The University of Zambia

School of Mines

Evaluation of mine design options for the Nkana Synclinorium Mining Project: A case study of Mopani Copper Mines, Zambia.

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Declaration

I Hugo Nicacio, declare that this work is my own and that the work of other persons utilized in this report has been duly acknowledged. This work presented in this report has not been previously submitted at the University of Zambia or any other university for similar purposes.

Signed: ______________________ Date: ______________________

HUGO NICACIO

Abstract

Mining method selection is one of the most critical activities of Mining engineering with the ultimate goal of maximizing profit, mineral recovery and arrive at a method with the least problems among feasible alternatives. The geology at the Synclinorium is complex with four main lithologies i.e. SOB Shale, Hanging Wall Argillite, Near Water Sediments and the Upper Quartzite, folded with variable thickness. A critical geological feature is the foliation or bedding that replicate the folding along the strike of the ore body. This poses a challenge in terms of selecting a suitable mining method for this ore body that will be less costly with high recoveries and low dilution.

The main aims of the research were to select a suitable mining method, identify the current challenges encountered in the existing mining methods and carry out an economic evaluation of the Synclinorium mining project.

The methodology used in data collection involved underground visits to various sections of the mine to understand the geology, rock types, orientations and geological discontinuities, current mining methods and associated challenges. Others involved a review of research works on underground mine planning and design, literature on mining method selection, mining journal and research done by Murray and Roberts and SRK consulting firms on the design. Detailed discussions on the input parameters for mine design with mine planning, geologists, geotechnical engineers, senior mining officials and cost accountants were undertaken to understand the current cost of mining. Diamond drilling and logging of borehole core was also conducted.

The results gathered were then subjected to mine design criteria for selecting a mining method. In this research, University of British Colombia (UBC) mining method selection criteria, online mining method selection tool (MMST) and fuzzy dominance methods were used for the selection of a suitable mining method. These methods reviewed that, Sublevel stoping, Cut and fill, Sublevel caving and Block caving were selected in that order.

However, after subjecting the selected mining methods to geotechnical, technical and economic evaluation, Sublevel Open (SLOs) stoping with fill in the anticline, Sublevel caving (SLC) in the limbs and VCR in the synclines were recommended. The current SLC and Vertical crater retreat methods require modification to address the current challenges due to hang-ups, delayed hanging wall exposure and creation of a huge void prior to hanging wall collapse as mining progresses and hole deviations in VCR at 50m that results in re-drilling and ultimately increase the costs of mining. The economic evaluation indicates that the Synclinorium mining project is viable with a projected average annual income of $340million.

Dedication

This thesis is dedicated to my wife

Mrs. Yesenia Trujillo De Nicacio

To my sons Bryan Nicacio, Jhean Nicacio my only daughter Yetzira Nicacio. You have really sacrificed to see me through the course of my studies. I would like to thank my parents Mr. and Mrs. Nicacio for their guidance and support they have continued to render.

I wish to thank the almighty God for giving me life and ensuring my safety during my long journey to and from Lusaka.

- This is for you -

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Chapter 1

1.0 Introduction

1.1 Background of the Synclinorium mining project

Mopani Copper Mines (Nkana mine) is anticipating a fifty percent decline in ore production in the coming years due to the projected depletion of copper ore at the Central Shaft and Mindola North Shaft. Currently, Nkana produces between 3.5 and 4.0 million tpa (tonnes per annum) of ore from its four underground mines. Due to depleting ore reserves in the Central Shaft and Mindola North Shaft, it is projected that the total amount of ore to be mined from the SOB and Mindola Main shafts will be 1.8 million tpa by 2018.

Mopani Copper Mines (MCM) intends to compensate for this decline in production by mining the copper ore in the Synclinorium ore body by sinking one rock-hoisting shaft and one ventilation shaft. SOB and Central shafts cannot handle the desired tonnage of 4.0 million tpa due to the limited shaft capacities (1.1 million tonnes for Central and 1.6 million tonnes for SOB).

MCM is currently sinking a 7m diameter rock hoisting shaft, 1,277m deep to hoist 4 million tpa of ore. The rock-hoisting shaft is being constructed by a conventional shaft sinking method. Murray & Roberts is the main contractor.

The increase in the mining activities in the Synclinorium will require improving the ventilation in the area. Hence, one Ventilation Shaft is proposed to be raise bored to expel used air from underground in order to ensure the health, safety and wellbeing of the workers. The ventilation shaft will be located at 75 metres away of the rock-hoist shaft described above. The shaft will be 6 metres diameter and 1,180 metres deep.

1.2 Project Location

Mopani Copper Mines (MCM) started operating in Zambia in the year 2000 after privatisation. MCM purchased the Zambia Consolidated Copper Mines Limited ("ZCCM") assets at Mufulira and Nkana in Kitwe from the Government of the Republic of Zambia (GRZ). The assets comprised of Underground Mine, Concentrator, Smelter, and Refinery at Mufulira and Underground Mine, Open Pits, Concentrator and Cobalt Plant at Nkana. It is an integrated copper and cobalt producer located on the Copper-belt Province of Zambia. MCM is a joint venture company comprising of Glencorextrata International AG (73%), First Quantum Minerals Limited (17%) and Zambia Consolidated Copper Mines Limited (10%). Since inception, Mopani Copper Mines Plc has demonstrated its unwavering commitment to Zambia by investing over US$2 billion in redevelopment of the old mine assets and implementation of new projects aimed at enhancing, productivity and promoting safety and environmental sustainability. The study was conducted at Nkana mine site (Synclinorium) in Kitwe, see Figure 1.1 below;

Figure 1.1 Location of Synclinorium Underground Mine, Kitwe. (Google maps, 2018)

1.3 Historical background

Nkana mine deposit was discovered sometime in 1910 when mineralized block shale was found along Wusakile stream by Mr. Moffat Thompson. In 1914, Mr. Donald Mclean, a railway employee first registered a block of claims known as Nkana's Copper. In 1916 another railway employee by the name of W. C. Winniort repegged Nkana's Copper and in 1922 the copper prospect was sold to Bwana Mkubwa. Diamond drilling was carried out under the directive of Mr. Austin J. Bancroft and in 1931 Nkana's first copper was produced at Central Shaft. In 1933 work began on the mine's second shaft at Mindola Sub-vertical and in the same year an electrolyte refinery was erected. In 1956, SOB shaft was officially opened.

By 1964, all copper mines in Zambia were privately owned either by Anglo American Corporation (AAC) or Roan Selection Trust (RST). However, in 1972, government nationalized all the mining assets and were divided into two companies namely Nchanga Consolidated Copper Mines (NCCM) and Roan Copper Mines Limited (RCM). NCCM consisted of all mines previously owned by Anglo American Corporation while RCM consisted of all mines previously owned by Roan Selection Trust mines. Nkana, then called, Rhokana was under Anglo American Corporation and thus became a constituent of NCCM.

In 1982, government merged NCCM and RCM into one to form the Zambia Consolidated Copper Mines (ZCCM) Limited. Unfortunately ZCCM suffered many problems resulting in the decline in copper production revenues. The loss of revenues from copper combined with other economic and political factors adversely affected the country's economic performance, leading to a decision in 1989 to go to a private sector driven economy. This is the privatization that led to the birth of Mopani Copper Mines (MCM) and other currently existing privately owned mine companies.

The Nkana mine, located on the outskirts of Kitwe, has been in production since 1931. Nkana mine hoists ore to surface through three operating shafts, the Mindola Shaft, the Central Shaft and the South Ore body Shaft.

Vertical crater retreat is the predominant mining method while sublevel open stoping and sublevel caving methods are also used.

Nkana's long-term future lies in the Synclinorium (SYNC), an ore body of some 90-million tonnes, situated between the central and South Ore body (SOB) shafts. Once the Synclinorium mining project comes into production, Nkana mine will have an extended life span of up to 20 years.

The upper levels of the Synclinorium are currently being reached by means of the South Ore body (SOB) Shaft. The Nkana Synclinorium is effectively a new mine and is, in some respects, similar to the Mufurila Deep project in that it is a reserve that has been known for some time and has not been developed due to lack of capital. See Figures 1.2, 1.3, 1.4 below;

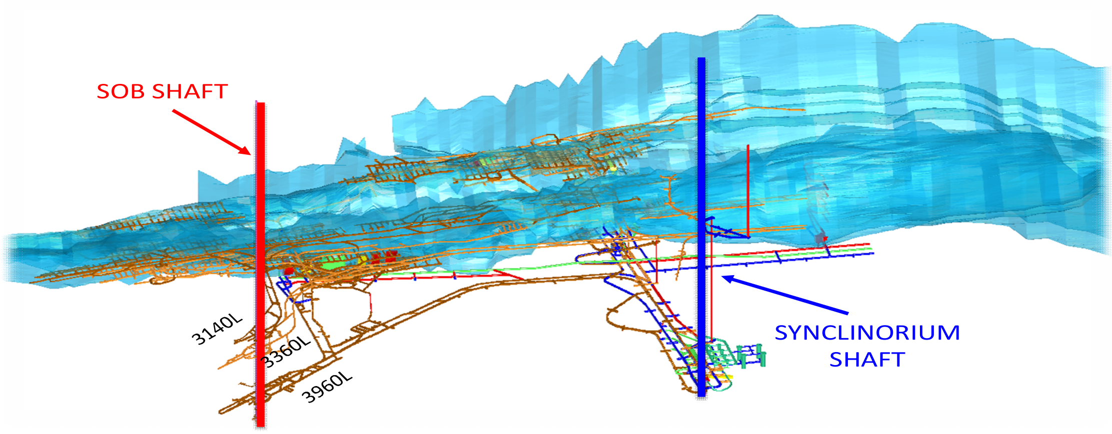


Figure 1.2 Ore body structure of Synclinorium Shaft (Mopani geology, 2013)

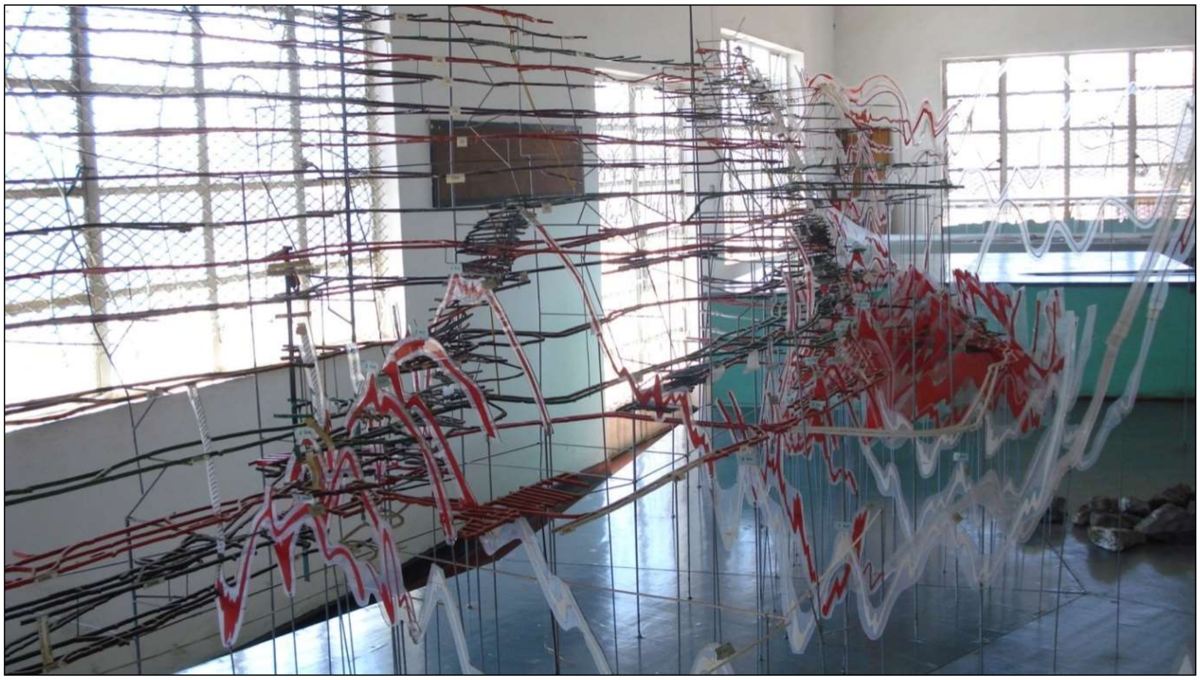


Figure 1.3 1:1000 Scale Model of Nkana Synclinorium Model (Nkana Model Shop)

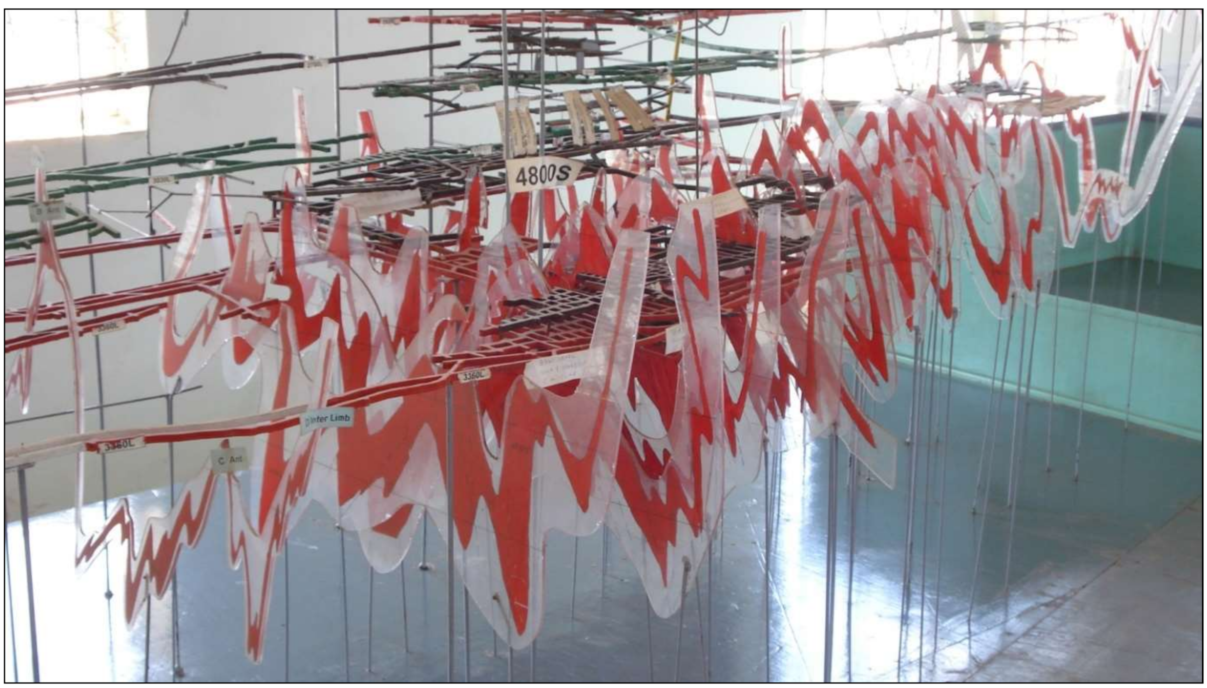


Figure 1.4 1:1000 Scale Model of Nkana Synclinorium Model (Nkana Model Shop)

1.4 Problem Statement

Mining method selection is one of the most critical activities of Mining engineering with the ultimate goal of maximizing profit, mineral recovery and arrive at a method with the least problems among feasible alternatives.

However, the geology at the Synclinorium is complex. There are four main lithologies within the Synclinorium. These are, SOB Shale (SOBS), Hanging Wall Argillite (HWA), Near Water Sediments (NWS), and the Upper Quartzite (Upper-Q). Each unit is folded with variable thickness and rock mass conditions. Another critical geological feature in the Synclinorium is the foliation or bedding that replicate the folding along the strike of the ore body. This ultimately pose a challenge in terms of selecting a suitable mining method for this ore body. See the overview of the main lithologies in Figure 1.5 below:

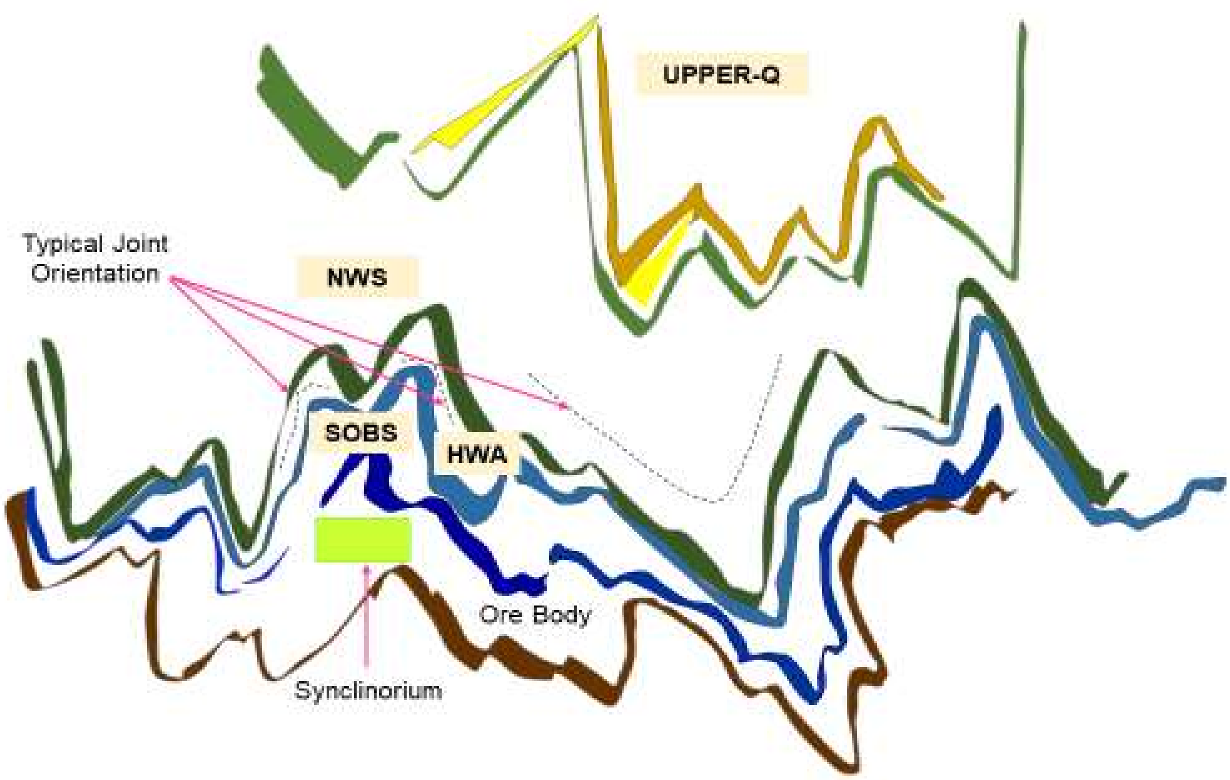


Figure 1.5 Geological section showing the main lithologies within the Synclinorium (Mopani geology, 2013)

1.5 Research Questions

- What are the current problems encountered with the current mining method?
- Which method can be used to mine complex deposit at MCM?
- What are the economics of the new mining method?

1.6 Research objectives

The main objectives of the research were to:

- Identify the current challenges encountered in the existing mining methods and recommend solutions.
- Design a suitable mining method for Nkana Synclinorium Mining Project.
- Carry out economic analysis of the design.

1.7 Significance of the study

The study is important because it highlights the most economical and safe mining methods to be used at Synclinorium project following the complexity nature of its ore body. The methodology chosen will help reduce ore loses from dilution and premature collapse of the stopes and mine excavations and proper plans for mine ventilation will be done. This will ultimately provide enough ore to increase production from the current 1.2million tonnes to planned 4million tonnes per annum by 2020.

1.8 Methodology

In order to achieve the objectives, there was need to collect sufficient data on each of the set objectives. Therefore, the methodology used in collecting data involved underground visits to various sections of the mine to understand the geology, rock types, orientations and geological discontinuities, current mining methods and associated challenges.

Other methods involved a review of research works done on underground mine planning and design, literature on mining method selection, mining journal and research work done by Murray and Roberts and SRK consulting firms on the design.

Detailed discussions on the input parameters for mine design with mine planning, geologists, geotechnical engineers, senior mining officials and cost accountants were undertaken to understand the current cost of mining. Furthermore, diamond drilling and logging of borehole core was also conducted.

Therefore, geological and geotechnical data of the ore body, hanging wall, foot wall required for mine design was collected. Furthermore, challenges encountered with the current mining methods such as Sublevel caving (SLC) and Vertical Crater Retreat (VCR) were noted. Information on tonnages, capital and operating costs, development and long hole metres, dilution, recoveries and projected metal prices were collected to do an economic analysis of the deposit.

The data collected was then subjected to mine design criteria for selecting a mining method. In this research, university of British Colombia (UBC) mining method selection criteria and online mining method selection tool (MMST) were used for the selection of a suitable mining method. The mining methods selected were further subjected to geotechnical, technical and economical evaluation to arrive at the most suitable mining method.

1.9 Thesis structure

The outline of this thesis is as follows:

Chapter 1 serves as an introduction to the study, it discusses the background of the Synclinorium project, the objectives of the research, its significance and the approach taken to achieve the stated objectives.

Chapter 2 presents theoretical basis of study by detailing and evaluating mine design criteria, mining methods and other related literature on mine and stope design.

Chapter 3 explains the methodology used in collecting data for the study. Research methodology used included; Site investigations, desktop study, diamond drilling, logging of borehole core and Software application for mine design and analysis

Chapter 4 presents data collected from the research methodology employed in chapter 3. Data collected included geometry of deposit, geology and hydrology conditions (mineralogy, petrology, uniformity, alteration and weathering), geotechnical properties of rocks and ore (elastic properties, state of stress, competency and other physical properties), economic considerations (tonnage of reserves, production rate, mine life, productivity and mining cost), technological factors and environmental concerns such as ground control and subsidence.

Chapter 5 discusses how the data obtained was processed and analyzed. In this research, university of British Colombia (UBC) mining method selection criteria and online mining method selection tool (MMST) were used for the selection of a suitable mining method. The mining methods selected were further subjected to geotechnical, technical and economical evaluation to arrive at the most suitable mining method.

Chapter 6 summarizes the findings of this research and recommends areas of improvement in the current mining methods and how best the selected mining methods may be applied to mine the deposit at the Synclinorium economically.

1.10 Summary

This Chapter has provided detailed background of the Synclinorium project, the objectives of the research, its importance and the methodology undertaken to achieve the stated objectives. It has also provided the overall structure of the study.

Chapter 2

2.0 Literature Review

2.1 Introduction

This Chapter reviews existing literature on mining methods selection criterion and underground mining methods in use. It also provides background information on parameters required for mine and stope design since they relate to research objectives. This chapter has extensively reviewed data from mining journals, research works done on mine planning and design and available mining engineering books.

2.2 Classification of mining methods

All mining methods can be classified as either self-supported, supported, or caving and the determination of which method is most applicable rests upon evaluation of parameters that can be grouped into seven basic categories summarised in Table 2.1:

Table 2.1. Underground Hard-Rock Mining Method Selection Criteria

Evaluation Parameter	Considerations
Geotechnical	• Lithology
	• Groundwater
	• Geophysics
	• Ore genesis
Mineral Occurrence	• Continuity of ore zones within mineralized strata
	• Occurrence of mineral within ore zone(geologic grade)
	• Economic mineral occurrence within ore zone(mining grade)
Ore body configuration	• Dip
	• Plunge
	• Size
	• Shape
Safety/ regulatory	• Labor intensity of method
	• Degree of mechanization
	• Ventilation requirements
	• Refrigeration requirements
	• Ground support requirements
	• Dust controls
	• Noise controls
	• Gas controls
Environmental	• Subsidence Potential
	• Groundwater contamination
	• Noise controls
	• Air quality controls
Economic	• Minable ore tons
	• Ore body grade
	• Mineral value
	• Capital costs
	• Operating costs
Labor/Political	• Costs, influences

2.2.1 Geotechnical evaluation

Geotechnical considerations in mining method selection include an evaluation of lithology, groundwater, geophysics, and ore genesis of the deposit. This analysis should occur concurrent with the exploration drilling phase and must include an evaluation of the ore zone hanging wall and footwall host formations and general surface topography. As a result, a percentage of the drilling must provide core.

2.2.1.1 Lithology

Drill core structural mapping provides critical information on faulting, jointing, foliation, shears, and oxidation that could affect the integrity of both ore and host formation zones. As a three-dimensional model of the ore body is built, so must all structural features be included. These become the parameters by which future method selection and subsequent design is influenced. For example, a highly sheared hanging wall would certainly preclude open stoping as a viable option and, depending upon geophysical testing and ore body configuration, could favour sublevel caving as the optimal method. The degree of foliation and oxidation within an ore zone can have a significant impact on the amount of ground support required or whether the ore zone is cavable. Additionally, detailed evaluation of the surface topography from aerial photographs and magnetic surveys will reveal geologic structures, such as major faults or magnetic anomalies. These allow a broad overview of the area and can subsequently be tied to detailed lithologic core mapping.

2.2.1.2 Groundwater

Initial exploratory drilling will usually define the existence of groundwater. It is important to determine water levels within the formation, permeability of the formation, initial flows, and sustained flows. Initial quantification during drilling can be made of the water-bearing strata interval and a selected number of holes then designated for piezometer installation. This will monitor water pressure within the borehole and become an indicator of whether the system is recharging itself. Flows can be measured from drill holes with packers and flow meters, and evaluation of flows over a period of time will provide data on the system's recharge rate. Finally, formation permeability can most simply be analysed by pumping through drill holes into the formation.

2.2.2 Ore body configuration

The physical parameter of an ore body will often preclude use of many mining methods. Orientation considerations such as dip, plunge and strike along with the size and shape of the ore body are key evaluation factors. Many methods, such as shrinkage stoping, open stoping, VCR mining, and sublevel caving, depend upon gravity ore flow to extraction points. However, deposits with footwall dips less than the broken ore rill angle will not allow complete extraction with these methods. Generally, ore bodies with dips less than 55° are not amenable to gravity flow. Additionally, hanging-wall dip must be considered in any method with structurally incompetent host rock. The hanging wall will often begin caving during extraction of a gravity system, waste is soon pulled through the extraction cone, progressively diluting and finally displacing ore. Those deposits with flat footwall dips are best mined with cut and fill or panel longwall methods. Where applicable, room and pillar is certainly preferred for its high mechanization and productivity amenability. Plunge is the vertical angle component along strike and it affects the number of mineral units per vertical and horizontal linear distance. In this respect, plunge has a significant impact on the amount of development required for each ton of ore. A flat-plunging ore body will ultimately require much more development than its steeper counterpart. Methods such as sublevel caving or undercut and fill stoping may be chosen because mining can progress down-plunge and minimize preproduction development.

The strike or azimuth of an ore body in the long dimension can affect a mining method by its length. Open voids have a critical geophysical length-to-width parameter beyond which they are no longer stable. Open stoping, shrinkage stoping, or VCR mining that creates large openings are often designed in a series of smaller panels to avoid hanging wall failure in deposits with a long strike dimension. These panels are then mined and backfilled in an alternating pattern along strike or complete extraction sacrificed and permanent ore-bearing pillars left as support.

2.2.3 Safety/regulatory factors

The health and safety of personnel is critical to the selection of any mining method. Irrespective of which commodity is extracted, a mine's most valuable asset is its people. Consequently, a number of safety considerations need to be included in any mining method evaluation. Primarily, those methods that are the least labour intensive offer an inherent safety advantage by minimizing the exposure of personnel to hazards. Replacement of shrinkage stoping with more recently developed VCR mining has been due, in part, to safety benefits derived by elimination of mining work on a broken ore pile in the active stope. Maximizing mechanization not only increases efficiency and reduces labour requirements but also reduces the physical work effort and resultant occupational injuries. Many previously termed "labour-intensive" mining methods have been adapted to mechanization. Typical cut and fill stoping that utilized jacklegs and slushers have now incorporated rubber-tired jumbos and load haul-dump (LHD) units. This has reduced instances of back injury, carpel tunnel syndrome, and a multitude of accidents. Mechanized mining systems do contain unique inherent safety considerations.

2.2.4 Environmental factors

Most underground mining, except the caving methods, have the distinct advantage of creating a minimal disturbance to the environment. Many underground operations exist below communities, and through proper choice of method and sound operating procedures, surface impact from mining is virtually non-existent. Where consideration must be given to minimizing impact on the surface environment, any mining method that could result in subsidence should be avoided. Both block caving and sublevel caving are systems employed in ground where either the ore and/or the host rock will naturally cave once extraction is initiated. These methods usually result in significant surface subsidence. In cases where block or sublevel caving is applicable, geotechnical considerations may preclude use of any other unsupported method. Supported methods must then be evaluated, and depending upon ground competency, either cut and fill or timbered cut and fill stoping utilized.

2.2.5 Economic considerations

The economic feasibility of an ore deposit is dependent upon Minable tons, ore body grade, mineral value, Capital costs and operating costs. Method selection plays an integral role in these considerations since it impacts all factors except mineral value. As a result, proper extraction method design dictates a project's profit margin, and in this sense, mineral value influences the mining method. An ore body's minable inventory is a reflection of the tons and grade that can be mined at a desired profit. The mining method will significantly influence this inventory by affecting selectivity. For example, open cut and fill (OCF) stoping offers a high degree of extraction control and will optimize the mineral content of every ton mined. Unfortunately, selective methods generate higher operating costs because they are more labour intensive and consequently less productive than bulk methods.

2.2.6 Labour and political considerations

The choice of any mining method is influenced by the availability and cost of labour. While the preceding discussion has assumed a relatively high labour cost, the economic situation in many countries results in an abundant work force, which is often unskilled. Depressed labour costs make mechanization a less economically attractive alternative in this case. Additionally, governments in these situations may be less profit motivated and more interested in maintaining maximum employment. Economic considerations take a diminished role in favour of providing consistent production. A mechanized method requiring trained personnel may not be a practical alternative since introduction of machinery will probably not reduce the work force. The political climate in many countries can result in relatively unstable governments. Significant risks associated with these conditions favour a mining method that minimizes capital outlay and provides the quickest return on investment. For instance, sublevel caving may be rejected in favour of cut and fill stoping because of the latter method's ability to generate revenue in significantly less time with less capital investment. In these cases, mitigation of risk on investment becomes a predominant.

2.3 Mining method selection by multiple criteria decision making tools

Mining method selection is one of the most critical and problematic activities of mining engineering. The ultimate goals of mining method selection are to maximize company profit, maximize recovery of the mineral resources and provide a safe environment for the miners by selecting the method with the least problems among the feasible alternatives.

Selection of an appropriate mining method is a complex task that requires consideration of many technical, economic, political, social, and historical factors. The appropriate mining method is the method which is technically feasible for the ore geometry and ground conditions, while also being a low-cost operation. This means that the best mining method is the one which presents the cheapest problem. There is no single appropriate mining method for a deposit. Usually two or more feasible methods are possible. Each method entails some inherent problems. Consequently, the optimal method is one that offers the least problems.

Each ore body is unique, with its own properties, and engineering judgement has a great effect on the decision in such the versatile job of mining. Therefore, it seems clear that only an experienced engineer who has improved his experience by working in several mines and gaining skills in different methods, can make a logical decision about mining method selection. Although experience and engineering judgement still provide major input into the selection of a mining method, differences in the characteristics of each deposit can usually be perceived only through a detailed analysis of the available data. It becomes the responsibility of the geologist and engineer to work together to ensure that all factors are considered in the mining method selection process.

Each of these criteria can become the principal determining factor in method selection, but the obvious predominance of one consideration should not preclude careful evaluation of all parameters. In order to determine which mining method is feasible, we need to compare the characteristics of the deposit with those required for each mining method; the method(s) that best match should be the one(s) considered technically feasible, and should then be evaluated economically.

Several methodologies have been developed in the past to evaluate suitable mining methods for an ore deposit, based on the physical and mechanical characteristics of the deposit such as shape, grade, and geomechanical properties of the rock. A group of mine scientists such as Boshkov and Wright (1973), Morrison (1976), Laubscher (1981) and Hartman (1987), suggested a series of approaches for suitable mining methods. These studies were neither enough nor complete, as it is not possible to design a methodology that will automatically choose a mining method for the ore body studied. The uses of numerical systems to evaluate the appropriateness of a mining method for a particular ore deposit have been in use for some time.

In 1981 Nicholas suggested for the first time a numerical approach for mining method selection. The Nicholas methodology follows a numerical approach to rate different mining methods based on the rankings of specific input parameters. A numerical rating for each mining method is arrived at by summing these rankings. The higher the rating, the more suitable the mining method. One of the problems of this approach was that all selection criteria had the same relevance. A recent modification involves the weighting of various categories, such as that of ore geometry, ore zone, hanging wall, and footwall (Nicholas 1992).

The wrong definition of some scores and the small domain between favourable and unfavourable scores prompted Miller, Pakalnis and Paulin (1995) to investigate the

UBC approach. The UBC mining method selection is a modification of the Nicolas approach, which places more emphasis on stoping method, thus better representing typical Canadian mining design practices. Unfortunately, in the UBC approach, the importance of each selection criteria has not been considered. In addition, neither of these methods takes account of the uncertainty associated with boundary conditions of the categories used to describe input variables.

2.3.1 Fuzzy set theory

Acquiring the information necessary for mining method selection is an elaborate process, to say the least, and once obtained, data is likely to be ambiguous. In addition, decision makers must often apply rules of thumb or incorporate their personal intuition and judgement when deriving performance measures based on indefinite linguistic concepts, e.g. 'high', 'low', 'strong', 'weak', and 'stable'. Such terminology is common and is caused by imperfectly defined problem attributes. Fuzzy sets have vague boundaries and are therefore well suited for representing linguistic terms such as 'very' or 'somewhat' or natural phenomena such as temperatures. Fuzzy set theory is used to describe fuzzy sets, and was developed as an alternative to ordinary (crisp) set theory. Fuzzy logic is used to derive the set membership function fora fuzzy set, which is used for fuzzy logic decision making.
The problem of constructing meaningful and suitable membership functions involves a lot of additional research. A number of empirical ways to establish membership functions for fuzzy sets are known.

2.3.2 Fuzzy multiple attribute decision making

Multiple attribute decision making deals with the problem of choosing an alternative from a set of alternatives, which are characterized in terms of various attributes. Usually multiple attribute decision making involves the observance of a single goal. Two distinct aspects need to be considered. The first is the selection of an alternative on the basis of a set of scores determined from the levels of attributes of each alternative. The second is the classification of alternatives using a role model defined on the basis of similar-case outcomes. Multiple attribute decision making usually involves the use of a framework that applies subjective criteria. Goals require information about the preferences with respect to each attribute measure, as well as trade-off preferences among selected attributes. The assessment of these preferences is either provided directly by the decision maker or determined on the basis of past choices. Further, the decision maker might express or define a ranking (weighting) for the criteria to reflect their importance. There are many forms for expressing the relative importance of criteria, but the most common are: utility preference functions; the analytical hierarchy processes; and fuzzy version of the classical linear weighted average. Notability for any fuzzy decision criteria could be fuzzy or crisp. The aim of multiple attribute decision making is to derive the best alternative as the one that shows the highest degree of satisfaction for all pre-elected attributes and predefined goals.

In order to obtain the best alternative, a ranking process is required. If the rating for alternative Ak is crisp, there is no problem and the best alternative is the one with the highest support. When the rating is itself a fuzzy set, a more sophisticated ranking procedure is required.

Describing multiple attribute decision making problems, only a single objective is considered, namely the selection of the best alternative from a set of alternatives. The decision method assumes the max-min principle approach. Formally, let A = $\{A1, A2,...,$ $An\}$ be the set of alternatives, C = $\{C1, C2,..., Cm\}$ be the set of criteria, which can be given as fuzzy sets in the space of alternatives. Hence, the fuzzy set decision is the intersection of all criteria:

$$\mu D\,(A) = \mathrm{Min}\{\mu C1(Ai),\ \mu C2(Ai),...,\mu Cm\,(Ai)\}. \qquad (2.1)$$

For all $(Ai) \in A$, and the optimal decision is yielded by,

$\mu D\,(A^*)$ = Max $(\mu D,(Ai))$,

Where $A*$ is the optimal decision.

The main difference in this approach is that the importance of criteria is represented as exponential scalars. This is based on the idea of linguistic hedges. The rationale behind using weights (or importance levels) as exponents is that the higher the importance of criteria, the larger should be the exponent, giving the minimum rule. Conversely, the less important a criterion, is the smaller its weight. This seems intuitive. Formally:

$$\mu_D(A_i) = \mathrm{Min}\{(\mu_{C_1}(A_i))^{a1}, (\mu_{C_2}(A_i))^{a2}, \ldots, (\mu_{C_m}(A_i))^{am}\} \tag{2.2}$$

Consider the problem of selecting an alternative from a set of alternatives $\{A1, A2, A3\}$ for which the set of criteria $C1$, $C2$, and $C3$ is defined. The judgement scale used is as follows: 1 means equally important, 3 means weakly more important, 5 means strongly more important, 7 means demonstrably more important, and 9 means absolutely more important. The values between 2, 4, 6, and 8 indicate some compromised judgment (Saaty, 1990). Yager (1978) suggests the use of Saaty's method for pair-wise comparison of the criteria (attributes). A pair-wise comparison of criteria (attributes) could improve and facilitate the assessment of criteria importance. Saaty developed a procedure for obtaining a ratio scale for the elements compared. To obtain the importance, the decision maker is asked to judge the criteria in pair-wise comparisons and the values assigned are $Wij = 1/Wij$. Having obtained the judgements, an m x m matrix B is constructed so that $bii = 1$, and $bij = wij$ and $bji = 1/bij$.

Yager(1978) suggests that, with respect to a decision problem, the use of the resulting eigenvector expresses a decision maker's empirical estimate of the level of importance of alternatives for a given criterion (Basetin and Kesimal,1999).

If $C2$ and $C3$ are three and two times as important as $C1$, respectively, and $C2$ is three times as important as $C3$, the pair-wise comparison reciprocal matrix will be expressed as:

$$
\begin{array}{cc}
 & \begin{array}{ccc} C_1 & C_2 & C_3 \end{array} \\
\begin{array}{c} c_1 \\ c_2 \\ c_3 \end{array} & \begin{bmatrix} 1 & \frac{1}{3} & \frac{1}{2} \\ 3 & 1 & 3 \\ 2 & \frac{1}{3} & 1 \end{bmatrix}
\end{array}
\tag{2.3}
$$

Hence, the eigenvalues of the reciprocal matrix are λ = [03.053 0] and, therefore, λmax = 3.054. The relative weights of the criteria are finally achieved in the eigenvector of the matrix, i.e. eigenvector = {0.157, 0.594, 0.249}, with λmax. The eigenvector reflects the weights associated with each attribute, feature and goal of a decision problem. Thus the exponential weightings are α1 = 0.157, α2 = 0.594, α3 =0.249; the final decision expressed in a membership decision function, can be determined as follows:

$\mu D(\mathrm{A}) = \mathrm{Min}$

$\{\mu C10.157,\ \mu C20.594,\ \mu C30.249\}.$ (2.4)

If the relative levels of importance of criteria for alternative $A1$ are 0.75, 0.4 and 0.7, respectively, those for alternative $A2$ are 0.8, 0.95 and 0.73, respectively, and those for alternative $A3$ are 0.54, 0.32 and 0.4 respectively, then the applicable membership decision functions of alternatives

$A1$, $A2$ and $A3$, respectively, can be defined as follows:

$$\mu D(\mathrm{A1})=\min\{(0.75)0.157,(0.4)0.594,(0.7)0.249\}=0.58 \tag{2.5}$$

$$\mu D(\mathrm{A2})=\min\{(0.8)0.157,(0.95)0.594,(0.73)0.249\}=0.92 \tag{2.6}$$

$$\mu D(\mathrm{A3})=\min\{(0.54)0.157,(0.32)0.594,(0.4)0.249\}=0.51 \tag{2.7}$$

Consequently, the optimal solution, corresponding to the maximum membership level of 0.92, is given as:

$$\mu_D(A'') = \left(\frac{0.92}{A_2} \right) \tag{2.8}$$

2.3.3 Fuzzy dominance method

Hiple (1982) has proposed a flexible comparison technique known as the dominance matrix concept. Usually a dominance matrix is constructed for a typical element, *dij* , of the dominance matrix D, where *dij* is the number of the factor for which the value of alternative *i* dominates i.e. is greater than alternative *j*. A typical element *dij* of the dominance matrix is explicitly defined as follows:

$$dij = \sum_{k-l}^{n} \{[\mu_i(X_k) - \mu j(X_k) \geq 0\} \tag{2.9}$$

Because the entries in the rating matrix are calculated using information which is often fuzzy or imprecise, a threshold level value can be chosen to represent the minimum amount by which one alternative must be greater than the other, for a given factor. In order for an alternative to be considered dominant for the factor the dimensionality of D is equal to the number of alternatives under consideration and dashes are entered for the diagonal elements since these elements have no meaning in the discrimination process (Hiple, 1982 and 1983). In the dominance matrix, the sum of the *k*th row indicates the number of times the *k*th alternative dominates all other alternatives. The sum of *k*th column represents the number of times the *k*th alternative is dominated by the others. Hence, the more preferable alternatives possess relatively high row sums and low column totals. These two attributes can be combined into a single measure by subtracting the column sum from the row sum for each alternative. The preferable alternative will have the highest difference

2.4 Mine planning and design

Underground mine planning and design has its goal in integrated mine systems design, whereby a mineral is extracted and prepared at a desired market specification and at a minimum unit cost within acceptable social, legal, and regulatory constraints. A large number of individual engineering disciplines contribute to the mine planning and design process, such that it is a multidisciplinary activity. Given the complexity of the mining system, planning assures the correct selection and coordinated operation of all subsystems, while design applies to the traditionally engineering design of subsystems.

A mining operation should be more correctly viewed as a system, because of the diversity of technological processes, facilities, and personal skills required, the large capital invested, the mutual relations that exist between subsystems, etc. The planning process necessitates a systems engineering approach suitable for complex design problems.

Mine planning can be compared with organizational strategic planning; although the scale may differ, corporate strategy management (Thompson and Strickland, 1989; Steiner, 1979) is very similar to plan development and implementation. A planning process that considers both external (e.g., inflation, technological, industry, and market changes, environmental and political effects, competitor practices) as well as internal forces at work during the mine life will have an improved chance of success and a competitive advantage in the market place. Planning in this context could be considered an entrepreneurial activity while design is more administrative in context. Although many of the basic principles of mine planning and design have not changed over many years, advancements made in individual systems such as rock mechanics or equipment design have either made or have the potential for making a significant impact on mining.

Design guidelines developed as empirical rules based on experience or derived from regulatory requirements are no longer adequate for mine design. A proactive posture regarding mine planning and design is mandated for successful mining in today's complex and competitive minerals market. Two of the major driving forces in this trend are the application of technical computing and the introduction of a systems engineering approach. The widespread acceptance of technical computing in the context of computerized mine design has expanded the scope of many older design techniques, yet has permitted the introduction of new techniques.

This trend has been driven by the increasing sophistication of readily available software and hardware, augmented by the decreasing cost of hardware (personal computers, workstations (spreadsheets through comprehensive mine planning systems utilizing interactive graphics), and the increasing applications of databases.

The planning process will, in general, move through four steps, irrespective of the design phase: (1) baseline assessment, (2) reserve determination, (3) premine planning, and (4) subsystem design.

2.4.1 Baseline assessment

In any mining project, a baseline assessment of all available data precedes any planning efforts. The baseline assessment is a comprehensive initial review of all available information on the potential reserve or mine from geographic, geologic, environmental, technical, and economic standpoints. For example, the geographic location of a resource will have a great influence on economics and may dictate the mining method due to equipment and power availability, labour availability and skills level, supplies transport, etc. As the size of the resource is defined, it may be large enough to overcome the negative aspects of its location, but other problems may preclude its development. The major factors to consider in a baseline assessment can be classified as geologic, environmental, geographic, and economic as explained in Section 2.1.

2.4.2 Reserve determination

From a mine planning and design perspective, the characteristics of a reserve are as crucial as the reserve magnitude or grade; the depth, inclination, geometry (delineating shape and extent), type and properties of host and deposit rocks, quality, etc. may play a key design role. For example, concerns such as wall rock characteristics or structural features of a deposit strongly influence the choice of mining method and the mine operating costs (Barnes, 1980; Anon., 1987).

2.4.2.1 Criteria

A mineral deposit/resource can be classified as an ore deposit or reserve only when it can be successfully mined at a profit. Although the planning and design process attempts to identify the method of exploitation that allows this to occur, all or part of a deposit may not meet the reserve criteria. The impact of the geographic and geologic environments have been previously discussed. Criteria are also a function of market (demand) and mining technology. Categorization changes occur as the level of inventory and knowledge of a particular deposit increase. Reclassification will also occur corresponding to geographic and economic factors, technological advances in either mining, preparation, transportation, or utilization (impacting market or demand), and depletion. Classification methodology is often mineral- or country-specific. For example, several books have been written on the assessment and classification of coal reserves (Todd, 1982; Wood et al., 1983). From the planning and design perspective, reserve determination can best be expressed as a qualitative assessment of overall potential, expanded to include engineering and economic aspects. For details of mineral resource estimation.

2.4.2.2 Data Presentation

Reserve/resource data typically are presented in a matrix form, with one side of the matrix indicating the degree of certainty and accuracy of the existence of the ore or mineral, based on the distance from a point of measurement, and the other side indicating the comparative economic recovery viability. Although this format is adequate for data summary, it is of necessity compromise presentation. Distance categories chosen to represent the degree of certainty may be derived from published sources or government agencies (e.g., Wood et al., 1983); however, categorizations specifically derived for a given deposit using techniques such as geostatistics or depositional modelling may yield more valuable information. The widespread use of computers often leads to a comprehensive reserve summary of tabulation that is too voluminous for practical utilization. Effective graphics can often maximize communications.

2.4.2.3 Mathematical Methods

Reserve estimation involves taking point data (samples from drilling or prospecting) and extrapolating those data to blocks or grids for calculation purposes. Intermediate steps, such as mapping (determination of the sample point coordinates), compositing of sample data into a common unit or horizon, or determining categorization boundaries (reserve classification, leasehold boundaries, etc.), are often required. The point data are then extended using a wide range of mathematical techniques, of which the most common are (1) polygonal, (2) inverse distance weighing, and (3) kriging. Block size in terms of length and width may be defined on the basis of geologic structure, deposit variability, and data spacing (determining grid size in computer modelling) or may be dictated by mining methodology or quality forecasting requirements. Mapping of the block data and comparison with in-mine sampling is critical to confirm the extension methodology.

2.5 Pre-mine planning

A mine plan relates a detailed view of a complex process in action at only one specific point in time. The plan constantly evolves as physical changes occur to the existing infrastructure and as new conditions are encountered. Engineering science and mining technology are constantly advancing, while the mine, once constructed, is essentially locked in its initial physical framework. Although fixed assets such as equipment change with time, the basic design remains the same. This fact causes an ever-widening gap between existing and new mines. When possible, a plan is chosen that will minimize the construction or development time required to get an operation into production. The shorter schedule will allow a more rapid cash flow and consequently better return on the capital investment. Although it may be preferable to drive to the property or deposit extremes and mine back to the plant for ground control or logistics reasons, this may not be economical. Plans are made to access the ore/resource soon after development is started, and to sequence development and production until the deposit is completely mined.

The mining method selection may be a trade-off between capital investment, time from development to full production, and production costs. As an example, in metal/ non-metal mining, a mining method such as block caving may require extensive capital and development time, yet yield lower production costs. Similarly, in coal mining, a selection of long wall mining may require extensive, high-cost development before lower operating costs, as compared with continuous mining, are realized. Conversely, in either coal or metal/non-metal mining, a room and pillar mining method might be chosen to yield immediate production, although at a higher mining cost than other methods.

Most plans start with a feasibility study to provide an engineering assessment of the potential viability and minability of the project. This study generally is an overview of the project, making reasonable assumptions about the physical and other key operating factors, to see if the project justifies further effort. Variations in the critical assumptions should also be evaluated to determine the sensitivity of the results to the corresponding input data. The only way to obtain accurate cost forecasts for the complete project is to develop a life-of-mine plan for the reserve block, including costs associated with post mining reclamation and final land use. Post mining reclamation costs can be of such a magnitude that the project feasibly may be questionable. Potential damage to the land, water system, or community can preclude mining and can relegate a promising reserve to a resource classification. Despite the fact that an almost limitless list of factors goes into a mine plan, the following is recommended as a checklist of concerns. Lineberry and Adler (1989) offer another organization of the complex array of concerns for underground coal mine design, as an a typical mine design model.

2.6 Review of underground mining methods

2.6.1 Sublevel stoping

Sublevel stoping, also known as blast hole or long hole stoping, is an open stoping, high-production, bulk mining method applicable to large, steeply dipping, regular ore bodies having competent ore and rock that require little or no support. The method is often selected as an alternative to sublevel caving when dilution levels must be kept to a minimum.

Sublevel stoping is very development intensive, although the cost of development is compensated by the fact that much of it is done in ore. It is currently limited to steeply inclined ore bodies where both ore and country rock are competent and broken ore flows under the influence of gravity. Production drilling is accomplished using long hole equipment. New developments in sublevel stoping involve the utilization of large-diameter DTH (down-the-hole) drills that, because of their directional accuracy, are revolutionizing the mining method (Pandey, 1984). The large, highly mechanized drilling equipment used restricts the minimum-width ore body that can be mined, while high development costs associated with sublevel stoping require that a high production rate be maintained. Efficient use of large-scale blasting makes sublevel stoping one of the lowest-cost underground mining methods available. Sublevel stoping also finds widespread application for pillar recovery in cut and fill and other types of mining methods (Irvine, 1982; Bharti, Lebl, and Cornett 1983). A variation of sublevel involves blasting the ore in horizontal instead of vertical slices with long stope holes drilled from raise climbers (Robertson, Vehkala, and Kerr, 1989).

2.6.1.1 Ore body type

The typical ore body required for successful sublevel stoping must be regular, large, strong to fairly strong, and competent, and the wall rock must be self-supporting. Rock strengths vary widely and can be compensated for in the design, but typically range from a minimum of 8000 psi 55MPa with no upper limit. The dip of the ore body footwall must be such that it exceeds the angle of repose of the broken ore, which permits gravity flow of blasted ore through to draw points and chutes. Ore bodies are typically a minimum of 6 m wide to afford efficient application of long hole blasting. Ore bodies less than 6 m wide realize a higher cost per ton of ore because of lowered production per blast, and as widths go less than 1.5 m, hand drilling and overhand and underhand methods must be resorted to. In wider weaker ore bodies, transverse stoping may be practiced to afford additional stope support (Matikainen, 1981). No upper size limit exists for ore bodies mined using this method. However, in large ore bodies, support pillars must often be left in place during the entire mining cycle on that level. These pillars are usually recovered after adjacent completed stopes have been backfilled (Boshkov and Wright,1973; Hamrin, 1982).

Long hole drilling and large-volume production blasting require ore bodies that are even and fairly well defined. Stope boundaries must be regular, because irregular ore bodies and those containing large waste inclusions cannot easily be avoided. Waste from irregular ore bodies and inclusions dilutes the ultimate grade of mined ore and thus increases the cost per ton of ore produced. A smooth ore-to-footwall contact allows for easier flow of blasted ore to draw points and chutes. Rock must be structurally competent and self-supporting as large openings may be left unfilled for extended periods of time. Additionally, repeated shocks from large blasts require an ore of high compressive strength and minimal structural weaknesses such as joints, faults, and bedding planes. Failures resulting from collapse of incompetent ground cause excessive dilution, loss of sublevels, and large chunks blocking draw points, and necessitate reconditioning of stopes. Small, localized ground failures result in ground movement and displacement, and cracking of blast holes. This in turn makes loading of blast holes difficult and in some cases necessitates extensive redrilling of a block

(Morrison and Russell, 1973; Mitchell, 1981; Lawrence, 1982). Sublevel stoping is used to depths of 900 m (Misra, 1983).

2.6.1.2 Stope Layouts

Mine development normally starts from a shaft sunk in the footwall to avoid any subsequent caving effects from the stopes. The ore body is divided vertically by driving crosscuts and haulage levels every 45 to 120 m. Access raises driven in the ore body are used to further subdivide the ore body into blocks for stoping. A collection system is constructed, during which time the stope block is all or partially undercut. Sublevels are driven through the proposed stope block every 10 to 55 m as shown in Figure 2.1 and more than one sublevel may be used on each level, depending on the width of the ore body.

Some typical dimensions for a wide variety of ore bodies are shown in Table 2.2. Stoping is carried out by blasting vertical slices into an expansion slot as in Figure 2.2, the height and width of the proposed stope, usually by enlarging a slotting raise with long hole drilling and blasting. Shrinkage stoping has also been used to form the starting slot that may be developed at the end or middle of the stope. Slotting has been shown to be a very expensive part of the stoping operation, accounting for 20 to 30% of total stoping costs (Matikainen, 1981).

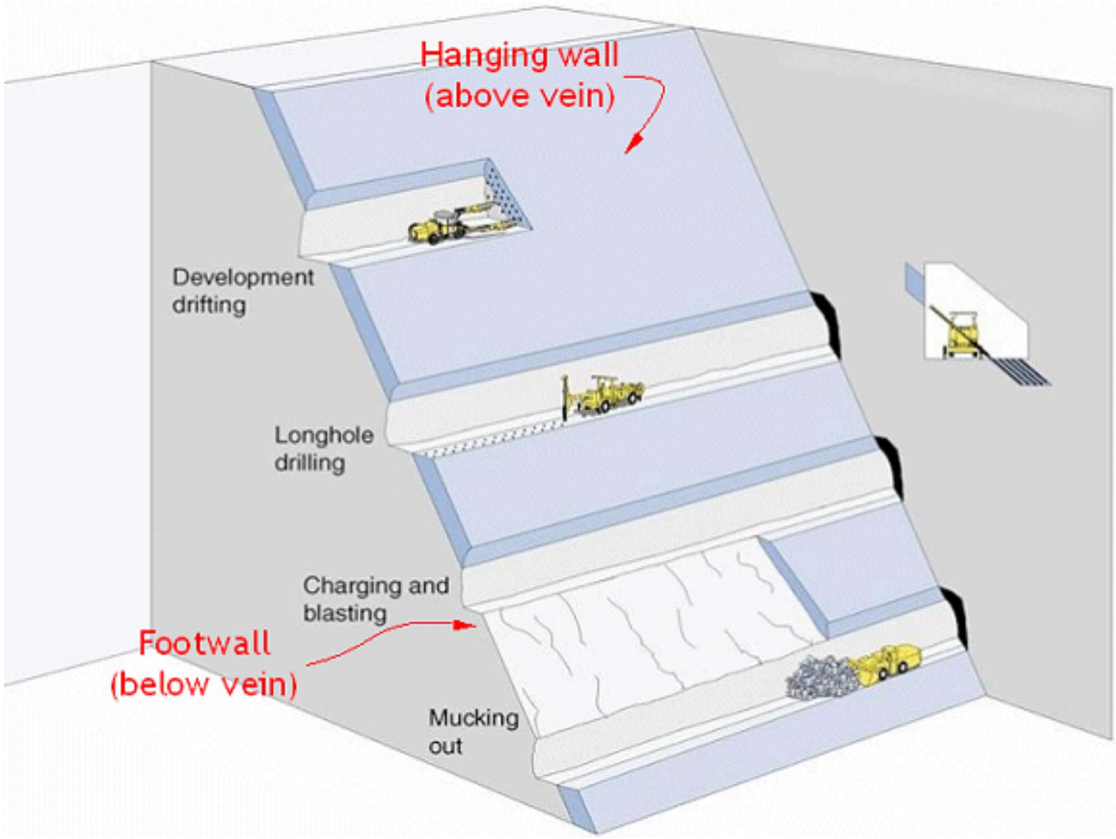


Figure 2.1. Sublevel stoping at Outokumpu Oy Vihanti mine showing the relationship between development and production drilling (Matikainen, 1981).

Table 2.2. Sublevel Stoping Basic Dimensions

Mine	Ore Body	Stope Dimensions, ft				Pillars, ft	Haulage, Interval, ft
		Width	Length	Height	Sublevel		
Kidd Creek (Belford, 1981)	Massive base metal sulfide	79	98	299	98	70–98	397
Torman (Matikainen, 1981)	Massive limestone	148–164	328–492	328	49–164	148–164	—
Rio Tinto (Botin and Singh, 1981)	Massive sulfide	66	66–164	131–236	131–236	41	174–276
Mt. Isa (Goddard, 1981)	Bedded sulfide	82–164	98	410–820	66	82	574–984
Burra Burra (McNaughton, 1929)	Massive sulfide	39	328	164	43	43	197
Luanshya (Mabson and Russell, 1981)	Bedded sulfide	39	39	115	36	16–32	164–230

Conversion factor: 1 ft = 0.3048 m.

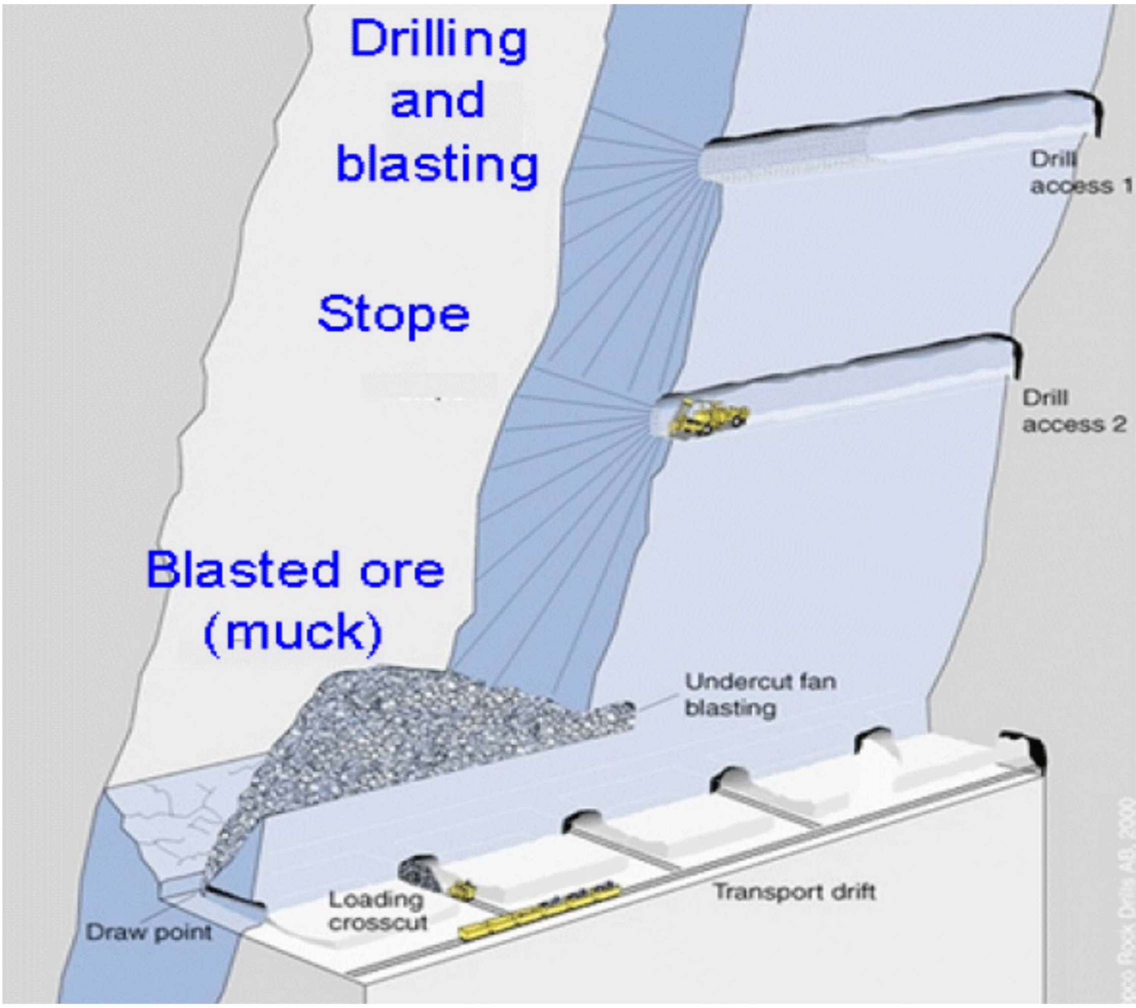


Figure 2.2 Production blasting in sublevel stope showing ring drilling patterns and loading system (Hamrin, 1982)

Stopes are typically contained by a crown pillar, which protects the level above, rib pillars, and a sill pillar through which the ore collection system is cut as shown in Figure 2.3. Substituting filled stopes for rib pillars has been achieved successfully under some conditions as shown in Figure 2.4 (Belford, 1981). Pillars are typically removed through large-scale blasting when stoping is exhausted, and up to 100% of the ore can be recovered under ideal conditions. Stopes can be filled where caving is unacceptable or must be controlled to reduce surface effects or minimize underground stress effects. Filling of stopes also facilitates pillar removal under some conditions.

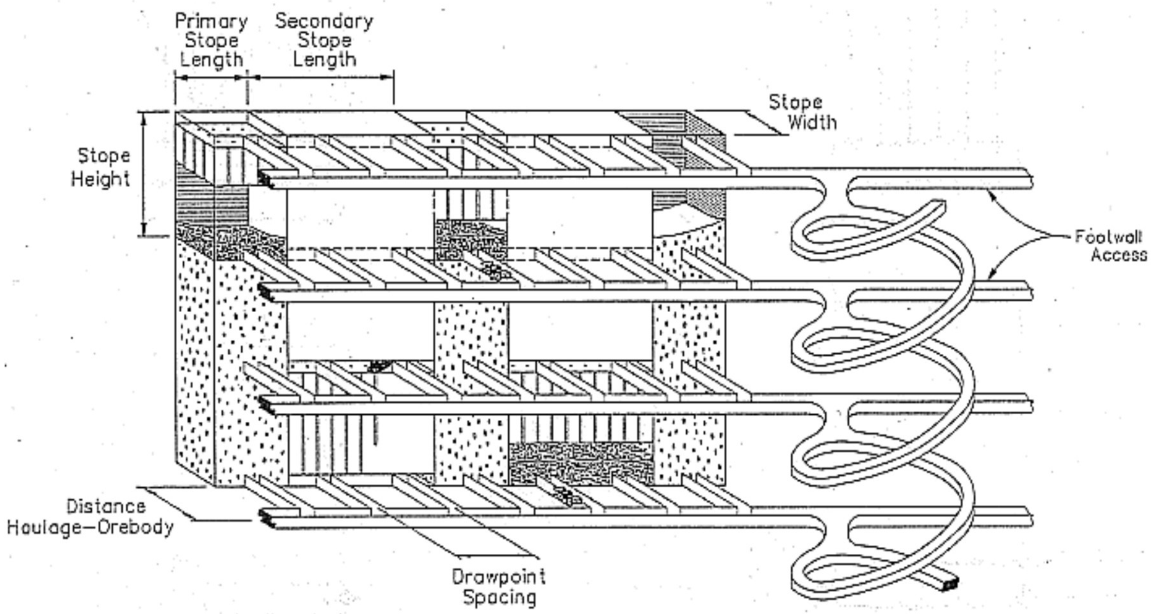


Figure 2.3 Pillar system around a sublevel around a sublevel overhand stope

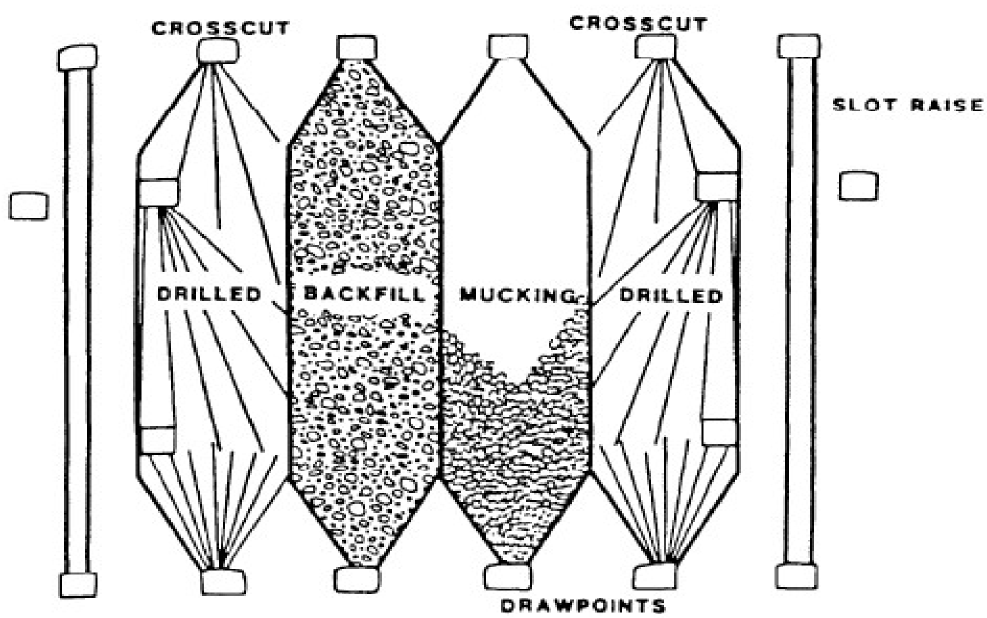


Figure 2.4 Cemented backfill used as an alternative to pillars (Belford, 1981).

2.6.1.3 Stope Extraction Systems

The base of the stope may be slotted horizontally prior to production blasting with the base of the slot being a collection trough or series of 45° cones. These are cut through the sill pillar to collect the ore. Alternative to this, no separate slot may be used, and stope blasting will expose the collection system as the stope face retreats. Excavation of the troughs or cones is typically carried out by drilling from the proposed draw point and mining upwards using small 55 to 94 mm long holes as in Figure 2.5.

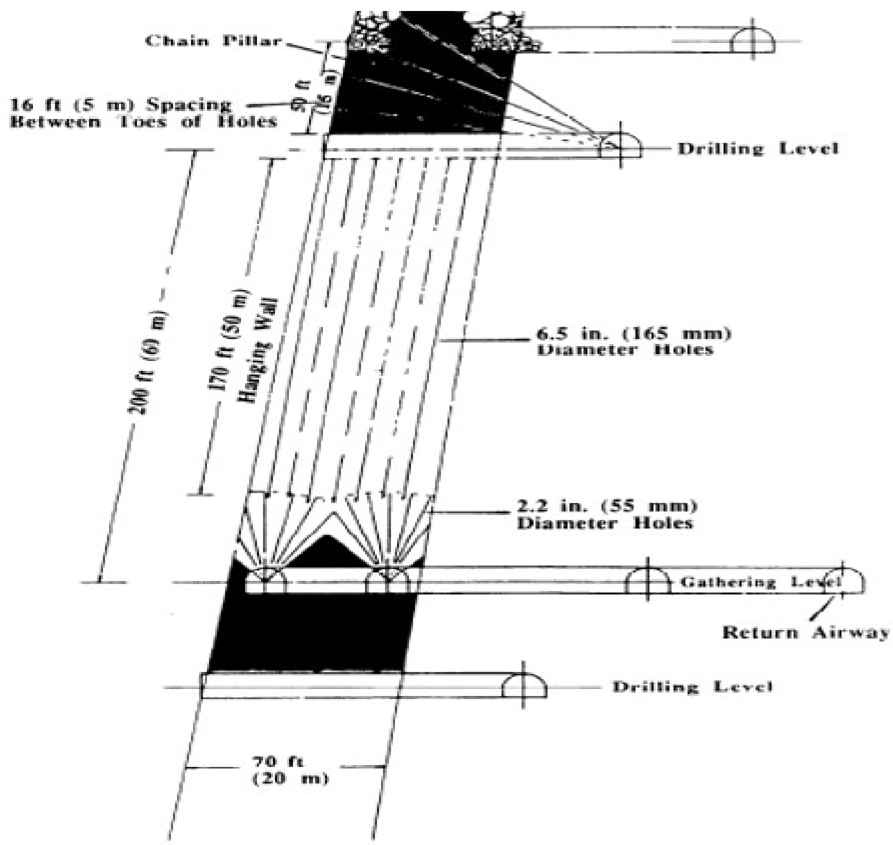


Figure 2.5 Small-diameter holes for cutting collection cones (Mabson and Russell, 1981)

2.6.1.4 Stoping action

Stoping typically commences by blasting the lowermost faces first, then moves progressively upwards from the overlying sublevels as shown in Figure 2.14 above. The overall stope face is carried overhand to facilitate ore flow in the stope. The faces may be carried vertically where the ground is very weak.

2.6.1.5 Production drilling

Consideration must be given to a number of factors before production drilling can commence. Drillability of the ore, required fragmentation size, hole size, hole length, orientation and spacing of drill holes, and desired drilling accuracy must be established. The above factors contribute to the selection of drilling equipment and design of production drilling patterns. Typically, production drilling is accomplished using mechanized long hole drilling.

Recent developments in drilling technology include high-efficiency column-and-arm long hole drills, and DTH drill rigs have revolutionized sublevel stoping. These units incorporate electric over hydraulic drive and feed systems, high pressure pneumatic DTH hammers, or rotary-percussion drilling systems. This type equipment is capable of drilling holes up to 229 mm in diameter to depths of 120 m. Alignment errors of holes exceeding 120 m is generally less than 2%. Drilling accuracy can have important economic implications for any proposed mining operation (Almgren and Klippmark, 1981).Many drill rigs have the capability of drilling a 360° ring of holes. This capability allows for up-hole and down-hole drilling, which in turn results in lower development times and costs. The best drilling efficiency, however, results when a pattern of vertical parallel holes can be drilled from a drill sill developed in the stope as shown in Figure 2.6.

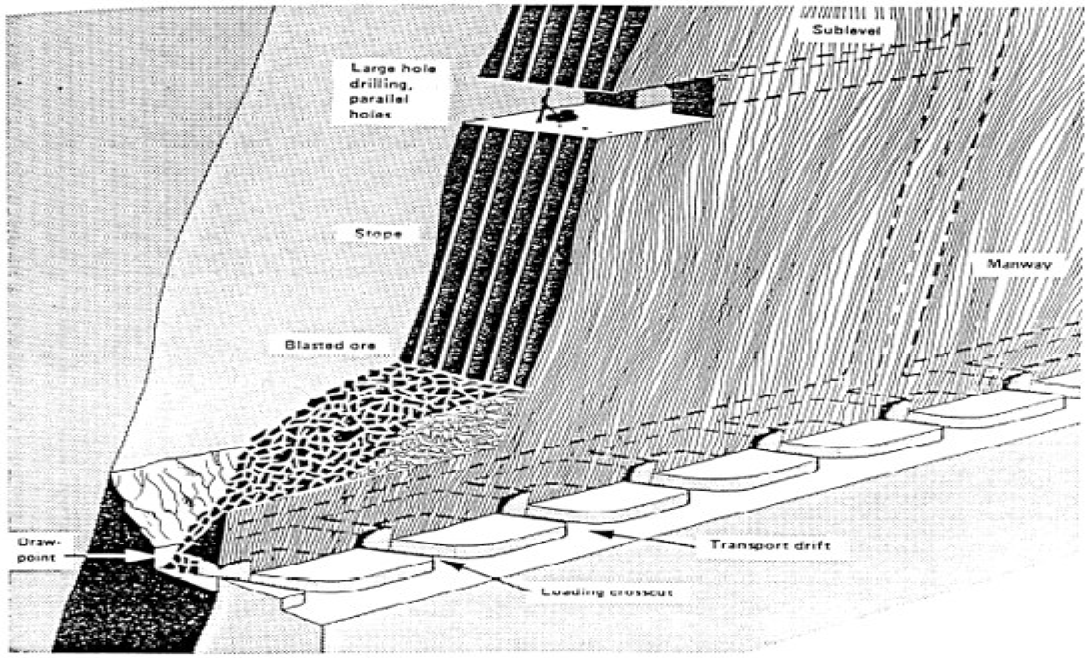


Figure 2.6 Parallel hole system of stope drilling from a drill sill (Hamrin, 1982)

Holes patterns, whether parallel or in rings, are drilled from sublevels that are 18 to 20 m apart. Generally, steeply dipping deposits are drilled with parallel holes, whereas ring drilling is usually adaptable to massive ore bodies. With the introduction of up-hole and down-hole rigs, the drilling sill no longer has to be developed to the full width of the ore contact. A fan-shaped array of holes drilled from a narrow drill sill can effectively delineate the required width of the ore body. This is probably the oldest method, predated only by the now obsolete overhand and underhand short hole bench and trail (Hamrin, 1982). The second parallel hole system evolved from the old short hole bench and trail method and utilizes 170-mm, large-diameter, accurate DTH drills. This method is much more efficient than ring or fan drilling and results in consistently acceptable fragmentation. The distance between sublevels can be extended to 45 to 55 m (Hamrin, 1982).

2.6.1.6 Backfilling

The large openings created by sublevel stoping typically require that some sort of backfilling program be practiced. Backfill includes uncemented rock and sandfill, cemented rock fill, cement hydraulic tailings fill, and a high density tailings or alluvial fill. Backfilling allows for future recovery of support pillars.

Recovery of pillars permits up to 90% recovery of ore. Backfilling also minimizes the occurrence of subsidence and allows for redistribution of stresses created by the mining cycle. This in turn minimizes the frequency of rock bursting. Backfilling has also been used successfully to eliminate rib pillars between the stopes. In this case the backfill contains sufficient cement to form a self-supporting unit. Cementing the backfill is not always economical, in which cases pillar recovery may not be practical, and the fill is used to control surface movement (Matikainen, 1981).

2.6.1.7 Ground control

Ground control is typically minimal with this system because the rock must be inherently strong, although different types of bolts are used for local support, which may or may not include mesh. Regional stresses have been shown to affect stope design and dimensions and are an important consideration. The large open stopes can serve to concentrate high horizontal stresses and cause severe deterioration in development openings in close proximity to the stopes (Goel and Page, 1981). Cable bolting has proved very successful for full stope wall support, using cables up to 45 m long. Cables are normally installed from the blasting sublevels during blast hole development, and resin anchored bolts have proven most successful although cement grout is also used (Goddard, 1981).

2.6.1.8 Economics

Sublevel stoping is an inherently high-production, low-cost method and is frequently selected as a primary underground method when surface mining of a deposit is no longer economical (Hedberg, 1981). The key to minimizing costs is mechanization, using as large-capacity machines as the ore body will permit in terms of production capacity and opening size. The utilization of large-diameter DTH machines can reduce total development compared to smaller-diameter long hole drills that are limited to hole lengths less than 30 m by accuracy restrictions. According to study, the method is development-intensive with development accounting for one-third of total mining costs (Lawrence, 1982). Labour cost typically average 40 to 50% of total stoping costs (Matikainen, 1981). Cost analyses are now possible for sublevel systems, involving such factors as stope sizes, drilling and blasting parameters, number of drill points, and loading machines (Chatterjee and Just, 1981).

2.6.2 Vertical crater retreat mining

VCR (vertical crater retreat) mining is a horizontal, flat-back variation of sublevel stoping using spherical crater charges to break the ore. It is the only patented stoping method (Livingston, 1973). Blasting is carried out at the base of vertical holes, making horizontal cuts and advancing upward (Mitchell, 1981). Shrinkage can be utilized in the stopes for wall support. Because the method requires less development than sublevel stoping, it has the potential for lower costs and is finding increasing application not only for pillar recovery but also for primary stoping (Lang, Roach, and Osoko, 1982).

2.6.2.1 General methodology

The first step in VCR mining is to define a block (stope, panel, section, pillar) of ore that is amenable to this type of mining. The initial criteria are dip and plunge, as the ore must be able to flow down to draw points under the influence of gravity. The second factor assessed is ore block shape and consistency. The ore block must have a shape that can be basically defined from two sills spaced a significant vertical distance apart. Once an ore block is defined, the blasting characteristics of the rock can be assessed. The assessment can be done theoretically or tests on similar ore blocks can be done for empirically developing the data as shown in figure 2.7 and 2.8 below;

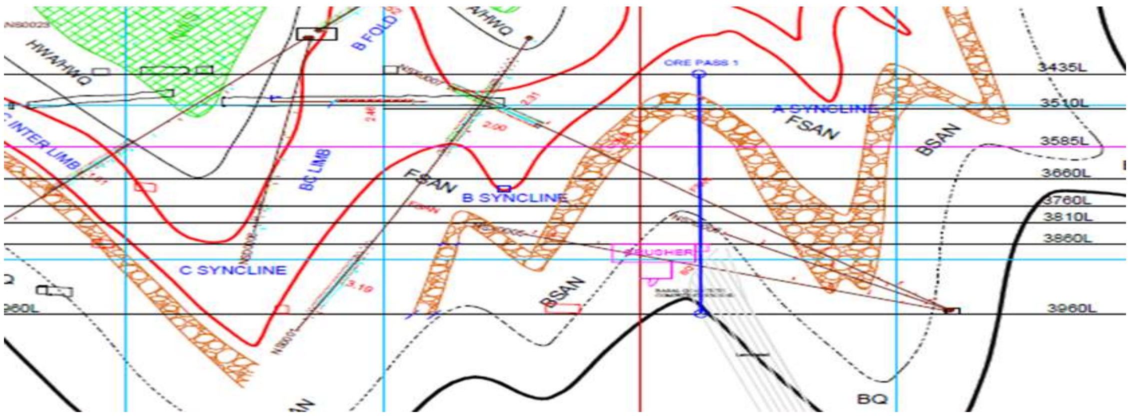


Figure 2.7 Vertical cross sections with diamond drill holes and ore intercepts

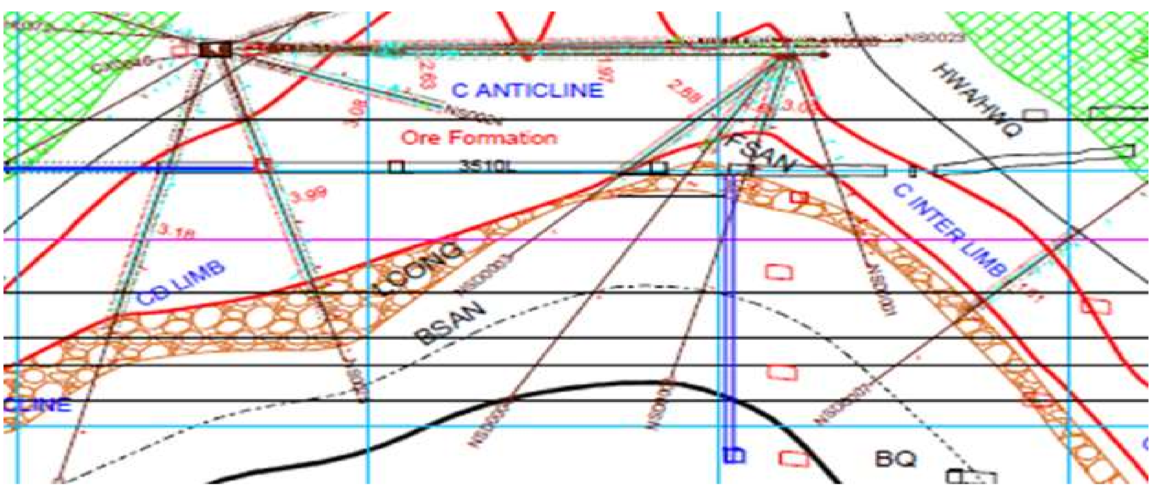


Figure 2.8 Expected VCR stope outlines from plotted diamond drill intercepts

Selection of hole size and drilling system can be made once blasting characteristics are defined. The extraction system is often predefined by existing extraction techniques in use at the mine; if not, then ground control, block size, production needs, and availability of equipment will determine the size and layout of the mucking scrams and haulage system (VCR is a bulk method, and high rates of production are common).

Once the design is completed, the top sill and bottom sill are cut. The vertical separation between the sills is dependent upon ore consistency, drilling accuracy, accessibility, and hanging wall competency. After the sills are cut, any secondary ground control measures (such as cable bolts) necessary are installed, and the stope is drilled. The holes are drilled from the top sill down to the bottom sill, usually with a down-hole hammer to minimize deviation. With the drilling complete, the stope should be ready to blast. Blasting is done to take off vertical slices of ore progressing from the bottom sill to the top sill. During blasting, only enough broken ore is mucked from the stope to open up the volume necessary for successive blasts. This keeps the open stope full of broken rock to support the walls until blasting is completed (basically, a form of shrinkage stoping without miners having to crawl in on top of the pile of broken rock each cut). Many variations of the blasting sequence have been used successfully at different operations. Once blasting is complete, the ore block is a large inventory of broken rock that can be extracted as fast as the extraction system will allow. After the ore is extracted, the bottom sill accesses can be closed off, and the stope can be backfilled the top sill to maintain rock stability in the area.

2.6.2.2 Development sequence

Top sill development consists of driving an access to the ore body and then cutting out the ore on that horizon, together with any waste mining necessary to form the drilling sill. The sill is now ready for ground control or down-hole drilling. The drilling sill must be cut at least 3.4 m high and 4.5 m wide for drill mast clearance and drill manoeuvring.

The extraction system is developed with the bottom sill, which is accessed via the designed mucking system. The ore zone is accessed as soon as possible so that it can be cut out and compared to original estimates and design modification made as necessary. If the ore extends below the sill, but only by a small amount, the development crew may mine a 3.6-m cut out of the bottom of the sill. Once done, the crew will drive the extraction system and place the waste into the area just mined.

As the crew mines the drifts, any major slips or ground control problems are noted and plans are changed when necessary. The development crew usually mines the bottom sill cut 3 m high. Often another cut is made above the sill cut. This helps minimize blast damage to scrams and also reduces the length of the down-holes to be drilled, thereby improving accuracy.

2.6.2.3 Ground control

There are several stages of ground control in the development and extraction of a VCR stope. First, pillars may be designed into the sill cut to keep widths minimized and the back stable. Second, is the immediate ground control of each round blasted by the development crew. Here 1.5 and 2.4m long bolts are installed to secure the immediate back and walls. Cable bolts are the most common form of secondary reinforcement. Cable bolts are used to stabilize the back and hanging wall of the top sill during the blasting and mucking cycle. This helps to maximize the safety of blasters and to minimize dilution. Bolt length varies from 9 to 18 m and is determined from sill size, ground conditions, and additional mining plans for ore above the top sill. Bolts are twin 16-mm wire rope cables installed parallel in a 57-mm diameter hole and grouted with cement.

The standard pattern of cables is 10 by 10, with 3 m between rows and 3 m between bolt points. Sequencing of mining by panelling a large stope is done to minimize the amount of open hanging wall at any given time. This has been effective in minimizing hanging wall slough. It does necessitate delays in stope production for backfilling of panels.

2.6.2.4 Drilling

The drill crew is provided with a layout of the sill with the grid overlaid on it as shown in figure 2.9 below. Surveyors mark the location of each row numbered and reference strike lines lettered. The drill crew will then drill plug holes at each location indicated by survey. (Plugs are wooden hole plugs, driven into drilled holes, and spads inserted.) Strings are pulled across rows to line up the drill mast and along strike lines to locate holes.

Ore bodies and sill configurations vary widely, and patterns are adapted to each stope. General guidelines are to utilize a 2.4 by 2.4m pattern, to minimize breaking into the hanging wall, and to minimize drill footage. Situations often force departures from standard such as employing 3 by 3m patterns, the need to splay holes into the hanging wall to recover ore, or utilizing a high density of drill footage near the top sill due to the changing geometry of the ore body between the sills. Holes that are drilled from the top sill and drilled through to the bottom sill cut are called “breakout” holes. Three breakout holes per row to start the blasting is the accepted minimum. Accuracy is a necessary part of the drilling. The grid developed in the top sill provides the basis for accurate drilling. Once a drill is lined up on the collar location of a hole, the inclination indicator mounted on the mast is used.

The inclination indicator is set for the specified angle of the hole, a green light then indicates when the mast is correctly aligned, and a red light indicates improper alignment. When the drilling is nearing completion, the location of holes in the bottom sill is checked. If there appears to be a problem, surveyors may be asked to check actual vs. planned hole breakthrough locations. If major deviation has occurred, then additional holes may be needed to cover the stope correctly. Once set up, drilling proceeds fairly rapidly. Many crews can average 60 m/shift. Tons blasted per foot of hole drilled (total drilling) runs about 11.5 t/m. Stope drilling is done with down-hole hammer drills and 165-mm diameter bits.

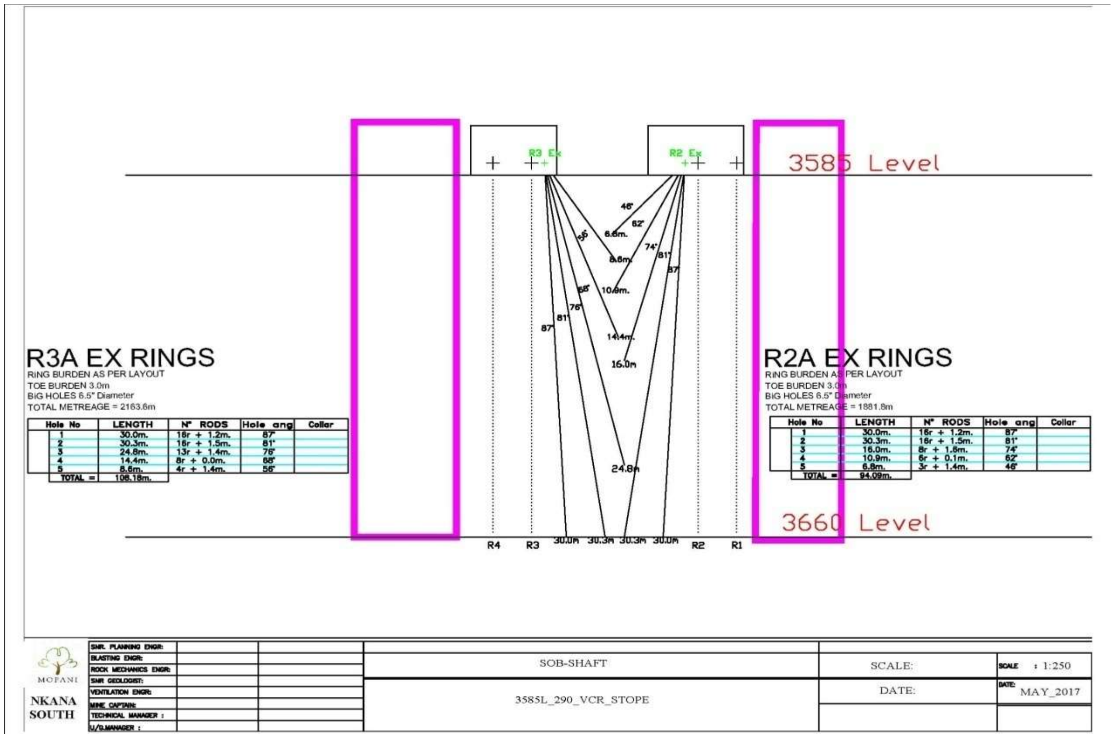


Figure 2.9 Cross section of standard row of VCR pattern layout (Mopani SOB Planning, 2017)

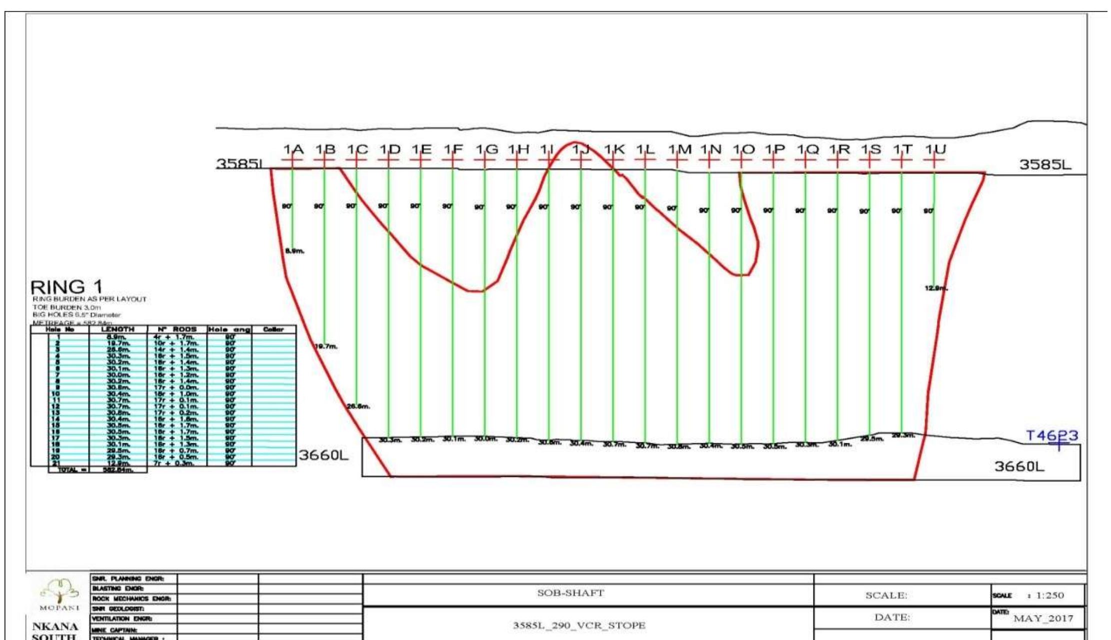


Figure 2.10 Cross section of standard row of VCR pattern layout ((Mopani SOB Planning, 2017)

2.6.2.5 Blasting

Prior to blasting, each hole is measured and recorded. Holes are measured by attaching a 38 by 38 by 914mm piece of mahogany to the end of plastic coated 6.4mm cable. The cable is approximately 45m in length. The cable is stamped every 1.5 m from the end with the corresponding distance on the stamp. Each hole is measured by lowering this device down the hole until it reaches the floor or rock pile on the bottom sill. Next the cable is pulled up until the mahogany stick hits the bottom of the blast hole. In the case of no breakout holes, a steel weight may be substituted for the mahogany if the hole is wet. Once all holes have been measured, it is necessary to estimate the volume of void available to determine if there is sufficient room for expansion of the rock from the blast. An area of the stope is selected which is equivalent to a volume of at least one third the volume of the stope. This is called the slot and is the first section of the stope to be blasted. When cratering the slot, 2.1-m average distance from rock pile to back of stope is the accepted minimum open space to allow successful blasting.

The next step is determining where to block the blast hole. Blocking is the process of securing two wedges at the desired location near the bottom of the blast- hole. The explosives are placed on top of the blocking. Angles of the blast hole determine where the hole is to be blocked. Generally accepted values for blocking heights are shown in Figure 2.11 below;

Hole Angle	Blocking Height (above hole bottom)
80–90°	4 ft (1.2 m)
57–79°	5 (1.5)
50–56°	6 (1.8)
Less than 50°	7 (2.1)

Figure 2.11 Accepted values for blocking heights for different hole angles (Andrews, 1988)

Loading the hole can begin now that blocking height and hole distance are available. A 12-grain prima cord is measured according to desired blocking location, and a wooden wedge is tied to the prima cord. The prima cord is marked to correspond to the blocking location. Once the mark on the prima cord reaches the hole collar, a second wedge is dropped down the hole and forms a friction lock with the first wedge at the desired blocking location. Next small rocks and drill cuttings are dropped down the hole to seal the space around the wedges. The end of the prima cord is tied to a rock on the top sill until needed.

Holes in the slot are loaded with two 13.6 kg cartridges of water gel. The cartridges are cut lengthwise and one dropped down the hole. Next a sliding primer with the appropriate delay is assembled. The prima cord is fed through the cord well of the booster, which is slid down the hole on the top of the explosives. The final 13.6 kg of explosives is placed in the hole on top of the booster.

Top stemming consists of a sand column of 2.1 m. The sand stemming is topped off with 4.5 m of water. The break holes are shot first, and the remaining holes are capped in a concentric pattern. The slot will continue to be shot in this fashion until within 7.5 to 9 m of the top sill. This crown pillar will require double-decking two explosive charges in each hole. Delay time between the double decks is 25 to 50 ms.

2.6.2.6 VCR theory

C. W. Livingston (1973) derived equations modelling rock breakage from crater blasting. The basic theory is that a spherical charge placed at an optimal distance from the back of a stope will break a maximum volume of rock in the shape of an inverted crater. The theory correlates placement of a given charge weight in varying blast hole depths (height above back) to the volume of rock broken by its detonation. The actual charge size and placement is dependent upon the size of the hole and the density of the explosive as well as the characteristics of the rock. Charge geometry can alter blasting results—as the length of the charge increases relative to the diameter (defined by hole diameter), the blasting results adhere less rigorously to the theoretical results. Testing determined that a ratio of explosive column length L to hole diameter D of 6 or less will work similar to a spherical charge. Thus either higher-density explosives or larger diameter holes will reduce the ratio for the same charge weight. Practice has shown that in most rock types, high-density/high energy slurry explosives provide superior results. A good explanation of the mathematical equations and a description of a practical test is provided by Orr (1983).

2.6.2.7 Adaptivity

Originally used to take out large pillars where only limited access was practical, VCR is now being applied to many situations. Some very large blocks have been mined in successive stages with VCR where adjacent ore blocks are filled with cemented backfill once mined out to provide an adequate wall for the next stope to mine against. Some very small blocks that would not support extensive development have been mined with VCR. Effectively using VCR means adapting it to the specific situation. Many variations on the basic blasting sequence are also being used, the most common being multiple-face crater blasting where the early shots in a round are used to open up a vertical free face for successive shots to utilize. Another common blasting variation is to utilize a mass blast that may break up to 50% of the ore block's tonnage in a single round. Mucking methods have varied from slusher extraction to mechanical continuous muckers, although extraction by load-haul-dump (LHD) is most common.

2.6.3 Sublevel caving

Sublevel caving is a mass mining method based upon the utilization of gravity flow of the blasted ore and the caved waste rock. As with any other mining method, sublevel caving has advantages and disadvantages that must be carefully considered and evaluated for mine design and planning. Major advantages of sublevel caving are as follows.

1. Safety. Because all the mining activities are executed in or from relatively small openings, sublevel caving is one of the safest mining methods. Drifts are the primary

working places. They are distributed in a uniform pattern on all levels. Normally, the maximum dimensions of the sublevel drifts are about 5 to 6 m wide and 3.7 to 4.0 m high. Transportation drifts can have the same cross section, or the height may be increased to about 4.5 m when trucks are loaded in the transport drifts. The stability and safety of such drifts in competent rock can be easily controlled by smooth blasting alone or by a combination of both smooth blasting and shotcreting. In less competent rock masses, stability can be achieved by a combination of smooth blasting, shotcreting, and rock bolting.

2. Mechanization. As shown in Figure 2.12, when using the sublevel caving method, major mining activities can be broken down into four groups of unit operations: (1) drifting and reinforcing,(2) fan drilling, (3) production blasting (i.e., fragmentation),and (4) ore drawing, loading, and transportation. Because of the repetitive nature of the mining system, almost all mining activities can be standardized. This means that a high degree of mechanization is possible. In modern sublevel caving, cross sections of the drifts and tunnels are sufficiently large to allow the introduction of large trackless mining equipment. The advantages of a trackless system can then be broadly utilized not only for direct mining but also for all services.

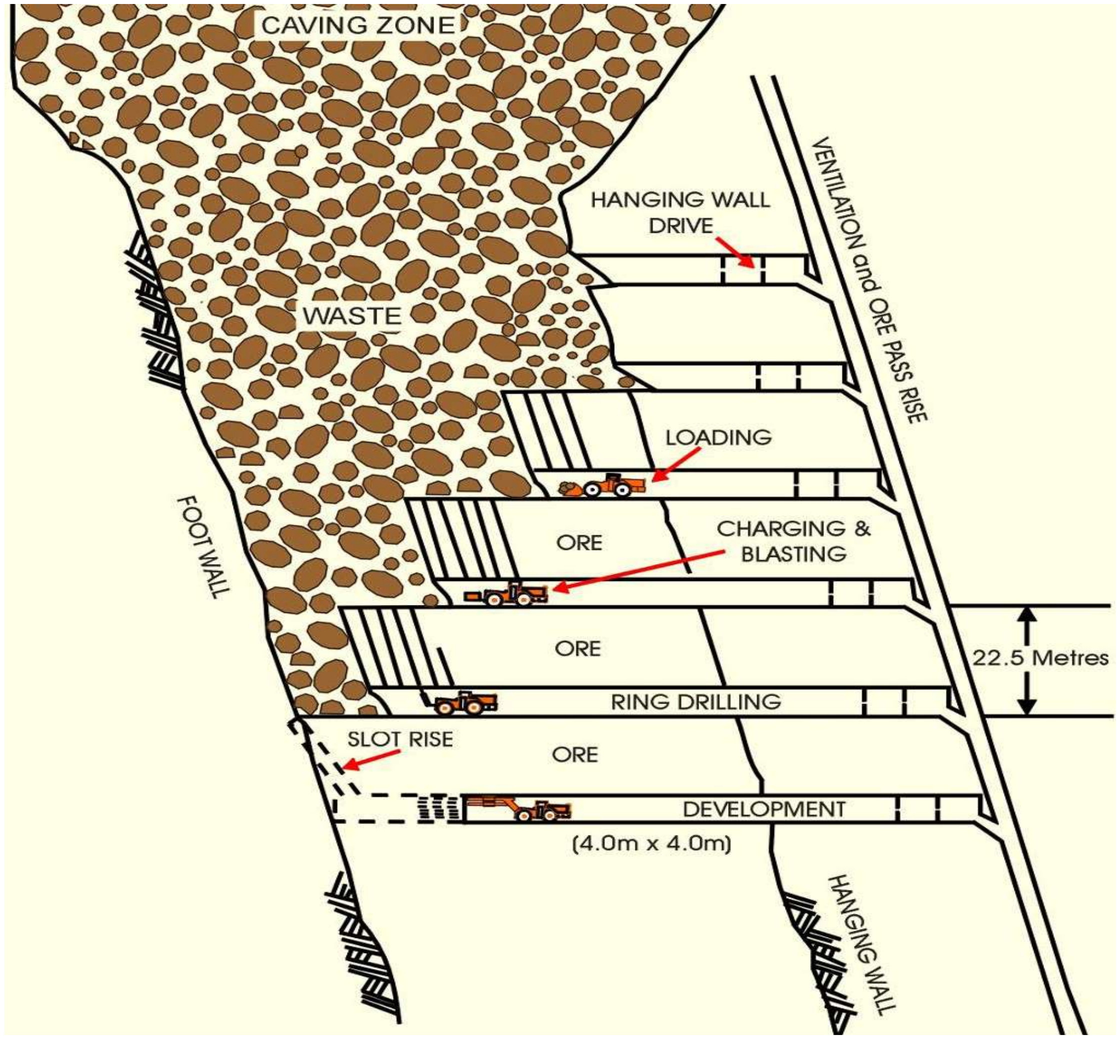


Figure 2.12 Typical schematic of sublevel caving visualized on a simplified transverse section and longitudinal section (after LKAB information brochure, 1989).

3. Flexibility. Standardization and specialization of mining activities and equipment on separate levels (lower level or levels in development, upper level or levels in production mining) together with a trackless haulage system creates a high degree of flexibility. This allows a rapid start-up of mining and flexibility when making production rate changes.

4. Work Organization. This method lends itself to good work concentration, organization, and working conditions. Normally, on the lower levels, various phases of development are underway. Upper levels are in various stages of extraction.

Therefore, the work can be easily organized into a system which excludes interference between mining activities.

Safety of mining (in small-dimension openings), good flexibility, work organization, and high mechanization using large modern mining equipment provide very good working conditions. The method also facilitates a high work concentration and rationalization of separate specialized mining activities. Therefore, mining by sublevel caving can be effective and relatively inexpensive. Major disadvantages of sublevel caving are

1. There is a relatively high dilution of the ore by caved waste, especially when higher recovery is requested.
2. All ore must be drilled and blasted in order to obtain a coarse material suitable for extraction by gravity flow.
3. Various types of ore loss can occur. When the extraction limit (that point yielding the maximum acceptable amount of dilution) is reached, the remaining highly diluted ore represents an ore loss. Some ore is lost in passive zones located on the level of extraction between the active zones of the gravity flow. A small part of the ore forming these passive zones can be recovered together with ore extraction on the lower sublevel, but some undiluted ore located in passive zones above the plane of the footwall is lost. In general, these losses are larger as the inclination of the ore body and the footwall is reduced.
4. A relatively large amount of development is required. This includes transport drifts, usually located in the footwall waste rock on each sublevel, and sublevel drifts, which connect the active mining areas in the ore body to the transport drifts. The sublevel drifts are partially in ore and partially in the waste rock. The waste rock length increases as the inclination of the ore body and footwall decreases. The system also needs ore passes, used to transport the ore or waste from the separate sublevels downward the main haulage level, and these openings are located in waste.
5. Mining generates progressive caving in overlying rock and results in subsidence and damage to the surface.

6. To maximize ore recovery, minimize dilution, and achieve a high efficiency of mining by sublevel caving, good data regarding the gravity flow parameters for the blasted ore and the caved waste are of utmost importance.

The exact type and amount of data required depend upon the purpose and needs of the mine design. For the first feasibility study, it may be sufficient to utilize the data from other sublevel caving operations with similar conditions and circumstances. For any higher level of mine design and planning, it is clear that more exact data, including analytical and experimental analyses up to full-scale in situ testing, are necessary.

In sublevel caving, all ore must be drilled and blasted in order to utilize the gravity flow of fragmented ore, which after blasting forms so-called “coarse material”. The objective is to obtain by minimum drilling and blasting a coarse material with a fragmentation that will permit gravity flow and undisturbed ore extraction. The coarse material is characterized by different size distribution and shape. The size distribution and shape of fragments depends on the density of the drill holes and on “blasting/tectonics” of the ore. Coarse materials can be very heterogeneous. With certain simplification, we could distinguish (with respect to their behaviour) four basic types Figure 2.13. Type I shows coarse material with large “spherical” pieces of more or less the same size and shape. Type II represents a material almost of the same size but different shape. Type III indicates a coarse material composed of large fragments, chippings, and sand. Type IV represents a coarse material that is characterized as a mixture of large blocks, medium-sized fragments, chippings, sand and/or rock powder, earth or clay components, etc. The coarse material, Type III, and especially Type IV, could change considerably the mechanical properties and behaviour depending on the percentage of fine particles (sand, clay, etc.) and on the presence of water. If water is present, then especially in coarse material, Type IV, the fine particles could create plastic and sticking components. This could change very unfavourably its properties and behaviour. Naturally, the coarse materials could have very different properties. The simplest characterization is shown in Figure 2.13, indicating the angles for gravity transportation of coarse material. Up to about 40° the coarse material requires mechanical transportation. With larger angles of inclination, gravity transportation could be used in open chutes or in ore passes. In Figure 2.13, GF represents the range of gravity transportation applicability, where the range A is for open chutes for transportation of Types I and II only, while the Types III and IV (in inclination range B) must be transported in steep passes. Type III requires inclination BI, and Type IV, especially when wet, needs very steep passes in the inclination range BII (Figure. 2.13). Almost all blasted ores (and waste) are similar to Types III and IV. The composition, properties, behaviour, etc., of these materials could be very different because they could consist of very large boulders down to the finest particles.

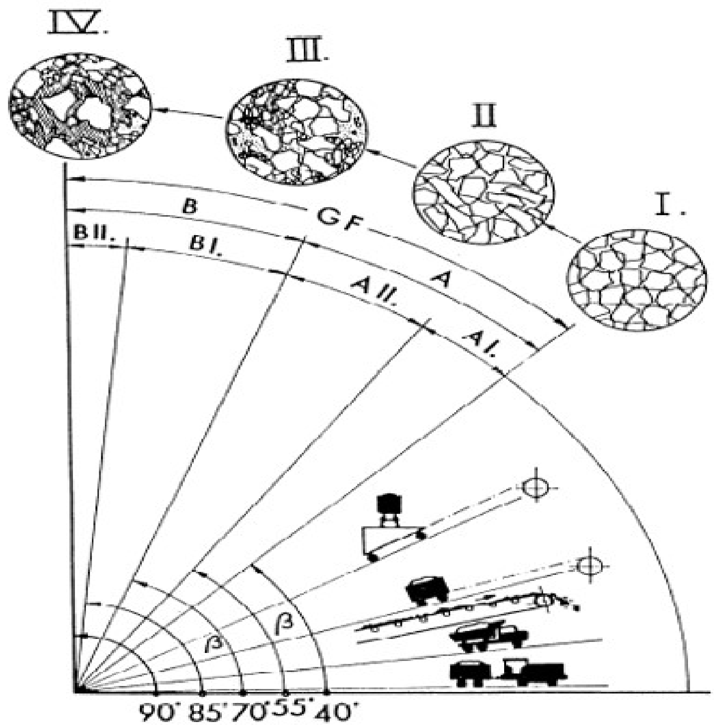


Figure 2.13 Four basic types I through IV of coarse material and their mobility as a function of inclination of chute or ore passes.

2.7 Summary

This literature review provided an insight of parameters required for mine design and the various mining methods currently being applied at SOB and across mines. Also the major challenges encountered in the current mining methods applied at SOB have been fully identified. Some mining methods reviewed included room and pillar, sublevel caving, sublevel stoping, shrinkage stoping and vertical crater retreat (VCR). Chapter 3, reviews the methodology that was used to collect sufficient data to achieve the objectives set out in section 1.5 of chapter 1.

Chapter 3

3.0 Research Methodology

3.1 Introduction

This Chapter discusses the various methodologies that were used to collect sufficient data required for mine planning and design. This data included; Geometry of the ore deposit, Geological and hydrology conditions , Geotechnical properties of rocks and ore , Economic considerations (tonnage of reserves and mining cost), Technological factors (mine recovery, dilution) and environmental concerns such as ground control, subsidence and atmospheric control (Samimi, Shahriar, Karimi2004; Hartman 2002) to achieve the objectives.

Research methodology used included;

- Site investigations;
- Desktop study;
- Diamond drilling;
- Logging of borehole core;
- Software application for mine design and analysis.

3.2 Site investigations

This methodology involved underground visits to various sections of the mine at SOB and Central shafts to have an initial understanding of the geology, rock types, ventilation, hydrology, available equipment, tramming distances, size of excavations, orientations and size of major geological discontinuities and the current mining methods and associated challenges.

3.3 Desktop study

During desktop study, previous research works that were done on underground mine planning and design and related literature available on mining method selection from mining engineering books and mining were reviewed, Furthermore, technical reports which were done by Murray and Roberts consultants and SRK consulting firms on the design of the Synclinorium mining project were also reviewed and provided useful data for this research.

Consultations on the input parameters for mine design with mine planning, geologists, geotechnical engineers, senior mining officials and cost accountants at SOB and central Shaft were also undertaken to understand the current cost of mining.

3.4 Diamond drilling

The primary exploration method used in the mine is diamond drilling. It was used to perform the following tasks; (Mopani, 2013d)

(i.) Provide advance Geological information on ore distribution and structural nature;

(ii.) Using the same geological data to generate new resource base which was used to replenish future Nkana mine site reserve base;

(iii.) Investigate extensions to the existing ore bodies;

(iv.) Explore further ore bodies within the immediate vicinity of the current mine workings;

(v.) Dewatering of aquifers to ensure a flood free mine environment prior to production;

(vi.) Special drilling for adverse ground conditions, pilot cover holes when need arises.

The diamond drilling was being carried out by a contractor and supervised by the mines geologists. The geologists provided the contractor with data regarding drill sites, core size, machine type and other drilling details. When drilling the contractor is expected to follow these standards; (Mopani,2013d); Supply core of a minimum of

95 % core recovery. Lower performance entails re-drilling at contractor' cost. Well stacked, clean and labelled core is delivered to the client's core shed and signed off. Boreholes in excess of 100m are camera surveyed.

Underground diamond drilling was being done from excavations called cubbies which are developed in haulages, water drives, footwall drives or the crosscuts depending on where drilling is needed. In order to generate the ore body, an initial drilling pattern is set with a spacing of 150 – 200m between the drill holes. The block generated by the drilling has a vertical height of 50 - 80m. The 28 metres drilling is centered at 60m in the later stages of mining in order to improve the confidence of the mineral resources.

The last step in classifying the mineral resource as a reserve is to drill closely spaced horizontal holes with 30m spacing. All drill cores are collected and logged for rock type, structure, ore minerals and alteration characteristics. The cores are split with a diamond saw and one part is taken to the laboratory for analysis (Mopani, 2013d).

3.4.1 Channel Sampling

A secondary way of defining the ore resource is chip sampling. The chips were collected inside the ore body and are generally very accurate. Channel sampling is performed at a spacing interval of 30m in order to correspond to the stope dimensions. Most often only the first crosscut is sampled and chisels are used to collect the samples from the exposed ore in the crosscut. The intervals in which samples are collected is determined by observing the lithology and density of the ore mineralization and the minimum interval is 0.25m (Mopani, 2013d).

3.4.2 Sample Handling

The validity and integrity of collected samples are very important as a diluted sample will give false readings. Therefore all samples are wrapped in a plastic bag and marked with a tag to make sure the samples are not mixed up before they are delivered to the laboratory (Mopani, 2013d).

3.4.3 Preparation, Analysis and Security

A laboratory on site handles copper and cobalt assays and a quality control program is used to ensure the accuracy of the laboratory work. Both ore and milled samples are tested (Mopani, 2013d).

3.4.4 Processing and Validation

Personnel from the mine are performing the mineral resource estimation for both the underground and open pit operations. Data from both diamond drilling cores and channel sampling is used during the estimation process and the estimation is based on manual polygonal, triangulation and contouring methods. The value for copper and cobalt were determined using weighted grades/tonnage and estimated with planimetered volumes and then converted to a tonnage by using a gravity of 2.56. The fringes of the ore body were determined using a combination of triangulation, grade and thickness contouring (Mopani, 2013d).

3.4.5 Grade Control

The mine planning department is basing their monthly schedules on data submitted by the geology department. The grades are controlled onsite by personnel from geology. Monthly estimates of stope tonnage and grades are based on data from working faces and are used to create the schedules. To further control the grades production sampling is carried out and samples are analyzed within 48 hours. Sampling is carried in the following areas;

(i.) Underground Stope sampling

(ii.) Conveyor belt sampling

(iii.)Tray car sampling

(iv.) Underground Stope Sampling is the most extensive, but also the most accurate way of grade control sampling (Mopani, 2013d).

3.4.6 Grade and Tonnage Reconciliation

The ore reserves and resources are constantly review and updated by the mine staff. Production and scheduling is evaluated taking into account tonnages, grades and new findings from exploration. Tonnages and grades are reconciled using the statistical method of tracking tonnages and grades back to the source shafts and consequently to stopes and development ends (Mopani, 2013d).

3.5 Logging of borehole core

Geotechnical parameters were gathered over a single core run (approximately 3m in length), however, were the core run resulted in the possible change in lithology or geotechnical property or character (such as strength, fracture frequency, and/or recovery), the features were measured and documented on the core logging sheet, either by breaking it out as a separate geotechnical logging interval (new domain), or writing a description in the comments column. Zones that consisted of broken or lost core were recorded in logging sheet as a broken core or lost core. Furthermore, the depth (meter marks) at any discontinuity, broken core, lost core, any zones tested or sampled and other 'information of interest' were recorded on the core box using a permanent marker to facilitate interpretation. Geotechnical parameters recorded during logging to classify the rock mass of the area were; Depth, Description of rock type, Core recovery, RQD, Strength of intact rock (UCS), Weathering or alterations, Fractures or discontinuity type, and Fracture frequency, Dip angle of structure with respect to core axis and dip direction in case of oriented core, and Discontinuity condition.

3.6 Computer Aided Designing (CAD)

A number of computer aided software packages have been developed for mine planning and design. However, in this research Surpac Germcom software was employed for design and analysis.

3.6.1 Mine Planning

The Software that was used for design are the package from Gemcom Softwares™ and AutoCAD. Gemcoms package includes SURPAC for design purposes and Mine Sched for midterm planning, long term planning and ore reserves generation. The use of Microsoft Office involves Excel for the short term planning, shift scheduling etc.

3.6.2 Production planning

The mine planning department is planning the production and developing the layout for the mine, based on data from the geology and survey departments. The work procedure for planning the mine is as follows; it starts with a geological section that's been divided into mining blocks with regard to geological reserves. The blocks are then divided into stopes. The next step is to include the main and sublevel development that is needed to access and extract the stopes. Geological and survey data determines the grades and tonnages for the stopes. The data is also the foundation for the design of VCR development and stope layouts. Planning the stopes also includes developing the drill layouts and including data such as; drilling positions, direction, depth and hole size.

3.7 Geotechnical modelling

Geotechnical modelling was done using Phase 2D and Flac 3D.

3.7.1 PHASE 2-D

Phase 2D was used to determine the best mining sequence and the safest size of stope blocks for the mining method.

In this method, the strength factor was calculated by dividing the rock strength based on the failure criterion used, by induced stress at every point in the mesh. PHASE 2-Dallows for graded meshing, and radial meshing, all of which were automatically generated.

The external boundary in this program was modelled which springs allows the far field rock mass to have a stiffness as well as using fixed/pinned displacement conditions.

The mesh setup consisted of 150 excavation node with a gradation factor 0.1. The mesh type was set to graded with the element type as a three-nodded triangle. The insitu field stress condition was set to constant.

One material type was used to represent the rock mass in each model. The rock mass was considered isotropic with an elastic modulus of 30,000Mpa, and a Poisson ratio 0.25.The strength parameters of the rock mass used. The rock mass was considered as an isotropic plastic material. The Mohr-coulomb failure criterion was used for the rock mass.

3.7.2 FLAC 3D

Flac3D is a three dimensional modelling program. It is a general analysis and design tool for geotechnical and mining engineers. In this research it was used to help sequence stope extraction and backfilling The results obtained using phase2D were then compared with the results of Flac 3D for accuracy. The results obtained are shown in appendix 2.

3.8 Summary

This Chapter discussed the methodology used to collect data required for mine design. The data collected is sufficient to complete the mining methods design and to assess the economic viability of the mining methods that will be selected. The next chapter will present the data that has been collected using the methodology presented in this Chapter.

Chapter 4

4.0 Data collection

4.1 Introduction

This Chapter presents data collected from desktop review, site investigations, diamond drilling and logging of borehole core.

4.2 Geology and Geometry of the ore deposit

There are four main lithologies within the Synclinorium. These are, SOB Shale, Hanging Wall Argillite – HWA, Near Water Sediments, and the Upper Quartzite – Upper-Q. Each unit is folded with variable thickness and has foliation or bedding that replicate the folding along the strike of the ore body as shown in Figure 1.5 of Chapter 1.

4.3 Hydrogeology

There are three major aquifers in the Nkana Syncline in the proximity of the Synclinorium Shaft. Two of these are in the hanging wall and one is in the footwall. The aquifers are classified according to their position relative to the ore body as Footwall Water, Near Water and Far Water.

The Footwall Water Aquifer occurs mainly in the Lower Conglomerate which is usually 20 metres below the ore formation. The base of Near Water Aquifer is stratigraphically 60 metres above the ore body. The aquifer thickness ranges from 30 to 45 metres. The base of the Far Water Aquifer is stratigraphically 100 to 300 metres above the ore body. The thickness of the Far Water formation ranges from 50 to 100 metres.

The footwall rocks are mainly Sandstones and Quartzite. These are generally competent and impermeable rocks. The ore body formation is made up of Argillites and Dolomites. Footwall water is easily drained as development for ore extraction is made. The lowest dewatering level for Footwall Water in the East Limb at Central Shaft is 3760 ft Level (1146m Level).

Near Water is also drained prior to ore extraction because of its proximity to the Ore body. The lowest dewatering level for Near Water in the East Limb at Central Shaft is 3760 ft Level (1146m above mean sea elevation).

The aquifers have been systematically dewatered as mining has progressed down-dip, and the phreatic surface in the Near Water and Footwall Water in the area of the Synclinorium shaft is 1146m below surface. The water from various aquifers is controlled through valves and channeled/piped to a system of pump chambers and hence to the surface. As the dewatering has progressed across strike and down-dip, sequential pump chambers have been established at lower elevations in the mine.

4.4 Geotechnical conditions

4.4.1 Rock mass data for mine design

From the lower levels of the SOB Shaft, Rock Mass Classification Data was collected and it showed that the mining rock mass rating (MRMR) of the ore varies between 63% and 77%. The MRMR results represents good ground conditions. The values of the Uniaxial Compressive Strength (UCS) and UTS are shown in the tables 4.1, 4.2 and 4.3.

Table 4.1 Laboratory UCS results and 25% and 75% percentile values of UCS (AMC curvability report,2007)

		UCS							
Rock Type	Beta Angle	Lower Value	Average (°)	Value (%)	SD (MPa)	Variance (%)	UCS (25%)	UCS (75%)	RMR
NWS	0	35	81	112	24	30	65	97	55
NWS	40 - 50	30	42	55*	13	30*	33	51	52
HWA	0	57	107	162	32	30	85	129	68
HWA	40 - 50	52	71	90	21	30*	57	85	65
SOBS	0	54	77	125*	26	34	59	96	66
SOBS	40 - 50	33	52	70*	18	34*	40	64	64

Table 4.2 Summary of UCS and UTS results by rock type and Drill hole (AMC curvability report,2007)

Drillhole	Rock Type	Density		UCS		UTS		
		Average g/cm^3	SD g/cm^3	Average Mpa	SD Mpa	Average Mpa	SD Mpa	mi
NSD21	HWA	2.72	0.07	78	27.8	10.6	2.3	7.2
	NWS	2.56	0.13	92	37.8	7.1	4.3	12.8
	SOBS	2.62	0.14	48	20.5	12.8	3.9	3.5
	UQ	2.54	0.10	94	19.5	-	-	-
NSD23	HWA	2.60	0.24	71	24.4	9.8	3.1	7.1
	NWS	2.73	0.05	52	16.8	7.3	3.6	7
	SOBS	2.63	0.10	135	58.8	12.2	4.1	11
	UQ	2.62	0.02	306	117	13	2.2	23.5
Combined	HWA	2.67	0.17	75	25.8	10	3	7.4
	NWS	2.61	0.14	81	37.5	7.2	4	11.2
	SOBS	2.62	0.11	100	63.6	12.2	4	8.1
	UQ	2.56	0.08	165	122.3	-	-	-

Table 4.3 Rock mass strength (AMC curvability report,2007)

			Hoek - Brown Strength Env.			Mohr - Coulomb Strength Env.				
Rock Type	Beta Angle	mi	mb	a	s	phi	coh (Mpa)	ten (Mpa)	E (Mpa)	Ei (Mpa)
NWS	0	11.2	1.878	0.506	0.0039	37	3.25	0.2	13900	45300
NWS	40 - 50	11.2	1.687	0.507	0.0028	31	2.39	0.084	11500	45300
HWA	0	7.4	1.974	0.502	0.0164	39	4.58	1.07	33900	57600
HWA	40 - 50	7.4	1.773	0.503	0.0117	35	3.4	0.563	30000	57600
SOBS	0	8.1	2.012	0.503	0.0131	37	3.76	0.626	32200	59300
SOBS	40 - 50	8.1	1.873	0.503	0.0105	34	2.97	0.359	29500	59300

4.5 Ground stability

4.5.1 Jointing

Instability in SLC loader cross cuts is uncommon, due to the favorable orientation of major weakness planes in respect to the long axis of the development. Significant instability is sometimes experienced along the loader drives resulting from slabbing on side walls due to high abutment stresses, when support is inadequate. Where side wall slabbing is excessive, serious roof stability is affected. Cross cut instability arises where thin highly weathered ore units are intersected along the cross cuts, and at geological contact. Where excessive mining in terms of driving and cross cutting, creating pillars, excessive ground deteriorate occurs.

4.5.2 Stress and Seismicity

The Synclinorium at both SOB and Central shafts structure is tightly folded resulting from tectonic stress loading, and the whole structure plunges generally to the north. Due to folding it is expected that the major principal stress would be oriented horizontally, and most likely to be > 1.2 times the value of the vertical stress. At depth greater than 1000m below surface the stresses would tend to be hydrostatic.

4.6 Economic considerations

4.6.1 Resources

The January 2012 resource Synclinorium has been estimated and classified according to the guidelines set out in the Joint Ore Reserve Committee ("JORC") 2004 code as Measured, Indicated and Inferred. The classification is based upon the continuity of mineralisation as demonstrated through rigorous diamond drilling and confidence bolstered by a successful exploitation of the Nkana copper-cobalt deposits ore bodies stretching over 80 years. At a copper cut-off grade of 1.00%, the Synclinorium resource is estimated at 121,786,000 tonnes at 1.99% Total Copper (TCu) and 0.08%

Total Cobalt (TCo) , made up of 90,982,000 tonnes at 2.08% TCu and 0.09% TCo of Measured Resources, 15,950,000 tonnes at 1.81% TCu ad 0.08% TCo of Indicated Resources and 14,853,000 tonnes at 1.64%TCu and 0.07% TCo of Inferred Resources. Table 4.4 is a summary of the Nkana Resource per mining level.

Upside potential in the Synclinorium, representing areas too far to be estimated by the variogram ranges, is estimated at 88.2 Mt at 1.84% TCu and 0.06%.

Table 4.4 Nkana Synclinorium Resource statement per mining level (Mopani Geology, 2012)

Level	Measured			Indicated			Inferred			Total		
	Tonnes	TCu(%)	TCo(%)	Tonnes	TCu(%)	TCo(%)	Tonnes	TCu(%)	TCo(%)	Tonnes	TCu(%)	TCo(%)
3140	2,076,938	1.86	0.10	965,263	1.75	0.11	976,525	1.09	0.08	4,018,725	1.65	0.10
3220	5,126,425	1.74	0.10	1,542,963	1.72	0.10	1,122,938	1.20	0.09	7,792,325	1.66	0.10
3295	6,528,275	1.80	0.10	989,113	1.67	0.13	1,113,663	1.38	0.12	8,631,050	1.73	0.11
3360	5,570,300	1.90	0.10	335,225	1.93	0.11	491,575	1.71	0.11	6,397,100	1.88	0.10
3435	12,809,438	1.93	0.09	1,087,163	1.75	0.09	567,100	2.17	0.08	14,463,700	1.93	0.09
3510	10,945,825	2.10	0.08	1,156,725	1.89	0.07	741,338	2.01	0.05	12,843,888	2.08	0.08
3585	13,361,300	2.25	0.09	1,988,825	1.83	0.07	1,300,488	1.95	0.04	16,650,613	2.18	0.08
3660	12,562,988	2.24	0.08	2,188,900	1.91	0.07	1,902,038	1.81	0.05	16,653,925	2.15	0.07
3735	7,704,213	2.22	0.08	1,540,975	1.97	0.08	1,639,025	1.78	0.06	10,884,213	2.12	0.08
3810	6,489,850	2.26	0.09	1,591,325	1.97	0.07	1,750,988	1.77	0.07	9,832,163	2.13	0.08
3885	3,068,700	2.15	0.09	824,813	1.77	0.07	878,475	1.57	0.06	4,771,988	1.98	0.08
3960	1,358,788	2.16	0.09	969,238	1.54	0.05	435,263	1.44	0.08	2,763,288	1.83	0.07
4035	160,988	1.75	0.07	180,200	1.60	0.06	327,938	1.35	0.09	669,125	1.51	0.08
4110	32,463	1.79	0.09	32,463	1.58	0.08	339,200	1.65	0.11	404,125	1.66	0.11
4185	1,325	2.13	0.10	9,275	1.54	0.08	121,238	1.66	0.07	131,838	1.65	0.07
4260	0						13,250	1.75	0.09	13,250	1.75	0.09
4335							1,988	1.90	0.09	1,988	1.90	0.09
D Limb	3,184,638	2.00	0.08	547,888	1.51	0.06	1,130,225	1.39	0.05	4,862,750	1.80	0.07
Total	**90,982,000**	**2.08**	**0.09**	**15,950,000**	**1.81**	**0.08**	**14,853,000**	**1.64**	**0.07**	**121,786,000**	**1.99**	**0.08**

4.6.2 Mineral reserves

The total ore reserves for the Synclinorium as at 1st January 2012 total 83.2Mt of underground sulphide ore at 1.96% TCu and 0.08% TCo and are split into Proven and Probable as shown in Table 4.5

Table 4.5 Synclinorium as at 1st January 2012 (Mopani Geology, 2012)

Area	PROVEN				PROBABLE				TOTAL			
	Tonnes	TCu(%)	TCo(%)		Tonnes	TCu(%)	TCo(%)		Tonnes	TCu(%)	TCo(%)	
SYNCLINORIUM	72,775,000	1.98	0.08		10,378,000	1.80	0.08		83,153,000	1.96	0.08	

4.7 Technological factors (mine recovery, dilution)

Extraction, Recovery and Dilution factors were applied per stope according to the different mining methods. Single Tonnage (TF) and Grade Factors (GF) were calculated for each structure as shown in Table 4.6. The factors used were agreed upon between Mopani and AMC mostly based on historical stope efficiencies of SOB and Central shafts.

Table 4.6 SOB and Central shaft historical stope efficiencies

STRUCTURE	MINING METHOD	EXTRACTION	DILUTION	RECOVERY	TF	GF
Limbs	SLC	0.95	1.15	1	1.1	0.9
Synclines	VCR	0.85	1.15	0.95	0.9	0.9
Anticlines	SLOS	0.85	1.05	0.95	0.9	1

4.8 Summary

This Chapter has provided data required for the design of mining methods for the Synclinorium mining project. Data collected included geometry of deposit, geology and hydrology conditions, geotechnical properties of rocks and ore, insitu stresses, tonnage of reserves, production rate, productivity, mining costs, extraction, dilution and recovery. In the next Chapter, the mining method is designed using the available mining design criterion and later subjected to geotechnical and economic considerations.

CHAPTER 5

5.0 Data analysis and discussion of results

5.1 Introduction

In this Chapter the data obtained from Chapter 4 was processed and analyzed for possible mining methods using the mining method selection criterion. It further discusses the problems encountered in the current mining methods and their solutions, application of the chosen mining methods and the economic viability of the Synclinorium mining project.

5.2 Mining method selection using the UBC approach

This methodology is the modified version of the Nicholas approach. It followed a very similar pattern to the Nicholas method. A value of -10 was introduced to strongly discount a method without totally eliminating with the -49 value. Moreover, the rock mechanics ratings were adjusted to reflect improvements with ground support and monitoring techniques. Instead of the RQD in the Nicholas method it uses the RMR and it does not focus on fracture strength. This methodology gives a scoring for suitability of a mining method to the predefined parameters. Adding up all the scoring will results in a final score, the mining method scoring the highest will be ranked highest and is considered most suitable although it is further subjected to economical, technical and other factors. The support table for the UBC method is shown in Appendix A to D.

5.2.1 Input parameters for determination of mining method using the UBC approach

The general and geotechnical characteristics for the ore and surrounding rocks of the Synclinorium were determined as shown in tables 5.1 to 5.4.

Table 5.1 General characteristics of the ore deposit

INPUT DATA	DESCRIPTION
General deposit shape	Irregular
Ore thickness	60 metres
Ore plunge	70 degrees
Grade distribution	uniform
depth	1170 metres

Table 5.2 Geotechnical information of the ore deposit

INPUT DATA	DESCRIPTION
Rock substance strength	175Mpa
RQD	60%
Joint spacing	0.2-0.6 mtrs
Joint conditions	Slightly rough surface, separation<1mm,soft joint wall rock, dry conditions
Principal insitu stress	17.1MPa
RMR	63%-77%
UCS/Principal stress	10.2

Table 5.3 Geotechnical information on Hanging wall

INPUT DATA	DESCRIPTION
Rock substance strength	300Mpa
RQD	71%
Joint spacing	0.6 metres
Joint conditions	Slightly rough surface, separation<1mm,soft joint wall rock, dry conditions
Principal insitu stress	17.1MPa
RMR	67%
UCS/Principal stress	17.5

Table 5.4 Geotechnical information on Footwall

INPUT DATA	DESCRIPTION
Rock substance strength	322Mpa
RQD	71%
Joint spacing	0.8 metres
Joint conditions	Slightly rough surface, separation<1mm,soft joint wall rock, dry conditions
Principal insitu stress	17.1MPa
RMR	68%
UCS/Principal stress	18.8

Using this input data in tables 5.1 to 5.4, the UBC method selection process was calculated and the results are shown in table 5.5 below;

Table 5.5 Results of the UBC mining method selection criteria

Geometry/Grade distribution											
		O/P	B/C	SST	S/C	LWM	R&P	SHS	C&F	T/S	SOS
General shape	Irregular	3	0	1	1	-49	2	2	4	0	4
Ore thickness	Thick	4	3	4	4	-49	-49	-49	1	2	0
Ore plunge	Steep	1	4	4	4	-49	-49	4	4	0	2
Grade distribution	uniform	3	3	4	3	4	4	3	2	2	0
Depth	Deep	-49	3	2	2	3	2	2	4	1	2
Rock mechanics characteristics											
		O/P	B/C	SST	S/C	LWM	R&P	SHS	C&F	T/S	SOS
Rockmass ratings											
Ore zone	Strong	3	0	4	1	2	5	3	3	1	0
Hanging wall	Strong	4	2	4	2	3	5	4	3	3	0
Footwall	strong	4	2	3	3			3	2	2	0
Rock substance strength											
		O/P	B/C	SST	S/C	LWM	R&P	SHS	C&F	T/S	SOS
Ore zone	medium	3	1	4	3	2	3	3	3	1	1
Hanging wall	medium	4	2	4	2	2	2	3	4	2	1
Footwall	Strong	4	1	3	2			3	2	1	0
GRAND TOTAL		-16	21	37	27	-131	-75	-19	32	15	2

From table 5.5, the top four mining methods are,

(i.) Sublevel open stoping (37),

(ii.) Cut and Fill (32),

(iii.) Sublevel caving (27),

(iv.) Block caving (21).

5.2.2 Mining Method online Selection Tool (MMST)

Results obtained using the online Mining Method Selection Tool as shown in Figure 5.1 also gave the following top four mining methods as;

(i.)Sublevel stoping (37)

(ii.)Cut and fill (32)

(iii.)Sublevel caving (27)

(iv.)Block caving (21)

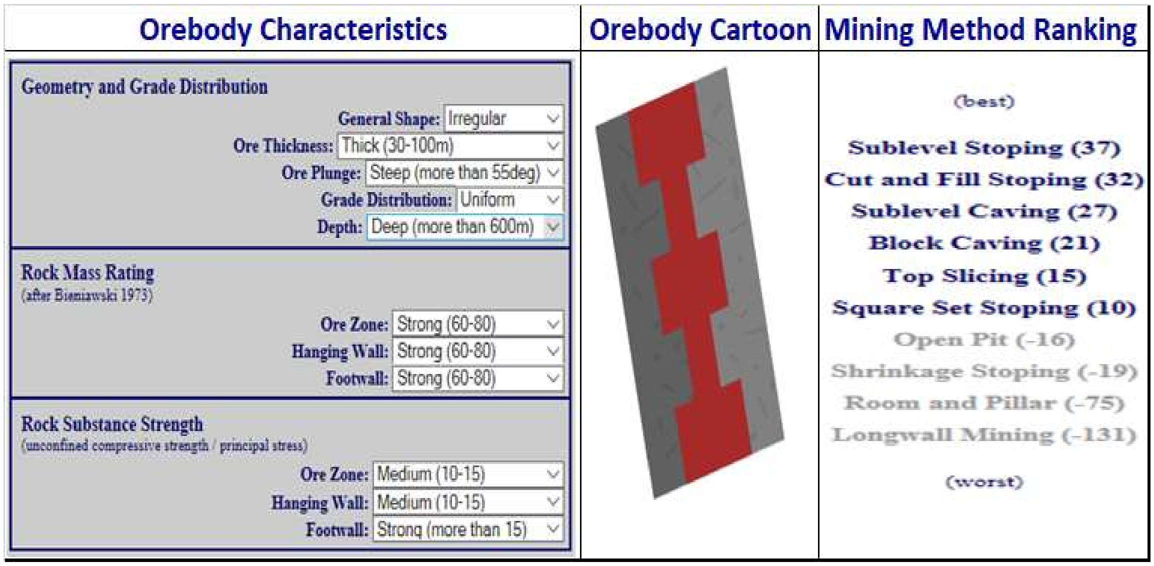


Figure 5.1 UBC windows results as depicted from online.

5.2.3 Mining Method Selection using Fuzzy dominance method

In this methodology, Membership functions of the Synchrinorium deposit were developed following the Yager (1978) approach. This is shown in table 5.6 below;

Table 5.6 Membership decision function of each criterion by the Yager method.

	CF	**OP**	**SQ**	**SLC**	**SLS**	**TS**	**BC**
C1	0.75	0.76	0.64	0.80	0.80	0.72	0.76
C2	0.79	0.79	0.79	0.66	0.66	0.74	0.98
C3	0.66	0.82	0.62	0.77	0.77	0.66	0.82
C4	0.77	0.77	0.63	0.70	0.84	0.63	0.70
C5	0.77	0.87	0.8	0.87	0.9	0.8	0.80
C6	0.94	0.94	0.89	0.90	0.94	0.9	0.90
C7	0.95	0.98	0.97	0.97	0.96	0.97	0.98
C8	0.78	0.84	0.88	0.88	0.84	0.84	0.81
C9	0.95	0.92	0.92	0.93	0.86	0.86	0.92
C10	0.98	0.97	0.97	0.98	0.97	0.97	0.98
C11	0.83	0.54	0.70	0.70	0.70	0.91	0.77
C12	0.98	0.98	0.98	0.96	0.97	0.96	0.96
C13	0.95	0.97	0.95	0.98	0.94	0.91	0.91
C14	0.97	0.96	0.98	0.96	0.96	0.97	0.97
C15	0.98	0.96	0.98	0.98	0.98	0.98	0.98

Further, a dominance matrix was created for the factors from table 5.6, in this regard a 7x7 matrix was created by first associating each method with a corresponding row and column of the matrix as shown in table 5.7. The ratings from table were then paired and examined. Furthermore, the last row and column cumulative values were calculated and indicated that, Sublevel stoping (36,36),Cut and fill (34,32), Sublevel caving (33,30), Block caving (30,27) were the chosen mining methods in that order as they had the highest scores amongst the set of alternatives (mining methods).

Table 5.7 Dominance matrix, mining method selection

	CF	OP	SQ	SLC	SLS	TS	BC	SUM
CF	0	2	7	7	7	5	6	**34**
OP	6	0	2	6	6	0	5	**25**
SQ	4	4	0	3	7	2	4	**24**
SLC	6	7	2	0	5	8	5	**33**
SLS	6	5	5	6	0	8	6	**36**
TS	4	3	2	3	4	0	1	**17**
BC	6	5	4	5	7	3	0	**30**
SUM	**32**	**26**	**22**	**30**	**36**	**26**	**27**	

5.3 Discussion on exclusion of mining methods

The mining method selection using the UBC method resulted in sublevel stoping as most suitable for the synclinorium ore body.

a. Cut and fill stoping scores just below Sublevel stoping. The shape and the dip of the ore body are in favour of cut and fill. However, this mining method is not workable for the scale of operations due to the following;

(i) It is labour intensive and there less skilled workers to undertake it.

(ii) It will be very expensive and will take long to set up a backfill plant. It will also be difficult to source for waste and cement. The amount of cement produced in Zambia is only enough to cater for road and structural construction in the country.

(iii) The activity of filling complicates the mining cycle causing reduction in production. For these reasons, this mining method is not suitable for this deposit.

b. Sublevel caving would be possible as the ore body shape, volume and dip are favorable for sublevel caving mining method.

c. Block caving is suitable mining method owing to the nature of the ore body. However, the method has more disadvantages compared to the new existing mining methods being applied at SOB. These include

(i) High capital investment,

(ii)High subsidence,

(iii)Low selectivity and flexibility,

(iv)High dilution from hanging wall when overburden fragmentation is higher than expected.

(v) Slow development rate and very high development cost. This will make the project take longer than projected before the areas may come into production compared to other new available mining methods. Based on these reasons, block caving may not be applied for this deposit.

5.4 Included mining methods

After excluding the unsuitable mining methods the sublevel stoping, sublevel caving options are the only mining methods likely to be viable for this deposit. However, there are multiple stoping methods available in modern day mining. These methods are variations on conventional sublevel stoping. The difference between them is the direction of mining, the design of the stopes or the method of backfilling. These four are used throughout the world and are proven to work for deposits like that of the Synclinorium. These include; Conventional Sublevel stoping, Longitudinal Bench Stoping, Transverse Bench Stoping and Vertical Crater retreat (VCR).

Therefore, the three different mining methods for different parts of the Synclinorium, were based on an attempt to minimize and manage any risk that may arise due to depth and ore body geometry as: This is shown in Figure 5.2 below;

(i) Sublevel Open Stoping (SLOS) in anticlines,

(ii) Sublevel Caving (SLC) in the limbs,

(iii) Vertical Crater Retreat (VCR) in synclines.

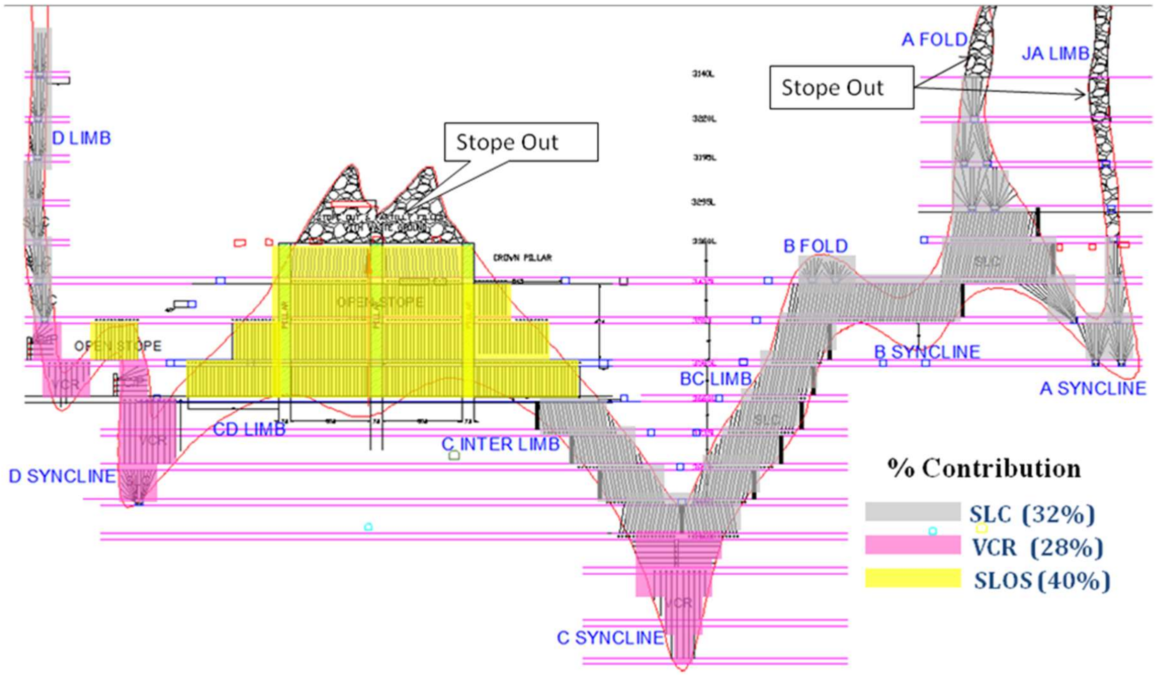


Figure 5.2 Section of the synchlinorium taken at 4800S

5.5 Designs of the selected mining methods

5.5.1 Primary development

In order to access the ore body at different levels a 5mx5m ramp has been designed from 3960L to 3360L as shown in Figure 5.3. The total number of metres required from the design is 680m. The applied design parameters are shown in appendix J.

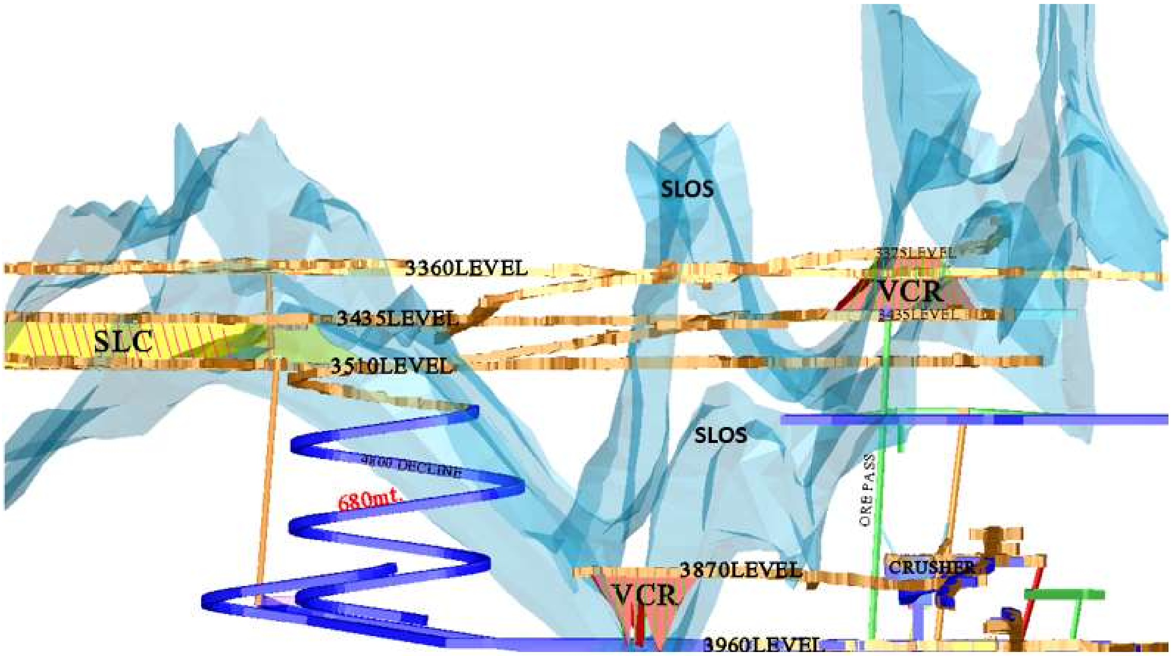


Figure 5.3 Profile of the Synclinorium showing proposed development and applicable mining methods.

5.5.2 Secondary development

This will involve the mining of crosscuts, raises and drives to facilitate mining of the ore body. The sizes and location of these development ends will depend on the mining method in each part of the ore body. The three mining methods will be applied in different parts of the ore body as shown in the profile of the design in Figure 5.3.

5.5.3 Open stope with delayed backfill

5.5.3.1 Stope design

Stope design parameters used for this mining method design has been shown in Appendix G. Sublevels will be placed at intervals of 20 to 22m with 8m x 4m crosscuts spaced at 25m centre to centre. Each such crosscut will serve as drilling crosscut or draw point or both during different stages of the extraction process. Waste from waste development will be used to fill the stopes and will help resolve the problems relating to waste handling at the shaft. The stope sizes were chosen to conform to the existing development on 3435L. The rock mass is competent with moderate strength and will support the excavation dimension. The planned development metres have been estimated using auto card software whilst the tonnages and grades have been obtained from surpac block model. A summary of the development, long hole and production are shown in table 5.8

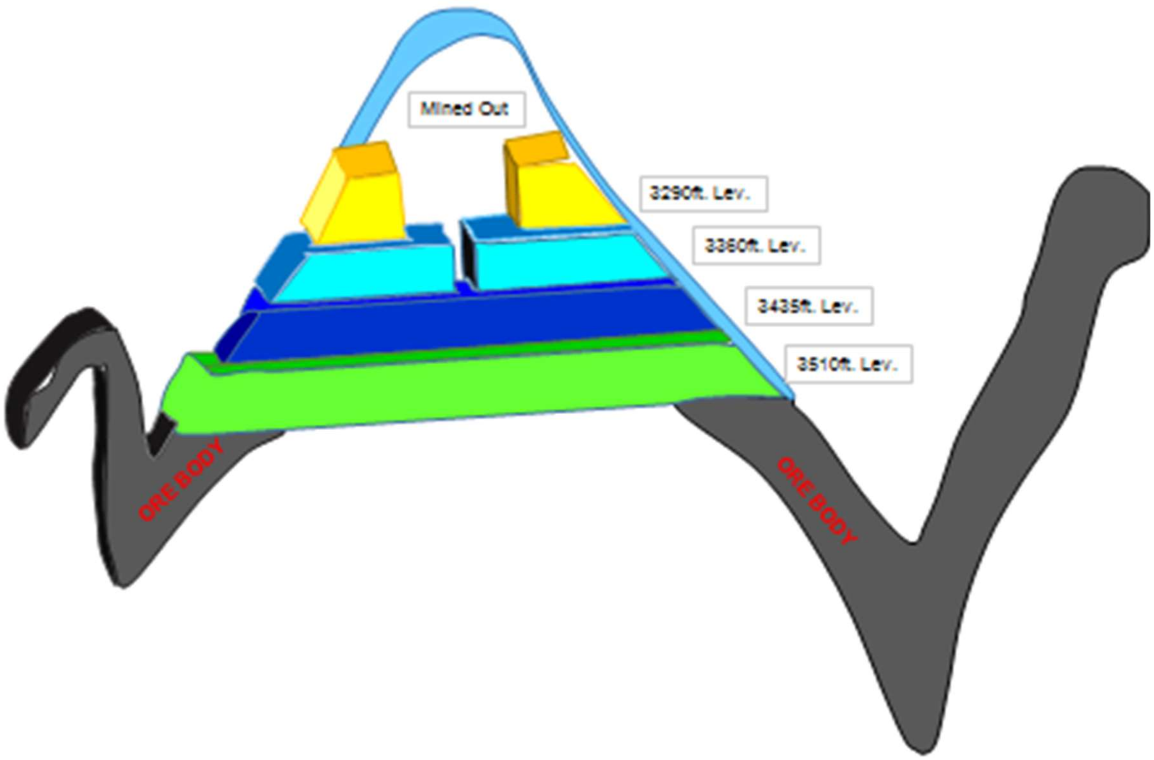
Table 5.8 Schedule of mining activities in the anticline

SLOS Production		2022	2023	2024	2025	2026	2027	2028	2029	AV 2030-2038
Tonnes hoisted	'000t		271	843	1,325	1,600	1,600	1,600	1,600	1,600
Cu Grade	%		1.93	1.96	1.96	2.21	2.20	2.21	2.21	2.21
Co Grade	%		0.04	0.04	0.05	0.05	0.05	0.05	0.05	0.05
Copper in ore hoisted	Tonnes		5,236	16,528	25,928	35,321	35,171	35,321	35,321	35,321
Cobalt in ore hoisted	Tonnes		120	326	573	855	854	855	855	855
Secondary development	metres	2,284	2,494	6,470	10,360	12,500	12,500	12,500	12,500	12,500
Long hole drilling	metres		46,132	104,910	165,000	202,700	202,700	202,700	202,700	202,700

5.5.4 Sublevel Caving (SLC)

The current SLC implies that mining starts from the center and advances outwardly towards the economic boundaries. An isometric view in Figure 5.4 shows the current SLC mining sequence practiced at the SOBS.

Figure 5.4 An isometric view of the current SLC mining sequence practiced SOB



The current challenges with this practice include:

– Delayed hanging wall exposure,

– Creation of a huge void prior to hanging wall collapse, as mining advances from the centre toward the economic boundaries,

– Potential dynamic hanging wall collapse which could result in an air blast

5.5.4.1 Proposed SLC Methodology

5.5.4.1.1 Stope design

Stope design parameters used for this mining method design has been shown in Appendix H. The crosscuts will be 4mx4m spaced at 15m center to center and loader drives positioned at 30m from the ore body drives. The slot drives dimensions will be 3.5m x 3.5 m to ensure stability of the drives in ore. The bock design of the proposed SLC is shown in Figure 5.5

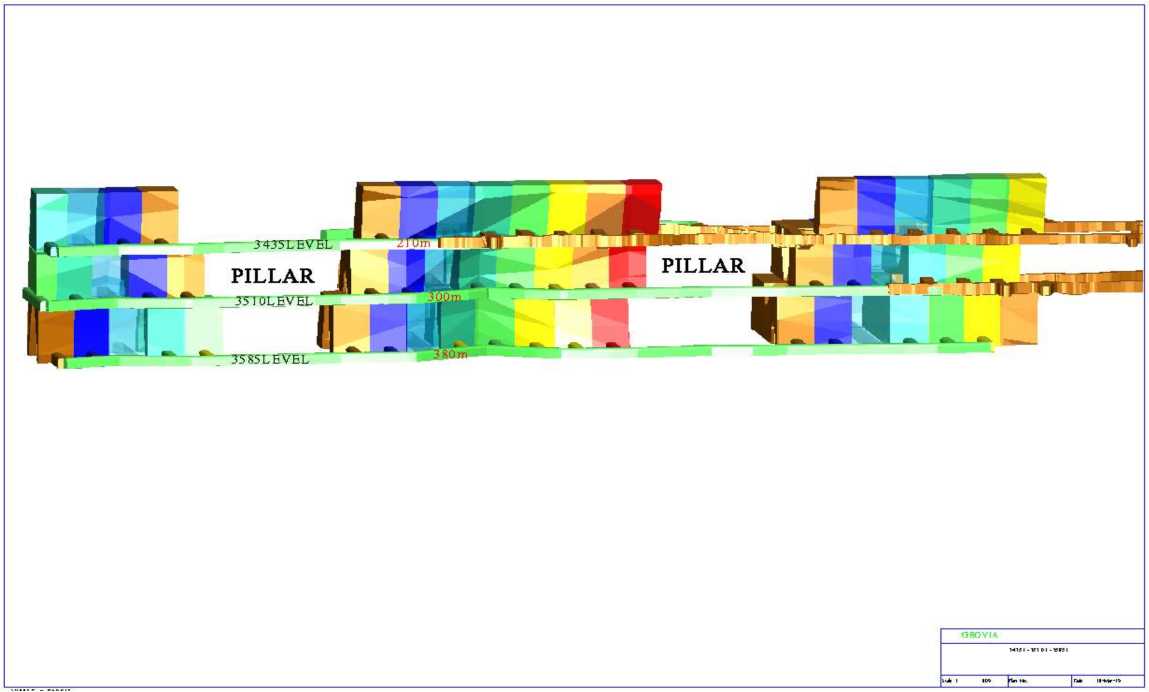


Figure 5.5 Profile of the proposed SLC stope designs

In order to avert the risk posed by the current SLC arrangement it is proposed to change the position of the slot drive. Preferably the slot drive should be positioned closer to the hanging wall of the eastern limb. Mining should first proceed along the slot drive just to open up the hanging on the various sublevel. The hanging wall of the eastern limb is systematically undercut to induce natural cave prior to transverse multiple front retreat.

Though the undercut will create a very limited opening along hanging wall, it will completely undercut the eastern limb hanging wall creating the required HR or footprint to induce complete hanging wall collapse. Considering the volume of the undercut the potential for a catastrophic hanging wall collapse and the subsequent air blast could be minimized if not completely avoided. Following the hanging wall cave transverse SLC could resume. This will progressively and gradually induce further hanging wall cave which is likely to link up with the eastern “B” limb cave as shown in appendix F.

It should be noted that the hanging of the current SLC stopes have been caving naturally. Considering the medium and long term production schedules, by the year 2023 and beyond, the production stope faces are likely to extend deeper. This is likely to increase the potential for the current cave and the SOBS cave to merge. The direction and extend of cave is dictated by geological structural defects – nature, direction and continuity of structures. In the real world, the rock masses are inherently variable and do not conform to an ideal pattern.

The schedule of activities obtained from AutoCAD and Surpac block models are shown in tables 5.9.

Table 5.9 Schedule of mining activities in the limbs

SLC Production		2022	2023	2024	2025	2026	2027	2028	2029	AV 2030-2038
Tonnes hoisted	'000t		135	453	879	1,300	1,300	1,300	1,300	1,300
Cu Grade	%		1.64	1.83	1.91	1.54	1.57	1.59	1.59	1.59
Co Grade	%		0.07	0.10	0.05	0.05	0.05	0.05	0.05	0.05
Copper in ore hoisted	Tonnes		2,214	8,290	16,789	20,020	20,410	20,670	20,670	20,670
Cobalt in ore hoisted	Tonnes	-	95	453	440	650	650	650	650	650
Secondary development	metres	1,731	1,890	3,478	6,980	10,011	10,011	10,011	10,011	10,011
Long hole drilling	metres		19,243	56,380	109,450	162,340	162,340	162,340	162,340	162,340

5.5.4.2 Cavability Assessment

This assessment was based on an empirical approach. It was deduced that to induce a complete cave in the hanging wall requires a hydraulic radius in the ranges of 28m – 37m. The Laubscher’s stability graph was used for cavability assessment as shown in Figure 5.6

Hanging wall collapse is a function of hanging wall exposure or footprint and it’s measured in terms of hydraulic radius. It could be inferred from table 5.10, the hydraulic radius matrix that should mining extent along strike over a distance of 130m and down dip towards or beyond 3585L the HR of the eastern hanging wall (c-fold) will exceed 37m, and it's likely to induce complete cave as read from Laubscher's stability graph shown in Figure 5.6. The strike length under consideration at the SOBS is about 180m. Thus, though the Synclinorium plunges to the north, there exists the potential to create a footprint likely to induce hanging wall cave as the hanging wall undercut mining advances down dip.

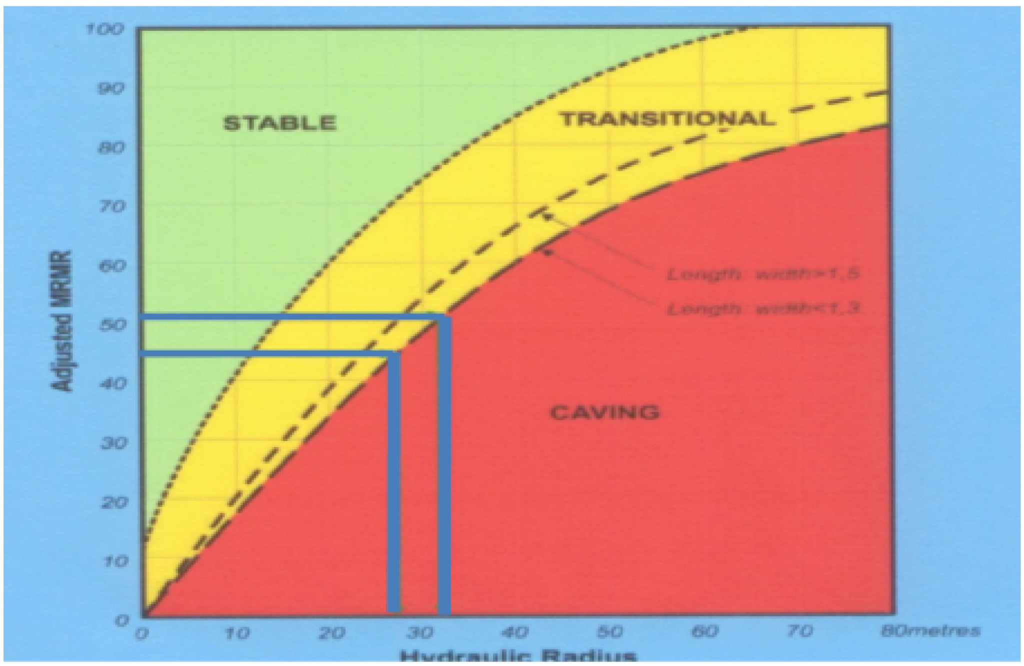


Figure 5.6 Laubscher's Stability Graph – showing the average HR values required for the Synclinorium ore and Hanging wall Argillite to completely cave naturally.

Table 5.10 Hydraulic Radius matrix – Synclinorium.

MINING ALONG STRIKE

MINING DOWN DIP

	HW EXP. [M]	100	110	120	130	140	150	160	170	180	190	200	210	220	230	240	250
3360L	78	22	23	24	24	25	26	26	27	27	28	28	28	29	29	29	30
3435L	112	26	28	29	30	31	32	33	34	35	35	36	37	37	38	38	39
3510L	142	29	31	33	34	35	36	38	39	40	41	42	42	43	44	45	45
3585L	175	32	34	36	37	39	40	42	43	44	46	47	48	49	50	51	51
3660L	213	34	36	38	40	42	44	46	47	49	50	52	53	54	55	56	58
3735L	251	36	38	41	43	45	47	49	51	52	54	56	57	59	60	61	63
3810L	289	37	40	42	45	47	49	51	54	55	57	59	61	62	64	66	67
3885L	327	38	41	44	47	49	51	54	56	58	60	62	64	66	68	69	71

A cross section showing the proposed SLC mining sequence and the projected mined out section of the "B" fold is shown in Appendix F. The potential for both caves to merge could be high should the "B" mining induce complete cave.

5.5.5 Vertical Crater Retreat (VCR)

Vertical Crater Retreat applies to ore bodies with steep dip and competent rock in both ore and host rock. Part of the blasted ore will remain in the stope over the production cycle, serving as temporary support of stope walls.

5.5.5.1 Stope design

Stope design parameters used for this mining method design has been shown in Appendix I. This involves setting up a top drilling level and the holing level. Two crosscuts will be mined from loader drive on the top and bottom levels. On the top level, the two crosscuts will be connected together and slyped to provide the drilling chamber where a DTH machine will be positioned. At the bottom, similarly the two crosscuts are connected to form a holing chamber.

The stope will be taken first followed by the rib and crown pillar. The crown drilling will be done using a drifter machine as per the current practice at SOB shaft.

The designed spacing between the 165mm holes is 3.5m x 3.5m. In an ideal situation it would be desirable to drill all the holes parallel to each other along the entire height of a stope. The width of the stopes will vary depending on the ore body but stope strike length and rib pillar will be 15m and 8m respectively. See Figure 5.7 and 5.8 for VCR stope profile and sections respectively. In order to optimize VCR blasting, regular blast hole surveys will be conducted in order to ensure drilling accuracy is within acceptable limits.

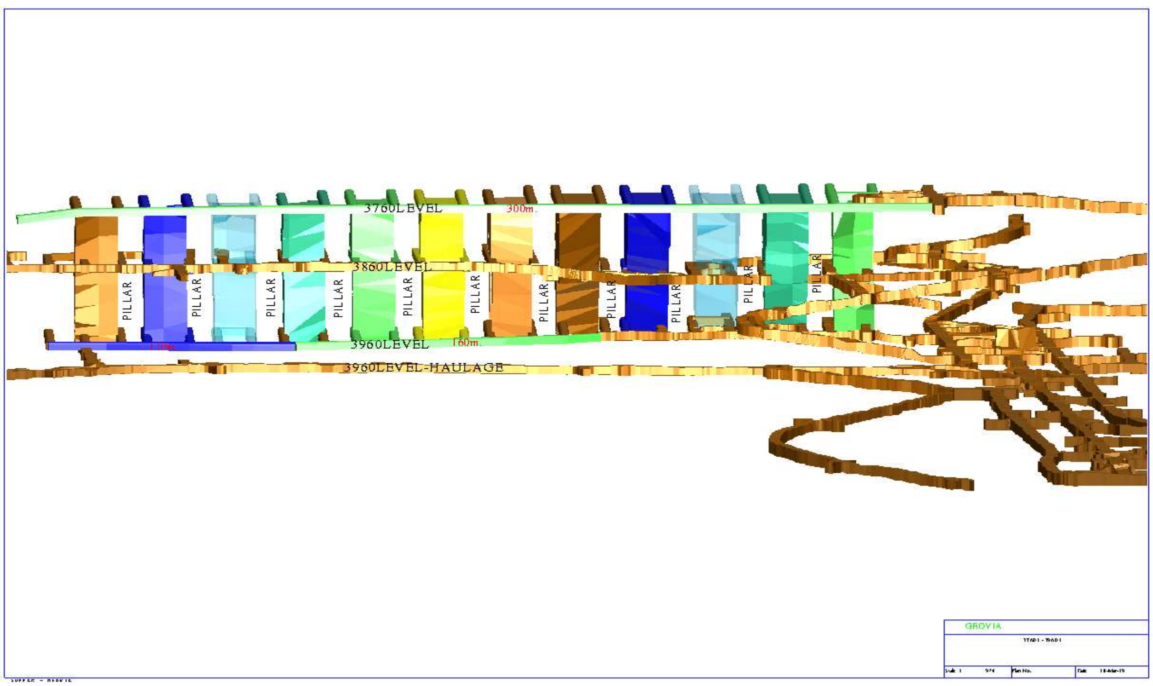


Figure 5.7 Profile of the proposed VCR stope designs

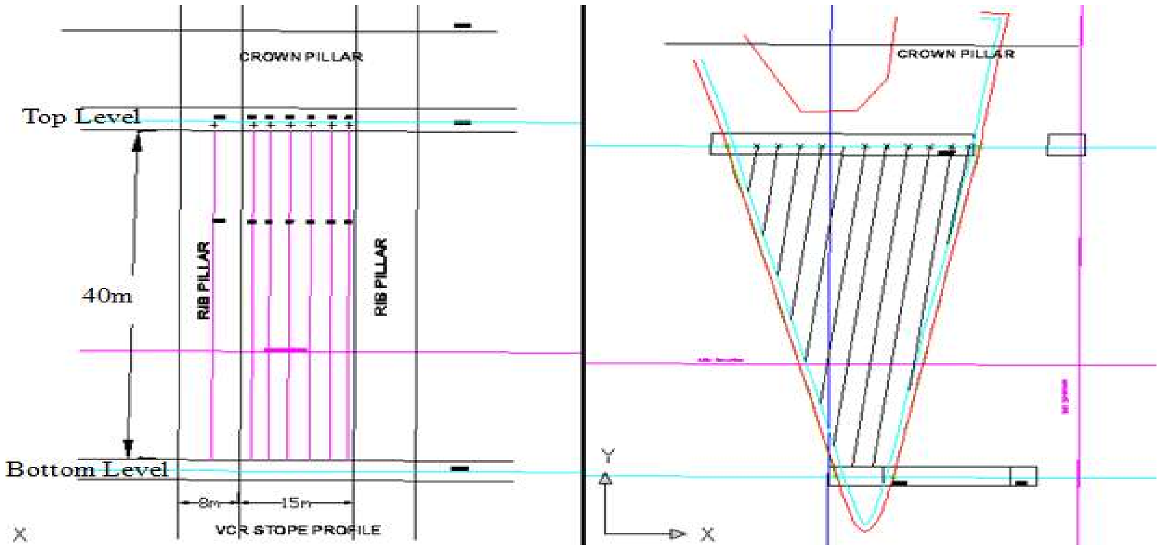


Figure 5.8 VCR stope profile and sections

The use of a 165mm diameter holes drilled with the down the hole (DTH) hammer allows accurate drilling of 50m. However, the method encounters challenges in drilling due to hole diversions that lead to improper charging and re-drilling at 50m. However, in case of Synclinorium this will be reduced to 40m to allow for increased accurate in drilling.

Another advantage of VCR is that the blasted ground (ore) is dropped uniformly across the stope to leave only sufficient void for the next blast. Therefore, the rock mass in the stope continuously supports the stope walls. This reduces waste dilution from hanging wall and footwall. The schedule of mining activities in the limbs are shown in Table 5.11

Table 5:11 Schedule of mining activities in Synclines

VCR Production		2022	2023	2024	2025	2026	2027	2028	2029	AV 2030-2038
Tonnes hoisted	'000t		271	468	811	1,100	1,100	1,100	1,100	1,100
Cu Grade	%		1.88	1.65	1.58	1.64	1.63	1.64	1.64	1.64
Co Grade	%		0.06	0.05	0.05	0.07	0.07	0.07	0.07	0.07
Copper in ore hoisted	Tonnes	-	5,095	7,722	12,814	18,040	17,930	18,040	18,040	18,040
Cobalt in ore hoisted	Tonnes	-	163	234	406	770	770	770	770	770
Secondary development	metres	1,192	1,301	1,874	3,530	4,348	4,348	4,348	4,348	4,348
Long hole drilling	metres		20,655	21,570	37,400	50,000	50,000	50,000	50,000	50,000

5.6 Economic analysis

5.6.1 Capital expenditure

The profile of the full design of the Synclinorium project is shown in Appendix H. From this design, an estimated capital cost of $95.633 million (SRK consultants report, 2008) is required to set up the areas for production. This expenditure will push production to 4.0 million tonnes per annum by 2023. The capital costs are the costs incurred in capital development, buying and replacing of equipment and setting up of the fill plant. See the table 5.12 below of the capital costs for the period 2019 to 2022 and the projection beyond.

Table 5.12 Projected Capital expenditure (SRK consultants report, 2008)

CAPEX ACTIVITY		2019	2020	2021	2022	2023	2024	2025	2026	AV 2026-2048
Stope chutes for Dump truck	USD'000	350	350	350	350	350	350	350	350	350
Ventilation fans	USD'000	325	325	325	325	325	325	325	325	325
Pumping system	USD'000	731								
loaders	USD'000	2,800	2,800	3,500	4,900	2,800	2,800	3,500	4,900	4,900
Dewatering & Diamond drilling	USD'000	469	469	469	469	469	469	469	469	469
Conveyors	USD'000			3,600						
Fill plant	USD'000			2,000	2,000					
Services	USD'000	828	847	794	766					
Labour cost(MCM)	USD'000	819	1,025	1,025	1,025					
Sumps,settlers,pump chamber	USD'000		659	4,058	446					
Crusher system	USD'000		620	5,445						
66KV Substation	USD'000	4,500	3,000							
Primary development	USD'000	6,048	1,765	3,487	4,357	4,357	4,357	4,357	4,357	4,357
Dump trucks	USD'000				1,600	1,600	3,200	1,600	2,400	2,400
Total	USD'000	**16,870**	**11,860**	**25,053**	**16,238**	**9,901**	**11,501**	**10,601**	**12,801**	**12,801**

5.6.2 Operating costs

The operating costs are the costs incurred in extracting the ore from stopes. The process includes development carried out to gain access to the stope from the main ramps, stoping and transportation of the ore. It also includes labour, support, consumables and mechanical equipment spare. See tables 5.13 to 5.15 for the calculated operating costs for each proposed mining method and Tables 5.16 to 5.19 for expected revenue and expenditure for each mining method.

5.6.2.1 Parameters

To work out the estimated cost for each year, the following parameters were considered:

- Tonnage to be produced each year;
- Development and long hole metres;
- Current contractor rates for development, long hole drilling and support;
- Current salaries for Mopani employees;
- $186 per hour cost of diesel loader;

- 50 tonnes per hour per loader;
- 75% equipment availability; and
- 85% equipment utilization.

Table 5.13 Schedule of annual operating costs SLOS.

OPEX ACTIVITY SLOS	Description		unit cost	Quantity	Total cost
Development	4m* 4m	USD	967.62	12500	12,095,250
Stoping	76mm holes	USD	22.35	246,154	5,501,538
Support	Rock bolts	USD	21.31	80088	1,706,675
	Cable bolts	USD	34.99	30033	1,050,855
	Wire mesh	USD	30.38	50005.5	1,519,167
Tramming		USD	3,307,667	1	3,307,667
Shafts		USD	3,760,133	1	3,760,133
Auxillary services		USD	5,854,000	1	5,854,000
Technical		USD	771,667	1	771,667
Total		USD			35,566,952

Table 5.14 Schedule of annual operating costs SLC.

OPEX ACTIVITY SLC	Description		unit cost	Quantity	Total cost
Development	4m* 4m	USD	967.62	10011	9,686,844
Stoping	76mm holes	USD	22.35	200,000	4,470,000
Support	Rock bolts	USD	21.31	80088	1,706,675
	Cable bolts	USD	34.99	30033	1,050,855
	Wire mesh	USD	30.38	50005.5	1,519,167
Tramming		USD	3,307,667	1	3,307,667
Shafts		USD	3,760,133	1	3,760,133
Auxillary services		USD	5,854,000	1	5,854,000
Technical		USD	771,667	1	771,667
Total		USD			32,127,008

Table 5.15 Schedule of annual operating costs VCR.

OPEX ACTIVITY VCR	Description		unit cost	Quantity	Total cost
Development	4m* 4m	USD	967.62	4348	4,207,212
Stoping	165mm holes	USD	42.56	50000	2,128,000
Support	Rock bolts	USD	21.31	34784	741,247
	Cable bolts	USD	34.99	13044	456,410
	Wire mesh	USD	30.38	13578	412,500
Tramming		USD	3,307,667	1	3,307,667
Shafts		USD	3,760,133	1	3,760,133
Auxillary services		USD	5,854,000	1	5,854,000
Technical		USD	771,667	1	771,667
Total		USD			21,638,835

Table 5.16 Expected revenue and expenditure of the SLOS mining method.

SLOS Production		Annual
Tonnes hoisted	Tonnes	1,600,000
Cu Grade	%	2.21
Co Grade	%	0.05
Copper in ore hoisted	Tonnes	35,360
Cobalt in ore hoisted	Tonnes	800
Final processed cu	Tonnes	31,117
Final processed co	Tonnes	704
Projected $/Tonne CU	$	6,450
Projected $/Tonne CO	$	55,000
Revenue CU	$	200,703,360
Revenue CO	$	38,720,000
Total revenue	$	239,423,360
Operating cost	$	35,566,952

Table 5.17 Expected revenue and expenditure of the SLC mining method.

SLC Production		**Annual**
Tonnes hoisted	*Tonnes*	1,300,000
Cu Grade	*%*	1.59
Co Grade	*%*	0.05
Copper in ore hoisted	*Tonnes*	20,670
Cobalt in ore hoisted	*Tonnes*	650
Final processed cu	*Tonnes*	18,190
Final processed co	*Tonnes*	572
Projected $/Tonne CU	*$*	6,450
Projected $/Tonne CO	*$*	55,000
Revenue CU	*$*	117,322,920
Revenue CO	*$*	*31,460,000*
Total revenue	*$*	*148,782,920*
Operating cost	*$*	32,127,008

Table 5.18 Expected revenue and expenditure of the VCR mining method.

VCR Production		**Annual**
Tonnes hoisted	*Tonnes*	1,100,000
Cu Grade	*%*	1.64
Co Grade	*%*	0.07
Copper in ore hoisted	*Tonnes*	18,040
Cobalt in ore hoisted	*Tonnes*	770
Final processed cu	*Tonnes*	15,875
Final processed co	*Tonnes*	678
Projected $/Tonne CU	*$*	6,450
Projected $/Tonne CO	*$*	55,000
Revenue CU	*$*	102,395,040
Revenue CO	*$*	37,268,000
Total revenue	*$*	*139,663,040*
Operating cost	*$*	21,638,835

Table 5.19 Summary of Expected revenue and expenditure of the Synclinorium project.

Annual Revenue	$	**527,869,320**
Capital Expenditure	$	95,633,000
Annual Operating costs	$	89,332,794
Revenue less costs	$	**342,903,526**

From Table 5.19 it's clear that the Synclinorium mining project is viable with an average annual projected revenue of $340 million.

5.6.3 Calculation of Net present Value of the project

A present value is a future profit discounted to the inflation over the years, the NPV was calculated by the formula;

$$NPV = -[CF_0] + \sum CF(\frac{P}{F}, i\%, j) \qquad (5.1)$$

Where;

NPV = the net present value;

CF_0 = initial investment;

$\frac{P}{F}$ = present worth factor;

CF_t =Cash flow for the jth year

i= rate of return

J= 1, 2, 3………….n years

NPV= ∑ {Net Period Cash Flow (1+i)∧ n} – Initial investment

Taking i at 15%

N= 1 year

NPV= 527869320(1+0.15)^-1 -184965794

NPV= $274051006

From the calculations the NPV is greater than 0 and is a clear justification that the project is viable.

5.7 Summary

In this Chapter, University of British Colombia (UBC) mining method selection criteria and online mining method selection tool (MMST) were used for the selection of a suitable mining method. Both methods reviewed that, Sublevel stoping, Cut and fill, Sublevel caving and Block caving were selected in that order. However, after subjecting these methods to geotechnical, technical and economic factors only sublevel stoping, sublevel caving and the included VCR mining methods were selected.

The next Chapter, summarizes the findings of this research and recommends areas of improvement in the current mining methods and chosen mining methods.

CHAPTER 6

Conclusions and recommendations

6.1 Conclusions

The following conclusions have been drawn;

According to the traditional, online UBC and fuzzy dominance mining method selection tools, Economical and geotechnical evaluation, the mining methods applicable in the Synclinorium, are Open stoping with fill in the anticline, SLC in the limbs and VCR in the synclines. However, the current SLC and Vertical crater retreat methods require modification to address the current challenges due to hang-ups, delayed hanging wall exposure and creation of a huge void prior to hanging wall collapse as mining advances from the centre toward the economic boundaries in SLC and hole deviations in VCR at 50m that results in re-drilling and ultimately increasing the costs of mining.

The empirical analysis, indicated that to induce a complete cave in the hanging wall required a hydraulic radius in the ranges of 28m – 37m. The Laubscher's stability graph was used for cavability assessment.

The modifications to the two mining methods will increase production, minimize dilution and costs. Furthermore, sublevel open stoping will help address the challenges of waste handling and will reduce costs of crushing, transporting and hoisting of waste to surface. The economic evaluation indicates that the Synclinorium mining project is viable with a projected average annual income of $340million.

6.2 Recommendations

i. In order to avert the risk posed by the current SLC arrangement it is proposed to change the position of the slot drive. Preferably the slot drive should be positioned closer to the hanging wall of the eastern limb (c-fold). Mining should first proceed along the slot drive just to open up the hanging on the various sublevels

ii. Reduce the VCR height between the drilling and holing chambers from 50m to 40m and ensure the mast of the cubex machine is firmly anchored to reduce hole diversions due to vibrations

iii. Significant additional research into the cavability of the Synclinorium hanging wall should be undertaken immediately mining commences to quickly address challenges that may be observed Cave assistance programs should also be set up for use in the event that hanging wall caving does not occur naturally.

iv. Until the hanging wall caves, it is recommended that drawing from the cave must be restricted to 40% to ensure that an air blast does not result from catastrophic caving of the hanging wall. A policy of interactive draw should be followed to ensure that the cave is drawn down evenly, allowing maximum recovery of ore and minimum dilution entry. This means that upwards of 20 crosscuts should be in production at any one time, retreated back in a straight line. This reflects a complete change to current 'sub level caving' practice at SOB Shaft. Draw plans detailing required tonnages from each draw point will have to be followed rigorously to ensure achievement of production and grade targets.

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Appendices

Appendix A: Ranking of geometry/grade distribution for different mining methods.

Ranking of geometry/grade distribution for different mining methods

mining method	general shape			ore thickness					ore plunge			grade distribution			depth		
	M	T/P	I	VN	N	I	T	VT	F	I	S	U	G	E	SH	I	D
open pit mining	4	2	3	1	2	3	4	4	3	3	1	3	3	2	4	0	-49
block caving	4	2	0	-49	-49	0	3	4	3	2	4	3	2	2	2	3	3
sublevel stoping	3	4	1	-10	1	3	4	3	2	1	4	4	4	3	3	4	2
sublevel caving	3	4	1	-49	-49	0	4	4	1	1	4	3	2	2	3	2	2
longwall mining	-49	4	-49	4	3	0	-49	-49	4	0	-49	4	1	0	2	2	3
room and pillar	0	4	2	4	3	1	-49	-49	4	0	-49	4	2	0	3	3	2
shrinkage stoping	0	4	2	4	4	0	-49	-49	-49	0	4	3	2	2	3	3	2
cut & fill stoping	1	4	4	3	4	4	1	0	1	3	4	2	3	4	2	3	4
top slicing	1	2	0	1	1	0	2	1	4	2	0	2	1	1	2	1	1
square set stoping	0	1	4	4	3	2	0	0	2	3	2	0	1	3	1	1	2

M = massive; T/P = tabular or platy; I = irregular

VN = very narrow; N = narrow; I = intermediate; T = thick; VT = very thick

F = flat; I = intermediate; S = steep

U = uniform; G = gradational; E = erratic

S = shallow; I = intermediate; D = deep

Appendix B: Rock mass strength

Rock substance strength

mining method	ore zone				hanging wall				footwall			
	VW	W	M	S	VW	W	M	S	VW	W	M	S
open pit mining	4	3	3	3	3	3	4	4	3	3	4	4
block caving	4	2	1	0	4	3	2	0	4	3	2	1
sublevel stoping	0	2	4	4	0	1	4	5	0	1	3	3
sublevel caving	2	3	3	2	4	3	2	1	1	2	2	2
longwall mining	6	5	2	1	6	5	2	2	-	-	-	-
room and pillar	0	0	3	6	0	0	2	6	-	-	-	-
shrinkage stoping	0	1	3	4	0	1	3	4	0	2	3	3
cut and fill	0	1	3	3	3	5	4	2	1	3	2	2
top slicing	3	2	1	0	3	2	2	2	2	2	1	1
square set	4	3	1	0	4	2	1	0	3	2	0	0

RSS ratings VW = very weak, W = weak, M = medium, S = strong.

Appendix C: Rock mass ratings for different mining methods.

Rock mechanics characteristics

Rock mass ratings

mining method	ore zone					hanging wall					footwall				
	VW	W	M	S	VS	VW	W	M	S	VS	VW	W	M	S	VS
open pit mining	3	3	3	3	3	2	3	4	4	4	2	3	4	4	4
block caving	4	3	2	0	-49	3	3	3	2	2	3	3	3	2	2
sublevel stoping	1	3	4	4	4	-49	0	3	4	4	0	0	2	3	3
sublevel caving	3	4	3	1	0	4	4	3	2	2	1	2	3	3	3
longwall mining	6	6	4	2	2	6	5	4	3	3	-	-	-	-	-
room and pillar	-49	0	3	5	6	-49	0	3	5	6	-	-	-	-	-
shrinkage stoping	0	1	3	3	3	0	0	2	4	4	0	0	2	3	3
cut and fill	0	1	2	3	3	3	5	4	3	3	3	3	2	2	2
top slicing	3	2	1	1	0	0	0	2	3	3	0	0	1	2	2
square set	4	4	1	0	0	4	4	1	0	0	3	1	0	0	0

RMR ratings: VW = 0-20, W = 20-40, M = 40-60, S = 60-80, VS = 80-100

Appendix D: Description of mining parameters in the UBC mining method selection

UBC mining method selection

1) general shape/width	
equi-dimensional	all dimensions are on the same order of magnitude
platy-tabular	two dimensions are many times the thickness, which does not usually exceed 35m
irregular	dimensions vary over short distances
2) ore thickness	
very narrow	<3 m
narrow	3-10 m
intermediate	10-30 m
thick	30-100 m
very thick	>100 m

3) plunge

flat	<20 degrees
intermediate	20 - 55 degrees
steep	>55 degrees

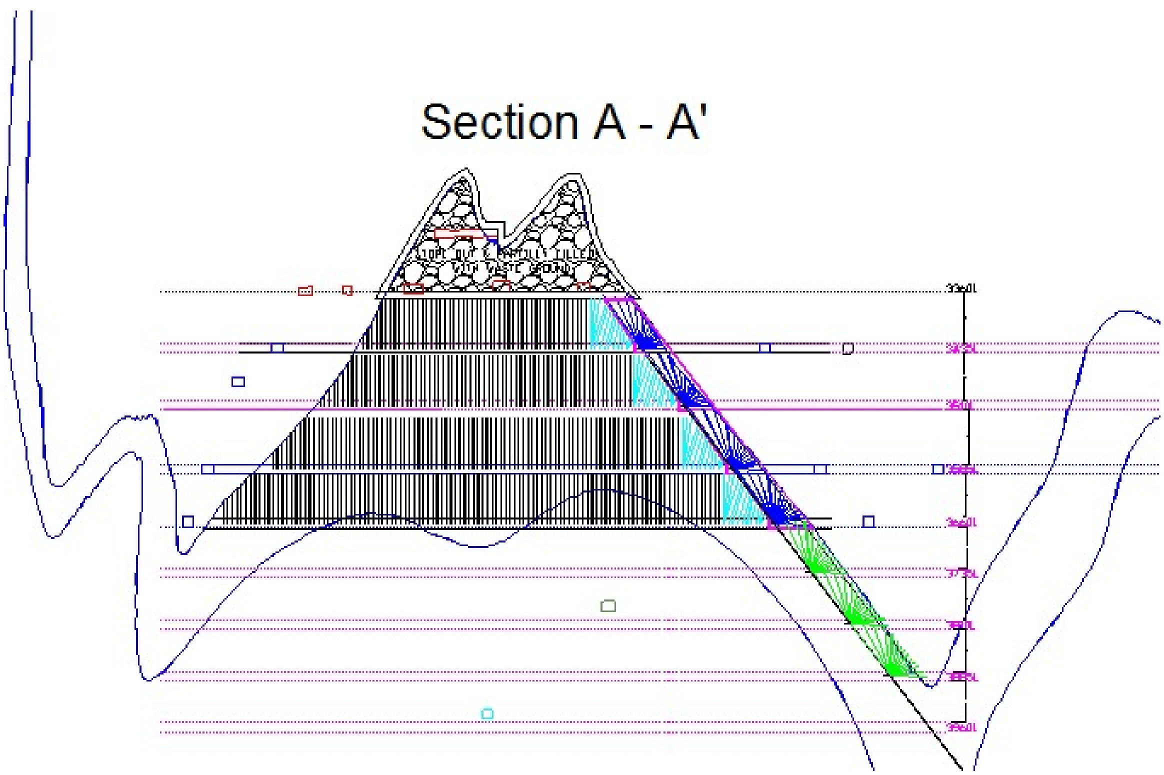
4) depth below surface

shallow	0-100 m
intermediate	100-600 m
deep	>600 m

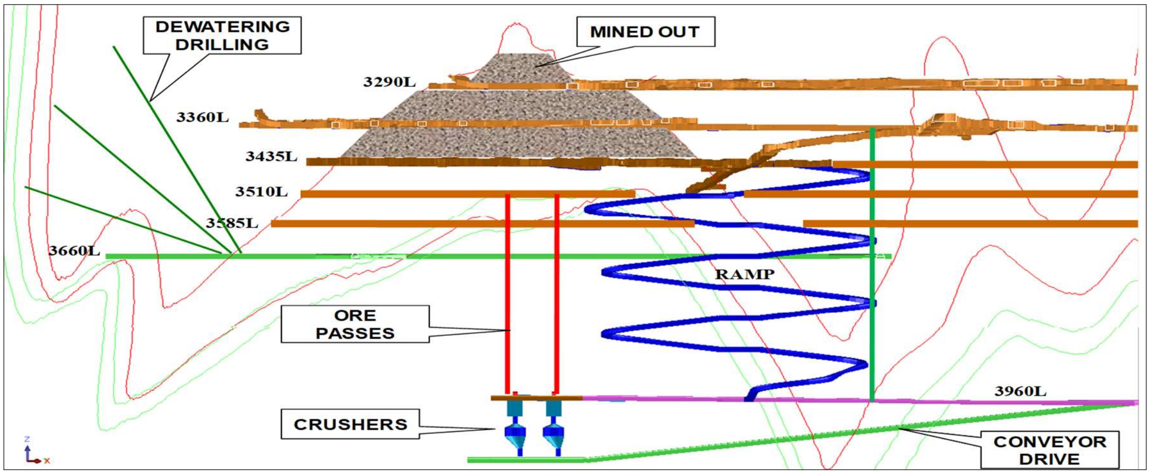
6) rock mass ratings:	very weak	0-20
	weak	20-40
	moderate	40-60
	strong	60-80
	very strong	80-100

7) rock substance strength:	very weak	<5
(uniaxial strength/principal stress)	weak	5-10
	moderate	10-15
	strong	>15

Appendix E: Proposed SLC mining sequence for the Synclinorium.



Appendix F: Profile of the proposed design of the Synclinorium



Appendix G: Mine design parameters – SLOS

Development	Unit	W	H	Criterion / Comment
End size of X/Cut	m	4	4	
End size of O/Body Drive	m	4	4	O/Body < 10.0m
End size of Slot Drive	m	4	4	O/Body > 10.0m
End size of F/W Drive [SOB > 3360 & Central]	m	4	4	
End size of F/W Drive [SOB 3360 -3960 - Sync]	m	4.5	4.8	
X/Cut Spacing	m	25		O/Body > 10.0m
No. of X/Cuts mined	No.	3 to 5		Dependant on O/Body Strike Extent
Type of Rig - Double boom		Boomer		
Footwall drives				
Distance away from ore body	m	30		
Stoping				
Stope Span - centre to centre - [X/Cut Spacing for longitudinal SLC]	m	25		
Stope Vertical height	m	20 - 22		
General Sequence - Lag	m	45°	Echelon	Top Down - Top Leading
Cut-off Ore Body Thickness (for Planning)	m	5		To exclude if occur at extremity
Long Hole Drilling				
Long hole drilled diameter	mm	76		
Ring Burden	m	2		
Toe Burden	m	2.3		
Type of long hole machine		Simba		
Loaders				
Type of loader required		R 1600		
Bucket capacity	tons	9		
Dump Trucks				
Type of Dump Truck required		AD 30		
Bucket capacity	tons	25		

Appendix H: Mine design parameters – SLC

Development	**Unit**	**W**	**H**	**Criterion / Comment**
End size of X/Cut	m	4	4	
End size of O/Body Drive - Strike (Longitudinal) SLC	m	4	4	O/Body < 10.0m
End size of Slot Drive - Transverse SLC	m	4	4	O/Body > 10.0m
End size of F/W Drive [SOB > 3360 & Central]	m	4	4	
End size of F/W Drive [SOB 3360 -3960 - Sync]	m	4.5	4.8	
X/Cut Spacing - Strike (Longitudinal) SLC	m	45		O/Body < 10.0m
X/Cut Spacing - Transverse SLC	m	15		O/Body > 10.0m
No. of X/Cuts mined	No.	3 to 5		Dependant on O/Body Strike
Type of Rig - Double boom		Boomer		
Footwall drives				
Distance away from ore body	m	30		
Stoping				
Stope Span - centre to centre - [X/Cut Spacing for longitudinal SLC]	m	15		
Stope Vertical height	m	20 - 25		
General Sequence - Lag	m	45°	Echelon	Top Down - Top Leading
Cut-off Ore Body Thickness (for Planning)	m	5		To exclude if occur at extremity
Long Hole Drilling				
Long hole drilled diameter	mm	76		
Ring Burden	m	2		
Toe Burden	m	2.3		
Type of long hole machine		Simba		
Loaders				
Type of loader required		R 1 600		
Bucket capacity	tons	9		
Dump Trucks				
Type of Dump Truck required		AD 30		
Bucket capacity	tons	25		

Appendix I: Mine design parameters – VCR

Development	**Unit**	**W**	**H**
Type of Rig - Double boom		Boomer	
End size of X/Cut	m	4	4
End size of O/Body Drive	m	4	4
End size of F/W Drive [SOB > 3360 & Central]	m	4	4
End size of F/W Drive [SOB 3360 -3960 - Sync]		4.5	4.8
No. of X/Cuts per stope	No.	2	
Footwall drives			
Distance away from ore body	m	30	
Stoping			
Stope Span - including pillar	m	23	
Pillar Size (Rib Pillar)	m	7	
Pillar Size (Crown Pillar)	m	18	
Stope Vertical Height – average [60m as Max.]	m	40	
General Sequence		Top Down	
Type of Fill			
Development Waste			
Long Hole Drilling			
Long hole drilled diameter	mm	165	
Long hole drilled hole spacing	m	3	
Type of long hole machine		Cubex	
Loaders			
Type of loader required		R 1 600	
Bucket capacity		9	
Dump Trucks			
Type of Dump Truck required		AD 30	
Bucket capacity		25	

Appendix J: Development design parameters

Collection Names						
Excavation Description / Name	**Width (m)**	**Height (m)**	**Length (m)**	**Excavation Type Assigned (SURPAC)**	**Comment**	**Primary / Secondary**
Conveyor Drive	5	5		**condr**		Primary
Water Drive	5	4		**wdr**	Footwall designed to maximise sediment carry	Primary
Access drive / Material Drive / Intake Drive / Haulage	5	5		**adr**		Primary
Connection	5	5		**conn**		Primary
Reverse bay	5	5	25	**rbay**		Primary
Muck bay	5	5	12	**mbay**		secondary
Pump cubby	2	2	1	**pcub**		secondary
Electrical cubby	2	2	1	**ecub**		secondary
Mini-sub cubby	5	5	Various	**msub**		Primary
Ore Pass	3.4		Various	**opass**	Ø	Primary
Waste Pass	3.4		Various	**wpass**	Ø	Primary
Service raise	3.4		Various	**srse**	Ø	Primary
Main Station Silo	10		41.3	**Silo**	Ø	Primary
Ventilation cubby	5	5	5	**vcub**		Primary
Chairlift	5	5		**clft**		Primary
Ventilation pass / raise	3.1			**vpass**	Ø	Primary
Main vent shaft	6.1			**ventilation shaft**	Ø	Primary
Main shaft	7			**mat shaft**	Ø	Primary
Decline	5	5		**decl**		Primary
Ramp	5	5		**ramp**		Primary
Main access ramp	5	5		**accramp**		Primary
Explosives store	5	5	10	**estore**		Primary
Clinics	5	5	7	**clinic**		Primary
Loader drive / Footwall drive [3360 & up] & Central	4	4		**ldr1**		Secondary
Loader drive / Footwall drive [3360 - 3960 - Sync]	4.5	4.8		**ldr2**		Secondary
Ore Body Drive	4	4		**odr**	on all levels	Secondary
Cross-cut	4	4		**xc**	on all levels	Secondary
Return airway	5	5		**raw**		Primary
Raise (pass) cubby	5	5	Various	**rcub**		Primary
Storage / material bay	5	5	18	**sbay**	Crusher storage bay	Secondary
Cubby	Various	Various	Various	**cub**		Secondary
Escape way	3.1			**escway**	Ø	Primary
Station drive - Top of Silos	8	5	108	**stndr**		Primary
Station drive - Top of Silos to Return Airway	6	5		**stndr**		Primary
Vent cross-over	4	4		**crossover**	Or airway size + 10% whichever is greater	Primary
NX hole	0.5			**annex**	Ø	Primary
Tip Access Cross-cut	5	7		**tipacc**	to accommodate AD30 tipping	Primary
Transformer Bays	6	4		**tbays**		Primary