115	- 9 164	-
Personnel	- 44 676	- 146 099	- 15 722	-
Backfill	- 4 618	- 29 449	-	-
Subtotal	- 137 540	- 435 110	- 47 647	- 260 442
Sustaining Capex	65 465	69 954	5 277	-
Subtotal	- 72 075	- 365 156	- 42 370	- 260 442
Distr. General	- 39 139	- 198 294	- 23 009	-
Total	**- 111 214**	**- 563 450**	**- 65 379**	**-**

Table 3.2. Total costs in SEK

Account	Kvarnberget [kkr]	Lappberget [kkr]	Dammsjön [kkr]
Mining	- 111 214	- 563 450	- 65 379
Processing	- 50 365	- 258 647	- 18 383
Surrounding	- 22 769	- 116 927	- 8 310
Total	**- 184 348**	**- 939 024**	**- 92 072**

3.2 Cut-off Grade Calculation

With the costs distributed over three deposits, the cut-off grade could be calculated. This was achieved by using formulas 2.2 through 2.4. These formulas were not followed strictly, as the difference between produced ore and milled product was not clearly divided between orebodies. After mining, the streams of ore from various orebodies blend and the origin cannot be traced. However, the difference between total produced ore and milled product in 2018 is negligible with only 0,13% less milled product than produced ore. This difference is due to stockpiling of ore. The produced amounts of ore and waste are visible in table 3.3.

Table 3.3. Garpenberg production 2018

Deposit	Ore [Kt]	Waste [Kt]	Total [Kt]
Kvarnberget	402	233	635
Lappberget	2 062	374	2 436
Huvudmalmen	2	0	2
Dammsjön	146	23	169
Kaspersbo	12	0	12
Total	**2 625**	**631**	**3 256**

When using the production data given above, cut-off grades for each element of the operation could be calculated. After which they were summed to obtain the total cut-off grade for the three deposits. The resulting cut-off grades are shown in table 3.4 below. Costs for processing and surrounding have been distributed based on ore production of each deposit, and therefore the cut-off grade for those elements are equal for each deposit.

Table 3.4. Cut-off grades

Element	Kvarnberget [kr/t]	Lappberget [kr/t]	Dammsjön [kr/t]
Mining	278	273	441
Processing	125	125	125
Surrounding	56	56	56
Total	**459**	**454**	**622**

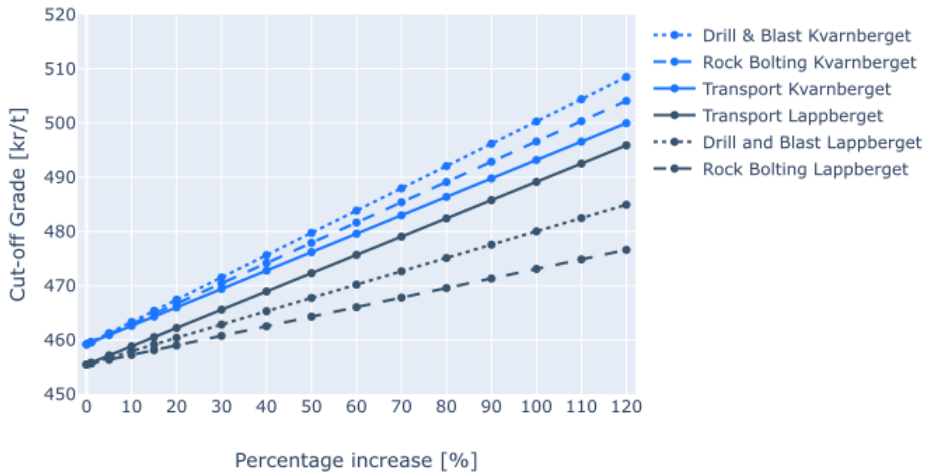
3.3 Parameter Influence

This section will display the results of the influence of the parameters on the cut-off grade and on the created stopes.

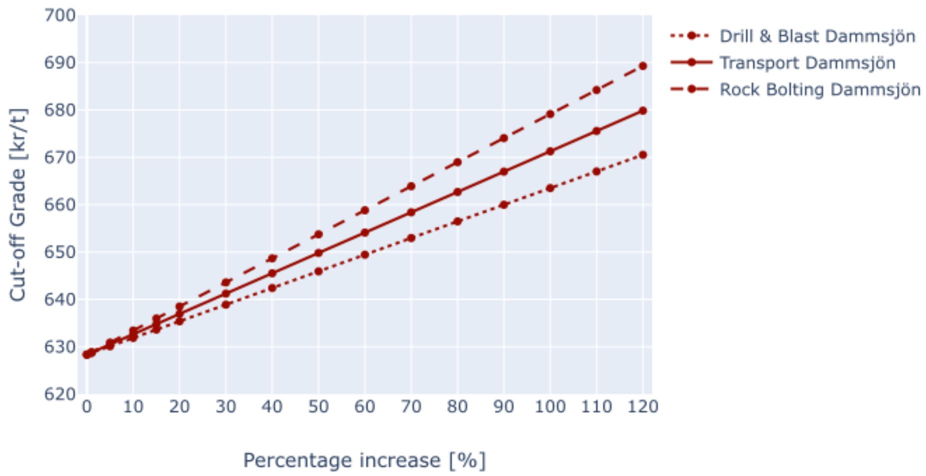
Figure 3.1 a) and b) show the results on the cut-off grade of Lappberget, Kvarnberget and Dammsjön, by an increase in mining cost by a parameter. These figures show the influence of rock bolting, drill and blast and ore transport and they clearly shows the varying influence each parameter has on the cut-off grade. An increase in ore transport cost will have a larger influence in Lappberget than in Kvarnberget. Which may be due to the large quantities of ore transported and the longer hauling distances. This figure clarifies the varying influence of parameters throughout the deposits.

What would be more interesting is to see how the stopes are influenced by increasing parameter costs. Figure 3.2 shows the results of multiple scenarios from the SO for an arbitrary area in Kvarnberget. The optimised regions comprises ten stopes, with settings used as shown in table 2.6. No optimisation runs were performed for complete deposits as this would be too time consuming. The parameter costs were increased from 0% to 120% which lead to higher cut-off grades. These grades were implemented into the SO to obtain new stopes.

The bars in the figures show how the sum of tonnages of all stopes changes with a percentage increase of the parameter. The dotted blue line shows the sum of the stope widths. The general pattern of the graphs is that tonnages and widths decrease as the cut-offs increase. Which is in accordance to what is explained in section 2.6. Figure 4.3 also visualises this. A higher cut-off grade will lead to less



(a) Parameter influence Lappberget and Kvarnberget



(b) Parameter influence Dammsjön

Figure 3.1. Parameter Influence on cut-off grade for Kvarnberget, Lappberget and Dammsjön

ore being mineable, which will in turn cause the stopes to become smaller and the tonnages to within the stopes to decrease. The influence the parameters have on the stopes varies however.

From figure 3.1, it can be seen that an increase in drill and blast costs has the largest influence on mining cost, and hence on cut-off grade, for all three deposits. This causes the tonnages of the stopes in figure 3.2 c) to show the steepest decline for an increasing cut-off grade.

A third observation which can be made is the varying behaviour of the plots. The sum of tonnages and widths are expected to decrease constantly for an increasing cut-off grade, as less ore will be mineable. However, this decrease is not very constant, as peaks and bottoms can be seen. Such as an increase in tonnages and widths while the cut-off parameters costs increase from 0% to 10%. A steep decrease in tonnages and widths is visible at a 70% increase of haulage costs, while the tonnages and widths increase again at an 80% increase in haulage costs. This varying behaviour does not align the expected values, but it can likely be explained due to the optimisation algorithm, which will be further explained in chapter 4. Similar results were obtained for the stopes created in an arbitrary position in Lappberget.

To asses the influence of a changing NSR on the process, the block models were altered and these were used as the optimisation field by the SO. As the difference in NSR was very variable with location, the optimisation was run for the complete lower Lappberget orebody. Figure 3.3 displays the tonnages and sum of the stope widths for changing metal prices. A 5% increment was chosen and the results are displayed in a similar manner as figure in 3.2. The figure clearly shows the stopes get larger with a higher metal price. This can be expected as more ore is above cut-off.

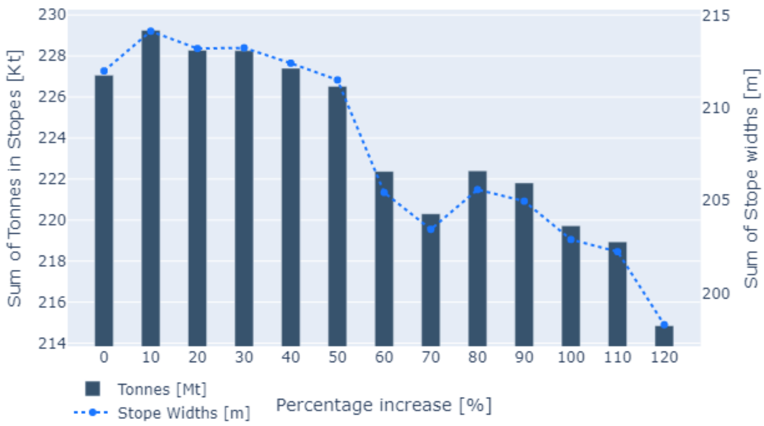
3.4 Parameter implementation

This section will describe the results of the parameter implementation into the various deposits.

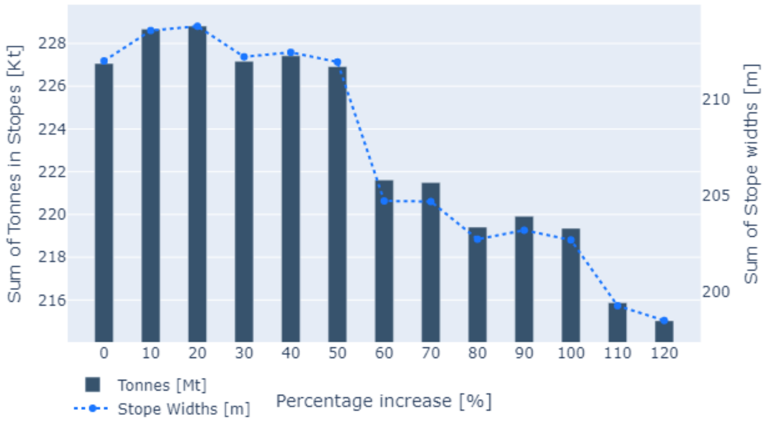
Implementing drill and blast costs as a locally varying parameter was not possible. The performance indicator for this parameter was drilled meters per ton of ore (m/t), but location-based trend was found. Section 4.2.3 will describe this in more detail.

The implementation of ore transport and rock bolting costs to adjust cut-off grades based on location was possible. The strong location-based characteristics of these parameter made it possible to allocate adjusted cut-off grades to various areas.

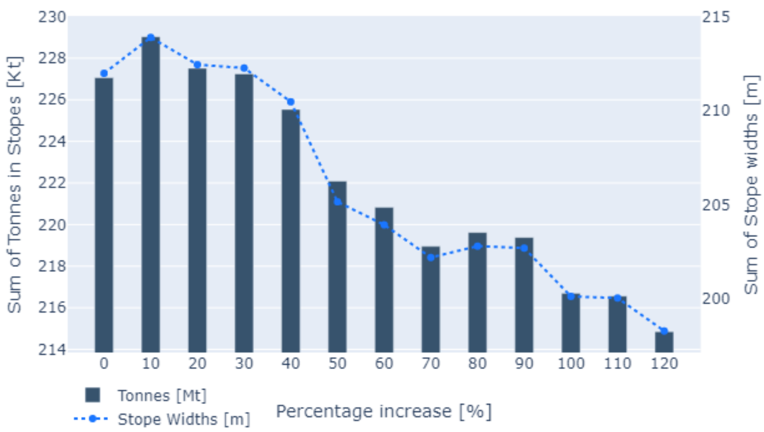
Ore transport and rock bolting costs were also used to obtain a static cut-off grade for Huvudmalmen and Kaspersbo, the two deposits which had not had sufficient mining activities in 2018, to deduce a cut-off grade. The average ore haulage distance for the deposits was used, as well as the planned rock bolting pattern. The changes in these parameters were compared to the cut-off grade of Kvarnberget. This orebody was, based on location and size, similar to Huvudmalmen and Kaspersbo. These calculation provided the following cut-off grades:



(a) Influence of haulage parameter in Kvarnberget



(b) Influence of rock bolting parameter in Kvarnberget



(c) Influence of drill and blast parameter in Kvarnberget

Figure 3.2. Stope widths and tonnages with changing parameter costs in an arbitrary area in Kvarnberget.

Kaspersbo : 480 kr/t

Huvudmalmen: 462 kr/t

This method was also used to calculate the cut-off grade for Dammsjön. Although sufficient data was acquired to calculate a static cut-off, this data was based on

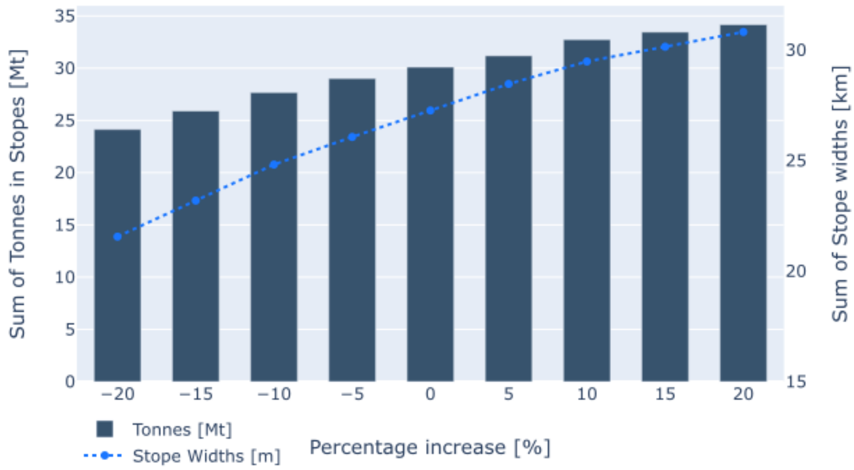


Figure 3.3. Stope widths and tonnages with changing metal prices in lower Lappberget

CaF mining, which has a higher cost per ton of ore, making it unsuitable for sublevel stoping. The cut-off grade for Dammsjön's stopes is: 465 kr/t.

The implementation of the parameters in the orebodies will be analysed as follows. Stopes created using a static cut-off, will be compared to stopes created using a variable cut-off, while variable costing has been applied to the orebodies.Tonnages and profits will be displayed to indicate the differences. Figure 3.4 illustrates Lappberget with variable mining costs implemented. Colours indicate the mining cost which was implemented in that area. As the cut-off grade is set at break-even, this cost is also the cut-off grade applied in that area.

Table 3.5 shows the results of scenarios with a constant cut-off and a variable cutoff. The results show an increase in tonnage and profit for Lappberget, Kvarnberget and Kaspersbo. Huvudmalmen and Dammsjön experienced a decrease in both tonnage and profit. The change in tonnage is less than the change in profit for all deposits, meaning the value improvement is not solely due to an increase in tonnage. A problem arose with Huvudmalmen and Dammsjön as these orebodies include more narrow areas, causing longitudinal stopes to fit the orebody best. The SO creates stopes perpendicular to the strike of the orebody, and this is more suitable for transversal stopes. This results in stopes being created in places which do not align with the infrastructure well. The longitudinal areas should be optimised separately form the transversal stopes, as the optimisation direction is perpendicular. Furthermore, the Huvudmalmen orebody has a more complicated shape, making it less suitable for bulk scenarios as is done in this thesis. This is

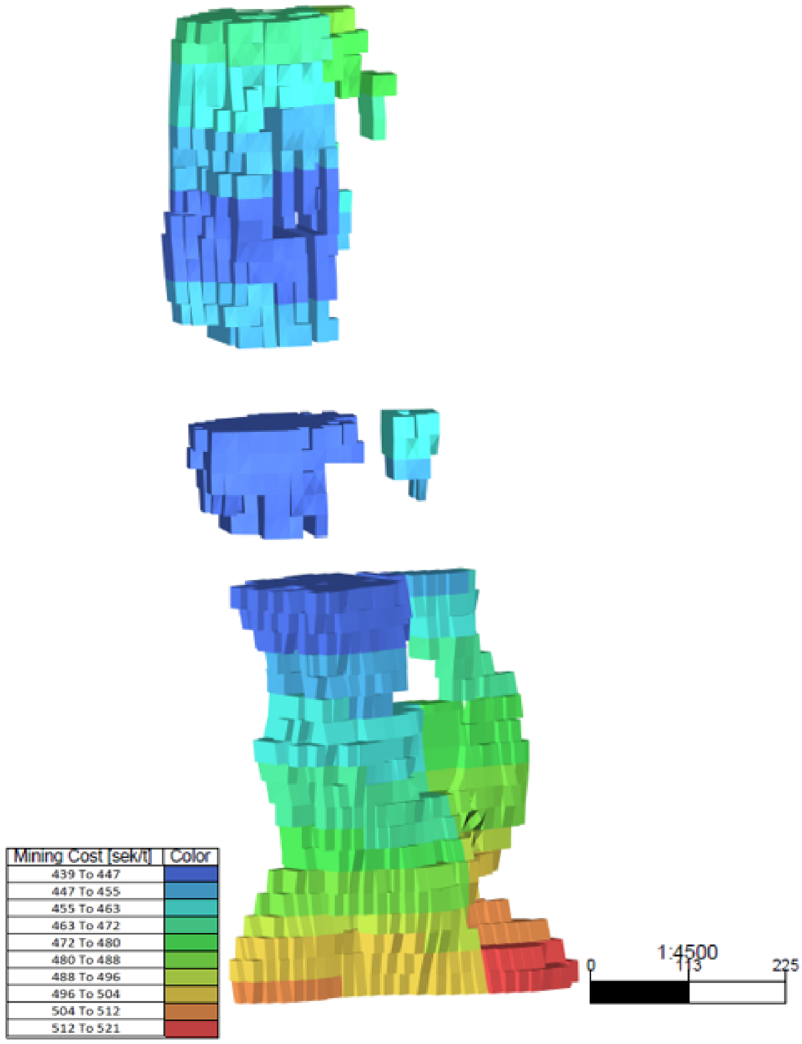


Figure 3.4. Illustration of mining cost implementation into Lappberget. Colours indicate the allocated mining costs.

why Huvudmalmen has been researched less.

Table 3.5. Tonnages and profits for optimisation with/without variable cut-off grade

Deposit	Cut-off	Tonnage [Mt]	Profit [Bkr]	△ Tonnage [%]	△ Profit [%]
Lappberget	Constant	53,07	29,98	1,91	3,06
	Variable	54,08	30,89		
Kvarnberget	Constant	6,23	7,29	0,54	3,98
	Variable	6,48	7,58		
Dammsjön	Constant	18,03	9,86	- 1,93	- 0,35
	Variable	17,67	9,83		
Huvudmalmen	Constant	26,19	10,42	- 1,32	- 1,50
	Variable	25,85	10,26		
Kaspersbo	Constant	1,36	0,77	2,68	3,69
	Variable	1,39	0,79		

Table 3.5 show that with a variable costing applied to mining areas, the cut-off should also be adjusted accordingly. The NPV of these five deposits were calculated using the extraction schedules which are in place for the current stopes and can be found in table 3.6. It shows similar results for Lappberget, Kvarnberget and Dammsjön. However, Huvudmalmen shows a positive NPV difference, while Kaspersbo shows a negative NPV difference. This indicates the profit distribution has changed from the original stope design, and the extraction scheduling does not match the newly designed stopes. This problem will be described more thoroughly in section 3.6. The following paragraph will describe the results obtained from optimising cut-off grades.

Table 3.6. NPV of optimisation with/without variable cut-off grades

Deposit	Cut-off	NPV [Mkr]	△ NPV [%]
Lappberget	Constant	8 137	3,63
	Variable	8 432	
Kvarnberget	Constant	1 921	3,99
	Variable	1 998	
Dammsjön	Constant	1 046	- 0,17
	Variable	1 044	
Huvudmalmen	Constant	1 417	0,44
	Variable	1 423	
Kaspersbo	Constant	421	- 1,03
	Variable	417	

3.5 Cut-off optimisation

The previous section described the variable cut-off grade implementation in each deposit, this section will show the results of the optimisation of these cut-off grades. The variable cut-off grades were elevated 5%, 10%, 20% or depressed if the break-even cut-off grade was too high for that area.

Lappberget consists of three distinct ore zones: upper, middle and lower Lappberget. Figure 3.4 clearly shows the three ore zones within Lappberget. Figure 3.5 indicates the changes in tonnage and NPV for lower Lappberget elevated cut-off grades, which were increased with a 5% step until 20%. It shows the tonnages decrease, while the NPV rises until an elevation of 15%, after which it decreases. The tonnage of the area has decreased by around 10%, while the NPV increases by 1%. Table 3.7 summarises the elevated or depressed cut-off which provided the highest value for each deposit. All other scenarios can be found in appendix A, table 1.1.

Another notable result is for middle Lappberget, this deposit yield the highest NPV for a 30% depressed cut-off which causes a tonnage increase of 27 %, and an NPV increase of 12%.

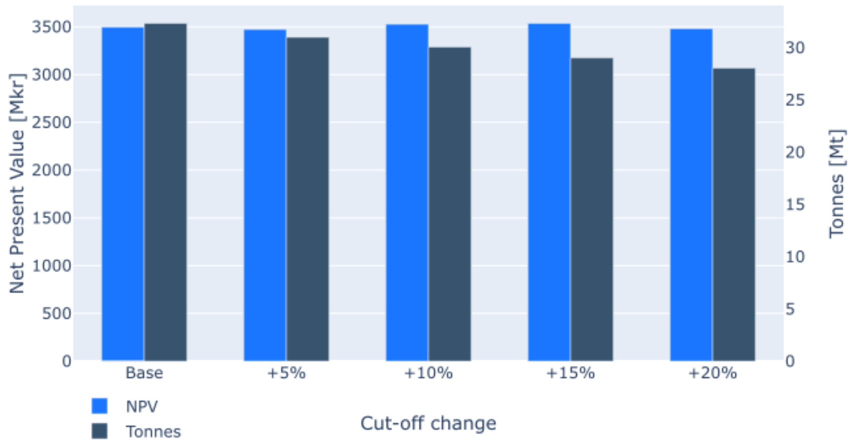


Figure 3.5. NPV and tonnages for lower Lappberget with elevated cut-off scenarios

Table 3.7. NPV and tonnages for best elevated cut-off scenarios

Deposit	**Scenario**	**Tonnage** [Mt]	**NPV** [Mkr]	**△ NPV** [%]
Upper Lappberget	Base	16,30	4 169	1,37
	+25%	13,98	4 186	
Middle Lappberget	Base	5,29	1 272	12,06
	-30%	6,86	1 426	
Lower Lappberget	Base	32,33	2 989	1,11
	+15%	29,03	3 022	
Dammsjön	Base	17,68	1 044	~ 0
	-20%	18,99	1 045	
Huvudmalmen	Base [1]	25,85	1 423	-
Kvarnberget	Base	14,46	2 025	0,77
	-10%	15,79	2 039	
Kaspersbo	Base [2]	1,39	416	-

[1]Only elevated Cut-off
[2]Not optimised

3.6 Scheduling

Section 2.9 described the varying profit distribution due to the newly implemented mining costs. This section will describe how scheduling of extraction of stopes was performed to cope with this shift. Lower Lappberget has been scheduled to maximise the value and it served as an example for the other deposits.

Several scenario's have been created by adjusting the starting dates of all stopes within a sector. The largest value improvement was found by extracting stopes with the highest profit first. This unrealistic scenario improved the NPV of lower Lappberget by 32% to 3 986 Mkr, compared to 3 022 Mkr as seen in table 3.6. This scenario was used as a basis for further scheduling. The year-by-year production rates were levelled out to reach similar values as the current LOMP. Figure 3.6 shows ore production levels of the current LOMP, the highest value yielding scheduling scenario, and the adjusted scenario. It shows the volatile production levels of the highest NPV scenario, which have been adjusted to align the current production levels.

The adjusted schedule yields an NPV of 3 611 Mkr, an increase of 18% compared to the newly created stopes using the original scheduling. This example shows the necessity and opportunities of scheduling of stope extraction.

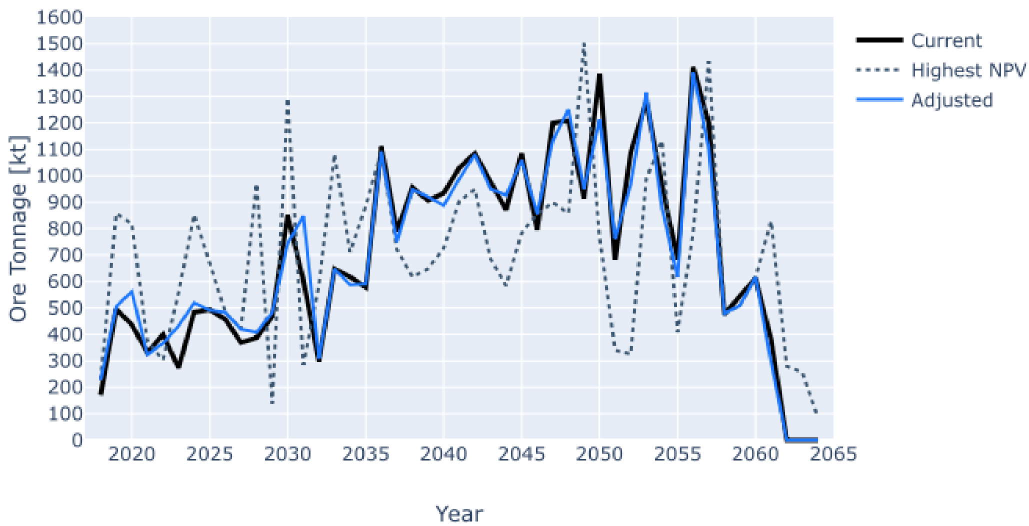


Figure 3.6. Yearly ore production rates of lower Lappberget for the current schedule, highest value yielding scenario, and adjusted scenario

3.7 Double Stoping

This section will describe the results of using 50 m high stopes in lower Lappberget to replace 25 m high stopes.

Lower Lappberget was the most suitable orebody for the analysis of double stoping, as it has the largest ore tonnages, longest LoM and the rock quality potentially allows for double stopes to be constructed. Two scenarios were created, one of three levels of double stopes, the other with four levels of double stopes. The scenarios were designed to be implemented in the lower areas of lower Lappberget, to maximise the benefit of the double stopes. The development of ramps and drifts did not reach this depth yet, hence the benefit of eliminating the middle drifts was most prominent and these depths. The two proposed designs of double stopes are shown in figure 3.7.

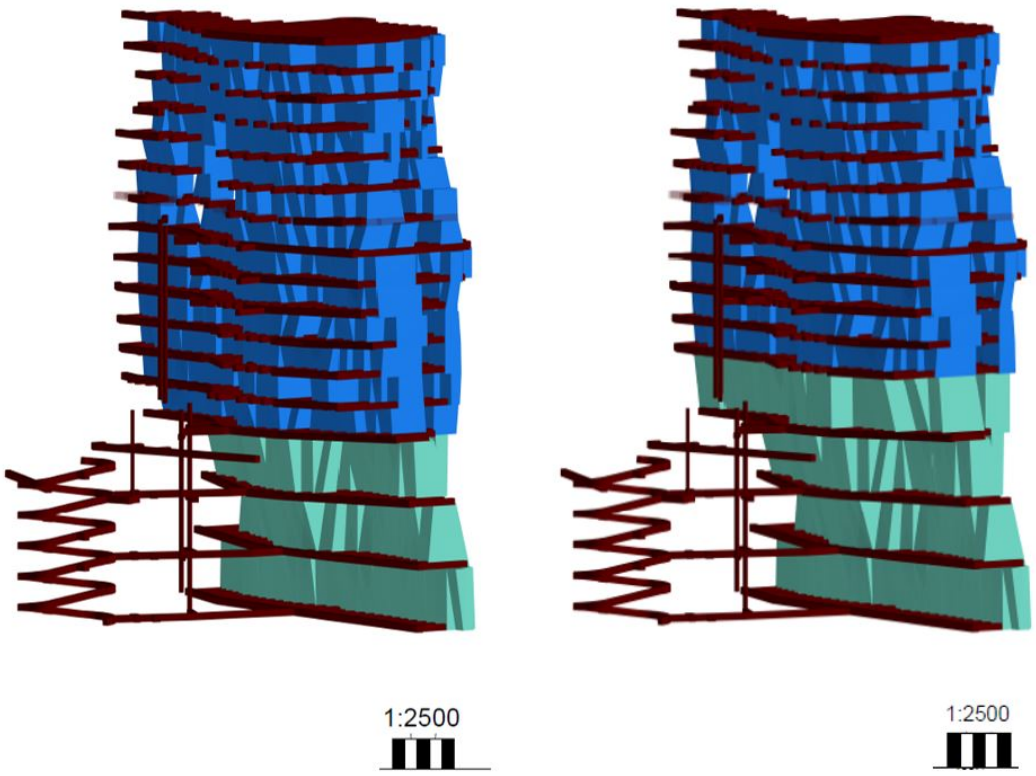


Figure 3.7. Double stoping scenarios. Left figure represents three double stoping levels, while the right figure illustrates four levels of double stoping. Colours indicate double stopes (green), regular stopes (blue) and drifts (brown).

Scenario I - Three Levels

The upper level is located at level 1432 and the lowest stope is at level 1582. This scenario allows the elimination of three production levels which decreases the required development meters by 9 610 m.

Scenario II - Four Levels

This scenario has four double stoping levels with the upper stope at 1382, and the lowest at level 1582. Four production levels can be omitted resulting in a decrease in development meters of 12 317 m.

By using costs known for various development types, such as ramps, production drifts, connecting drifts, the resulting cost benefits could be calculated. However, the method also has the disadvantage of being less selective than the smaller stopes. Figure 3.8 indicates some stopes which cannot be mined due to the elimination of several production levels in red for scenario II. This leads to a decrease in ore production, as is shown in table 3.8. The maximum wall angles will also limit the ore which can be reached in a stope of 50 m, as two smaller stopes can be placed off centre, and reaching further away ore. The table displays the tonnages, savings in development costs and profit for the area of lower Lappberget which was used for the implementation of double stopes (green area in figures 3.7 and 3.8). The table shows that double stopes have a lower selectivity, which has a negative impact on the profit of the stopes. As less ore can be mined, less value is obtained. However, the elimination of three and four production levels for scenario I and II respectively, overcome this decrease in profit. Table 3.9 shows the result of both scenarios, in combination with regular stopes from level 1432 and 1382 upwards for scenario I and II respectively. These scenarios use cut-off grades which are based on the 10% elevated cut-off grade which was most profitable for lower Lappberget (see section 3.6).

Table 3.8. Tonnages and profit for double stoping area. 25m indicates area of implementation with regular stopes. 50 m indicates double stopes.

Scenario	**Tonnage** [Mt]	**Profit** [Mkr]	**∆ Dev. costs** [Mkr]	**Total Profit** [Mkr]
Scenario I - 25 m	10,27	6 392	0	6 392
Scenario I - 50m	10,18	6 220	281	6 501
Scenario II - 25m	13,96	8 590	0	8 590
Scenario II - 50m	13,86	8 375	361	8 736

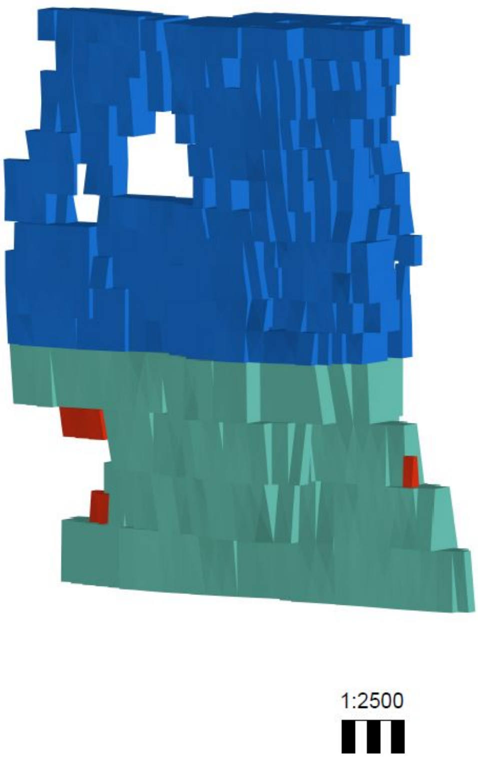


Figure 3.8. Four double stoping levels in lower Lappberget. Colours indicate double stopes (green), regular stopes (blue) and unmineable stopes in red.

Table 3.9. Tonnages and profits of double stoping scenarios compared to regular 25 m stoping.

Scenario	Dev. Meters [m]	Dev. costs [Mkr]	Tonnage [Mt]	Profit [Mkr]
Regular	54 587	1 605	29,954	18 083
Scenario I	44 976	1 324	29,950	18 192
Scenario II	42 271	1 244	29,922	18 229

4. Discussion

This chapter will summarise and interpret the results of this research, and implications for the Garpenberg mine will be shown. Limitations will be stated and recommendations for improvements in Garpenberg are given, as well as ideas for further research. The chapter will be divided based on the division as is found in the results chapter.

To be able to maximise the value of the Garpenberg mine, the influence several parameters had on the cut-off grade needed to be known. This was achieved by performing cost calculations which led to cut-off grade calculations for three of the five deposits. It showed a similar cut-off grade for Kvarnberget and Lappberget which was significantly higher than the current cut-off grade of 316 kr/t. The main parameters which were found were ore transport and rock bolting, which were used to assign variable mining costs to various areas. The cut-off grades were adjusted to obtain a break-even situation. The resulting shift in profit distribution required rescheduling of stope extraction.

4.1 Costs and cut-off grades

The cost calculations performed in section 3.1 show that around 30% of mining costs could not be distributed based on activity. Therefore this amount was distributed over the three deposits based on ore tons produced in 2018. This causes Lappberget to incur almost 80 % of the general costs. For future research, this 30% could be reduced by distributing more costs based on activity, such as more accurate maintenance division, personnel and more.

The cut-off grades presented in table 3.4 show that Kvarnberget and Lappberget have similar cut-off grades of around 460 kr/t, while Dammsjön's cut-off is higher. This clear difference is due to mining activity in Dammsjön in 2018, which consisted entirely of CaF, which is a smaller scale mining method. Therefore, the mining costs will be higher, as explained in section 1.2.2. Similarly, the large-scale production in Lappberget caused mining costs of this orebody to be even lower than Kvarnberget. A larger scale operation will in general have lower operation costs than a smaller deposit.

Cut-offs for Garpenberg stopes in the LOMP are significantly lower than the calculated break-even cut-off grades (table 3.4). Currently, sublevel stopes have a cut-off grade of 316 kr/t. Figure 4.3 shows how the reserves of Lappberget are influenced by such an increase in cut-off grade. The grey line indicates the current cut-off grade of 316 kr/t, while the dashed grey line indicates the increases cut-off of 454 kr/t. As a result, ore reserves decreased by around 10 Mt. Even though a new break-even cut-off grade for all stoping areas has been calculated by Boliden, it is not implemented in the current designs yet. This is why the results obtained in this research are quite contrasting compared to the current designs, as the current cut-off grade is an older value, which does not represent the actual costs well.

4.2 Parameters

This section will interpret the results regarding the influence and implementation of parameters. The drawbacks of the method will also be described.

4.2.1 Parameter Sensitivity

Figure 3.1 displays the varying behaviour of cut-off grades to an increase in parameters for three deposits. This influence can be traced back to the costs found in table 3.1. Costs which are incurred in 2018 by an orebody, will dictate the influence each parameter has on the cut-off grade. Rock bolting accounts for around 6% of mining costs in Kvarnberget, while it accounts for 3% of mining costs in Lappberget. If maintenance and personnel costs are included, these percentages rise to 10,5% and 8,4% respectively. It becomes clear rock bolting has a larger influence on Kvarnberget than on Lappberget, which leads to a sharper increase in the break-even cut-off grade for Kvarnberget. However, these values are based on data from only one year. The data shows Kvarnberget had relatively more rock bolting activities in 2018 than Lappberget. This could be due to the required rock bolting patterns in the constructed drifts, or due to the amount of constructed drifts.

The bolting factors for Lappberget and Kvarnberget are calculated and are 1,16 and 1,02 respectively. These numbers represent the weighted average of bolting patterns used in 2018, and the difference in costs can therefore not originate from the bolting patterns. The second possible source of the cost difference is the amount of development meters constructed. 5465 m of drift was constructed in Lappberget, and 2944 m in Kvarnberget. To be able to compare these value, the size of the production must be taken into account, and therefore the indicator of meters of development per kilo ton of ore (m/kt) is used. By dividing development meters by the amount of ore produced from table 3.3, the values obtained are 7,32 for Kvarnberget and 2,65 for Lappberget. Which is likely to be the source of increased rock bolting costs. This difference indicates relatively more drifts were constructed in Kvarnberget, which led to the higher costs. As mentioned earlier, this value is only true for 2018. The planned m/kt for 2019 in Kvarnberget is 5,4, whilst Lappberget will be 3,4.

This example shows how sensitive the individual parameters are to calculations made in one single year. The overall cut-off grade is much more robust, due to the sustaining capital which are not included in cost calculations. When using the same data as in the previous paragraph, but including sustaining capital, the data becomes much more reliable.

4.2.2 Sustaining Capital

Section 3.1 briefly mentioned sustaining capital. Sustaining capital are expenses which are required to maintain existing capabilities (Hall 2014). This is made up of some larger maintenance projects, such as truck overhauls, but it mainly comprises the costs of development drifts. All costs of drifts and installations which are not 'entirely' permanent, are assigned as sustaining capital. As most of the development drifts meet this criterion, these costs are defined as sustaining capital. Figure 4.1 illustrates the sustaining capital drifts for Huvudmalmen, which will not be included into the final sum of cost calculations, hence their positive sign in table 3.1.

If the meter development per kilo ton indicator is limited to operational costs, which are directly related to ore production, the 2018 values for Lappberget and Kvarnberget are 1,7 and 2,7 m/kt. The values for 2019 are 1,8 and 1,9 respectively. Some differences between planned production and development and actual values

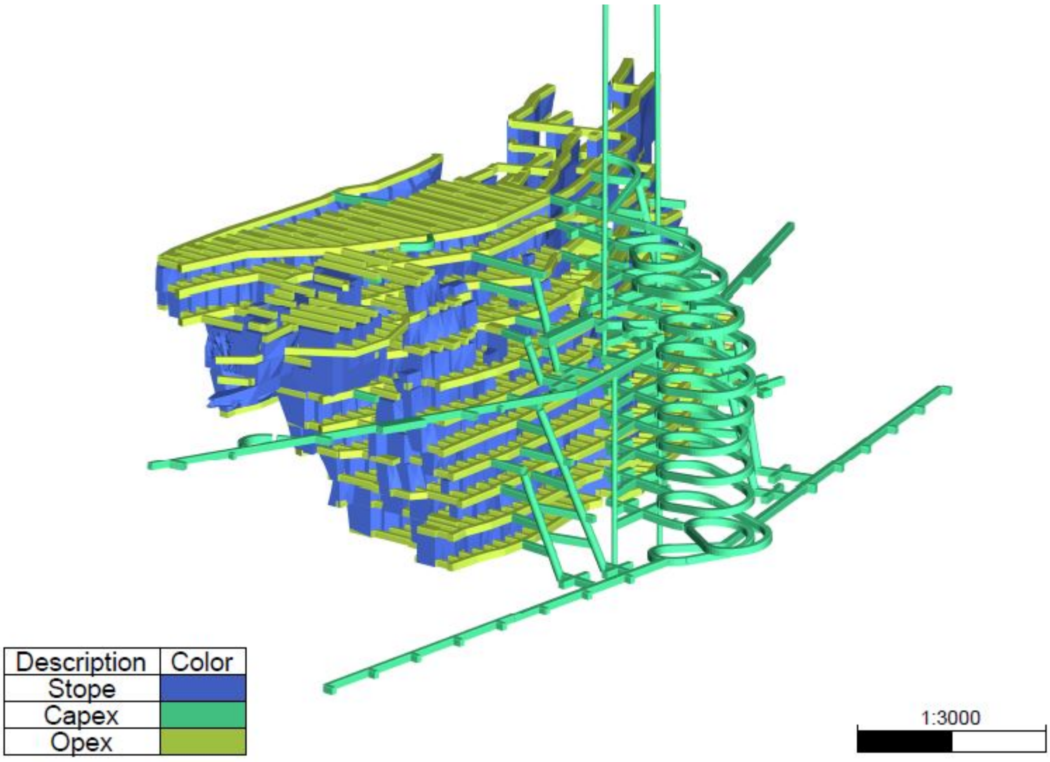


Figure 4.1. Illustration of sustaining capital for Huvudmalmen. Blue represents stopes, green depicts sustaining capital developments, yellow shows operating expenses development.

will still occur. However, these adjusted values clearly show the differences are not as large as depicted in the example above, which omits the sustaining capital.

4.2.3 Drill and blast

Section 3.4 mentioned drill and blast costs were not suitable for implementation into deposits. Figure 4.2 illustrates m/t values of 53 stopes slices in Lappberget in 2018. The stopes are arranged from highest to deepest production level and it clearly shows no location-based trend regarding required meters per ton of ore was found. Distinct areas within Lappberget which required more meters drilled per ton of ore could not be identified.

A possible reason for this is that the indicator relies on detailed stope and drilling designs. These designs are specific for each stope and are made shortly before extraction. These designs take very local factors such as rock fragmentation and stability issues into account and are therefore based on factors unknown during overall stope shape design. Therefore it not suitable for this research.

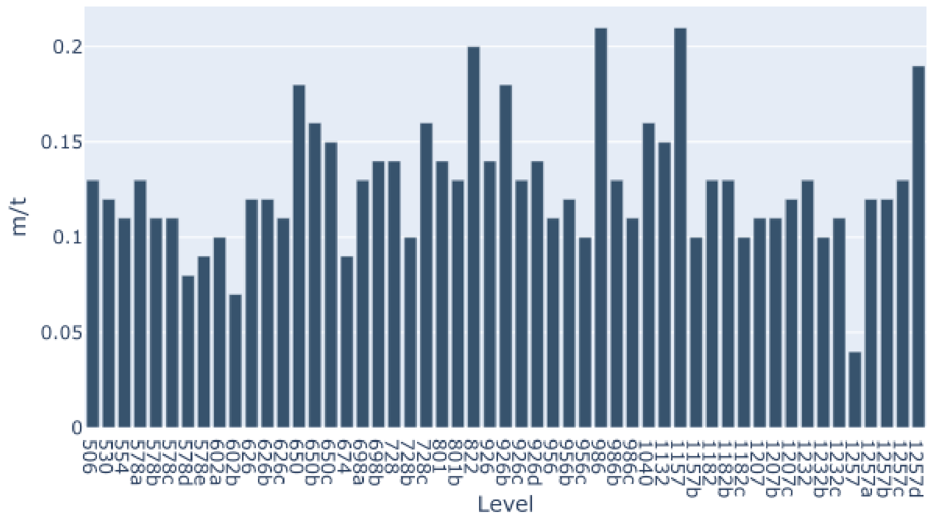


Figure 4.2. Meters drilled per ton of ore for stope in Lappberget in 2018. Letters indicate different stope on same level.

4.2.4 Net Smelter Return

Unfortunately, block models used in Garpenberg were not suitable for the addition of detrimental elements to value, into the NSR. Furthermore, the NSR method uses recovery values based on the LOMP, which are adjusted by a an empirical factor. Costs for processing ore from one deposit are not different from the other. Ideally, this should be combined into the NSR, which would make the cost models more accurate and stope designs better adjusted to this differences in costs.

This thesis solely adapted metal prices to assess the influence it had on stopes created. However, this could be seen as a sensitivity analysis towards market fluctuations, instead of an actual parameter.

4.2.5 Rock Bolting and Transport

Rock bolting and ore transport costs were found to be suitable for implementation into the deposits as locally varying parameters to adjust the mining costs. The resulting cut-off grade adjustments provided more value than a constant cut-off for Lappberget, Kvarnberget and Kaspersbo. Particularly ore transport cost implementation is aligned with results described by Hills et al. (2014). Although the example in section 4.2.1 above describes the vulnerabilities of parameter implementation, the method as such is a good way forward. It proved that straightforward calculations on cost data from the past can be used to develop a cost model which can be implemented into plans for the future. This method can be improved and developed further by analysing data from more years, and producing a more robust costing model. Furthermore, more costs could be divided as section 4.1 above described.

Due to the capabilities of the software, stopes were bundled in groups and no distinction between primary and secondary stopes was made, while they require varying bolting patterns. By adjusting these costs per stope, a more accurate cost model is presented which will improve the allocation of costs into deposits. Another addition to this method would be including the distance from stope to shaft or stope to storage. As each part of equipment will have to travel further to each stope, especially for the lower parts of the orebodies, this could have an influence on the costs in that area.

4.2.6 Disadvantages

However, the addition of more details to the costing model also has its drawbacks. Firstly sensitivity to errors. More errors could occur as more details are added to the costing model. Several steps during the cost distribution require assumptions to assign costs, which creates possibilities for errors. More errors could arise as more details are assigned to the cost model.

Secondly, the model creates a less adaptive LOMP. The LOMP will be created using detailed costs, but over time these costs could change due to new advances in technology, or different ways of mining, similar to the changes that may occur due to the use double stoping in the future. These changes will affect the LOMP more than a constant cost LOMP, which could lead to some areas which were too expensive to mine, to be surpassed.

Plans which are created using a costing model are good to have, but costs in an underground mine are not well predictable, and accidental costs are likely to occur. These accidental costs could become continuous costs if, for example, stresses in the rocks were underestimated, and a production level requires more reinforcement. In this case the implemented costs do not reflect the reality.

Finally, a general drawback of implementing variable costing and adjusting the cut-off to this is that it could lead to the cut-off strategy not being well aligned with the overall mining strategy. Ensuring the processing plant is operating at optimal capacity is an example of a overall mining strategy as well as extending the LoM. As described in section 4.1, implementing a new static break-even static cut-off grade in Lappberget causes the mineable reserves to diminish by around 10 Mt. This effect of a cut-off increase is shown in figure 4.3. Assuming all else remains equal, this cut-off grade increase shortens LoM by around three years. The combination of cut-offs in different areas of the mine could also cause issues to provide sufficient ore with the right grades for the processing plant.

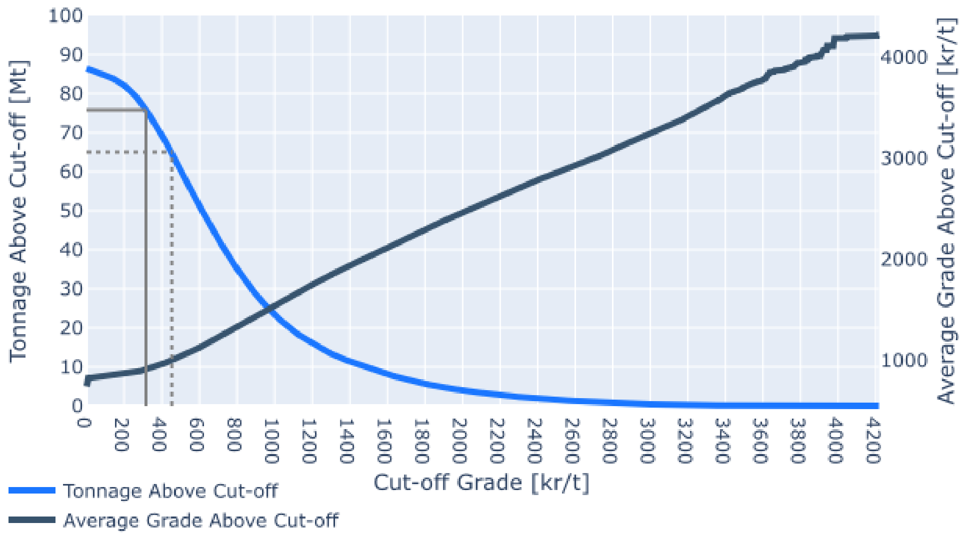


Figure 4.3. Grade-tonnage curve Lappberget. Grey lines indicate change in reserves due to new cut-off

4.3 Cut-off Optimisation

Optimising the variable cut-off grades was done using scenario analysis, and each deposit showed different results. Table 1.1 of the Appendix shows cut-off grades which yielded the highest NPV for each deposit. Huvudmalmen was not suitable for bulk analysis of cut-off grade differences and cut-offs for this orebody were only elevated. Kaspersbo, the smallest deposit, was not suitable for cut-off alterations as the amount of stopes was too small and removal of stopes had a large impact.

Even though scenario-analysis of cut-off optimisation yields some extra value, it is not a reliable method to improve overall value. Cut-offs vary per area or deposit and each deposit is handled as a separate entity, however, adjusting each deposit to it's optimal cut-off does not necessarily yield the highest value for a mine as a whole. A consequence of lowering cut-off is an increase in tonnage and a decrease in average grade above cut-off. Consequences of elevating cut-offs are vice versa, a decrease in tonnages while increasing the average grade above cut-off. For the Garpenberg mine, but also other multi- deposit mines, it is key to combine cut-offs of various areas to maximise total value of the mine, while obeying the constraints of maintaining a constant head grade and ore production. Combining cut-offs to obey these constraints could mean one deposit operates below break-even cut-off, to ensure the head grade is not too high, and sufficient production levels are achieved. Operating below break-even cut-off means leaving the philosophy of Mortimer stating "each tonne pays for itself" behind. Another deposit may yield more value when it focuses on high grade ore, instead of larger ore volumes. The cut-off of the latter deposit could be lowered once a hypothetical third deposit is mined out.

A combination of contrasting cut-off strategy's in these hypothetical deposits could provide more value for this mine as a whole.

Even though the above optimisation problem is too complex to be solved manually, some first steps can be taken using data from this research. Looking at table 3.7 shows some opportunities in Lappberget. If the assumption is made Lappberget is the sole deposit of Garpenberg. Example 10 below displays some combinations which are possible:

EXAMPLE 10

The following tables 4.1 to 4.3 show tonnages, profit, NPV and average grade per ton for three scenarios. Table 4.1 shows results of applying break-even cut-off grade for locally varying mining costs in Lappberget. Table 4.2 demonstrates the results of combining cut-off grades calculated as the optimal cut-offs for each deposit (from table 3.7). Lastly, table 4.3 demonstrates a third combination of calculated cut-off options to illustrate the opportunities of combining elevated and depressed cut-off grades.

Table 4.1. Tonnages, NPV and average kr/t for Lappberget for base Case

Base Case				
Area and Scenario	**Tonnage** [Mt]	**Profit** [Bkr]	**NPV** [Mkr]	**Average Grade** [kr/t]
Upper Lapp - 0%	16,3	10 654	4 169	893,8
Middle Lapp - 0%	5,4	2 199	1 272	728,8
Lower Lapp - 0%	32,3	18 008	2 989	880,6
Total	**54,9**	**31 024**	**8 487**	**896,8**

Table 4.2. Tonnages, NPV and average kr/t for Lappberget for optimal cut-offs from table 3.7

Optimal Cut-offs				
Area and Scenario	**Tonnage** [Mt]	**Profit** [Bkr]	**NPV** [Mkr]	**Average Grade** [kr/t]
Upper Lapp +25%	13,9	10 270	4 186	1015,8
Middle Lapp -30%	6,9	2 345	1 426	671,3
Lower Lapp +15%	29,1	17 973	3 022	934,4
Total	**49,9**	**30 587**	**8 634**	**921,0**

The table above shows NPV increases by around 2% while the amount of ore decreases by almost 10%, compared to a break-even scenario.

Table 4.3. Tonnages, NPV and average kr/t for Lappberget for combination of cut-offs

Combination of Optimal Cut-offs				
Area and Scenario	**Tonnage** [Mt]	**Profit** [Bkr]	**NPV** [Mkr]	**Average Grade** [kr/t]
Upper Lapp +20 %	14,3	10 286	4 144	1001,3
Middle Lapp -30%	6,8	2 345	1 426	671,3
Lower Lapp 0%	32,3	18 008	2 989	880,7
Total	**53,5**	**30 639**	**8 558**	**886,1**

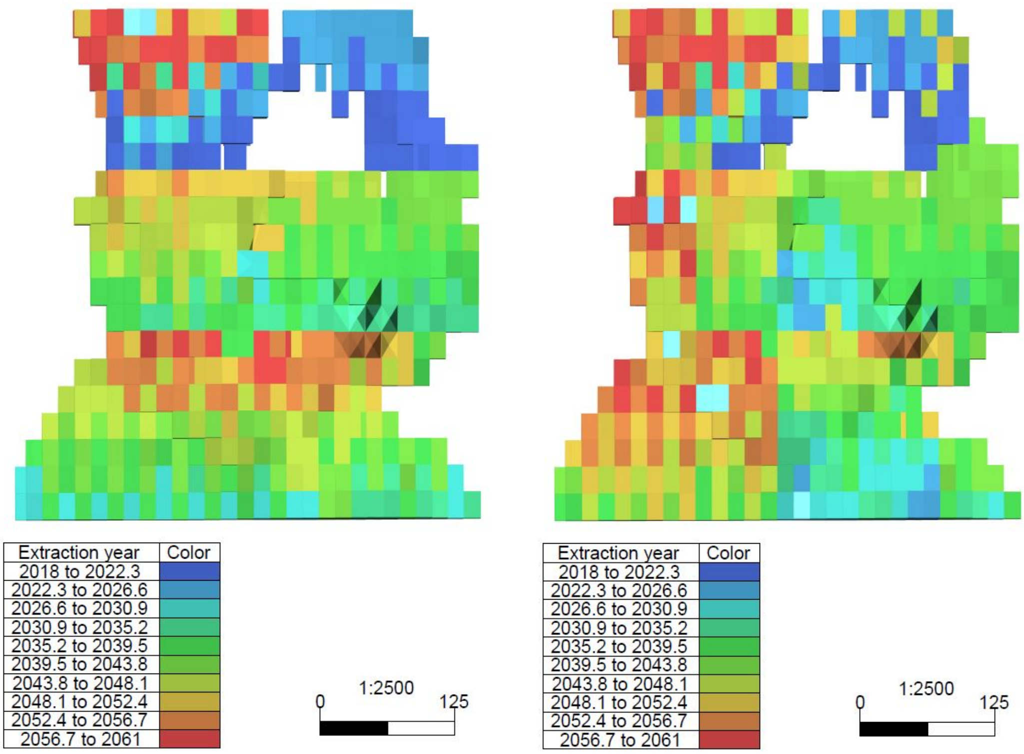
This last scenario of table 4.3 shows a combination of cut-offs to illustrate how the choice of these affects tonnages, value and average grade. This is an arbitrary combination which aims to increase ore tonnages, while maintaining a value increase. The latter scenario decreases ore tonnages by around 3%, while increasing NPV by around 1%, compared to a break-even scenario from table 4.1.

This example also shows upper Lappberget has the highest ore grade, which makes it suitable to elevate the cut-off substantially. Ore grade in each deposit has a large influence on the possibilities of cut-off grade combinations.

One can imagine the opportunities which arise when combining multiple deposits and areas. The addition of changing cut-off grades over time make the situation more complex, but could create more value for the operation as a whole.

4.4 Scheduling

Scheduling of lower Lappberget was performed to assess which influence the time value of money has on value obtained for a deposit. This process was done on a smaller scale for areas within a deposit. The method showed a substantial increase in value could be obtained by rescheduling newly created stopes. The schedule was adjusted to represent the yearly produced tonnages of Garpenberg's current LOMP, to validate results, as is shown in figure 3.6. However, another crucial constraint is maintaining a triangular retreat shape of stope extraction. Figure 4.4 a) represents the current extraction schedule for lower Lappberget copied onto newly created stopes. The triangular retreat is visible in the lower levels of each area. However, maintaining this shape was difficult as the scheduling was performed manually. The result of scheduling lower Lappberget is shown in figure 4.4 b)



(a) Current LOMP schedule for new stopes **(b)** Adjusted schedule for new stopes

Figure 4.4. Illustration of scheduling in lower Lappberget. Colours indicate extraction year of each stope.

Figure 4.4 b) shows newly scheduled stopes do not respect the triangular retreat shape constraint well. However, the fact the NPV increased by over 18% for a conservative settings which obeyed yearly ore targets proves the effect of rescheduling of stopes is large. By using suitable software and setting tighter constraints, a safe extraction schedule could likely be implemented, while still benefiting from increased value.

4.5 Double Stoping

The results of section 3.7 on double stopes described the financial feasibility of this method. It showed the majority of cost reduction was based on eliminating production levels and connecting drifts. However, this analysis omitted several extra costs and benefits, as these could not be stated with sufficient certainty. One omitted benefit is a potentially lower mining cost. Equipment will be operating in the same location for longer times, which is beneficial for costs. Secondly, less preparations for backfill are necessary, such as bulkheads. These extra factors could lead to cost reductions.

On the other hand, as section 1.4 described, the possibility of increased dilution is substantial. This will limit benefits obtained from using double stopes. If dilution percentages increase, the processing plant will be less efficient and more worthless rock will need to be transported. Preventing extra dilution could lead to increased drilling costs, as more directed drill holes are required.

Tests need to be performed to assess the consequences of double stoping on geotechnical stability, dilution and practical issues.

4.6 limitations

Even though the sections above already put forward several limitation to this research, this paragraph will describe other limitations which arose during this research.

Comparisons between newly created stopes using a variable cut-off and the LOMP of Garpenberg were performed, but these results were always more positive for the current LOMP. These stopes have been designed using the Stope Optimizer and have been adjusted to represent infrastructure and orebody shapes better. As this is a manual and time-consuming task, it was unsuitable for this research. Proper adjustment of stope shapes will likely provide more value than output of the Stope Optimizer. However, no proof for this could be provided.

The hill-climb algorithm implemented in the stope annealing stage of the Stope Optimizer (section 1.6.2) could get stranded in local maxima. Figure 4.5 illustrates the process of Simulated Annealing, but it also shows local minima in which a hill-climb algorithm could get stranded. Even though the figure mentions minima instead of maxima, it is identical. A hill-climb algorithm continues its search for a better solution until the next solution is not better than the previous. This means it could get stranded in a local maximum.

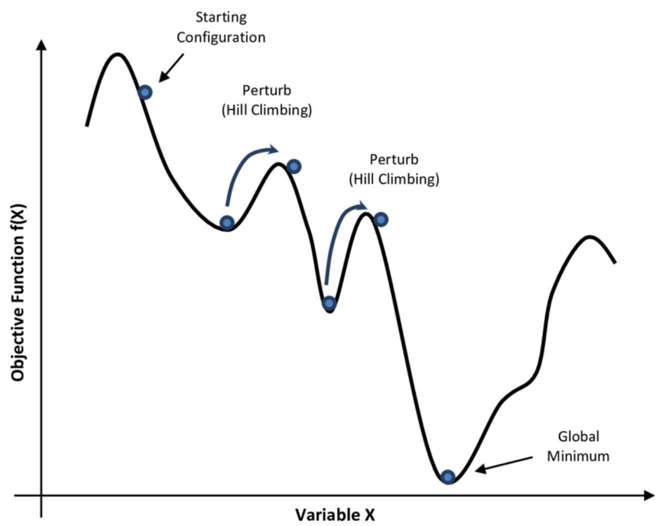


Figure 4.5. Illustration of Simulated Annealing which depicts the local minima. From Ghasemalizadeh et al. (2016)

This could also explain the results obtained by implementing variable cut-offs in an arbitrary area in Kvarnberget (figures 3.2 a-c). A constant downward trend of tonnages and stope sizes is expected, unlike the varying results obtained. The hill-climb algorithm used in the SO moves corners of stope shapes into all directions and interrogates the value. If value does not improve anymore, it assumes it has found its maximum, while this could be a local maximum.

A next cut-off grade which is used in the SO undergoes the same process, however, due to its slightly changed cut-off, the algorithm may find a new maximum. This could lead to the peaks which were identified in figure 3.2 a-c. This makes the SO unreliable for small optimisation regions. However, as optimisation regions for deposits are much larger, the inaccuracies become negligible. Additionally, the stope optimiser is used to create general stope shapes, not detailed stopes. Meaning smaller differences in tonnages and shapes will be adjusted when the final stope shapes are designed.

4.7 Recommendations

An improvement to this research would be to include more accurate costs regarding the NSR calculations. Poniewierski et al. (2003) showed the implementation of a variable processing cost was beneficial for the operation. This could also be beneficial for Garpenberg as the newly added costs would provide a more accurate image of the costs associated with mining a certain area.

Secondly, cost calculations should be performed more frequent to obtain a more robust costing model. Cost calculations, once set up, are straightforward to conduct, and will remove yearly outliers. Mining costs will most likely converge to a value over time. With this knowledge, cost calculations can be extended to further divide areas, such as Lappberget into three areas. Furthermore, cost can be distributed further to reduce the current 30% general costs.

Thirdly, with all mining costs implemented into orebodies, cut-offs should be deviated from their break-even values. Multiple scenario's could be created to assess the potential of this combination. However, combining sub-break-even cut-offs with elevated cut-offs will generate more value.

Scheduling was merely performed in this thesis to prove which influence this had on stoping. Effort should be put into acquiring a more competitive scheduling, instead of focusing on geotechnical and practical issues only.

Lastly, double stoping outputs should be analysed to assess the negative impact it possibly has on dilution. A time study could be used to locate the potential extra financial benefits, besides the elimination of production levels.

5. Conclusions and Recommendations

This thesis was aimed at determining the influence parameters had on cut-off grades of sublevel stopes, and using this to maximise value for the Garpenberg mine. This chapter will conclude the findings of this research and answers will be given to the research questions stated in section 1.8, which will also dictate the structure of this chapter.

Recommendations regarding further research are given in section 4.7, while recommendations in this chapter are aimed at how to use the findings from this research.

What influence do the parameters have on the stope design?

Each of the three locally varying parameters influenced cut-off grades differently. This influence was based on costs related to that parameter which were incurred in 2018. Ore transport and rock bolting are suitable to be used as a variable costing parameter. Calculations needed to assign a cost to an area are simple and can be adjusted easily, making it useful for implementation into deposits.

The parameter costs are included into break-even cut-off calculations and these either increased or decreased the cut-off grade of that area. Logically, an increased cut-off grade decreased the stope size, and therefore the ore tonnages. Conversely, a decrease in cut-off grade increased the stope sizes and the tonnages.

How does scheduling of stope extraction influence the value of the mine?

Stope extraction largely influences the value of the mine. Lower Lappberget was used to asses the influence of rescheduling, which became necessary after the implementation of variable costing and an aligned break-even cut-off grade. Assigning earlier extraction dates to stopes with a higher profit, yielded NPV improvements of up to 18%. However, even though the yearly ore production constraint was respected, a triangular retreat shape could not be achieved. Nonetheless, value obtained from implementing variable costing and variable cut-offs can be improved by rescheduling the stope extraction.

What is the best combination of cut-off grades and scheduling to be used to maximise value?

Scheduling has not been used to adjust the overall extraction dates of larger areas, or deposits. Therefore no conclusions can be drawn as to what the optimal combination of scheduling is, regarding different areas. However, as the answer to the previous research questions states, significant value can be obtained by rescheduling.

Cut-off grades were found by implementing variable costing into each deposit and by adjusted the cut-off grade to reach a break-even situation. These cut-offs underwent scenario analysis to optimise them. This was done for each area separately, however, the opportunities of combining contrasting cut-off strategies in different areas was proven. Using these combinations to obtain more value will require more time and research. This could eventually be combined with scheduling to maximise the value.

Nonetheless, implementing and optimising cut-off grades separately for each area yielded an NPV increase of around 134 Mkr compared to non-optimised cut-offs, and a 530 Mkr NPV increase compared to static cut-off grades. Which is an increase of around 4%. No clear comparisons between the newly created stopes and current LOMP stopes could be conducted as the latter are manually adjusted. This adjustment is a time-consuming task outside the scope of this research.

Is double stoping a feasible method to improve the efficiency of sublevel stoping?

Using a double stope of 50 m height compared to a regular stope of 25 m height costs less as it requires one production level less. For this research two scenarios were created for lower Lappberget. Both scenarios lead to significant decreases in development costs due to the redundancy of connecting drifts. The most profitable scenario made it possible to save 361 Mkr in development costs for four levels of double stopes. Combining these stopes with regular 25 m stopes in the upper areas of lower Lappberget, provided a profit increase of 146 Mkr. As double stopes are less selective than 25 m stopes, some ore is not accessible. This causes a decrease in the profitability of the stopes. Furthermore, dilation is expected to lower the benefit of double stopes further. However, potential benefits, besides a reduction in required development, are also not taken into account.

More research is required into the benefits and extra costs of double stoping to finalise the financial picture of this method. However, the method is likely to be more profitable than 25 m stopes for this area.

5.1 Recommendations for Garpenberg

The conclusions allow the author to recommend the following to the Garpenberg mine:

Adjust the static break-even cut-off grade from 316 kr/t for all stoping areas to the proper break-even cut-off for each area. Then implement a more robust variable costing system into each deposit. It will aid in the delineation of ore and prevent costly but low yielding stopes to be mined.

Secondly, optimise the combination of cut-offs as a whole. Substantial differences between orebodies are present, making it suitable to apply a cut-off strategy which can most likely respect all constraints and create more value. Ideally, this also includes scheduling from stope to deposit scale.

Lastly, implement double stoping from level 1400 downward in Lappberget and asses its influence on dilution and other geotechnical and practical consequences.

Bibliography

Akerstrom, M. (2010), 'Presentation garpenberg with different orebodies and mining methods', Boliden Powerpoint Presentation.

Alford, C., Brazil, M. & Lee, D. (2007), *Optimisation in Underground Mining*, Springer, Bosten, MA, USA.

Alford, C. & Hall, B. (2009), Stope Optimisation Tools for Selection of Optimum Cut-off Grade in Underground Mine Design, *in* 'Project Evaluation Conference 2009', Australasian Institute of Mining and Metallurgy Publication, Curran Associates, Inc., Victoria, Australia, pp. 137–144.

Ataee-Pour, M. (2000), 'A heuristic algorithm to optimise stope boundaries', *University of Wollongong Thesis Collection* .

Ataee-pour, M. J. (2005), 'A Critical Survey of the Existing Stope Layout Optimization Techniques', *Journal of Mining Science* **41**(5), 447 – 466.

Bai, X. (2013), Optimization of Underground Stope With Network Flow Method, PhD thesis, École Polytechnique de Montréal.

Boliden Group (2019), 'Metals for a sustainable society, Group Presentation', Company Presentation. http://intranet.boliden.internal/en/me/mediacenter/Pages/default.aspx.

Bootsma, M. T. (2013), Cut-off Grade Based Sublevel Stope Mine Optimization: Introduction and evaluation of an optimization approach and method for grade risk quantification, Master's thesis, Delft University of Technology.

Bootsma, M. T., Alford, C., Benndorf, J. & Buxton, M. (2014), Cut-off Grade Based Sublevel Stope Mine Optimisation, *in* 'Orebody Modelling and Strategic Mine Planning Symposium', Australasian Institute of Mining and Metallurgy, Springer International Publishing, pp. 281–290.

Cawrse, I. (2001), 'Multiple pass floating stope process', *Australasian Institute of Mining and Metallurgy Publication Series* pp. 87–94.

Dagdelen, K. (1992), Cutoff grade optimization, *in* 'Proceedings of the 23rd APCOM', Society for Mining, Metallurgy & Exploration, Society for Mining, Metallurgy & Exploration, Tucson, Arizona.

Dagdelen, K. (1993), An NPV maximization algorithm for open pit mine design, *in* 'Proceedings of the 24th APCOM', Institute of mining, Metallurgy and Petroleum, Institute of mining, Metallurgy and Petroleum, Montreal, Canada.

Elkington, T. (2009), Optimising mining project value for a given configuration, PhD thesis, The University of Western Australia.

Erdogan, G. & Yavuz, M. (2017), 'Application of Three Existing Stope Boundary Optimisation Methods in an Operating Underground Mine', *IOP Conference Series: Earth and Environmental Science* **95**, 042077.
URL: *https://doi.org/10.1088%2F1755-1315%2F95%2F4%2F042077*

Fjellström, P., Nyström, A. & Petersson, A. (2016), Rock mechanics feasibility study of the Dammsjön orebody at 1100-1250 meter level in Garpenberg, Technical report, Boliden Mineral AB.

Ghasemalizadeh, O., Khaleghian, M. & Taheri, S. (2016), 'A Review of Optimization Techniques in Artificial Networks', *International Journal of Advanced Research* **4**, 1668–1686.

Githiria, J. (2016), 'Cut-off grade optimisation to maximise the net present value using whittle 4X', *International Journal of Mining and Mineral Engineering* **7**, 313.

Hall, B. E. (2003), 'How Mining Companies Improve Share Price by Destroying Shareholder Value', *CIM Mining Conference and Exhibition Proceedings* .

Hall, B. E. (2014), *Cut-off Grades and Optimising the Strategic Mine Plan*, Vol. 1 of *20*, The Australasian Institute of mining and Metallurgy, Ground Floor, 204 Lygon Street, Carlton Victoria 3053 Australia.

Hall, B. E. & Poniewierski, J. (2016), 'Break-even is broken,Why the use of the break-even cut-off grade is flawed, and how it poses a fundamental problem for the mining industry', AusiMM Website. https://www.ausimmbulletin.com/feature/break-even-is-broken/.

Hall, B. E. & Stewart, C. (2007), *Optimising the Strategic Mine Plan - Methodologies, Findings, Successes and Failures*, Vol. 14, The Australasian Institute of mining and Metallurgy.

Hamrin, H. (1980), *Guide to Underground Mining Methods and Applications*, Mining and construction methods, Atlas Copco.
URL: *https://books.google.nl/books?id=VwHlAAAAMAAJ*

Henning, J. (2007), Evaluation of Long-Hole Mine Design Influences on Unplanned Ore Dilution, PhD thesis, McGill University.

Hills, A. J., Hall, B. E. & Poniewierski, J. (2014), Mine Scenario Analysis and Mine Variable Value, *in* 'Proceedings 12th AusIMM Underground Operators' Conference', Australasian Institute of Mining and Metallurgy Publication, The Australasian Institute of Mining and Metallurgy, pp. 111–118.

Horsley, T. P. (2005), Differentieal Cut-off Grades, *in* 'Proceedings Ninth AusIMM Underground Operators' Conference', The Australasian Institute of Mining and Metallurgy, The Australasian Institute of Mining and Metallurgy, Montreal, Canada, pp. 103–109.

Hou, J., Xu, C., Dowd, P. A. & Li, G. (2019), 'Integrated optimisation of stope boundary and access layout for underground mining operations', *Mining Technology* **0**(0), 1–13.
URL: *https://doi.org/10.1080/25726668.2019.1603920*

Högnäs, J. (2018), Boliden Summary Report, Resrouces and Reserves 2018 - Garpenberg, Technical report, Boliden Mineral AB.

King, B. & Newman, A. (2018), 'Optimizing the Cutoff Grade for an Operational Underground Mine', *Interfaces* **48**.

Kumral, M. & Villalba Matamoros, M. (2017), 'Heuristic stope layout optimisation accounting for variable stope dimensions and dilution management', *International Journal of Mining and Mineral Engineering* **8**, 1.

Lane, K. F. (1964), 'Choosing the optimum cut-off grade', *Colorado School of Mines Quarterly* **59**(4), 811–829.

Manchuk, J. & Deutsch, C. (2008), 'Optimizing stope designs and sequences in underground mines', *SME Transactions* **324**, 67–75.

Mc Elroy, A. (2016), Garpenberg Life of Mine Recovery Forecast, Technical report, Boliden Mineral AB.

Mokos, P., Glover, I. & Clauson, S. (2014), AMS Stope Shape Optimiser, Technical report, Alford Mining Systems.

Nhleko, A., Tholana, T. & Neingo, P. (2018), 'A review of underground stope boundary optimization algorithms', *Resources Policy* **56**, 59–69.

Ouchi, A., Pakalnis, R. & Brady, T. (2008), Empirical Design of Span Openings in Weak Rock based upon Support Type Employed, *in* 'Proceedings of the 99th annual ARMA conference', San Francisco, CA.

Ovanic, J. & Young, D. (1999), Economic Optimization of Open Stope Geometry, *in* 'Proceedings 28th APCOM-International Symposium', APCOM, Colorado School of Mines, Montreal, Canada, pp. 855–862.

Perron, J. (1999), Simple solutions and employee's involvement reduced the operating cost and improved the productivity at Langlois mine, *in* '14th CIM Underground Operators' Conference', Bathurst, NB, Canada, p. 20.

Poniewierski, J., MacSporran, G. & Sheppard, I. (2003), 'Optimisation of cut-off grade at Mount Isa Mines Limited's enterprise mine', pp. 32–37.

Poxleitner, G. (2016), 'Operating costs for miners', Presentation at MeMO, Sudbury.

Rendu, J. (2014), *An Introduction to Cut-off Grade Estimation, Second Edition*, Society for Mining, Metallurgy & Exploration, Incorporated.
URL: *https://books.google.se/books?id=jK1yAgAAQBAJ*

Sandanayake, D. S. S. (2014), Stope Boundary Optimisation in Underground Mining Based on a Heuristic Approach, PhD thesis, Wa School of Mines, Curtin University.

Tommila, E. (2014), Mining method evaluation and dilution control in Kittilä mine, Master's thesis, Aalto University.

Topal, E. & Sens, J. (2010), 'A new algorithm for stope boundary optimization', *Journal of Coal Science and Engineering (China)* **16**, 113–119.

Villaescusa, E. (2014), *Geotechnical Design for Sublevel Open Stoping*, CRC Press, Taylor & Francis Group.

Wellmer, F.-W., Dalheimer, M. & Wagener, M. (2008), *Calculation of the Net Smelter Return (NSR) of a Mine. In: Economic Evaluations in Exploration.*, Springer, Berlin, Heidelberg.

Wheeler, A. J. & Rodrigues, R. L. (2002), 'Cutoff-grade analysis at Fazenda Brasileiro: Mine planning for declining gold prices', *Mining Technology* **111**, 35–46.

Yao, X., Allen, G. & Willet, M. (1999), Dilution evaluation using Cavity Monitoring System at HBMS - Trout Lake Mine, *in* '101st CIM Annual General Meeting', Calgary, Alberta, Canada, p. 10.

A. Appendix 1

Table 1.1. NPV and tonnages for elevated cut-off scenarios. Part 1

Deposit	Scenario	Tonnage [Mt]	NPV [Mkr]
Lower Lappberget	Base	32,33	2 989
	+5%	31,00	2 968
	+10%	30,06	3 015
	+15%	29,03	3 022
	+20%	28,05	2 974
Middle Lappberget	-30%	6,86	1 426
	-20%	6,56	1 413
	-10%	5,98	1 565
	Base	5,46	1 275
	+5%	5,29	1 268
	+10%	5,05	1 218
	+20%	4,65	1 185
Upper Lappberget	Base	16,30	4 169
	+5%	15,68	4 010
	+10%	15,22	3 991
	+20%	14,33	4 144
	+25%	13,98	4 186
Dammsjön	-20%	18,99	1 045
	-10%	19,90	1 043
	Base	17,68	1 044
	+10%	17,01	1 041
	+20%	16,27	1 028

Table 1.2. NPV and tonnages for elevated cut-off scenarios. Part 2

Deposit	Scenario	Tonnage [Mt]	NPV [Mkr]
Huvudmalmen	-20%	18,99	1 045
	-10%	19,90	1 043
	Base	17,68	1 044
	+10%	17,01	1 041
	+20%	16,27	1 028
Kvarnberget	-20% [a]	17,07	2 032
	-10% [b]	15,79	2 039
	-5% [c]	15,02	2 032
	0	14,46	2 025
	+10%	13,25	1 865
	+20%	11,81	1 837

[a] Unadjusted
[b] Unadjusted
[c] Unadjusted