EXAMININATION OF THE GEOLOGICAL SET UP AND ROCK MASS CHARACTERISTICS OF NCHANGA UNDERGROUND MINE IN ORDER TO OPTIMIZE DRAW OF 'THE FELDSPATHIC-QUARTZITE' (TFQ) ORE IN THE CAVED AREAS

By

Hakakwale Bornwell

A dissertation submitted to the University of Zambia in fulfillment of the requirements for the degree of Master of Mineral Sciences

The University of Zambia

School of Mines

Mining Engineering Department

Lusaka

November, 2013

Declaration

I, Bornwell Hakakwale, do declare hereby that the work presented in this dissertation is my original research work with exception of quotes and citation of other authors' work duly referenced and acknowledged herein. No part of this dissertation has been presented or published for pursuit of any degree in this or any other university or college.

I, therefore, declare that this dissertation was written and presented according to the rules and regulations governing the award the of Master of Mineral Sciences degree of the University of Zambia.

Author's Signature: ________________ *Date:* ______________

Certificate of Approval

This dissertation, by Hakakwale Bornwell, is approved as fulfilling the requirements for the award of the Degree of Master of Mineral Sciences by the University of Zambia

Signature of Examiners

External Examiner: ……………………………………… Date: …………………………………..

Internal Examiner: ……………………………………….. Date: …………………………………..

Internal Examiner: ……………………………………….. Date: …………………………………..

Project Supervisor: ……………………………………… Date: …………………………………..

Dedication

To my family.

Acknowledgments

I want to thank Professor Mutale W. Chanda for his guidance, mentorship and untiring efforts to ensure the research was done properly and successfully. Special thanks go Mr. Muwindwa Lipalile, my site supervisor, for his significant input into this research work.

My sincere gratitude and special thanks go to Konkola Copper Mines (KCM) Plc management for the sponsorship opportunity accorded me to carry out my studies in conjunction with the University of Zambia - School of Mines and the Directorate of Research and Graduate Studies. Special appreciation goes to Nchanga Underground Mine management and staff for technical support and always ensuring that I was always accorded time as and when required to be in session at the University. Key to this was the role played by the personnel at KCM Central Training Department.

My cordial tribute goes to my wife and friend, Lilly, for her support and enduring care and attention especially for our children in my long absence from home.

Lastly, but not the least, I wish to thank all individuals that have played a role in one way or the other in rendering help towards this research work.

Abstract

Nchanga Underground Mine applies a variant of block cave mining to exploit the massive copper ore deposits with generally low ore grades fairly distributed within the orebody. Mining activities date as far back as 1937 when underground production commenced. Traditional block caving mining method is in use where the Jackhammers are used to drill the rock. Jumbo drifter machines are then used to drill long holes for caving. The ore is then caved and channeled through a series of transfer chutes and later hauled to shaft hoisting facilities to surface.

Block cave mining has proved to render a solution to massive orebody mining worldwide especially with low grades due to, mainly, its low mining costs (i.e. high profitability), high productivity per man-shift and generally safety advantages as one 'does not re-enter' the caved stopes. However, the major challenge faced with block cave mining is ore dilution as waste rock gets caved along with the ore.

The Nchanga sedimentary copper orebody comprises mainly of two major limbs of the orebody, i.e. the Lower Orebody (LOB) and the Upper Orebody (UOB). The minor limb, the Intermediate Orebody (IOB), exists between the LOB and UOB in some areas but is generally very thin. The two are separated by a weak but thick layer of banded sandstone (BSS). The upper orebody is made up of 'The Feldspathic-Quartzite' (TFQ) copper mineralized rock unit. During extraction of the ore, the LOB is drawn up to the point where the recovery reaches the BSS material. Once all the production drifts reach their draw into the BSS layer, the block is declared exhausted. This leaves out the copper mineralization in the TFQ above the BSS.

This project intended to examine the ore potential in the TFQ copper ore above the BSS layer as well as the BSS tonnage to be removed to recover this remnant ore. This has been achieved by reviewing the ground conditions of the old exhausted areas (rock mass characteristics), support requirements as well as draw parameters to enhance optimized recovery of ore left in the TFQ.

An economic analysis of the project has shown that a breakeven copper market price of US$5,737.22 per tonne above which the project becomes viable has been determined. The project objective to optimize draw of 'TFQ' ore in the caved areas of Nchanga has been achieved: BSS pulling to commence with from down-dip drifts and end with up-dip drifts in a block. The project should be embarked upon speedily to take advantage of the currently high copper prices.

Contents

List of Figures

List of Tables

Abbreviation and Acronyms

AFW	-	Assay Footwall
AHW	-	Assay Hangingwall
ARK	-	Arkose
ASCU	-	Acid Soluble Copper
AICU	-	Acid Insoluble Copper
BSS	-	Banded Sandstone
BSSL	-	Banded Sandstone Lower
BSSU	-	Banded Sandstone Upper
CDOL	-	Chingola Dolomite
CRO	-	Chingola Refractory Ore
DOLSCH	-	Dolomitic Schists
DORS	-	Dynamic Ore Reserves System
DRC	-	Democratic Republic of the Congo
E	-	East
eqn	-	equation
GSI	-	Geological Strength Index
HR	-	Hydraulic Radius
IOB	-	Intermediate Orebody
KCM	-	Konkola Copper Mines
LBS	-	Lower Banded Shale
LOB	-	Lower Orebody
LOM	-	Life of Mine
MRMR	-	Mining Rock Mass Rating
NGI	-	Norwegian Geotechnical Institute
NOP	-	Nchanga Open Pit
NRG	-	Nchanga Red Granite

NUG	-	Nchanga Underground
PQ	-	Pink Quartzite
RD	-	Rate of Damage
RC	-	Rate of Caving
RMR	-	Rock Mass Rating
RMS	-	Rock Mass Strength
RQD	-	Rock Quality Designation
RU	-	Rate of Undercutting
SG	-	Specific Gravity
SM	-	Shale Marker
SRF	-	Stress Reduction Factor
SWG	-	Shale with Grit
TCU	-	Total Copper
TFQ	-	The Feldspathic-Quartzite
TR	-	Transition
TR ARK	-	Transition Arkose
UBS	-	Upper Banded Shale
UCS	-	Uniaxial Compressive Strength
UOB	-	Upper Orebody
URD	-	Upper Roan Dolomites
W	-	West

CHAPTER 1

1.0 INTRODUCTION

1.1 Location and Background

Nchanga mine is one of the Konkola Copper Mines (KCM) Plc operated mines in Zambia. It is located on the Copperbelt Province in the town of Chingola which is about 410 kilometers North of Lusaka, the Zambian Capital City. KCM is a Zambian based company where major shareholder is VEDANTA Resources Limited of India which is a London Stock Exchange listed company. The company also runs other mines and business entities in Zambia such as the Konkola Mine in Chililabombwe, Nampundwe Mine in Lusaka West, Nchanga Open Pits in Chingola, Nkana Refinery in Kitwe and the newly commissioned Nchanga New Smelter in Chingola. KCM endeavours to be a world class producer of copper and cobalt by embracing vigorously the principles of doing business in the areas of good safety, health, environmental and quality practices as well as extending a corporate social responsibility in the communities cost effectively, (KCM Policy, 2010). Geographically, the Copperbelt region is located 12 degrees south of the equator and 28 degrees east of the Greenwich meridian and borders with Democratic Republic of the Congo (DRC). The Katanga Province or region of DRC shares the border with Copperbelt. The geographical location of the Copperbelt is as shown in Figure 1.1.

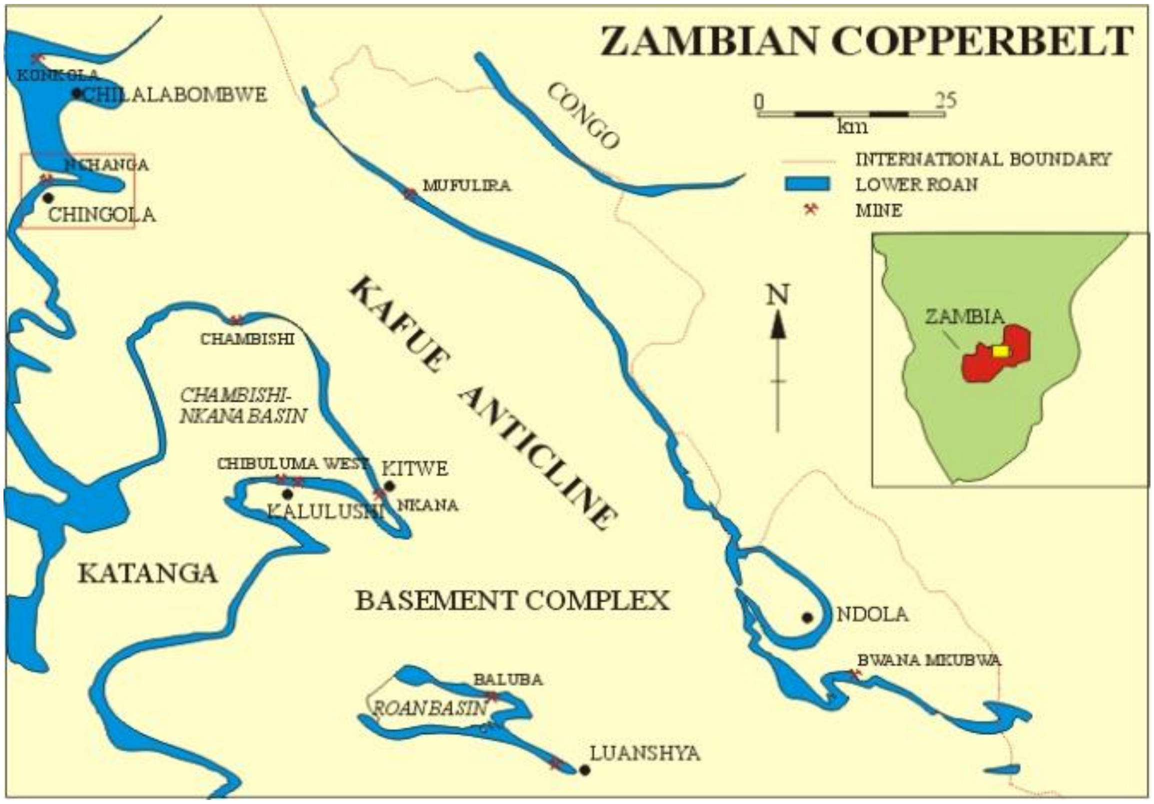


Figure 1.1: Location of Zambian Copperbelt (Gerrad, 1995)

Mining activities in the Nchanga area date as far back as 1931 following the discovery of the copper orebody in 1923 near Nchanga stream. The prospectors, James Beaton and Andrew Osterburg of Rhodesia Congo Border Concessions, led in this discovery of copper. The ore mineralization exists as both sulphide and oxide in mixed forms. The initial mining activity was done using surface mining methods, i.e. through open pits. This mining method is a method in which overlying waste rock material is removed and dumped somewhere to expose the ore. With this exposure of the copper ore and the dipping trend of the orebody exposed, extensive exploration works were carried out to establish the extent of the orebody. The orebody was found to be dipping in the south-north direction and continued along dip to a depth of about 1,000m. Therefore, it became clear that future mining to realize the copper ore would entail mining at depth, i.e. through underground mining methods down to a thousand metres.

1.2 Climatic Conditions

The climatic conditions experienced on the Copperbelt Province are generally subtropical and seasonal. These climatic conditions conform to overall climatic conditions in Zambia. The following are the distinct seasons prevailing annually [2]:-

- Rainy season, (i.e. generally warm and wet with temperatures in the range of 16˚C to 27˚C); runs from November to about mid April;
- Winter, (i.e. cool to cold with temperatures in the range of below10˚C to 23˚C); runs from mid April to about mid August and finally and
- Hot and dry season, (i.e. with temperatures in the range of 26˚C to 38˚C); runs from mid August to October.

In view of these climatic conditions, underground mining operations are generally not affected throughout the year. Only in the rainy season is there a minimal anticipation of rainy water seeping through subsidence sinkholes or cracks. This might result into a mud rush in working areas close to surface. Safety measures by way of weirs (barriers) are always installed in the haulages before the rainy season to hold back any influx of mud.

1.3 Ore Deposits

A total of 14 known conventional orebodies exist within the Nchanga Mining Licence Area. These occur at seven different horizons in a variety of sedimentary rocks. There are three superimposed stratiform orebodies locally known as Lower Ore Body (LOB), Intermediate Ore Body (IOB) and Upper Ore Body (UOB). These orebodies extend over a vertical span of 150m on the stratigraphic column as shown in Figure 1.2. The Lower Ore body is the one currently being mined through Block Caving mining method.

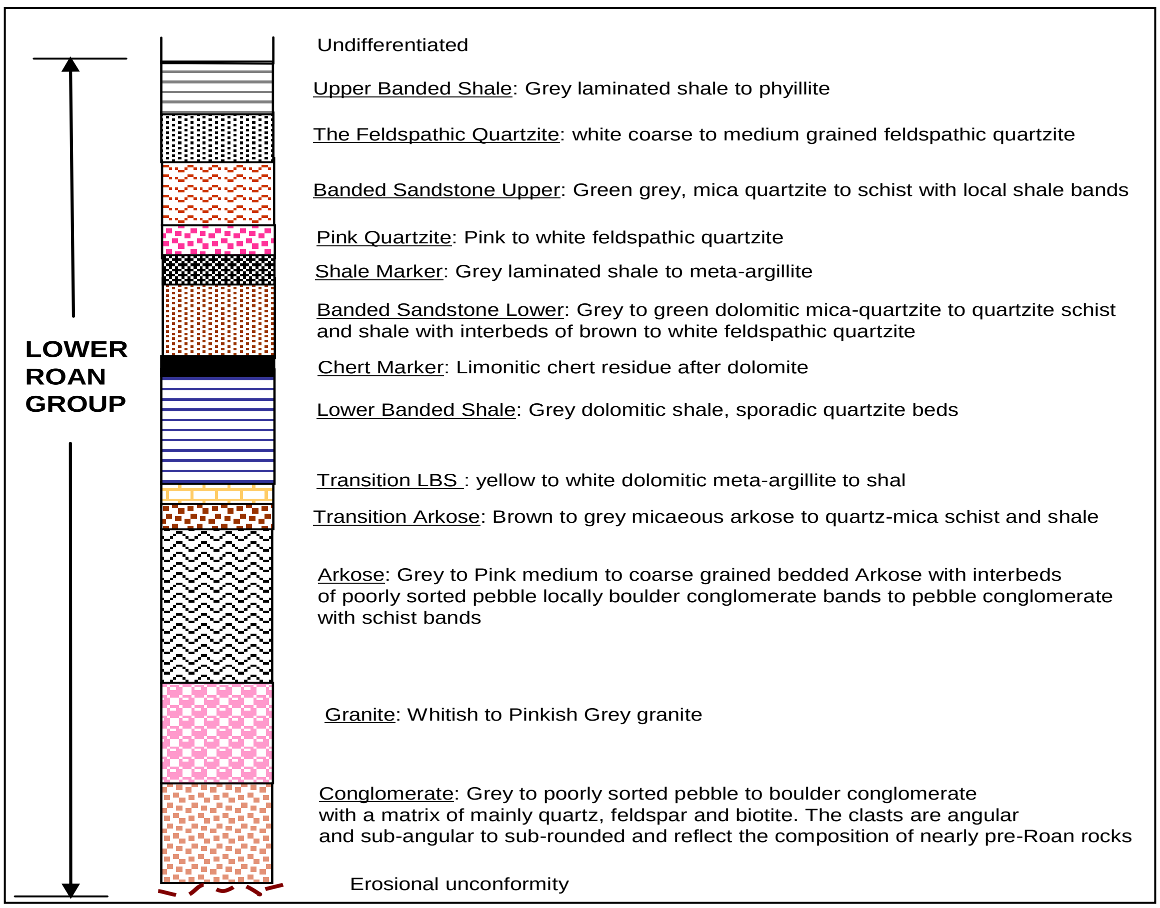


Figure 1.2: Stratigraphy of the Lower Roan Group within the Nchanga Syncline (Gerrad, 1995)

Three other horizons contain copper in a micaeous form. These are famously known as Chingola Refractory Ore (CRO). Copper distribution is widespread over the vertical range on the Copperbelt with most deposits confined to between one and three stratigraphic horizons. Over 20 million tonnes of copper deposition is found in these orebodies with a significant presence of cobalt as well before any exploitation. The copper deposits are mainly on the northern side of Chingola town with few other deposits on the southern and western side of the town. The orebodies include the following, (i.e. by local names); River Lode, Luano, Chingola (‘A’, ‘B’, ‘C’, ‘D’, ‘E’ & ‘F’) deposits, two Mimbula deposits, Fitula and the Nchanga main orebodies comprising the Lower Ore Body (LOB), Intermediate Orebody (IOB) and the Upper Ore Body (UOB) also known as ‘The Feldspathic-Quartzite’ (TFQ). Figure 1.3 shows the three Orebodies, (i.e. UOB, IOB and LOB) locations:

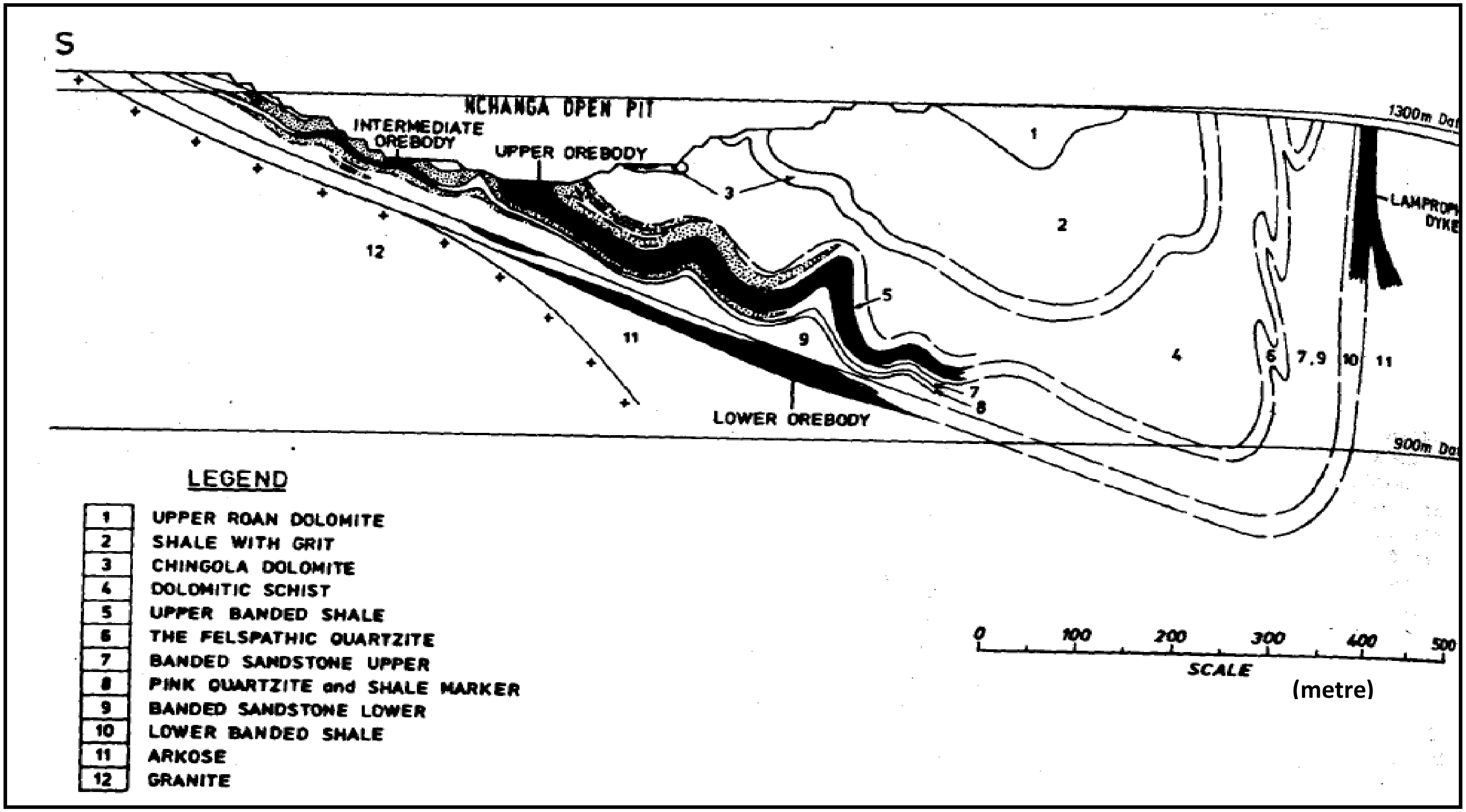


Figure 1.3: Showing the Three Orebodies, (i.e. UOB, IOB and LOB) of Nchanga mine (NUG Training manual, 2000)

1.4 Geology of the Zambian Copperbelt

Generally, the Zambian Copperbelt is hosted by the Neoproterozoic Katangan Supergroup bassinal succession. They are of predominantly marginal marine and terrestrial meta-sedimentary rocks. The basal sequence portion being Lower Roan Group consisting of continental sandstones and conglomerates. They are deposited in series of restricted sub-basins controlled by extensional normal faults and folding (Selley and Broughton, 2005).

1.4.1 Stratigraphy

The stratigraphic rock unit column of the Chingola mining area with rock unit descriptions comprises the Basement Complex through to the Upper Roan Group as illustrated in Figure 1.2:

1.4.2 The Succession

The rocks present in the Nchanga Mining License Area range stratigraphically from the Basement Complex to the Lower Kundelungu with copper deposists occurring in the Lower Roan subdivision of the late Precambrain Katanga sequence consisting in ascending order of the Roan, Mweshia and Kundelungu Groups (Selley, and Broughton, 2005).

An account of the stratigraphy in ascending order is as detailed below based on Gerrad (1995):

- Basement Rocks. The contacts observed to-date at Nchanga between Katanga Sequence and the Basement Complex are erosional. Lufubu Schists form part of the Basement Complex. Various granites like Nchanga Dark Granite, Nchanga Grey Granite and Nchanga Red Granite intrude the basement complex. These form part of a major structural ridge which influenced sedimentation of the Katanga as well as the Lufilian orogeny.

- Arkose (ARK). This rock unit is a cross-bedded coarse grained rock with interbeds of poorly sorted pebble locally boulder conglomerate bands to pebble conglomerate with schist bands. Principally arkose is composed of quartz, feldspar and sericite.

- Transition (TR) is a friable and soft incoherent rock solid recognizable Lower Banded Shale and Arkose. The rock unit has been divided into two members, i.e. the lower member known as Transitional Arkose (TR ARK) with the incoherent limonitic, micaceous, kaolinised arenites and the top member known as Transition lower Banded Shale (TR LBS) consisting of the bleached and silicified siltstone and shale. The TR LBS is an alteration of the Lower Banded Shale (LBS).

- Lower Banded Shale (LBS) is the equivalent of "ore shale" elsewhere in the Copperbelt. It is dominantly a black, carbonaceous, laminated, dolomitic shale but can be a grey, silty argillite to yellowish finely laminated quartzite. The formation marks the lower hanging wall aquifer above the orebody.

- Shale Marker (SM) or Chert Marker - overlies the Lower Banded Shale. It is a thin discontinuous layer of porous micro-crystalline quartz with muscovite flakes and hydrous iron-oxide.

- Banded Sandstone Lower (BSSL) in the Nchanga-Chingola region is divided in two parts by a lower shale marker and an upper Pink Quartzite (PQ) which contains, in a limited area, the Intermediate Orebody (IOB). The banded sandstone lower is composed of thinly bedded feldspathic sandstones and siltstones and a few feldspathic quartzites.

- Pink Quartzite (PQ) is a creamy faintly pink arkose flanked by less competent beds. It is a well-sorted feldspathic quartzite.

- Banded Sandstone Upper (BSSU) generally contains more vermiculite than the banded sandstone lower which is more dolomitic. Tectonic deformation of the BSSU developed a strong schistose character with a local growth of the metamorphic mineral scapolite. This unit forms the top limit of the lower hanging wall aquifer.

- The Feldspathic Quartzite (TFQ) - is a relatively competent rock formation consisting of well-sorted, medium-sized grains of quartz, microline and orthoclase. Cross bedding is wide spread. Its upper part is sometimes interbedded with dolomitic schist. TFQ forms part of the Upper Ore Body (UOB) of the Nchanga ore body.

- Upper Banded Shale (UBS) is a grey, laminated and finely bedded quart-microline muscovite-vermiculite shale or phyllite with sporadic carbonaceous material and in places a dolomitic cement. Its extent ranges from 15 to 40m.

- Dolomitic Schists (DOLSCH) occur above the ill-defined boundary with the upper banded shale with an alternating sequence of micaceous, quartzose dolomites, dolomitic mica-schists and argillites.

- Chingola Dolomite (CDOL) is a distinctive succession of light coloured quartzose and talcose dolomites. Also present near the top 2.5m is light green talc-sericite schists. This rock unit marks the start of the upper hanging wall aquifer.

- Shale with Grit (SWG) and Upper Roan Dolomites (URD) are predominantly argillaceous with zones of gritty, fine to medium grained feldspathic sandstones, scattered grits and succession of interbedded dolomites, sandy siltstones and argillites respectively. The Upper Roan Dolomite forms the top limit of the upper aquifer.

1.4.3 Structures

The prominent geological structures in the Nchanga Licence Mining Area are the famous Nchanga Synclines and Anticlines. The Lower Roan Group strata comprising, as mentioned in section 1.4.1, mainly arkose, wacke, quartzites, shale micaceous banded sandstone, feldspathic quartzite and schist is folded into major northwesterly plunging, asymmetrical synclines and anticlines draped round a hub of red granite. The three principal structural elements being the Nchanga Syncline, the Chingola Anticline and the Mimbula-Chabanyama Syncline. The two largest Nchanga orebodies are on the gently dipping south limb of the Nchanga Syncline. The Nchanga syncline plunges or dips in the north-south direction striking in an almost east-west direction. In the strike direction the orebody extends from about 13E (east) mine position to 21W (west) mine position taking the main Underground shaft at zero position. These mine direction positions are actually section divisions of the mine with each section averaging a length of 120m in the east-west direction. Figure 1.4 shows the limbs of the Nchanga orebody.

The other geological structure in the mining area is the Nchanga fault. The fault is believed to have resulted into the Nchanga stream which was passing through across the main Nchanga Open Pit (NOP). The Nchanga fault zone continues all the way from surface to 3020 feet working level of Nchanga Underground mine (NUG). The influence of the fault zone poses serious ground support challenges when ever mining activities underground are near the fault zone. Excavations mined in these conditions collapse. Other structures include joint sets and bedding planes.

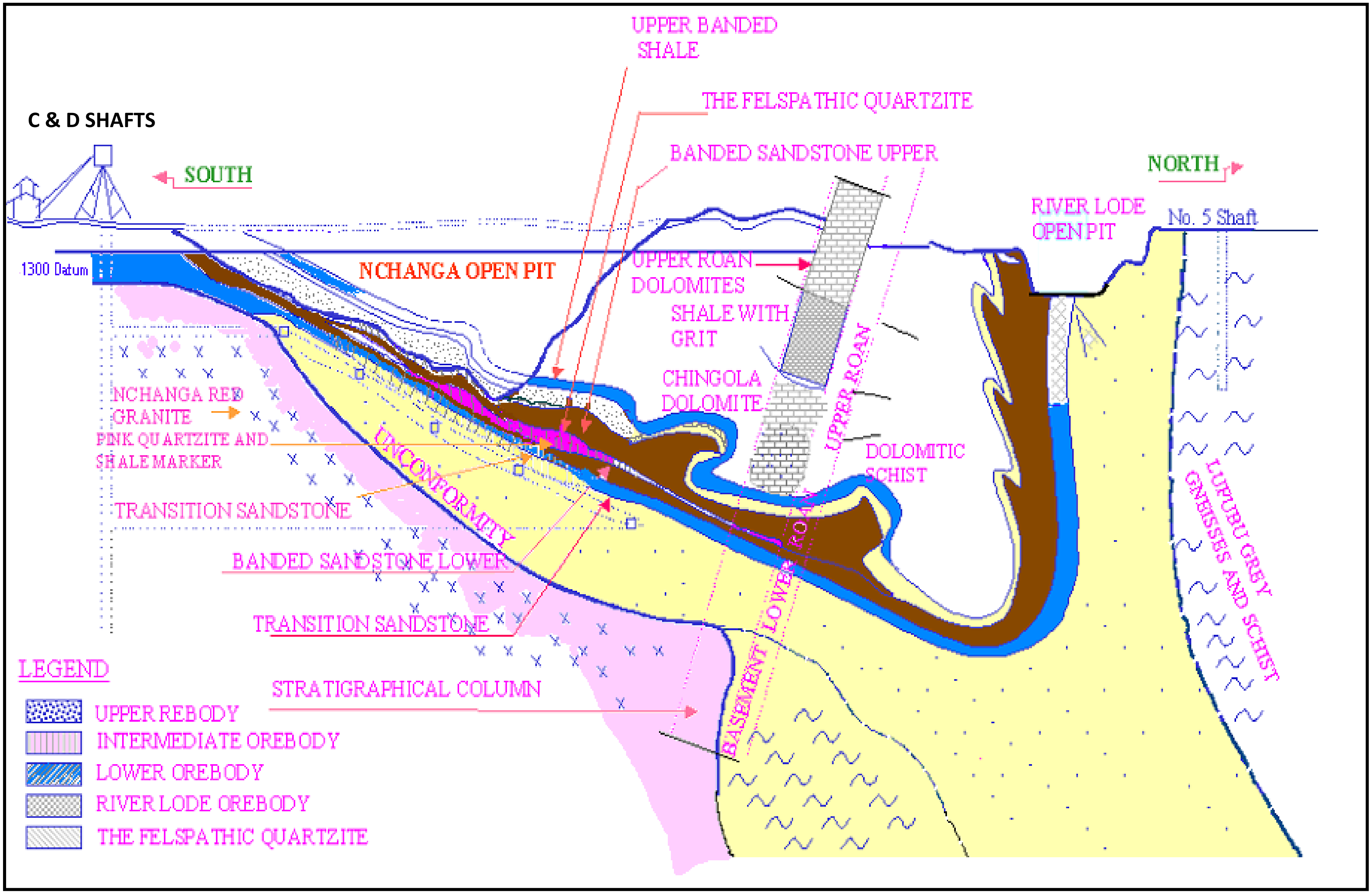


Figure 1.4: Showing the Nchanga Orebody Limbs (NUG Training manual, 2000)

1.5 History of Nchanga Mining Activities and Current Operations

The first mining activities, after the discovery of copper ore, date as far back as 1923. Initial mining was through a surface mining method known as Open Cast mining which later turned into Open Pit mining method. Due to the nature of the ore dipping in the range 15˚ to 30˚ further access to the copper ore at depth meant an underground mining method would be used. Returns realized from the copper mined through the Open Pit mining activities were used to fund sinking of Shafts and main haulages to access the orebody underground.

Block Caving is the main mining method employed at Nchanga Underground mine with Sublevel caving in some sections. The other mining method employed on a near surface orebody is Room and Pillar in the Chingola "B" area. Block caving takes advantage of gravity flow of generally weak ore by undercutting it using long-hole drilling into the ore and blasting.

In the initial underground mining trials, mining methods progressed from Top Slicing, Dip Scraping and then Block Caving. Due to numerous difficulties encountered with these initial mining methods coupled with high costs and low production outputs, the methods were abandoned and Block Caving was adopted. This mining method is being used to date. Part of the UOB and IOB have been mined in the range of about 500 metres below surface mainly in the Nchanga Open Pit (NOP) while the LOB has been mined through the Block Caving mining method in the NUG mine. The LOB is nearly exhausted with only about 5 years remaining in the life of the mine.

1.6 Problem Statement

NUG is faced with a short Life of Mine (LOM) currently, i.e. about 5 years. Once the mine closes the labour force, (i.e. about 3,000 permanent and contractor employees) working for the mine will lose employment. In the quest to extend LOM, the mining department at NUG has embarked on extensive initiatives and efforts to search for extra copper ore resources. Some of these efforts include going back to old areas which were not exhausted due challenges faced in the past like weak ground condition and influx of water. Mitigation measures are being adopted to overcome these challenges. One special opportunity involves 'The Feldspathic Quartzite' (TFQ) ore. TFQ occurs above the banded sandstone (BSS) in the LOB.

In the process of mining the LOB through Block and Sublevel caving at NUG mine, extraction of copper ore ends immediately the draw reaches the BSS lower rock unit shown in the stratigraphic column (Figure 1.2). The LOB comprises the following copper mineralized rock units, as shown in Figure 1.2: Arkose, Transition and Shale. This means that the other copper mineralized rock units above the BSS remain un-tapped or undrawn. The Feldspathic Quartzite rock unit hosts this copper ore resource for this un-tapped ore.

In the quest to increase ore resources for the company, TFQ mining has become an important factor as it would extend the life of mine once it is accessed. This research work seeks to evaluate the TFQ ore and provide an economical way to extract it.

1.7 Research Questions

The research questions that require to be addressed in this project are:-

- What copper ore grades and tonnages exist in the TFQ ore?
- Are the old working areas stable enough for safe extraction of the remnant ore, TFQ?
- How much of the banded sandstone (BSS) should be removed before any TFQ ore can be extracted economically and how (draw parameters)?

The answers to these questions set the basis for the project research work.

1.8 Aim and Objectives

The project aims and seeks to optimize recovery of ‘The Feldspathic Quartzite’ ore in the caved areas of Nchanga Underground Mine. The following objectives will help achieve this aim:-

- To review the geological setup (structures, features) in the caved areas;

- To establish and determine the ore tonnages to be recovered ;
- To establish ore grades in the Nchanga caved areas of interest of TFQ ore;
- To examine rock mass characteristics in the caved areas and ground conditions;
- To establish the drawpoint parameters and
- To conduct an economical analysis of the whole project to ascertain viability.

1.9 Description of the Research Site

The project research site was Nchanga Underground in the mine's caved and exhausted areas with high presence of copper mineralized rock units. These rock units are above the LOB, i.e. TFQ ore. Out of the total number of these caved areas with the TFQ ore, the research work had to segregate the blocks to be exploited on grounds of the TFQ ore tonnages and total copper ore grades obtaining in each block. Very low copper grade TFQ ore is uneconomical to recover. Therefore blocks underground were segregated on the basis of economic viability.

1.10 Justification

Life of Mine for NUG is a source of concern not only to management but to every employee. The question in mind is, "What happens after the year 2016/17 when the LOB ore is completely exploited? Close the mine?"

The future of Nchanga underground mine is an important aspect of consideration by every worker at the mine. Potential alternatives, in terms of extra ore resources at the mine, offer great hope for every underground employee as this would result in extended life of the mine. Extra ore resources would offer investors continued business opportunity. As for the employees this will be a source of sustained employment while to the government of the Republic of Zambia it will be a source of revenue through income and other mineral taxes which the investor pays to government.

CHAPTER 2

2.0 LITERATURE REVIEW

Nchanga Underground (NUG) mine uses Block Caving mining method to exploit the copper ore in its Lower Ore Body (LOB). The mining method utilizes the principles of caving. This takes advantage of the caveability of the ore material once it is undercut and its ability to cave and propagate caving onto the overlying layers of the ore. The general mechanism of the Block Caving mining method entails initiation and propagation of a caving boundary in both the orebody and overlying rock strata. This requires creation or generation of an undercut of sufficient width and height to induce failure in tension and shear. This results in progressive failure/collapse of the undercut crown causing the broken rockmass to fall by gravity into the undercut. The general principle is that the magnitude of the induced tensile and shear stresses should exceed the rock strength for failure to occur.

It is worth noting from the onset that caving is one of the low-cost and effective underground mining methods. This is more so when the drawpoint spacing, size and handling facilities are designed to suit the caved material, (i.e. in terms of fragmentation) and the extraction horizon is or can be maintained for the life span of the draw. Cave mining operations are generally associated with high daily production output around the world, (Laubscher, 2004).

2.1 Caving at Nchanga Underground Mine

Continuous caving mining system has been used from the early days of Nchanga underground mine to date. The mining approaches used, however, kept evolving and changing from one form to the other in the quest to achieve possible maximum production outputs at low cost. The orebody naturally caves on its own once the failure is initiated below by an undercut and rock falls by gravity onto a system of draw points called finger raises along a series of scraper drifts. This is illustrated in figure 2.1.

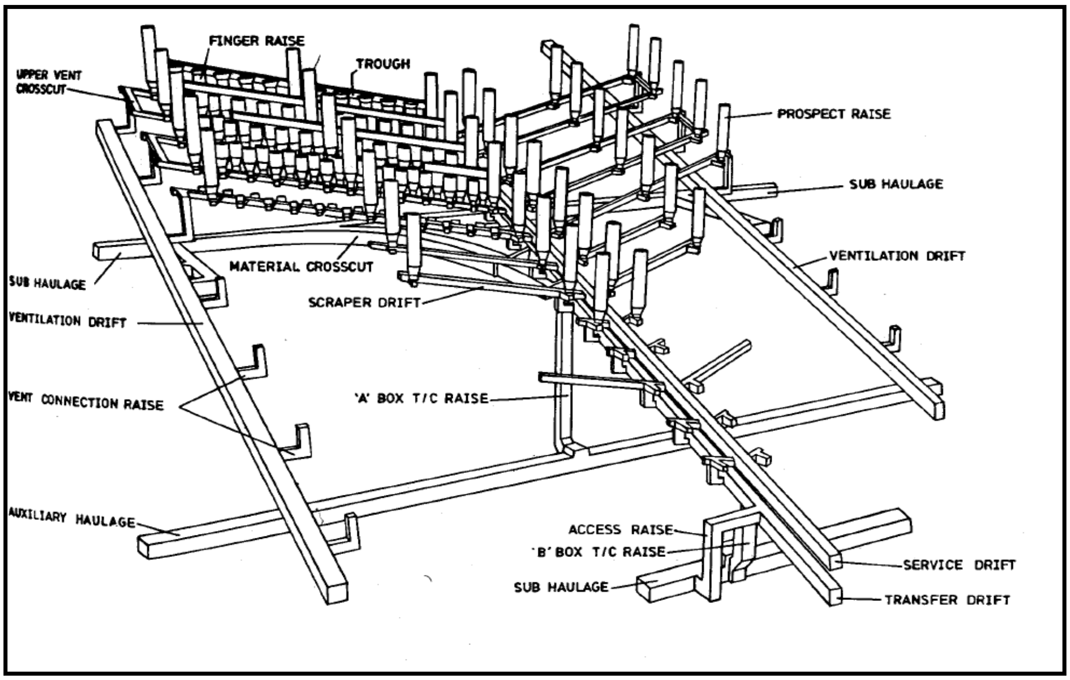


Figure 2.1: Nchanga Underground Mine Block Caving Layout (NUG Training manual, 2000)

The ore is then scraped into other draw excavations via loading boxes onto locomotive wagons enroute to shafts for hoisting to surface. A series of development phases is followed from primary development, (i.e. mining of bigger excavations such as haulages, sub-haulages and material cross-cuts) to secondary development (i.e. mining of transfer drifts, service drifts, scraper drifts, draw points and the undercut excavations). These set out the infrastructure to handle the ore right from the draw points all the way up to shafts and hoisting to surface. The major challenge being ore-waste segregation as ore and waste use the same collection points. This results in dilution of ore.

Variant caving mining methods were employed at Nchanga mine (Chileshe and Phiri, 1994). These include Top Slicing, Dip Scraping and the current Block Caving.

2.1.1 Top Slicing

Under this method, the copper ore was extracted in a series of timbered slices. Production started from the hangingwall of the orebody and progressed downwards into the subsequent slices below the timber mat made on the footwall of the slice. This is shown in Figure 2.2.

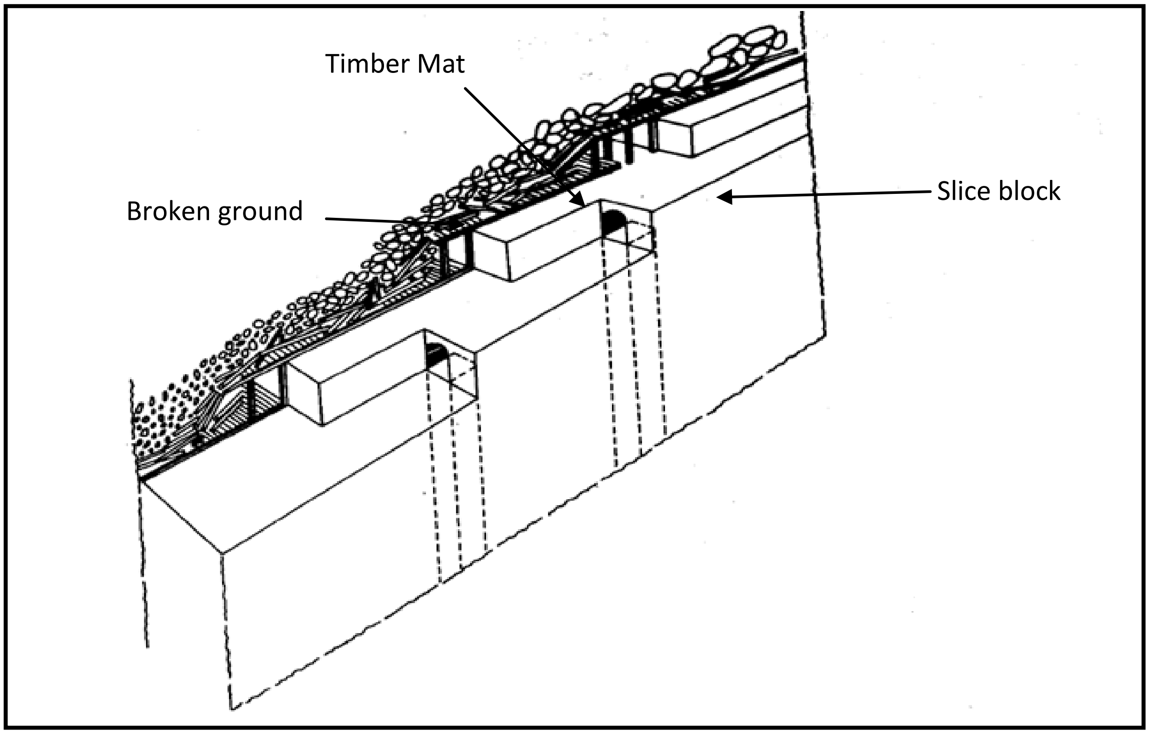


Figure 2.2: Top Slicing mining method (NUG Training manual, 2000)

Production was realized from some trial stopes. Generally, Top Slicing mining method was characterized by low production tonnages. The method was highly labour intensive. Timber consumption was high. Difficulties in ventilation and soft ores encountered posed a lot of challenges in premature rock falls or failure. The method proved too costly and was abandoned.

2.1.2 Dip Scraping Mining

This method made use of undercutting by blasting long holes drilled into the ore and then the ore caves and flows by gravity. Large working sections were planned with haulage levels at 90m below the assay footwall, 13m apart vertically and 50m horizontally. The ground was scraped directly from scraper drifts into the underground locomotive wagons or cars stationed in the haulage below. The drawpoints (i.e. finger raises, mined vertically) were positioned at 5m apart, on the sides of the scraper drifts. These finger raises were coned out near the ore elevation to increase their catchment span between them, i.e. to a diameter of 2.3m. The pillars between the raises were blasted in the final round to completely undercut the ore, allowing it to cave into the drawpoints. Figure 2.3 shows a scraper drift section in Dip Scraping mining.

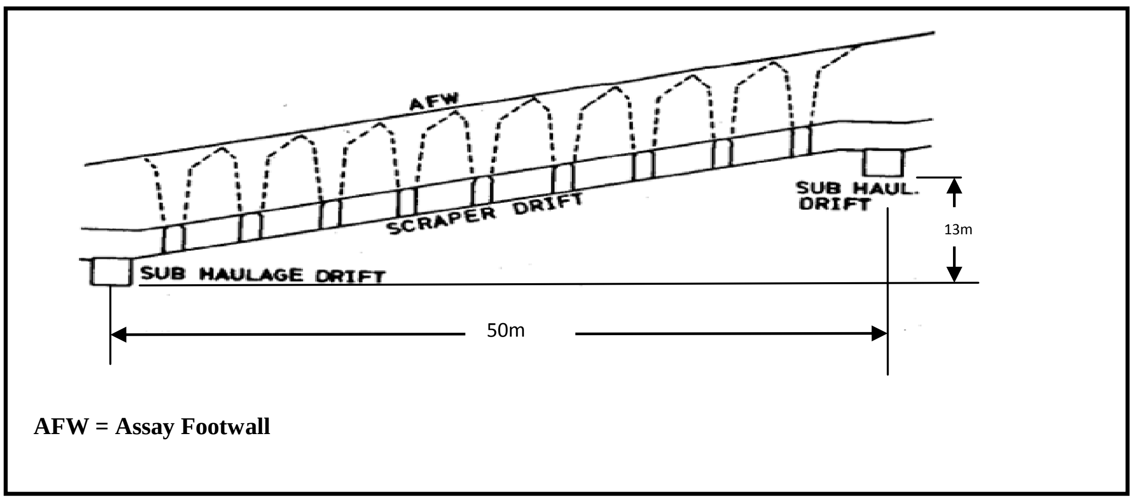


Figure 2.3: A Scraper Drift Section in the Dip scraping (NUG Training manual, 2000)

Production restrictions in this mining method resulted in low production outputs. This was due to scraper units' low outputs, break downs and lack of storage facilities as direct tipping was used into the wagons. Further delays came from poor positioning of wagons/cars in the haulages, delays in waiting for locomotives to arrive from tipping ore into box loading facilities, production delays and poor positioning of the scraper drifts during development phases. These setbacks resulted in the modification of the Dip

Scraping system introducing new excavations (i.e. grizzly drives). The grizzley drives were mined along the orebody strike linking all scraper drifts on one elevation and haulage positioned at 13m lower in the footwall. The loading boxes were also increased in number giving the system temporal storage capacity especially before the locomotive cars were positioned below for loading.

These modifications increased production output as each drift had temporal storage capacity. Adversely, the increase in the number of loading boxes meant increased maintenance costs. To resolve the problem, a transfer drift was mined on strike below the scraper drifts position. It served a number of drifts directly below the grizzly drive and linked each scraper drift by a sub-transfer chute into a single steel loading box. Each box was equipped with a box raise on the side mined for providing service access to the steel loading box and the box chute. Therefore the maintenance costs were controlled and production output increased tremendously.

These modifications only resulted into the current Block Caving system in use at Nchanga underground mine. Figure 2.1 illustrates the current Block Caving mining method.

2.1.3 Block Caving Mining Method

The block cave mining as practiced at Nchanga mine will be highlighted in this section.

2.1.3.1 Layout and Location of Excavations

Figure 2.1 shows a layout plan of the current Block Caving mining method as practiced at Nchanga Underground mine. The main access ways, haulages, equipped with tracks and all other service accessories like compressed air and water pipes, electrical cables and water drains are mined at approximately 45.42m from the assay footwall (AFW). The haulages are generally 4.3m wide and 3.9m in height, i.e. main haulages, sub-haulages and auxiliary haulages. Material crosscuts branch off from the sub-haulages to various sections and reduce in size to 4.3m wide by 3.0m high but at the same elevation. A pair of haulages is mined on main levels from the shafts to production working areas to provide easy travelling of the underground locomotives to and from the shafts using respective haulages in each direction. Sub-haulages are mined along strike of the orebody while service drifts are mined along dip. Service drift orientation follows the dip of the orebody in each section or mining block. The service drift is positioned 10.6m below the AFW and 120m apart on strike between two sections. A network of scraper drifts (i.e. herringbone system) with a set of 13 finger raises on each side is used to collect caved and broken ore material. This material is transferred into loading boxes.

An undercut drive is mined at a middling of 1.8m below the orebody above finger raises on the southern side of the scraper drift. This middling is left to prevent the weak ore from collapsing before being caved. It is from this drive that rings of long blast holes are drilled into the orebody and blasted to create an undercut to pave way for ore to cave and fall by gravity into the drawpoints. Among the northern finger raises in a scraper drift, three raises (i.e. one near the service drift, one mid-way and third one at end of scraper drift) are used as prospect raises. They are mined until they intercept the ore to provide height at interception and geology of the ore in terms of copper ore grades for layout confirmations. Slot crosscuts are mined from the trough drive towards these three prospect raises to hole into them. These slots provide breaking faces during blasting of the rings of long blast holes into the ore to undercut the ore. The generated ground from these blasts known as ‘Swell’ is scraped after each ring of long holes is blasted to create a void to allow further ring blasting to be conducted and prevent choke blasting.

As undercutting is completed, caving of the transition and shale beds commences. Figure 2.4 shows the major drives mined in the block.

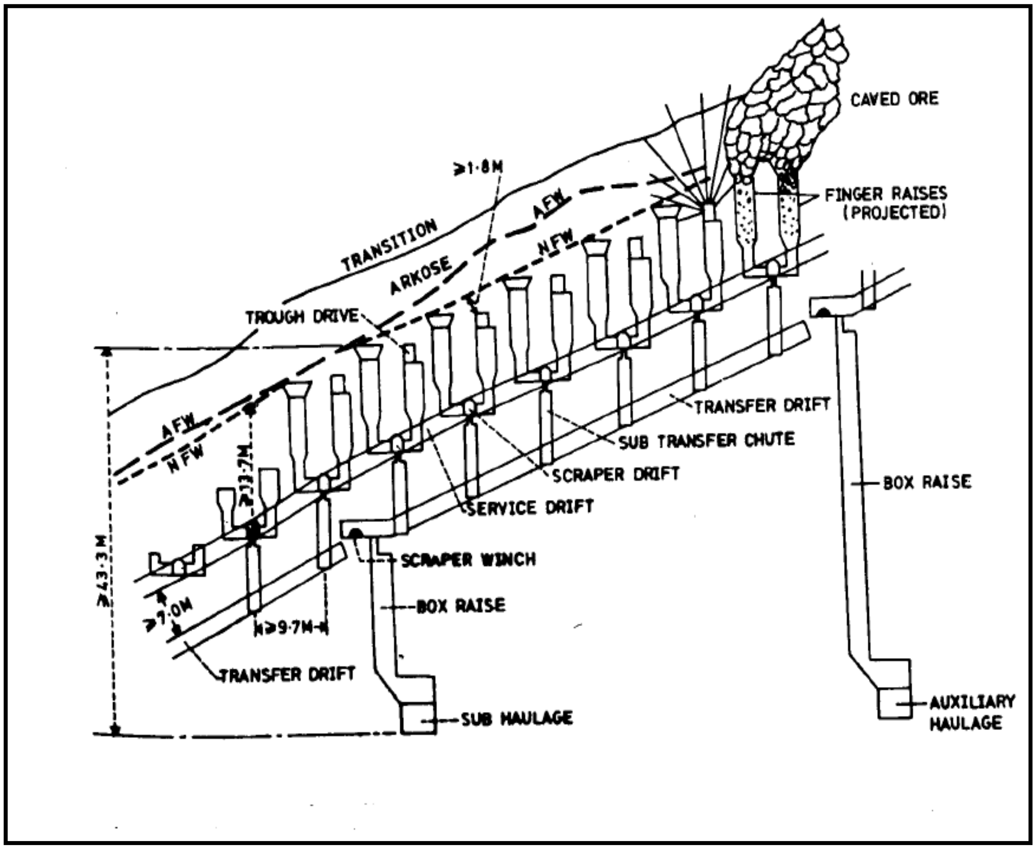


Figure 2.4: Major Drives Mined on the Block (NUG Training manual, 2000)

Positioning of the layout below the assay footwall of the orebody is very critical in achieving good recovery of the copper ore. If it is placed too far below the AFW, excessive dilution would be encountered from the increased middling (waste rock) between the ore and the trough drive. Again placing a layout too close to the orebody would result in loss of ore tonnages as it might fail and collapse before being fully undercut and caved to achieve good rock fragmentation for good flow of ground. This is because the ore (i.e. caving ground) is generally weak and stress equilibrium disturbance would result in ore failing and collapsing with poor fragmentation. Where this occurrence has been experienced in the mine, it resulted in excessive use of explosives to break the large chunks of collapsed rock through secondary blasting (Chileshe, 1992).

Limiting factors in terms of where to place the trough drive normally are obtained from the analysis of the prospect raises as they will show the natural footwall (i.e., lowest elevation at which copper mineralization is present within the Arkose) which has been geologically established. The trough drive is positioned 1.8m below in competent ground. The orebody stretches on the western side of the mine from 1W to 12W (west) while on the eastern side from 1E to 12E (east) mine positions in the lower levels. In the upper levels, it stretches from 1W to 21W mine position.

For each section, after the access ways have been developed, diamond drilling is done to establish dip angles of the orebody, assay footwall locations and the presence of water aquifers. Figure 2.5 shows locations of the diamond holes in relation to the haulages and orebody.

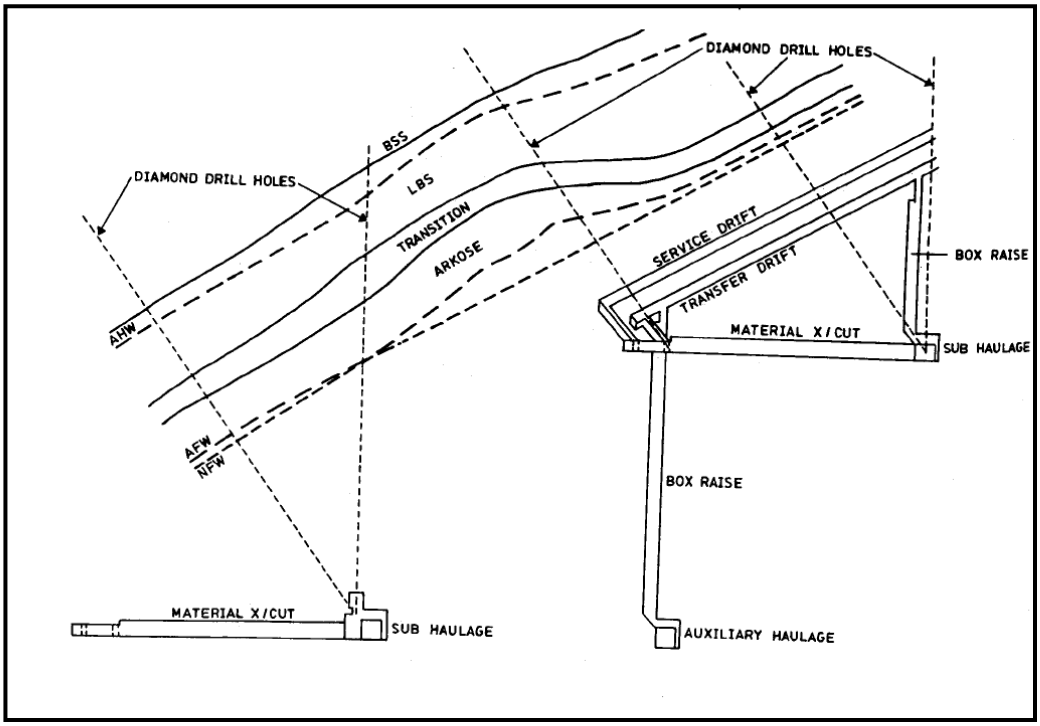


Figure 2.5: Idealized Locations of diamond drillholes (NUG Training manual, 2000)

2.1.3.2 Production Control

Production control involves physical inspection of the ore material flowing in the finger raises on a shift basis and collection of ore samples. The samples are analyzed in the Laboratory to ascertain copper content at every stage of draw. Physical inspection is done cave control officers from planning section of the mine. This physical inspection enables the officers to put off raises which advance in geology ahead of other raises in the same scraper drift. After the orebody is caved, the waste middling between the undercave excavation and the ore results in waste reporting in the finger raises, i.e. swell waste. This waste is cleared, (i.e. swell waste pulling) until all the finger raises reach in ore. The swell pulling phase would ensure that all the finger raises are in ore and all the waste from the middling ground is scraped off. The drifts then are put on production to extract all the caved and fragmented ore above. Strict physical inspection and sampling of finger raises are done during all production shifts by the cave control section to ensure the ore is drawn uniformly in all the drawpoints. This is done to prevent dilution through early introduction of the banded sandstone layer above the ore beds. The cave control section plays a vital role in monitoring tonnages drawn from each scraper drift and makes production forecasts monthly depending on the draw percentage achieved in each drift. Production scraping tonnages allocated to each drift monthly are calculated based on the reserve percentage drawn for each drift. For instance, until swell tonnage (i.e. 30% of the reserve) is pulled from a drift, only 8% of the ore reserve is pulled per drift per month. This translates to a maximum of up to 2,000 ore tonnes in each drift per month. Then from 30% to 100% draw, 2,500 ore tonnes reserves are pulled per month in each drift. When the draw goes above 100% in a drift, a maximum of 1,000 tonnes per month is pulled per drift. This reduction in tonnage is meant to avoid eventual exposure of the banded sandstone with potential to dilute any remaining ore in the drift. Section profiles showing the cave for each drift are made on a monthly basis. Each finger raise's geology changes during the draw period is constantly updated and recorded. The draw section profiles are sketched on assumption that the ore-waste interface will not move unless 30%, swell fraction of reserve tonnage has been pulled from the drift. The ore-waste interface should be kept near horizontal level to avoid early dilution. If ore is pulled more from one finger raise than the others in the same drift, it would introduce the BSS to dilute ore in the neighbouring finger raises (NUG Cave control manual, 2000). Figure 2.6 shows the ore-waste interface in relation to the intact ore beds.

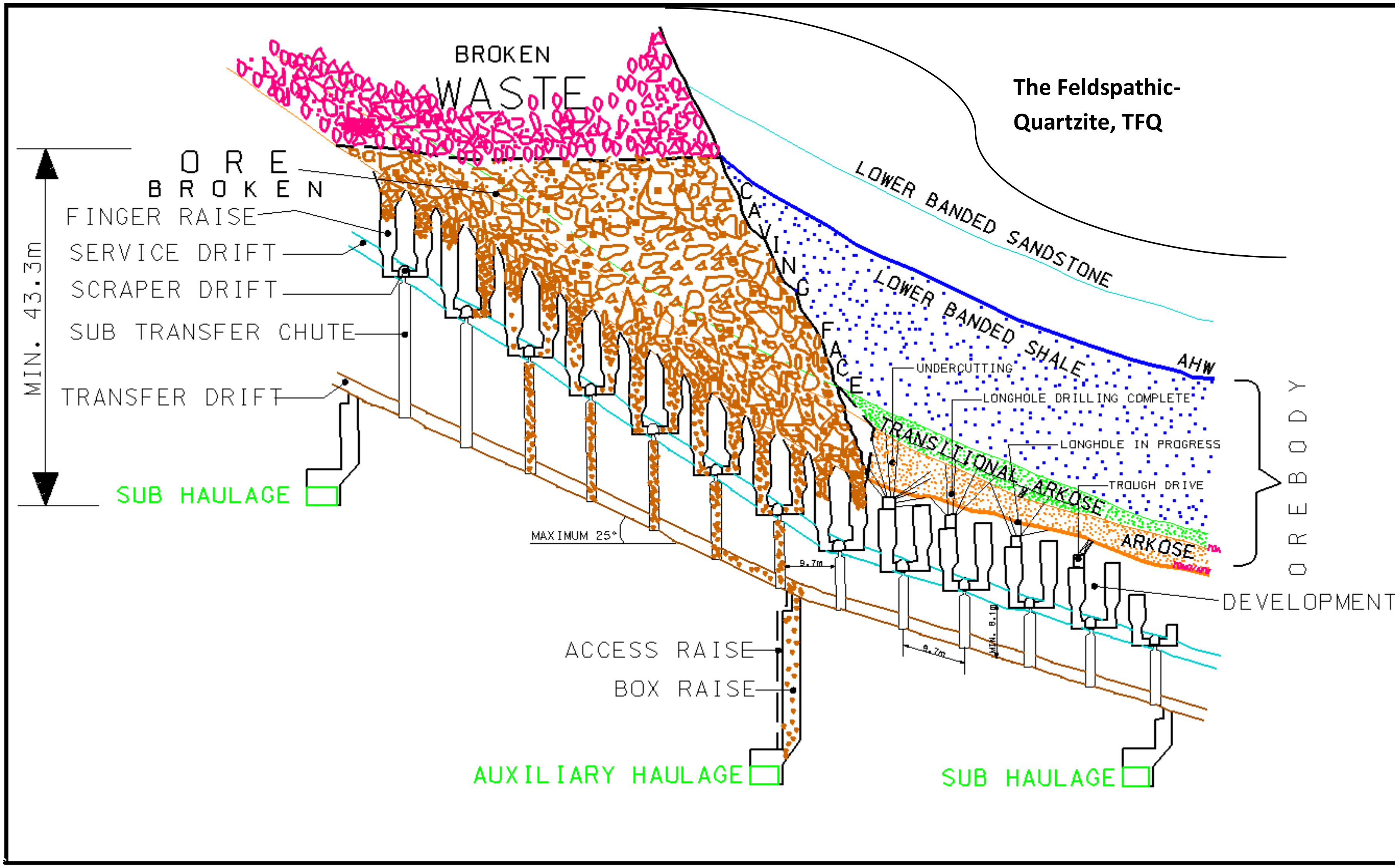


Figure 2.6: The Ore-Waste Interface (NUG Training manual, 2000)

Cave control monitoring ensures drawpoints are regulated to be almost at the same level or percentage of draw at all times. This is achieved by monitoring geology changes in the raises on a daily basis and comparing with borehole logs for each drift. The cave monitoring official estimates the stage of the draw in each raise. Sampling in all the raises is done to ascertain exhaustion of ore before drifts are declared completely exhausted.

2.1.3.3 Production Forecasting

The ore reserve calculation system has been computerized since 1972 (KCM Geology Dept, 2000). Lateral and vertical variations in ore grades throughout the orebody are taken into account and data stored. This is then used in calculations of more realistic production forecasts. The system is known as Dynamic Ore Reserves System (DORS) for both forecasting future production figures and assessing past performance records (Maxwell, 1978). It was particularly modified for Nchanga Underground mine by hired consultants from South Africa, from a company called Mineral Industries Computing Limited.

2.2 General Principles of Caving

Caving of underground orebodies works on the principle of initiating failure in the rock mass beneath by means of an undercut and then a cave is propagated through the orebody mass through the influence of gravity. The collapsed ore falls onto a network of drawpoints and is then scooped and channeled to other collection conduits to underground crushers, where present, or straight to shaft loading facilities for hoisting to surface. The undercut should have a sufficient width to height ratio in order for caving to be initiated. This ratio varies from rock type to another. Each rock unit or type will cave depending on whether the failure stresses induced exceed its rock mass strength (RMS). To ensure the cave back or column propagates vertically, a void must always be created into which fragmented rock will fall by extracting broken ground, (i.e. swell pulling). The caving process experiences hindrance when angular rock blocks form a stable interlocking configuration known as arched crown obstructing any further caving to the rock mass. This is shown in Figure 2.7.

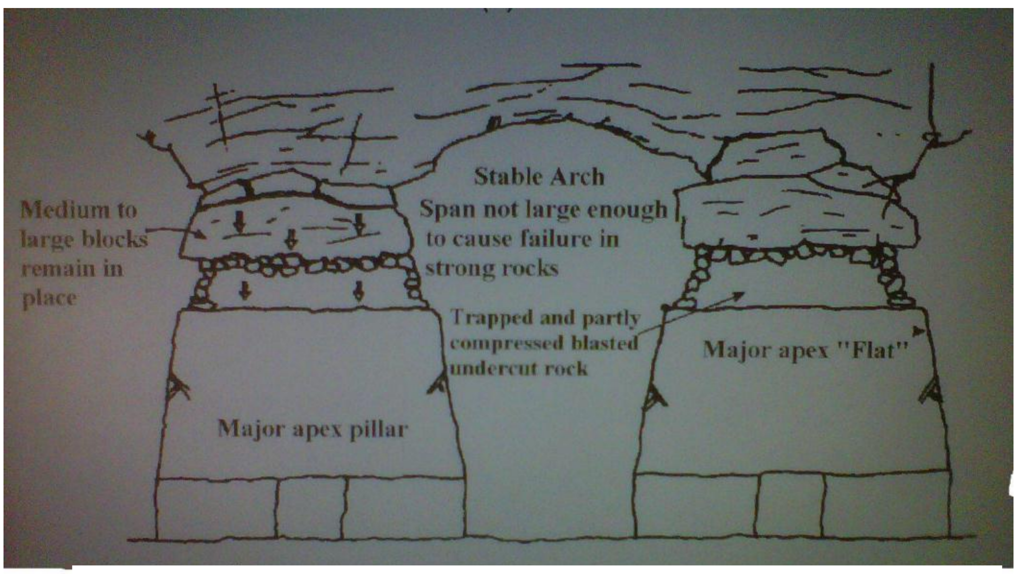


Figure 2.7: Stable arch illustration (Laubcher, 1994)

2.2.1 Factors which influence Caveability of Rockmass

Table 2.1 gives a brief discussion of parameters which should be considered when contemplating to implement cave mining (Laubscher, 2006).

Table 2.1: Parameters to be considered before the implementation of Cave Mining (Laubscher, 1994)

CAVABILITY	PRIMARY FRAGMENTATION	DRAWPOINT/DRAWZONE SPACING
Rockmass strength (MRMR) Rockmass structure *In situ* stress Induced stress Hydraulic radius of orebody Water	Rockmass strength (MRMR) Geological structures Joint/fracture spacing Joint condition ratings Stress or subsidence caving Induced stress	Fragmentation Overburden load and direction Friction angles of caved particles Practical excavation size Stability of host rockmass (MRMR) Induced stress
DRAW HEIGHTS	**LAYOUT**	**ROCKBURST POTENTIAL**
Capital Orebody geometry Excavation stability	Fragmentation Drawpoint spacing and size Method of draw	Regional and induced stresses Rockmass strength/modulus Structures Mining sequence
SEQUENCE	**UNDERCUTTING SEQUENCE (pre/advance/post)**	**INDUCED CAVE STRESSES**
Cavability Orebody geometry Induced stresses Geological environment Rockburst potential Production requirements Influence on adjacent operations Water inflow	Regional stresses Rockmass strength Rockburst potential Rate of advance Ore requirements	Regional stresses Area of undercut Shape of undercut Rate of undercutting Rate of draw
DRILLING AND BLASTING	**DEVELOPMENT**	**EXCAVATION STABILITY**
Rockmass strength Powder factor Rockmass stability (drillhole closure) Required fragmentation Height of undercut	Layout Sequence Production Drilling and blasting	Rockmass strength Regional and induced stresses Rockburst potential Excavation size Draw height Mining sequence
PRIMARY SUPPORT	**PRACTICAL EXCAVATION SIZE**	**METHOD OF DRAW**
Excavation stability Rockburst potential Brow stability	Rockmass strength *In situ* stress Induced stress Caving stresses Secondary blasting	Fragmentation Practical drawpoint spacing Practical size of excavation
RATE OF DRAW	**DRAWPOINT INTERACTION**	**DRAW COLUMN STRESSES**
Fragmentation Method of draw Percentage hangups Secondary breaking requirements	Drawpoint spacing Fragmentation Time frame of working drawpoints	Draw-column height Fragmentation Homogeneity of ore fragmentation Draw control Draw-height interaction Height-to-base ratio Direction of draw
SECONDARY FRAGMENTATION	**SECONDARY BLASTING/BREAKING**	**DILUTION**
Rock-block shape Draw height Draw rate—time-dependent failure Rock-block workability Range in fragmentation size Draw-control program	Secondary fragmentation Draw method Drawpoint size Size of equipment and grizzly spacing	Orebody geometry Fragmentation size distribution Fragmentation range of unpay ore and waste Grade distribution of pay and unpay ore Mineral distribution in ore Drawpoint interaction Secondary breaking Draw control
TONNAGE DRAWN	**SUPPORT REPAIR**	**ORE/GRADE EXTRACTION**
Level interval Drawpoint spacing Dilution percentage	Tonnage drawn Point and column loading Brow wear Secondary blasting	Mineral distribution Method of draw Rate of draw Dilution percentage Ore losses
SUBSIDENCE		
Major geological structures Rockmass strength Induced stresses Depth of mining		

The following are some of the major factors which affect caveability of rock mass (Laubscher, 2006; Brown, 2003 and 2007):

- Regional structures (i.e. faults, folds, discontinuities, etc) – magnitude and orientation ;
- Induced stresses in cave back;
- Mining Rock Mass Ratings (MRMR) of orebody and hangingwall;
- Structural domains – changes in density and orientation;
- Major and minor structures – discontinuities especially those with dipping angles near flat 0˚ - 45˚;
- Water presence;
- Sufficient void space into which caving material can flow; and
- Gravitational force displacing unstable blocks from the cave boundary.

2.2.2 Types of Caving

Caving occurs in two forms, i.e. stress caving and subsidence caving.

2.2.2.1 Stress Caving

Stress caving occurs in virgin cave blocks when the induced stresses exceed the strength of the rock mass. Stable arch formation may stop this caving in cave back. For further caving to occur or continue, the undercut must be increased in size or boundary weakening must be undertaken.

2.2.2.2 Subsidence Caving

Subsidence caving results from removal of lateral restraints on the advancing face of the block being caved. This kind of caving, normally results in rapid propagation of the cave with limited bulking. Subsidence caving occurs with lower values of hydraulic radius (HR) of a given rock system. (HR, also known as caving radius or stability index, is the ratio of area of a span to be caved to the wetted perimeter of the same span. The dimension is length, i.e. metre). In terms of propagation of the cave, hydraulic radius refers to the unsupported area of the cave back linearly (Laubscher, 1994). Subsidence caving is also characterized with wedge failures as the relaxation zone forms ahead of the cave front thereby affecting fragmentation

2.2.3 Hydraulic Radius

The hydraulic radius required to ensure cave propagation is achieved, is the ratio of area of a span to be caved to the wetted perimeter of the same span. Hydraulic radius is illustrated in Figure 2.8 and calculated using equation 2.1:

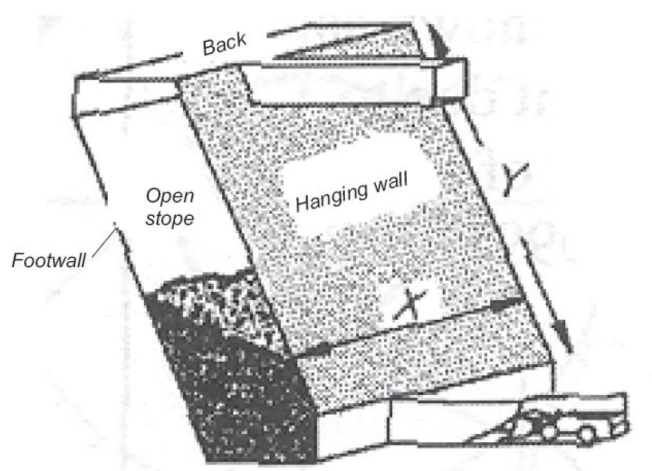


Figure 2.8: Hydraulic Radius of a Hangingwall (Laubscher, 1994)

Hydraulic radius = area/perimeter = XY/(2X+2Y); …………………………….... 2.1

Therefore, for a rectangular area span of width, X = 50m and length, Y = 200m, will give

Area = X*Y = 50m * 200m = 10,000m^2

Perimeter = 2*(X + Y) = 2*(50m + 200m) = 500m;

HR = area/perimeter = 10,000m^2/500m = 20m.

This implies for this particular kind of rock unit that a hydraulic radius of more than 20m will be required to initiate caving.

From different shapes of excavations, maximum area with minimum perimeter is achieved by circular shapes followed by square shapes. The HR to propagate the cave in a block must be based on the highest Mining Rock Mass Rating (MRMR) zone of the block. Figure 2.9 shows Laubscher's Caveabililty Chart with Lower Banded Shale (LBS) as an example.

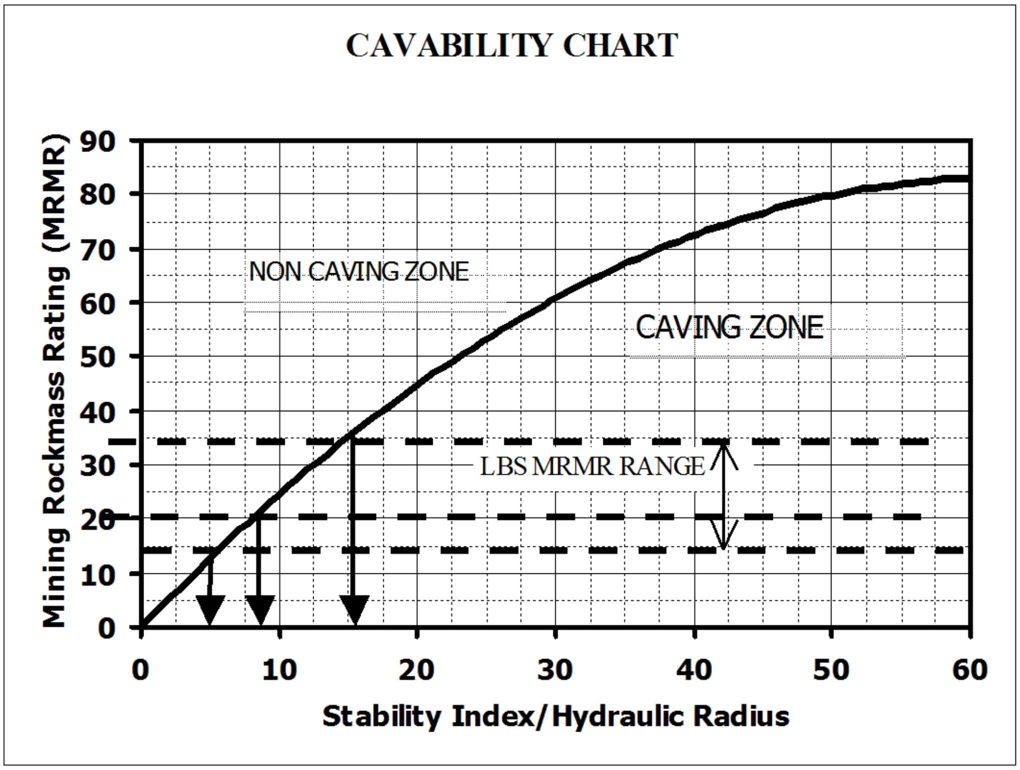


Figure 2.9: Laubscher's Cavability Chart for LBS Rock at Nchanga Underground Mine (KCM Geotech, 2006)

From Figure 2.8, HR for LBS rock formation ranges from 5m to about 16m (i.e. along the x-axis) intercepting the MRMR (i.e. y-axis) range 14 and 34 respectively. Conversely, for an MRMR value of 14, a minimum HR of more than 5m will be required to initiate cave failure. In this case therefore for LBS MRMR range, an HR value greater than 16m will induce caving in the LBS rock formation.

2.2.4 Rate of Caving

The rate of caving, i.e. rate which failure is induced in a rockmass, is regulated by controlling the draw as the cave can only propagate if there is space into which rock or caved ground can move.

The void should be created to accommodate the swell or caving ground. The rate of caving (RC) is increased by advancing the undercut more rapidly. This can only be hindered if an air gap builds up or forms over a large area between the caved material and the rockmass above which is caving. This would therefore imply that in such cases intersection of major structures, heavy blasting and/or influx of water, in a wet mine, can cause devastating airblasts. Rapid uncontrolled caving can result in early influx of waste thereby causing ore dilution.

This relationship between rate of caving (RC), rate of undercutting (RU) and rate of damage (RD) needs to be fine tuned or harmonized by good geotechnical information and monitoring. The relationship is shown in equation 2.2:

$$RC > RU > RD; \quad \text{……………………………………………..………………. 2.2}$$

According to equation 2.2, the rate of caving is greater than the rate of undercutting which is also greater than the rate of damage. Attention should be paid to all aspects of the caving process as the only control is achieved through the rate of undercutting and rate of draw. Draw rate creates voids into which caved material can flow thereby enhancing caving.

2.2.5 Fragmentation

According to Laubscher (2006), fragmentation achieved during the caving process has serious bearing on:-

- Draw point spacing;
- Dilution entry into the draw column;
- Draw control;
- Draw productivity;
- Secondary blasting/breaking costs; and
- Secondary blasting damage.

Input data in determining primary fragmentation, (i.e. particle sizes achieved during caving) include the following:-

- Joint spacing and orientation;
- Variation in mining rock mass ratings (MRMR);
- Geological structures;
- Orientation of cave fronts against joints; and
- Induced stresses.

Secondary fragmentation is reduction in size of the original particles which enter the draw column. The following factors will lead to secondary fragmentation:-

- Draw height;
- Attrition along the draw column;
- Slow draw rate allows or gives time for stresses to work on particles in the draw column (attrition);
- Shape of particles;

- Caving stresses; and
- Differential draw.

Finer fragments in the broken ground tend to move faster and easily along the draw column while the larger ones face higher friction and hence their flow is reduced. The fragmentation profile demonstrating this kind of sorting out of particles according to size in the draw column is shown in Figure 2.10.

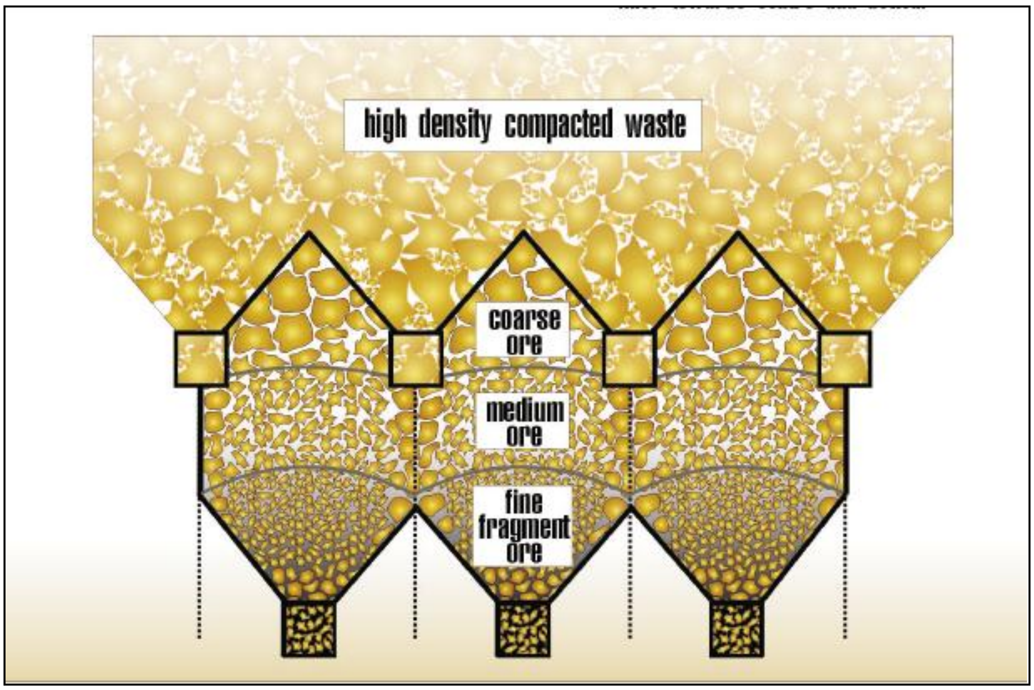


Figure 2.10: Blast fragmentation profile through typical Ring (Bull and Page, 2000)

2.2.6 Draw Mechanisms and Spacing

Once caving of the orebody is completed, the quality of ore drawn at the draw points will depend on:

- Characteristics of the broken material;
- Size and spacing of the grizzleys; and
- Draw strategy and control practice.

2.2.6.1 Material Flow

The characteristics of the material being drawn have influence on its mobility. Particle size and distribution in the ore or waste rock, shape or angularity of fragments, friction, cohesive properties, bulky density and extent of consolidation will generally govern how the material will flow in the draw column (Power, 2004). Smaller particles travel easily and faster than coarse ones as is shown in Figure 2.10. The shape of particles tends to increase or decrease their resistance to flow/draw. Loose ore material left without being drawn for some time tends to consolidate and compact resulting in a lot of difficulties in flow during draw.

2.2.6.2 Size and spacing of drawpoints

Generally flow requires large discharge points for fragments to pass. Literature survey on the topic shows that drawpoint size should be 3 to 6 times the size of largest fragments (Janelid and Kvapil, 1966). This adherence is vital to avoid ore fragments from choking drawpoints causing delays in the rate of draw and extra costs as explosives might be required to loosen or break the fragments further.

Interaction of drawpoints is very important in cave mining. This is achieved by ensuring that the draw points are closely (i.e. depending on the width of the ellipsoids of draw worked out based on drawpoint size) spaced to enable ellipsoids of draw from each drawpoint to overlap with the neighbouring ones. It avoids over stressing of the apex/crown pillars individually as the weight is spread across the network of drawpoints. Figure 2.11 illustrates the interaction of drawpoints through the ellipsoids.

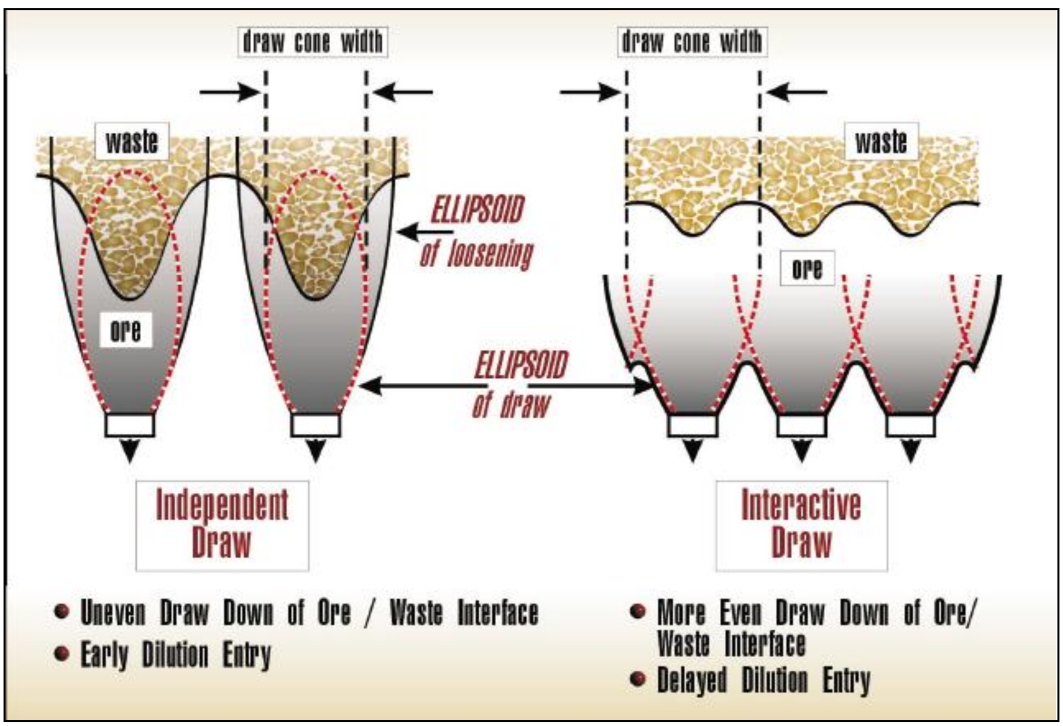


Figure 2.11: Ore/Waste interface with Interaction Draw (Bull and Page, 2000)

The interaction of the drawpoint ellipsoids takes care of early dilution as an even flow is achieved.

Numerical modeling literature (Janelid and Kvapil, 1966), indicated that a relationship exists between the flow width and flow height of a drawhole of 1.5 to 2.0m diameter as shown in equation 2.3:

$$\mathbf{H/D = \sqrt{(mH)};} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots \quad 2.3$$

Where:

D - is the biggest horizontal width of the ellipsoid (in metres);

H - is the height of the ellipsoid (in metres); and

m - is an empirical coefficient (unitless) depending on the loose condition of the material for medium hard ores. It varies from 1.2 for loose material to 1.7 for highly compacted ore.

Example:

Taking **m** as 1.45 for moderately loose material, the ellipsoid width **D** would be estimated from the following ellipsoid heights (**H**) of 15m, 17m, 19, 20m and 22m as follows from equation 2.3:

$$D = H/\sqrt{(mH)};$$

Table 2.2: Ellipsoid - Height and width relationship (Kvapil, 2008)

Height of ellipsoid H in (m)	**Width of Ellipsoid D in (m)**
15	3.2
17	3.4
19	3.6
20	3.7
22	3.9

Depending on the spacing of drawpoints, this relationship would indicate at what draw height the ellipsoids would be able to interact from their determined largest horizontal widths. Interaction of the ellipsoids of draw results in maximum ore recovery as less ore would be left out on the crown pillars. The chimneying effect of the material from draw of scattered drawpoints is ruled out, which if not avoided would introduce early dilution.

Generally a drawhole area forms an ellipsoid of draw above a drawpoint (Kvapil, 2008). Therefore, a series or network of ellipsoids of draw exists above drawpoints. The ellipsoid theory is illustrated in Figure 2.12.

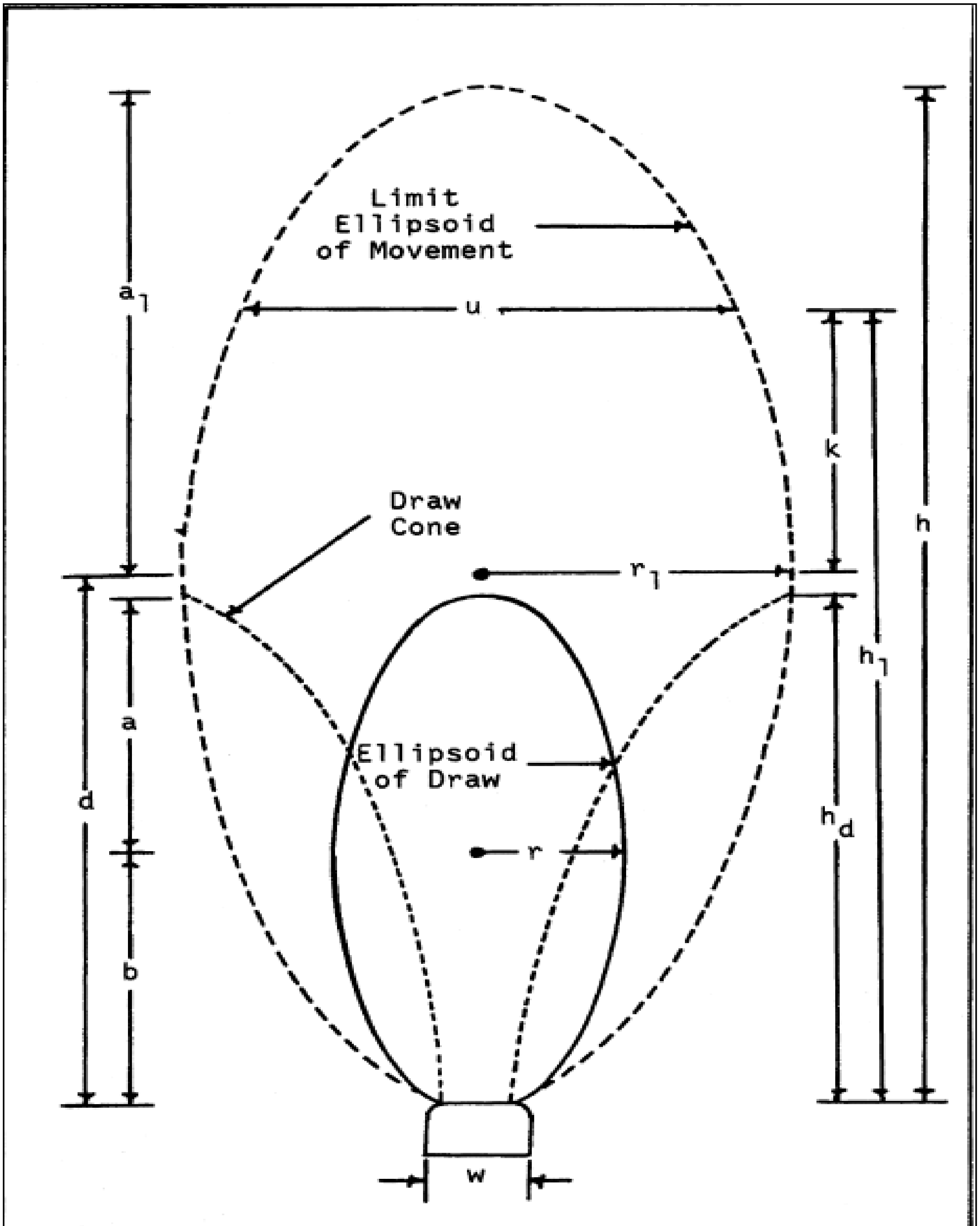


Figure 2.12: Draw shape based on ellipsoid theory (Kvapil, 2008)

The volume, V_d, of the ellipsoid of draw (from the parameters in Figure 2.12) is given by the equation 2.4:

$$V_d = \frac{\pi r^2 h_d^2}{a}\left(1 - \frac{h_d}{3a}\right) \qquad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots.\ 2.4$$

Where:

r = semi-minor axis of ellipsoid of draw

h_d = height of ellipsoid of draw

a = semi-major axis of ellipsoid of draw

d = distance from centroid of ellipsoid of draw to the height of the drawpoint (a>b).

In terms of r, h_d and opening width, w, of the drawhole; equation 2.5 shows the relationship:

$$V_d = \frac{\pi}{3} r^2 h_d \left[1 + \frac{w^2}{4r^2} + \left(1 - \frac{w^2}{4r^2}\right)^{\frac{1}{2}}\right] \qquad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots...\ldots\ldots\ldots\ldots\ldots\ldots\ldots.\ 2.5$$

For the loosened material/ground around the ellipsoid of draw, forms the limit ellipsoid which separates the zone of motion (i.e. inside the loosening ellipsoid) and the stationery zone (i.e.

outside the loosening ellipsoid). The general equation for the volume, V_l, of the loosening ellipsoid, as illustrated in Figure 2.12, would be given by the equation 2.6:

$$V_l = \pi r_l^2 \left[(k+d) - \left(\frac{k^3 + d^3}{a_l^2} \right) \right] \qquad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots \; 2.6$$

Where:

r_l = length of semi-minor axis of the loosening ellipsoid in metres;

k = distance from the centroid of loosening ellipsoid to any arbitrary level above the centroid in metres;

d = distance from the centroid of loosening ellipsoid to the height of the drawpoint in metres; and

a_l = length of semi-major axis of loosening ellipsoid in metres;.

The shape of the ellipsoid of draw is determined by its eccentricity, ε, which is given by the equation 2.7:

$$\varepsilon = \frac{1}{a\sqrt{a^2 - r^2}}$$
…………………………………………………………………. 2.7

NB: Smaller material particles produce larger eccentricity of the ellipsoid of draw than the larger particles (Kvapil, 1998).

2.3 Rockmass classification systems

In the field of civil engineering construction and mining industry, classification of rock mass characteristics is of vital and critical importance as decision making regarding the shape, size and orientation of any excavation or embankment is concerned. Any infrastructure that is put up in the rock mass should be able to stand for the period its services would be required and beyond. These aspects of the rock mass characterization play an important role in decision making (Hudson and Harrison, 2000).

A number of rock mass classification systems are in use in geotechnical, mining and civil engineering activities. A classification system is an important tool as it provides a communication between geological, planning, operating and managerial personnel. All personnel involved need to be familiar with the system to enable them make good decisions or judgments in the field. The rock mass classification systems have a long and old history of over 100 years since initial attempts to develop empirical approach to tunnel design by Ritter date as far back as 1879 (Hoek and Brown, 1990). However latest literature shows that research in the same field continues to generate new information and knowledge. These classification systems are discussed briefly in the following sub headings.

2.3.1 Terzaghi's rockmass classification

The work by Terzaghi (1946) is one of the earliest references to the use of rockmass classification for design of tunnel support where rock loads using steel sets were estimated on the basis of descriptive classification. The following is a list of rockmass classification according to Terzaghi:

- Intact rock – with no joints or hair cracks;
- Stratified rock – with little or no resistance against separation along boundaries between the strata;
- Moderately jointed rock – has joints/cracks with blocks between joints interlocked such that vertical walls do not require lateral support;
- Blocky and seamy rock – chemically intact rock or almost intact rock fragments imperfectly interlocked. May require lateral support on vertical walls;
- Crushed but chemically intact rock – where no re-cementation occurs, behaves as sand;
- Squeezing rock – of micaceous minerals or clay minerals which may slowly flow without volume increase (low swelling capacity); and

- Swelling rock – expands/swells; rocks with clay minerals with high swelling capacity.

Terzaghi's classification is not commonly used today. This is because rockmass classification has been identified as an engineering as well as a geomechanics classification.

2.3.2 Deere's Rock Quality Designation (RQD)

Rock quality designation (RQD) was developed by Deere (Deere et al, 1988; Palmstrom, 2005). It provides a quantitative estimate of rockmass quality from drill core logs. The RQD is defined as the percentage of intact core pieces longer than 100mm in the total length of the core. The core should at least be of the size 50mm or greater in diameter or thickness drilled with a double-tube core barrel. The quality of the rock is determined from equation 2.8:

$$RQD = \frac{100 * \Sigma(\text{Length of core pieces} > 100mm)}{\text{Total length of core run}} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots 2.8$$

Where no core is available but only discontinuity traces visible in surfaces exposures, Palstrom (1982) suggested that RQD may be estimated from the number of discontinuities per unit volume. The relationship for clay-free rock masses is given by equation 2.9:

$$RQD = 115 - 3.3J_v \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots 2.9$$

Where J_v is sum of the number of joints per unit length for all joint sets known as the volumetric joint count.

The orientation of the borehole is important as RQD is dependent on direction and its value changes with change in borehole orientation. It is intended to represent rock mass quality in situ. The importance of RQD is in its use in the rock mass rating and 'Q System' (discussed in section 2.3.5) for rock mass classification. Limitations of the RQD are:

- RQD = 0, where core pieces are 100mm or less, formula is not applicable; while RQD = 100, when core pieces are more than 100mm (single core piece length more than 100mm and is the total core run length).
- It gives no information or does not provide for the remaining core pieces less than 100mm in length.

2.3.3 Bieniawski's Geomechanics rockmass classification

Initial works on the geomechanics rock mass classification or rock mass rating (RMR) system by Bieniawski were done in 1976. With the passage of time, this work has been refined by Bieniawski with significant changes (1989). The following six parameters are used for classification of rock mass using the RMR system (Bieniawski, 1989):

i. Uniaxial compressive strength (UCS) of rock material;

ii. Rock Quality Designation (RQD);

iii. Spacing of discontinuities;

iv. Condition of discontinuities;

v. Ground water conditions; and

vi. Orientation of discontinuities.

A rating value is assigned to the various ranges of each parameter. The overall RMR value is obtained by adding the values of the ratings determined from individual parameters. Table 2.3 shows the parameter ranges and RMR ratings.

Table 2.3: Rock Mass Rating System (Bieniawski, 1989)

A. CLASSIFICATION PARAMETERS AND THEIR RATINGS

	Parameter		Range of values						
1	Strength of intact rock material	Point-load strength index	>10 MPa	4 - 10 MPa	2 - 4 MPa	1 - 2 MPa	For this low range - uniaxial compressive test is preferred		
		Uniaxial comp. strength	>250 MPa	100 - 250 MPa	50 - 100 MPa	25 - 50 MPa	5 - 25 MPa	1 - 5 MPa	< 1 MPa
	Rating		15	12	7	4	2	1	0
2	Drill core Quality *RQD*		90% - 100%	75% - 90%	50% - 75%	25% - 50%	< 25%		
	Rating		20	17	13	8	3		
3	Spacing of discontinuities		> 2 m	0.6 - 2 . m	200 - 600 mm	60 - 200 mm	< 60 mm		
	Rating		20	15	10	8	5		
4	Condition of discontinuities (See E)		Very rough surfaces Not continuous No separation Unweathered wall rock	Slightly rough surfaces Separation < 1 mm Slightly weathered walls	Slightly rough surfaces Separation < 1 mm Highly weathered walls	Slickensided surfaces or Gouge < 5 mm thick or Separation 1-5 mm Continuous	Soft gouge >5 mm thick or Separation > 5 mm Continuous		
	Rating		30	25	20	10	0		
5	Ground water	Inflow per 10 m tunnel length (l/m)	None	< 10	10 - 25	25 - 125	> 125		
		(Joint water press)/ (Major principal σ)	0	< 0.1	0.1, - 0.2	0.2 - 0.5	> 0.5		
		General conditions	Completely dry	Damp	Wet	Dripping	Flowing		
	Rating		15	10	7	4	0		

B. RATING ADJUSTMENT FOR DISCONTINUITY ORIENTATIONS (See F)

Strike and dip orientations		Very favourable	Favourable	Fair	Unfavourable	Very Unfavourable
Ratings	Tunnels & mines	0	-2	-5	-10	-12
	Foundations	0	-2	-7	-15	-25
	Slopes	0	-5	-25	-50	

C. ROCK MASS CLASSES DETERMINED FROM TOTAL RATINGS

Rating	100 ← 81	80 ← 61	60 ← 41	40 ← 21	< 21
Class number	I	II	III	IV	V
Description	Very good rock	Good rock	Fair rock	Poor rock	Very poor rock

D. MEANING OF ROCK CLASSES

Class number	I	II	III	IV	V
Average stand-up time	20 yrs for 15 m span	1 year for 10 m span	1 week for 5 m span	10 hrs for 2.5 m span	30 min for 1 m span
Cohesion of rock mass (kPa)	> 400	300 - 400	200 - 300	100 - 200	< 100
Friction angle of rock mass (deg)	> 45	35 - 45	25 - 35	15 - 25	< 15

E. GUIDELINES FOR CLASSIFICATION OF DISCONTINUITY conditions

Discontinuity length (persistence) Rating	< 1 m 6	1 - 3 m 4	3 - 10 m 2	10 - 20 m 1	> 20 m 0
Separation (aperture) Rating	None 6	< 0.1 mm 5	0.1 - 1.0 mm 4	1 - 5 mm 1	> 5 mm 0
Roughness Rating	Very rough 6	Rough 5	Slightly rough 3	Smooth 1	Slickensided 0
Infilling (gouge) Rating	None 6	Hard filling < 5 mm 4	Hard filling > 5 mm 2	Soft filling < 5 mm 2	Soft filling > 5 mm 0
Weathering Ratings	Unweathered 6	Slightly weathered 5	Moderately weathered 3	Highly weathered 1	Decomposed 0

F. EFFECT OF DISCONTINUITY STRIKE AND DIP ORIENTATION IN TUNNELLING**

Strike perpendicular to tunnel axis		Strike parallel to tunnel axis	
Drive with dip - Dip 45 - 90°	Drive with dip - Dip 20 - 45°	Dip 45 - 90°	Dip 20 - 45°
Very favourable	Favourable	Very unfavourable	Fair
Drive against dip - Dip 45-90°	Drive against dip - Dip 20-45°	Dip 0-20 - Irrespective of strike°	
Fair	Unfavourable	Fair	

* Some conditions are mutually exclusive . For example, if infilling is present, the roughness of the surface will be overshadowed by the influence of the gouge. In such cases use A.4 directly.

** Modified after Wickham et al (1972).

2.3.4 Modification to RMR for mining applications by Laubscher (1977)

The mining industry tended to regard Bieniawski's RMR earlier system as a little conservative because his case histories were drawn from civil engineering practices and applications. Several modifications have been done to make the classification more relevant to mining applications.

Laubscher (1977), Laubscher and Taylor (1976) and Laubscher and Page (1990) have described a Modified Rock Mass Rating system for mining (MRMR). This system takes the basic value of Bieniawski's RMR rating and adjusts it to account for in situ and induced stresses, stress changes and effects of blasting and weathering. Ground support recommendations accompany the resulting MRMR value too.

2.3.5 The Q System

Barton, Lien and Lunde (1974) proposed a Tunneling Quality Index (Q) for the determination of rock mass characteristics and tunnel support requirements. The Q value varies on a logarithmic scale from 0.001 to a maximum of 1,000. It combines six parameters in a multiplicative function as shown in equation 2.10:

$$Q = \frac{RQD}{\mathrm{Jn}} * \frac{\mathrm{Jr}}{\mathrm{Ja}} * \frac{\mathrm{Jw}}{SRF} \qquad \ldots\ldots\ldots\ldots 2.10$$

Where:

RQD - the rock quality designation;

J_n - is the joint set number;

J_r - joint roughness number;

J_a - joint alteration number;

J_w - joint water reduction factor; and

SRF - stress reduction factor.

The explanation of the quotient factors is as follows:

- RQD/J_n - is the measure of sizes of joint block surfaces;
- J_r/J_a - is shear strength of the block surfaces (friction and roughness); and
- J_w/SRF – is the influence of environmental conditions on the behavior of the rock mass (active stress).

An approximate correlation relationship between the RMR and Q systems was proposed by Bieniawski (1989) based upon studies of a large number of case histories as shown in equations 2.11a and 2.11b:

$$RMR = 9\ln Q + 44 \quad \ldots\ldots\ldots\ldots\ldots \quad 2.11a$$

or

$$RMR = 13.5\log Q + 43 \quad \ldots\ldots\ldots\ldots\ldots \quad 2.11b.$$

2.4 Geotechnical parameters for Nchanga mine rock units

Table 2.4 shows the rock unit description and rock mass ratings for the Nchanga lithographic column.

Table 2.4: Nchanga Rock Unit description (KCM Geotech, 2006)

Rock Type	**Lithological Description**	**General Description**
Nchanga Red Granite (NRG)	Coarse grained, pinkish white granite	Moderately weathered and jointed, massive. Good to fair ground conditions
Arkose (ARK)	Gray coarse to medium grained sandstone, grayish moderately weathered	Well bedded and moderately jointed. Good to fair ground condition
Transitional Arkose (TRARK)	Fine grained, kaolinized sandy silty rock	Highly weathered, friable and sheared. Very poor to poor ground condition
Banded Sandstone lower (BSSL)	Dark grey to brown fine grained, micaceous sandy material	Incoherent, loose material, may have strong bands in places. Extremely poor to poor ground conditions
Pink Quartzite (PQ)/ Shale Marker(SM)	Dark grey to brownish fine grained, silty to sand material for the shale marker. Dark grey pinkish medium grained quartzite	Highly fractured as a result of being a thin bed (3-6) sandwiched between two very weak rock formations. Poor to fair ground conditions
Banded Sandstone Upper (BSSU)	Dark grey to brownish, fine grained, highly micaceous sandy material	Incoherent, loose material, may have strong bands in places. Very poor to poor ground conditions
The Feldspathic Quartzite (TFQ)	Medium grained, white to creamish grey feldspathic quartzite	Massively bedded; steeply jointed resulting into the unit being weakly fractured and blocky in places to highly fractured in others. Joints are rough and generally filled with quartz. Variable strength as aresult of different degree of weathering
Upper Banded Shale (UBS)	Dark grey to brownish fine grained, silty material	Laminated. Highly layered and failure usually occurs along bedding planes. Poor ground condition

2.5 Rock Failure Criteria

Generally rock masses are characterized with micro to large fractures known as joints. The joints network defines blocks within the rock mass. The joints form zones of weakness in the rock along which failure normally occurs. The degree of the rock mass strength and its deformation ability largely depend on these joint networks or sets. Rock masses with no or minimal joints presence will tend to have a higher strength compared to one with a high presence of joints. Two rock failure criteria, i.e. Mohr Coulomb and Hoek-Brown (Brown, 2003) are discussed in the following subsections.

2.5.1 Mohr Coulomb criterion

The Mohr-Coulomb criterion expresses the relationship between shear and normal stresses at the failure plane. Prediction of potential failure can be achieved by using transformation equations represented by the Mohr circle as shown in Figure 2.13. The transformation equations indicate the principal stresses (major σ_1 and minor σ_3), normal stress (σ_n), shear stress (τ), cohesion (c) and the internal angle of friction ($\emptyset$). These values are obtainable in the laboratory Triaxial test.

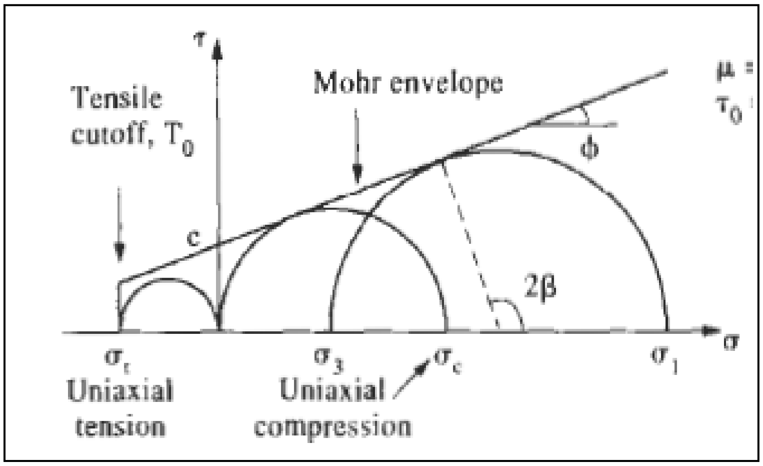


Figure 2.13: The Mohr-Coulomb failure criterion (Hoek et al, 2007)

The interpretation of the (σ – τ) curve or envelope is as follows:

Below the envelope	:	represent stable conditions or elastic;
On the envelope	:	represent limiting equilibrium or yielding; and
Above envelope	:	unobtainable conditions under static loading or impossible.

The general Mohr-Coulomb envelope is defined as shown in equation 2.12:

$$\tau_c = c + \sigma_n \tan\emptyset \quad \ldots\ldots \quad 2.12$$

where:

τ_c - Shear stress;

c - Cohesion;

σ_n - Normal stress; and

$\emptyset$ - Internal angle of friction.

Linearly, the major and minor principal stresses are related as shown in equation 2.13:

$$\sigma_1 = \frac{2c\cos\emptyset + \sigma_3(1 + \sin\emptyset)}{(1 - \sin\emptyset)} \quad \text{……………..……………………………….} \; 2.13$$

Where:

σ_1 - Major principal stress;

σ_3 - Minor principal stress; and

$\emptyset$ - Internal angle of friction.

The Mohr-Coulomb criterion is most suitable or applicable in high confining pressures when the material does fail through development of shear planes.

2.5.2 Hoek - Brown Criterion

The Hoek-Brown (Hoek and Brown, 1990; Brady and Brown, 1985) rock failure criterion, generalized version, for jointed rock mass is defined as shown in equation 2.14:

$$\sigma_1 = \sigma_3 + \sigma_{ci}\left(m_b \frac{\sigma_3}{\sigma_{ci}} + s\right)^{\mathbf{a}} \quad \text{……………………...………..……………....} \; 2.14$$

Where:

σ_1 and σ_3 - maximum and minimum principal stresses at failure

σ_{ci} - uniaxial compressive strength of intact rock pieces.

$\boldsymbol{m_b}$ - value of Hoek-Brown constant for the rock mass (a reduced value of the material constant $\mathbf{m_i}$) and is given by equation 2.15:

$$m_b = m_i \exp\left(\frac{GSI-100}{28-14D}\right) \quad \text{…………………………..………….... 2.15}$$

Where:

GSI - (Geological Strength Index) an index which describes the blockiness and the conditions of discontinuities; it is given by equation 2.16:

GSI = RMR – 5 ……………………...…..……………….. 2.16

D - a factor which depends upon degree of disturbance by blasting and stress relaxation. Details of factor D determination are presented in Appendix 1.

s & **a -** constants which depend upon the rock mass characteristics and are given by relationships as shown in equations 2.17 and 2.18 respectively:

$$s = \exp\left(\frac{GSI-100}{9-3D}\right) \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots \; 2.17$$

$$a = \frac{1}{2} + \frac{1}{6}[e^{-\frac{GSI}{15}} - e^{\left(-\frac{20}{3}\right)}] \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots \; 2.18$$

The uniaxial compressive strength is obtained by setting $\sigma_3 = 0$ in equation 2.14 giving:

$$\sigma_c = \sigma_{ci} * s^a \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots \; 2.19$$

For tensile strength:

$$\sigma_t = -\frac{s\sigma_{ci}}{m_b} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots \; 2.20$$

This is obtained by setting $\sigma_1 = \sigma_3 = \sigma_t$ in equation 2.14.

The equations 2.14 to 2.20 give smooth continuous transitions for the entire range of GSI values unlike when the “switch” at GSI = 25 for good quality rock for the coefficients **s** and **a**. This criterion indicates that the relationship between σ_1 and σ_3 is non-linear unlike one deduced by the Mohr-Coulomb criterion.

Three properties of the rock mass that have to be estimated in order to use the Hoek-Brown criterion to determine the strength and deformability of jointed rock masses are:

- σ_{ci} - Uniaxial compressive strength of intact rock pieces;
- m_i - Hoek-Brown constant for intact rock pieces and; and
- GSI (Geological Strength Index) for rock mass.

GSI is determined from the Bieniawski's rock mass rating system as shown in equation 2.16.

Rock mass failure initiates at the boundary of an excavation when critical stress (σ_c) is exceeded by induced stress on the boundary. This failure propagates from the ignition point into a biaxial stress field and eventually stabilizes when the local strength is higher than the induced stresses, ($\text{i.e.}\, \sigma_{1,\,3}$), (Marinos and Hoek, 2005).

CHAPTER 3

3.0 RESEARCH METHODOLOGY

In carrying out this research work, the procedure in acquiring data involved field observations, laboratory tests and data collection from existing Nchanga mine records. A comprehensive literature review was conducted. Further, data collected was analyzed using computer software available at the mine including other desktop analyses.

3.1 Review of Related Literature

Important to the topic of study was the review of related literature as it set the platform of carrying out the research. Literature review is presented in Chapter 2. All the principles that guide the theories on the topic required a thorough review.

3.2 Geological Setup

Review of prominent geological structural trends like faults, joints and discontinuities and their influences on the strength of, especially, the host rock is paramount. Stability of the host rock is very important as all infrastructures are positioned in it for the life span of each mining block.

3.3 Rock mass characterization

Determination of rock mass characteristics of the rock units involved in the study was significant as it guided in determining excavation sizes and appropriate support requirements. Other ore parameters reviewed were caveability and hydraulic ratio or radius of TFQ.

3.4 Ore Reserves and Waste (Banded Sandstone, BSS) Tonnages

Evaluation of the ore reserves and copper grades for all blocks in the area of interest was done with the help of personnel from Mineral Resources Department using the captured model in **Datamine software**. This was vital in determining TFQ ore tonnages and grades for each mining block (i.e. exhausted blocks). Eventually this helped to determine the overall quantity of ore sitting above developed and exhausted mining blocks using the overall model ore reserve tonnage. The ore reserves in the geological Datamine model included ore reserves even from areas with bad or poor ground conditions where it was not possible to develop the mining blocks. Conducting economical viability analysis of the project depended on this ore reserve evaluation. Another factor which was dependant on this evaluation is the ratio of TFQ ore to Waste (i.e. Banded Sandstone, BSS).

3.5 Field Observations and Sampling

The field observations included monitoring of TFQ production blocks on trial, fragmentation of TFQ ore and waste rock (BSS), establishing old areas’ stability and deterioration status as well as collection of samples for laboratory investigation.

3.6 Economic Analysis

Economic viability of the project was done as a last step after establishing the mineable ore reserves including all cost aspects associated with the project. This was done to ascertain viability of the project especially in view of the current copper market conditions.

3.7 Data Analysis and Research Conclusion

Finally after all the data was collected, it was analyzed to deduce conclusions and recommendations with regard to the viable implementation of the project.

CHAPTER 4

4.0 DATA COLLECTION AND ANALYSIS

This chapter is devoted to the presentation of results or data gathered and analysis of the findings. This will be based on the objectives of the research. The sub-objectives will be presented first and then finally the main objective. Each objective's results will be followed with a brief analysis or discussion.

4.1 Geological Setup

The Nchanga copper mineralization is hosted in the igneous hard and competent rock known as granite. It is also commonly referred to as the Nchanga Red Granite (NRG). The actual copper ore mineralization occurs in the following sedimentary rock units as described in Table 4.1:

- Arkose
- Banded Shale
- Pink Quartzite
- Shale Marker
- The Feldspathic Quartzite.

The overall ore body is generally weak. Copper ore grades are fairly distributed and the orebody dips at angles ranging 15 to 30 degrees.

Prominent geological structures encountered in the Nchanga Underground Mine include:

- Nchanga fault
- Joints and fractures
- Bedding planes
- Discontinuities and
- Folding of the ore limb as shown in Figure 4.1.

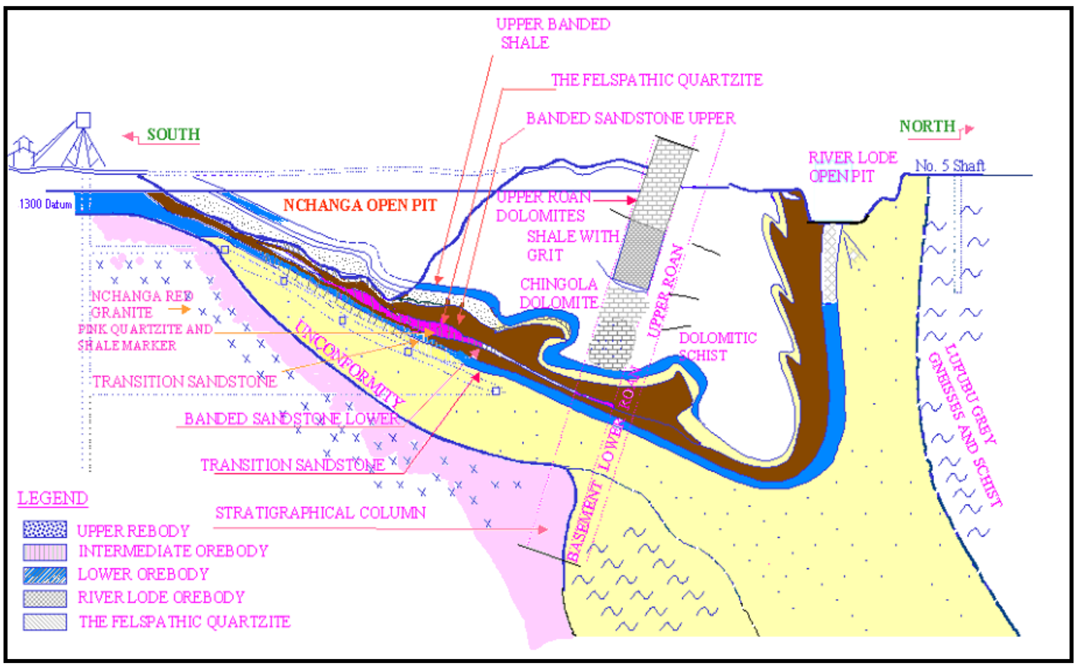


Figure 4.1: The Great Nchanga Syncline (NUG Training manual, 2000)

These structures pose challenges in mining as they lower the strength of the rock mass to stand when excavations are made in them. Heavy support might be required where the rock mass is heavily jointed. Fractured rock also gives advantage where the block caving mining method is used as the cave front would propagate easily. The TFQ and BSS Datamine model section is illustrated in Figure 4.2, larger print shown in Appendix 5.

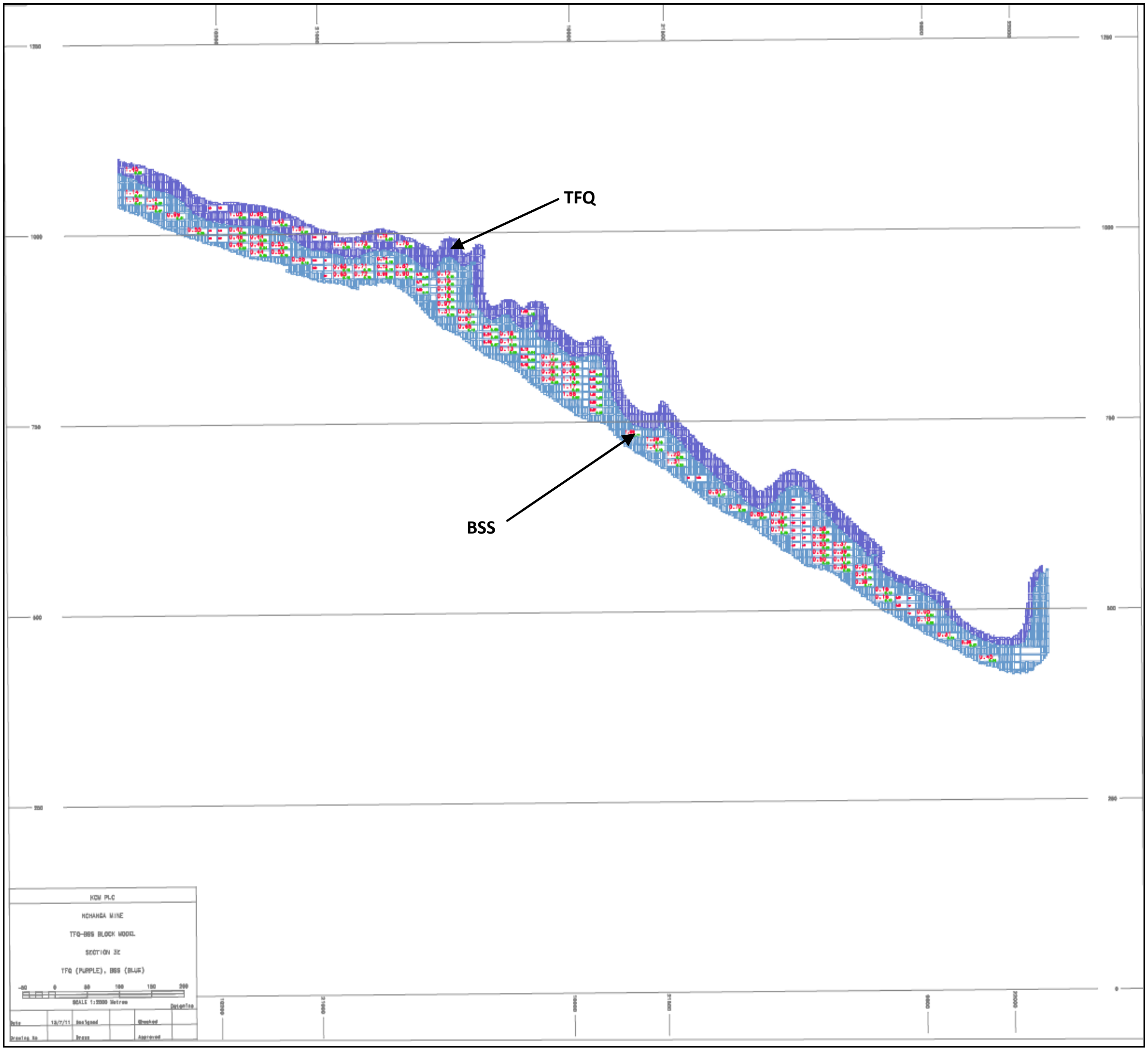


Figure 4.2: Datamine Model showing TFQ and BSS rock units (KCM Geology Dept, 2000)

The TFQ occurs above the banded sandstone layer. This is as shown in Figure 4.2. The model clearly shows the thicknesses of the two rock units being that of BSS is about twice that of TFQ.

4.2 Rock Mass Characterization

In terms of rock mass characterization of the rock units, geotechnical parameters findings are presented in the following sections courtesy of KCM Geotech department.

4.2.1 Rock Mass Parameters

The Nchanga Underground Mine rock mass ratings are as illustrated in Table 4.1:

Table 4.1 Nchanga rock unit descriptions and rock mass rating (KCM Geotech, 2006)

Rock Type	General Lithological Descriptions	Engineering Geology		Rock Mass Rating		
		General Description	UCS (MPa)	Q value	RMR_{89}	RMR_L
Nchanga Red Granite (NRG)	Coarse grained, pinkish white granite	Moderately weathered and jointed, massive. Good to fair ground conditions	>120	18	65	63
Arkose (ARK)	Gray coarse to medium grained sandstone, grayish moderately weathered	Well bedded and moderately jointed. Good to fair ground condition	80 - 150	14.85	60	55
Transitional Arkose (TRARK)	Fine grained, kaolinized sandy silty rock	Highly weathered, friable and sheared. Very poor to poor ground condition	<10	<0.003	<5	<3
Banded Sandstone lower (BSSL)	Dark grey to brown fine grained, micaceous sandy material	Incoherent, loose material, may have strong bands in places. Extremely poor to poor ground conditions	<10	<0.003	<5	<3
Pink Quartzite (PQ)/ Shale Marker(SM)	Dark grey to brownish fine grained, silty to sand material for the shale marker. Dark grey pinkish medium grained quartzite	Highly fractured as a result of being a thin bed (3-6) sandwiched between two very weak rock formations. Poor to fair ground conditions	40 - 80	<0.01	<15	<14
Banded Sandstone Upper (BSSU)	Dark grey to brownish, fine grained, highly micaceous sandy material	Incoherent, loose material, may have strong bands in places. Very poor to poor ground conditions	<10	<0.003	3	<3
The Feldspathic Quartzite (TFQ)	Medium grained, white to creamish grey feldspathic quartzite	Massively bedded; steeply jointed resulting into the unit being weakly fractured and blocky in places to highly fractured in others. Joints are rough and generally filled with quartz. Variable strength as aresult of different degree of weathering	50 - 120	0.833 – 5.377	20 - 45	18 - 40
Upper Banded Shale (UBS)	Dark grey to brownish fine grained, silty material	Laminated. Highly layered and failure usually occurs along bedding planes. Poor ground condition	25 - 55	<0.03	<10	<9

NB: TFQ rock unit falls in the range of very poor to poor rock mass rating (RMR) from Bieniawski, Laubscher and Barton' Q system ratings and hence the rock fails easily under caving.

Table 4.2 shows cohesion, friction angle and Young's Modulus for the same rock units presented in table 4.1.

Table 4.2: Other Rock Parameters (KCM Geotech, 2006)

Rock Type	Cohesion (KPa)	Friction angle (°)	Young's Modulus (MPa)
UBS	600	30	8000
TFQ	1000	55	30000
BSSU	15	43	3000
PQ/SM	1000	55	15000
BSSL	15	43	3000
LBS	200	30	10000
TRARK	0	25	2000
ARK	1000	50	25000

Properties of the Banded sandstone samples collected from underground are as shown in table 4.3 and details also shown in Figure 4.3. The sample analysis was done at Geotechnical Services laboratory based at Nchanga mine.

Table 4.3: Banded Sandstone Lower Shear Parameters

Sample No	Insitu Moisture Content (%)	S.G	Bulk Density	Dry Density	Shear Values		Void ratio, e
			kg/m^3	kg/m^3	C(kPa)	Φ	
1	12	2.6	2047	1834	15	43	0.42
2	13	2.6	1802	1590	6	35	0.64
3	9	2.6	1608	1470	133	31	0.77
4	16	2.6	1970	1706	15	42	0.5
5	12	2.6	2047	1834	15	43	0.4

Where **c** and Φ are cohesion and angle of internal friction respectively.

Konkola Copper Mines plc
Nchanga Mine
Geotechnical Services Laboratory
Triaxial Testing of Soil and Soft Rock log sheet

Project:	NUG Research Project by Hakakwale B.		Tesing Date	April 29, 2011	
Description:	Reddish brown, fine to medium and coarse grained soil (BSS)		Tested By	Madata C, Nyendwa P and Hakakwale B	
Sample:	1	Test Type:	Checked by	Mutawa Ackim	
Location:	NUG 2420 6WB	Saturated, Unconsolidated, Undrained			SUU
Coordinates: x					
y		Strain Rate:	0.024	inches/min	
z			0.6096	mm/min	

Test Number	Initial Cell Pressure	Conditions at Failure		Circle Geometry	
		Major Principal Stress σ_1	Minor Principal Stress σ_3	Centre	Radius
1	200 kPa	1075.1 kPa	200.0 kPa	637.57	437.57
2	400 kPa	2204.4 kPa	400.0 kPa	1302.22	902.22
3	600 kPa	3220.8 kPa	400.0 kPa	1910.42	1310.42
4	0 kPa	0.0 kPa	0.0 kPa	0.00	0.00
5	0 kPa	0.0 kPa	0.0 kPa	0.00	0.00
6	0 kPa	0.0 kPa	0.0 kPa	0.00	0.00

Insitu Density:	**2047**	kg/m^3	Insitu Void Ratio,e	0.4	
Dry Density:	**1834**	kg/m^3	Moisture Content	12	%

Cohesion:	**15 kPa**
Angle of Friction:	**43°**

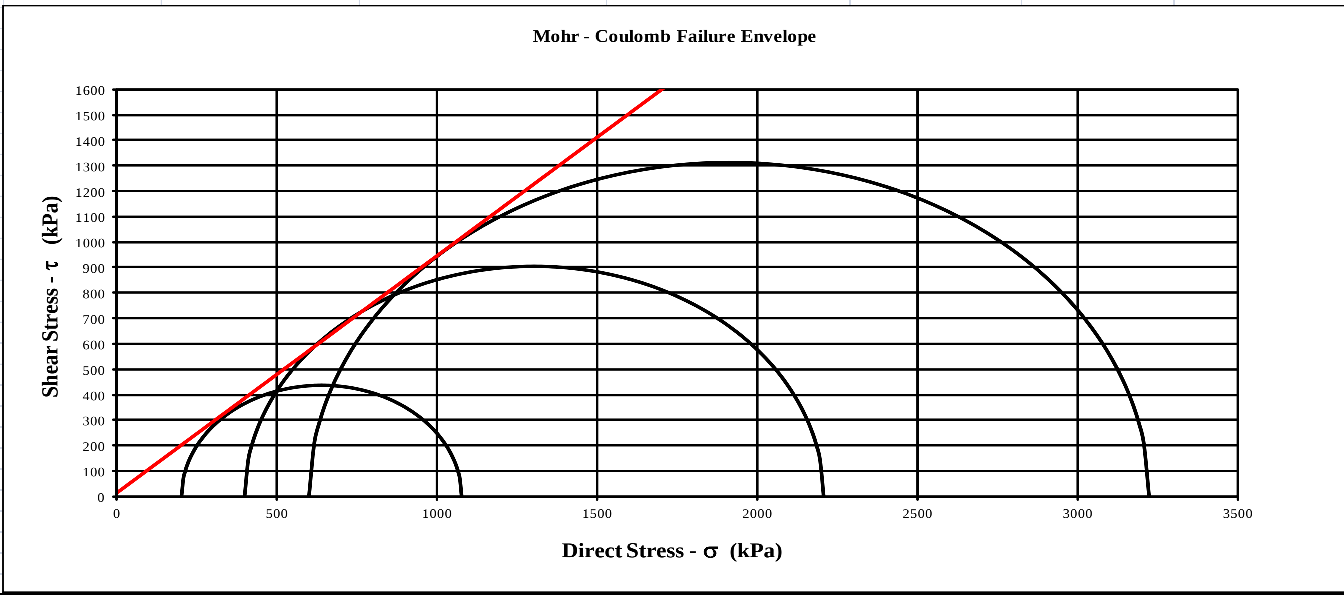


Figure 4.3: Mohr-Circles for BSS samples

Of interest from these tests is the average angle of internal friction (43˚) for the BSS material. Internal friction angle was determined from laboratory investigations as shown in Table 4.3. The value 43˚ was used as the highest or worst case scenario. This is because the BSS material forms funnel shaped sand heaps between the drawpoints above. This angle enables the ground above (TFQ) to flow to the drawpoints. Going by the spacing of the drawpoints, the funnels formed have the following geometrical parameters as shown in Figure 4.4:

$\Phi = 43^\circ$

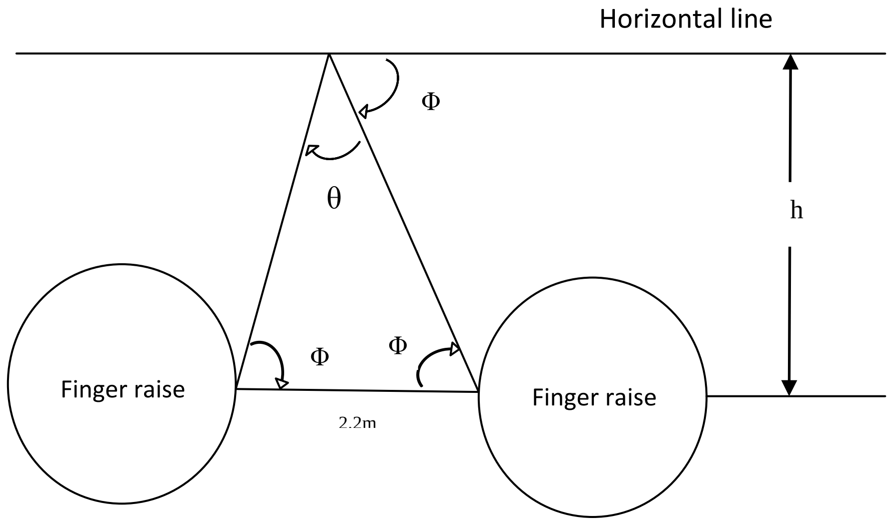


Figure 4.4: Frictional angle sketch

From the trigonometrical relationships:

$2\Phi + \theta = 180^\circ$ ………………………………………..………………………… 4.1

Since $\Phi = 43^\circ$; $\theta = 94^\circ$.

Height h is, h = 3.17m, since base length distance is known 2.2m from drawpoint spacing. The cone heaps between the drawpoints enables materials to flow into the drawpoints instead of accumulating above. The lower the internal angle of friction, the wider the zone of flow (Kvapil, 1982 and Craig, 1987).

4.3 Ore Reserves, Grades and Waste (BSS) Tonnages

In order to determine the TFQ ore reserves potential, an evaluation of the mining blocks was conducted to ascertain the ore tonnages sitting above the caved areas of the mine as well as the copper ore grades from the geological ore model in Datamine software (Appendix 5). The exhausted or caved areas of interest underground are shown in Appendix 2. The block by block detail TFQ ore and BSS evaluation are presented in Appendix 3A and 3B respectively.

In the Datamine model, TFQ ore and banded sandstone (BSS, i.e. waste) tonnages include the reserves and waste in the blocks which were not even developed due to poor ground conditions, i.e. as shown in Table 4.4. Hence the actual ore reserves in the exhausted areas (mining blocks) are less than the overall model reserves. This is shown in Table 4.5. The TFQ-BSS section model is shown in Figure 4.2.

Table 4.4 shows the ore tonnages and average total copper ore grade from the Datamine model while Table 4.5 shows the tonnage and grade from developed blocks:

Table 4.4: Model Ore and Waste Tonnages

BSS Tonnage	TFQ Ore	%TCu	%ASCU
395,577,509	198,257,760	1.63	0.67

Table 4.5: The evaluation from exhausted mining blocks

ORE TONNES	%TCU	%ASCU
75,743,960.80	1.8	0.74

The overall figures from the model show higher figures for the waste and ore but lower copper ore grades, i.e. in Table 4.4. This is because these figures include even tonnages from areas where the mining blocks were not developed due to very poor ground conditions, such as near the Nchanga fault zone, and those with no pay ore grades, i.e. total copper below 1.5%. Table 4.5 shows the actual TFQ tonnages and copper grade for the mining blocks which were actually mined and exhausted. Appendix 2 shows plan section of the mining blocks which were developed.

4.3.1 Ore to Waste Ratio

Tonnages of both TFQ ore and banded sandstone (BSS) were evaluated from geological model as carried in the datamine software as well as from the trial mining records. They are as follows:-

From geology estimations:

From the mineral evaluation by geology department, the ore to waste (banded sandstone, BSS) as presented in Table 4.1 is:

TFQ	**:**	**BSS**
198,257,760	:	395,577,509;
1	:	2

From trial mining statistics:

From the trial mining in a few TFQ-ore old blocks, the following figures have been obtained during the production period (May 2008 to July 2010):

Ore	**:**	**BSS**
149,603	:	346,950
1	:	2.3

This implies that to recover a tonne of TFQ ore, two tonnes of BSS have to be removed first.

4.3.2 Fragmentation of BSS and TFQ

BSS:

Banded Sandstone fragments range from fine particles in millimeter diameters to an average boulder size of 0.4m. This implies that BSS material flows easily and quickly due to smaller fragment sizes in the draw column.

TFQ:

As encountered from TFQ-ore trial mining, a spread of boulder sizes has been encountered. These range from small particles of few centimeters to about 0.6m. Large boulders bigger than 0.6m in size are rarely encountered and when present, they are reduced through secondary blasting to enable passage through the ore transfer chutes, i.e. 1.2m by 0.6m. Secondary blasting explosives consumption is the tune of 0.20Kg of powder explosives per tonne of ore produced.

4.4 Drawpoint parameters

In terms of draw parameters (i.e. shown in Figure 4.5); each mining block's development was set out on the pretext of the lower orebody conditions. This implies that to draw TFQ ore, the same development infrastructure would be used. This in fact is already happening in the trial mining which is going on. The herringbone system is in use as discussed under literature review in section 2.1.3.1. The spacing of drifts, finger raises and their dimensions were determined and are as shown in Figure 4.5:

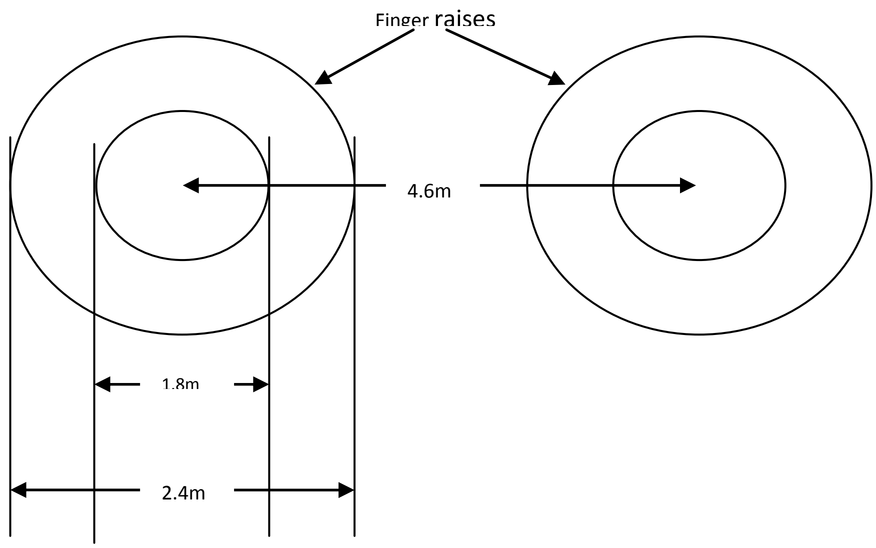


Figure 4.5: Drawpoint dimensions - sketch

Scraper drifts are positioned at 9.0m apart. The drawpoint sizes are 1.8m and 2.4m diameters at the bottom and on top respectively. Spacing of the draw points on the same side of the drift is 4.6m while spacing across the drift for the nearest draw points is 2.3m. This defines a network of draw catchments for the caved material sitting above. These are illustrated in Figure 4.5 and Figure 4.6. This has worked successfully in the extraction of the lower orebody at the mine and in the TFQ pulling trials too.

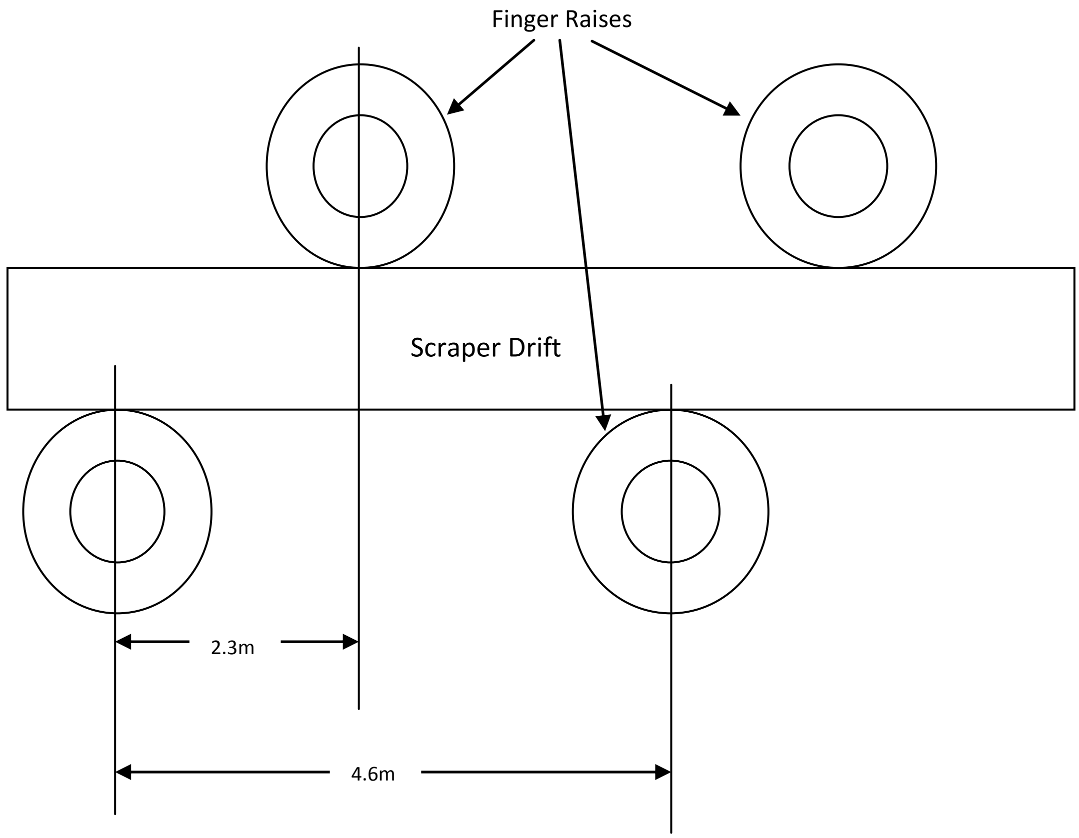


Figure 4.6: Plan view of drawpoint spacing along scraper drift - sketch

4.5 Hydraulic Radius

Hydraulic radius, as discussed under section 2.2.3, is the ratio of area of a span to be caved to the wetted perimeter of the same span. The undercut above a scraper drift has the following dimensions (Nchanga mine):

Length	=	64.0m
Width	=	9.6m.

Therefore, the resulting hydraulic radius from equation 2.1 is:

Hydraulic radius (HR) = area/perimeter,

i.e.; area of the span = 9.6m x 64.0m; and

perimeter of the span = 2 x (9.6m + 64.0m).

Hence;

HR = (9.6m x 64.0m) / [2*(9.6m + 64.0m)]

= 4.2m.

For this practice, a span of 4.2m (hydraulic radius) or more of the undercut advanced, will initiate the cave. All the other copper ore containing rock units in the lower orebody (LOB, i.e. Shale and Arkose) were caved from the same design parameters.

From the TFQ rock mass ratings range (18 – 40) as presented in Table 4.1, drawing lines on the Laubscher stability chart shown in Figure 4.7, a hydraulic radius of 8 to 21m would be obtained. Therefore, in the planning stage of a mine, undercuts should be designed to give a minimum hydraulic radius of 8.0m and above in order to achieve caving and 21.0m and above for the upper case.

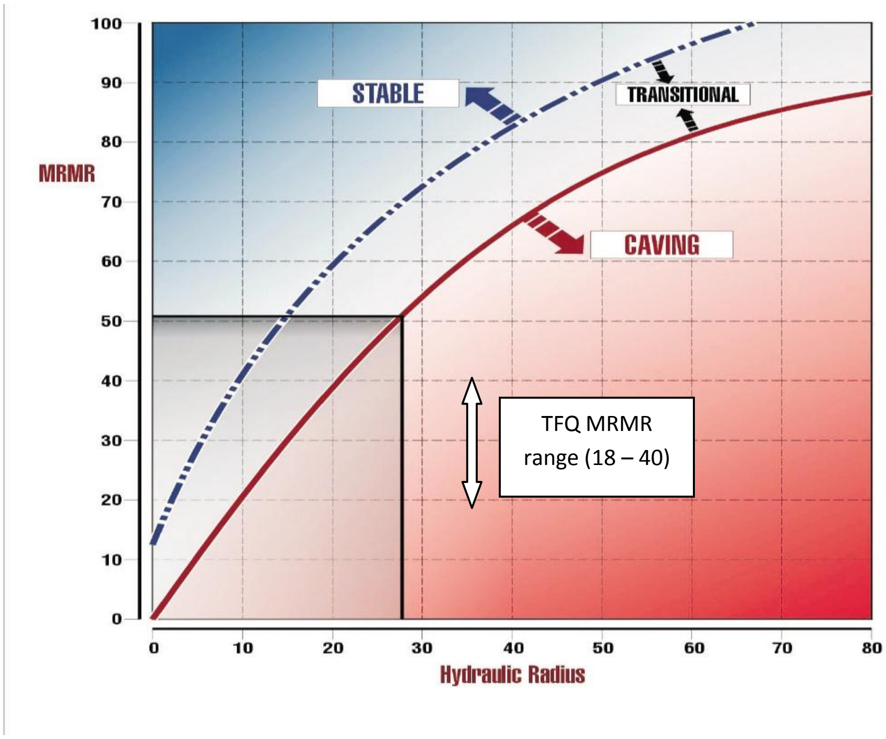


Figure 4.7: Stability Diagram (Laubscher, 1994)

4.6 Height of Caving

From the geological records of the various rock units' intersection points obtained for the 2420feet level 6W mining block, the caving height above the assay footwall during the TFQ ore trial mining in the block was observed as follows:

AFW to AHW of the LOB (thickness)	:	30m
Banded Sandstone Lower (BSSL, thickness)	:	35m
Feldspathic Quatrzite (TFQ ore thickness)	:	90m
Banded Sandstone Upper (BSSU)	:	above TFQ or 155m from AFW.

During the pulling of BSSL and eventually TFQ ore, in the exhausted drifts in the same 2420 6W block, the BSSU reported in the drawpoints. This indicated that the TFQ ore was completely extracted (i.e. exhausted) as the rock unit above it, BSSU, reported in the drawpoints. The height of caving, therefore, was found to have gone way beyond or above the BSSU which sits at 155m (30m + 35m + 90m) above AFW of the lower orebody.

4.7 Stability of old Working Areas

The support challenge faced in the old or exhausted mining blocks is that some drives (i.e. scraper drifts and service drifts) collapse in some areas. The extent of support has been estimated from the observed TFQ ore trial mining blocks; one out of six drifts requires full support (i.e. about 17%). This has been captured and included in the economic analysis of the whole project as presented in Appendix 4. The support system for the collapsed areas is by use of steel sets. Figure 4.8 shows a steel set supported drift in the block 2420 6E 7A scraper drift.

Figure 4.8: Steel set support (Photo: 2420 6E 7B scraper drift)

4.8 Economic Analysis of the TFQ Ore Project

The economic analysis of the whole TFQ project was done from the perspective of the two materials to be handled, i.e. waste (BSS) and ore (TFQ).

The costs involved with waste are extraction, tramming or transportation, hoisting as well as the overheads costs. For ore, the costs begin from extraction all the way to selling of finished copper. Appendix 4 carries details of the analysis in excel. The summary information/considerations on the economic analysis are as follows:

1. Recovery is at 80% (ore losses due to dilution).

2. Overhead, Divisional and Corporate unit costs; have been considered at 25% for the TFQ production as current management structure remains the same except for additional mine captains and other junior supervisors.

3. Support unit cost (US$ per metre); converted to US$ per tonne for support.

4. Conversion ratio 1tonne = 2204.623lb (pounds).

5. Determination of the total copper tonnes contained in the recovered ore and the finished copper tonnes after processing.

6. Ore costs: were divided into two categories:

   - Costs related to the hoisted and milled ore tonnes and
   - Costs related to finished copper tonnes.

7. Waste/BSS cost, used unit costs related to removal of BSS to surface as shown in the spreadsheet, Appendix 4.

8. With the overall cost (ore + waste), the breakeven copper price was determined. Hence any price above breakeven will render the project viable.

Table 4.6 presents a summary of the analysis figures:

Table 4.6: Economic analysis summary figures

BSS (Waste) Tonnes	151,487,921.60
Recoverable Ore Tonnes	60,595,168.64
Total Copper in Ore Tonnes	1,090,713.04
Finished Copper Tonnes	728,232.74
BSS (Waste) Costs (US$)	1,949,649,550.99
Ore Costs (US$)	2,228,384,061.12
Total Costs (BSS + Ore), US$	4,178,033,612.11
Breakeven Copper Price ($/t)	5,737.22
Revenue @ US$8,500/t Cu Price	6,189,978,262.08
Proceeds (Revenue less Costs)	2,011,944,649.97

Comments:

- Based on the analysis above, the TFQ ore project is viable for copper market selling prices above the breakeven price of US$5,737.22 per tonne of finished copper. This also implies that the project is non-viable for copper market prices below the breakeven price.

- The current international copper market prices prevailing are in the range US$7,000 to $10,000 per tonne would offer maximum proceeds from the project.

- Copper prices trend at London Metal Exchange (LME) is illustrated in Figure 4.9.

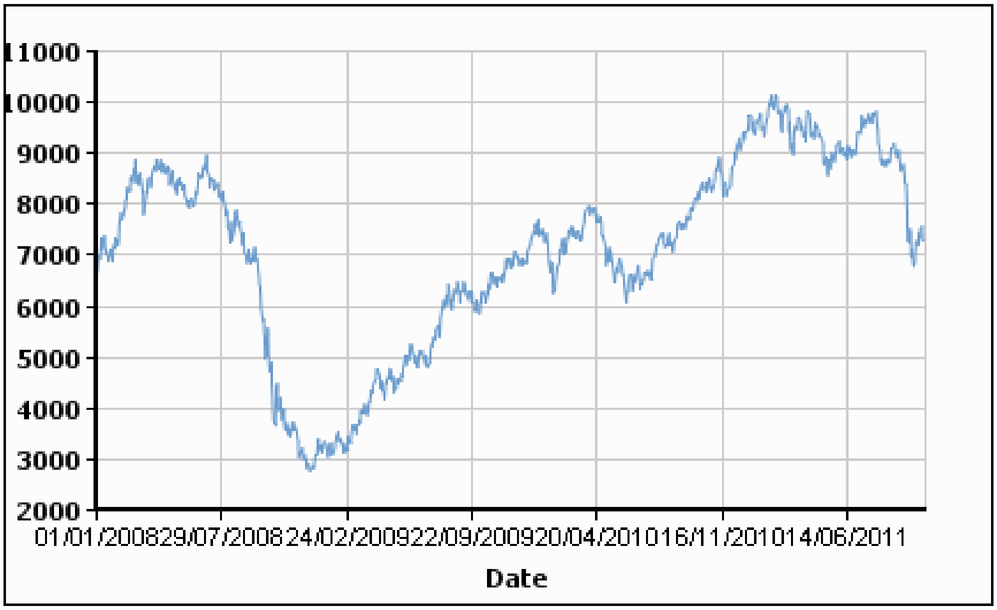


Figure 4.9: LME Copper price trend (US$ per tonne; www.lme.com)

Other Implications:

The contribution to the life of Nchanga Underground Mine by the TFQ ore project:

- Taking the current annual mine ore output of 2,300,000 tonnes and the recoverable TFQ ore tonnage of 60,595,168.64 tonnes: it would take the mine 26 years to exploit the TFQ ore potential (Appendix 4).

- Added together with the lower orebody (LOB) remaining reserves, with an estimated life span of 4 to 5 years currently, it would really add to a life span of close to 30 years for the mine. This would hence answer the question of sustained employment to local people in the town of Chingola and other service providers.

CHAPTER 5

5.0 DISCUSSION

Gathering of data involved mainly review of existing data on laboratory tests conducted previously and associated information from mapping and core logging records. Site visits and observations were also conducted in the TFQ-ore potential blocks especially where trial mining is already going on.

5.1 Geological Setup

The geological environment under which the copper mineralization (TFQ) occurs is well defined in the geological model of the orebody. Major geological features and structures associated with the geology of the orebody are highlighted in section 4.1. Prominent among these features is the famous Nchanga fault zone which is coupled with very weak and heavily weathered rock making it almost impossible to mine excavations which can stand up. This has caused a sterilization of ore reserves defined in the ore model as no mining blocks have ever been successfully developed around and near the fault zone. Other tonnages exist in the no pay ore zones with too low copper ore grades to be exploited profitably.

5.2 TFQ ore tonnages and grades

Block by block, ore evaluation from the TFQ ore model (i.e. Appendix 5) was done with help of personnel from the Mineral Resources department. All the TFQ potential blocks were evaluated to ascertain the ore and waste (BSS) tonnages and the associated copper ore grades. These are presented in Table 4.2.

The ore to waste ratio was determined to be 1 to 2. This implies that to recover a tonne of ore, two tonnes of waste, BSS, have to be removed first. A total of **75,743,960** tonnes of TFQ ore at 1.80% TCu was ascertained. Compared with the current production figures at Nchanga Underground mine with total copper grades averaging 1.65%, the TFQ ore project offers a big opportunity for continued operations.

5.3 Rock mass characteristics

Review of the rock mass parameters of the TFQ rock unit, as presented in section 4.2, revealed that it falls in the category of poor to very poor rock mass rating based on Bieniawski's Rock Mass Rating (RMR) and Laubscher's RMR. With this rock mass rating and high fracture presence, TFQ ore would fare well under the effect of caving and induced stresses through blasting. Details are presented in Table 4.1. TFQ falls under **Poor to Very Poor** in the rock mass rating based on Bieniawski, Laubscher and Barton's Q system ratings. With a high presence of joints, TFQ would easily give in to caving influence.

The banded sandstone (BSS) flow parameters (i.e. cohesion and angle of friction) are presented in Table 4.3 and found amenable to facilitate cave mining.

5.4 Drawpoint parameters

Owing to similar rock mass characteristics of the LOB and TFQ ore, which almost fall in the same rock mass rating of poor to very poor, same drawpoint spacing and size has been found to fit well for drawing of TFQ ore in the trial mining. Drawpoint spacing and size are presented in section 4.4. This, therefore, implies that no major changes are required to the drawpoint layout of the old infrastructure in the exhausted areas. Only minimal adjustments might be required in few drawpoints with broken and large brows. Concrete re-enforcements might be required to reduce the brows for easy control of flow of ground through the drawpoints.

Observed TFQ ore particle fragmentation, from the trial mining observations, ranges from few millimeter particles to about 0.5m in size. This is also similar to the particle range for the LOB ore. Going by drawpoint size determination guide (Kvapil, 1965, 1982), which indicates that it should be 3 to 6 times the size of the largest fragment, TFQ ore largest fragments of about 0.5m would require a maximum drawpoint size of 1.8m, (i.e. six times the size of largest fragments). This happens to be the size of the finger raises for the block cave mining as practiced at Nchanga Underground mine.

In terms of estimation of the height of caving, observations in the TFQ ore trial mining in the block, **2420 6W**, showed presence of the banded sandstone upper (BSSU) in drifts where TFQ ore was drawn to exhaustion. The BSSU lies above the TFQ rock unit in the stratigraphy. As illustrated in section 4.6, the BSSU is above the assay footwall (AFW) of the LOB at more than 155m. Hence the caving effect went above 155m above AFW of the LOB ore. This implies that the TFQ ore layer is caved completely. The TFQ ore therefore doesn't require any further caving operations.

5.5 Project economic analysis

The economic analysis of the TFQ ore project was done. The analysis estimated or found the project to be viable for copper prices above the breakeven price of US$5,737.22 per tonne of finished copper. At breakeven price, the project just pays back the cost incurred to realize the finished copper. As tabulated in Appendix 4, the costs involved comprise removal of the BSS up to surface and ore costs from extraction to realization including all attendant costs (i.e. support, overheads, etc) up to the point of sale.

Given the current copper prices at the international market (i.e. ranging from US$9,000.00 to US$10,000.00 per tonne), there is no better timing than now to realize maximum benefits or proceeds from the project.

NB: The BSS, which is generated as waste from the TFQ ore project, is another matter of concern or under investigation under another project at Nchanga mine, i.e. the Chingola Refractory Ore (CRO). BSS contains generally, low grade copper as demonstrated in Appendix 3B. The CRO project established an approach to treat and realize copper from the refractory ore and recommended construction on a new concentrator plant to treat it. Currently, this new concentrator is under plan for construction at Nchanga mine. Therefore the TFQ ore project waste (BSS), is actually an ore resource under the CRO project. This, therefore, would make the TFQ ore project render a double benefit to the company, (i.e. copper ore resource from TFQ and from BSS in CRO project).

CHAPTER 6

6.0 CONCLUSION AND RECOMMENDATIONS

The ‘The Feldspathic-Quartzite’ (TFQ) ore project has been assessed to be viable for Nchanga Underground mine. The discussion here is made by firstly a brief conclusion and then recommendations.

6.1 Conclusion

I. The geological environment in which the Nchanga copper mineralization occurs, i.e. various rock units, was as presented under section 4.1. An ore evaluation in the caved areas of Nchanga Underground mine for the potential TFQ ore resource was eventually done block by block. This is presented in Appendix 3A. Recognition was made regarding the presence of geological structures and features (section 4.1) which pose challenges to the mining activities, i.e. Nchanga fault and the attendant discontinuities.

II. The block by block evaluation of TFQ ore as well as BSS tonnages led to the determination of the actual ore available for exploitation, (i.e. 75,743,960 tonnes at 1.80% total copper). The ore to waste ratio was also determined from both the TFQ-BSS model tonnages and from the TFQ ore trial mining observations. The ore to waste ratio was determined to be one to two, (i.e. 1: 2), as tabulated under Chapter 4 of this report.

III. The TFQ copper ore grades were established for the various blocks and the weighted average for total copper content was at 1.80%. Results are as presented under Chapter 4 'Data Collection and Analysis' section and details are carried in Appendix 3A.

IV. With a view of ground characteristics in terms of ground conditions and stability of old working areas, were established through physical site inspections and appropriate support requirements estimated. These are carried under section 5.5. The rock mass classification of TFQ was found to be in the range Poor to Very Poor rating based on the three rockmass classification systems, i.e. Bieniawski, Laubscher and Barton's Q ratings. This gave implication that TFQ would easily give in to the effect of caving. This would be more so because of the presence of discontinuities or fractures in its rock mass. Estimation and determination of the frictional angle and cohesion factors in the BSS material flow was also done, i.e. 43˚ and 15kPa respectively.

V. Drawpoint size and spacing were also reviewed especially with TFQ ore characteristics established in terms of fragmentation spread (particle size distribution). Through the review it has been established that the same drawpoint size and spacing, (i.e. 1.8m and 2.3m respectively), used in the recovery of the LOB ore previously would give a good recovery of TFQ ore. This was deduced due to the similarities in the LOB and TFQ ore rock mass characteristics, (i.e. fragmentation and rock mass ratings). The height of caving was demonstrated in **2420 6W** block where drifts have been drawn to exhaustion. The drifts registered flow of banded sandstone upper (BSSU) which lies above TFQ.

VI. TFQ ore project was evaluated to establish its economic viability. It was established and determined as presented under the results section that the project is economically viable more especially at the current international copper prices ranging above US$7,000.00. The breakeven copper price was established at US$5,737.22 per tonne. Therefore, the TFQ ore project offers an additional copper ore resource potential and opportunity to Konkola Copper Mines Plc besides other ore resources being exploited at the moment. This would result in more revenue for the company as well as continued operations of Nchanga Underground mine whose life of mine was predicted to close by the year 2017 (Nchanga Underground Life of Mine reports, 2006). This is an opportunity to sustain employment for the many permanent and contractor workers based at Nchanga mine. Ultimately, the government of the Republic of Zambia would stand to benefit also from Nchanga Underground continued operations through taxes from the employees and other mineral royalties payable by the investor.

6.2 Recommendations

- Extraction and draw of TFQ ore should be continued with the same mined scraper drifts and drawpoints used to extract the LOB ore. As alluded to under results and discussion, LOB and TFQ ore have almost similar rock mass characteristics hence the design parameters of the drawpoints will enhance good ore recovery of the TFQ too. Increasing the spacing of drawpoints, e.g. by sealing a pair of drawpoints in every set of two pairs, would result in increased ore losses as the draw cones on top of the crown pillars between the drawpoints would be increased tremendously. This would reduce the recoverable ore tonnages and hence compromise the viability of the TFQ ore project. The other option of enlarging the drawpoint to more than 1.8m in size would reduce the crown pillars between drawpoints. This would compromise the strength of crown pillars and hence their ability to stand the load of caved material resting above.

- Since the sequence of recovery starts with drawing of BSS to expose the copper ore (TFQ) above, removal of BSS should begin at the down-dip drifts progressing to up-dip drifts. This is because material flow is more rapid down dip due to the force of gravity and hence higher waste tonnages would be recovered at the down dip drifts. This would minimize the effect of dilution if drawing of BSS is commenced from the up dip drifts.

- To avoid early dilution of the ore during production draw, all the drifts in a production block should be fully exposed in TFQ. Premature production draw in a scraper drift with neighbouring drifts still in BSS would lead to the BSS percolating through and diluting the drift on production. This is because BSS particles are generally finer than ore fragments and flow much quicker and with ease in a column of draw.

- Reduction of large drawpoint brows should be done to normalize them to standard opening sizes, i.e. to 1.8m. This would enhance control of material in each drawpoint especially to overcome early dilution.

- Constant monitoring of material drawn from the drawpoints should be in place to ensure drawing of BSS and TFQ is controlled evenly to avoid loss of ore due to early dilution.

- To get a more clear indication of material flow, recommendation is being made for further examination of all the rock units encountered in project and acquisition of softwares (i.e. Particle Flow Code 3D, PFC3D and others on the market) that can simulate the material flow once all parameters have been established.

REFERENCES

Barton, N.R., Lien, R. and Lunde, J. 1974. *Engineering classification of rock masses for the design of tunnel support*. Rock Mechanics; vol. 6, No. 4, pp189-239.

Bieniawski, Z.T. 1973. *Engineering classification of jointed rock masses*. Transactions of the South African Institution of Civil Engineers; vo1.5, pp335-344.

Bieniawski, Z.T. 1976. *Rock mass classification in rock engineering*. In Exploration for rock engineering, proceedings of the symposium, (ed. Z.T. Bieniawski), vol. 1, pp 97-106. Cape Town: Balkema.

Bieniawski, Z.T. 1989. Engineering rock mass classifications. New York: Wiley.

Brady, B.H.G. and Brown, E.T., 1985. *Rock Mechanics for Underground Mining*. London: George Allen and Unwin Ltd.

Brown, E.T., 2003. *Block Caving Geomechanics*. Brisbane: Julius Kruttschnitt Mineral Research Centre.

Brown, E.T., 2007. *Block Caving Geomechanics*. 2nd Edition. Brisbane: Julius Kruttschnitt Mineral Research Centre.

Bull, G. and Page, C.H., 2000. *Sublevel Caving – Today's dependable low-cost ore factory'*, in Massmin 2000 proceedings, Brisbane; 29th October-2nd November, pp537-556.

Chileshe, P.R.K., 1992. *An evaluation of stress development around mining excavations in the Zambian Copperbelt*. PhD Thesis (unpublished): University of Wales of Cardiff, Wales, UK.

Chileshe, P.R.K. and Phiri, S.N., 1994. *Reviewed production from previously exhausted block caving areas at Nchanga mine, Zambia*. In proceedings of 15th Congress of Mining and Metallurgy Institution, Johannesburg, South Africa, Vol. 1, SAIMM pp237-245.

Craig, R.F., 1987. Soil Mechanics. London: Van Nostrand Reinhold (International).

Deere, D.U. and Deere, D.W. 1988. The rock quality designation (RQD) index in practice. In Rock classification systems for engineering purposes, (ed. L.Kirkaldie), ASTM Special Publication 984, 91-101. Philadelphia: Am. Soc. Test.Mat.

Garrad P., 1995. *Geology of the Chingola Area: Explanation of degree Sheet 1227. SE Quart, No. 66. Lusaka*: Geological Survey Department of Zambia.

Hoek E. and Brown E.T., 1990. Underground Excavations in Rock. London: Institution of Mining and Metallurgy.

Hoek, E., 2007 Edition. *Practical Rock Engineering*. Unpublished; (http://www.rocscience.com/hoek/PracticalRockEngineering), accessed on 18th May, 2011.

Laubscher, D.H., 2000. *Block cave manual, design topic: drawpoint spacing and draw control*. Julius Kruttschnitt Mineral Research Centre, The University of Queensland, Brisbane, Australia??????

Hudson, J.A. and Harrison, P.J., 2000. *Engineering Rock Mechanics - An Introduction to the Principles*. 2nd Edition. London: Elsevier Science Ltd.

Janelid, I. and Kvapil, R., 1966. *Sublevel Caving*. International journal of Rock Mechanics and Mining Sciences. vol.3, pp129-153.

KCM, 2000. *Nchanga Underground (NUG) Training manual*. (Unpublished report): Chingola.

KCM Geology Department, 2000. *Nchanga Underground Mineral Resources reports for Konkola Copper Mines Plc*. (Unpublished reports). Chingola.

KCM Geotech Department, 2006. *Nchanga Underground geotechnical reports*. (Unpublished reports). Chingola.

KCM, 2010. *Konkola Copper Mines plc policy document*. (Unpublished report): Chingola.

Kvapil, R. (1965). Gravity flow of granular material in hoppers and bins. *International Journal of Rock Mechanics and Mining Sciences, 2*(1), 25-41 & *2*(3), 277-304.

Kvapil, R., 1982. *Sublevel Caving*. SME Mining Engineers Handbook. 2nd Edition (Ed: H L Hartman), pp1789-1814, (SME).

Kvapil, R., 1982. *The Mechanics and Design of Sublevel Caving Systems*. Underground Mining Methods Handbook (Ed: W A Hustrulid), pp880-897, (SME).

Kvapil, R. (2008). *Gravity flow in sublevel and panel caving – a common sense approach*. Luleå, Sweden: Luleå University of Technology.

Kvapil, R. (1998). *The mechanics and design of sublevel caving systems*. In R.E. Gertsch & R.L. Bullock, Techniques in underground mining. Selections from underground mining methods handbook (pp. 621-653). Littleton, USA: Society for Mining. Metallurgy, and Exploration, Inc.

Laubscher, D.H. 1977. *Geomechanics classification of jointed rock masses – mining applications*. Transactions of the Institution of Mining and Metallurgy. 86, A1-8.
Laubscher, D.H. and Taylor, H.W. 1976. *The importance of geomechanics classification of jointed rock masses in mining operations*. In Exploration for rock engineering, (ed. Z.T. Bieniawski) **1**, 119-128. Cape Town: Balkema.

Laubscher, D.H. and Page, C.H. 1990. *The design of rock support in high stress or weak rock environments*. Proc. 92nd Can. Inst. Min. Metall. AGM, Ottawa, Paper # 91.

Laubscher, D.H., 1994. *Cave Mining – state of the art*. The Journal of the South African Institute of Mining and Metallurgy, Vol. 94 No. 10; pp279-293.

Laubscher, D,H., 2006. *Cave Mining Handbook*, accessed from (http://www.brc.uq.edu.au/) on 11th August 2010.

Marinos, P. and Hoek, E., 2005. *The Geological Index: Applications and Limitations*. Bull Engineering Geol. Environment, Vol. 64. pp 54 – 65.

Nchanga Underground Planning Section, 2006. Life of mine reports. (Unpublished reports). Chingola.

Nchanga Underground, 2000. Cave control manual. (Unpublished report). Chingola.

Palmstrom, A. 1982. *The volumetric joint count - a useful and simple measure of the degree of rock jointing*. Proceedings of the 4th congress of International Association of Engineering Geology, Delhi Vol. 5, pp221-228.

Palmstrom, A., 2005. *Measurement of and Correlation between Block Size and Rock Quality Designation (RQD)*. The Tunneling and Underground Space Technology Journal. Vol. 20, pp 362-376.

Power, G.R., 2004. *Modeling Granular Flow in Caving Mines: Large Scale Physical Modeling and Full Scale Experiments*. PhD thesis (unpublished), University of Queensland, Brisbane.

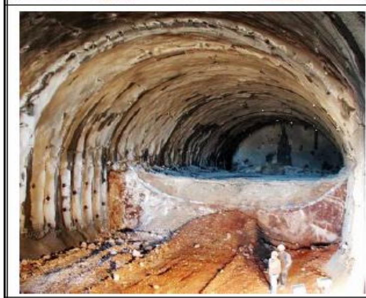
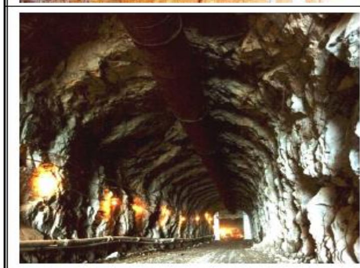
Selley, D. and Broughton, D., 2005. *Sequence and carbon isotropic stratigraphy of Neoproterozoic Roan Group Strata of the Zambian Copperbelt*; Precambrian Research, 190(1-4) pp 70-89.

Terzaghi, K. 1946. *Rock defects and loads on tunnel supports*. In Rock tunneling with steel supports, (eds R. V. Proctor and T. L. White) 1, 17-99. Youngstown, OH: Commercial Shearing and Stamping Company.

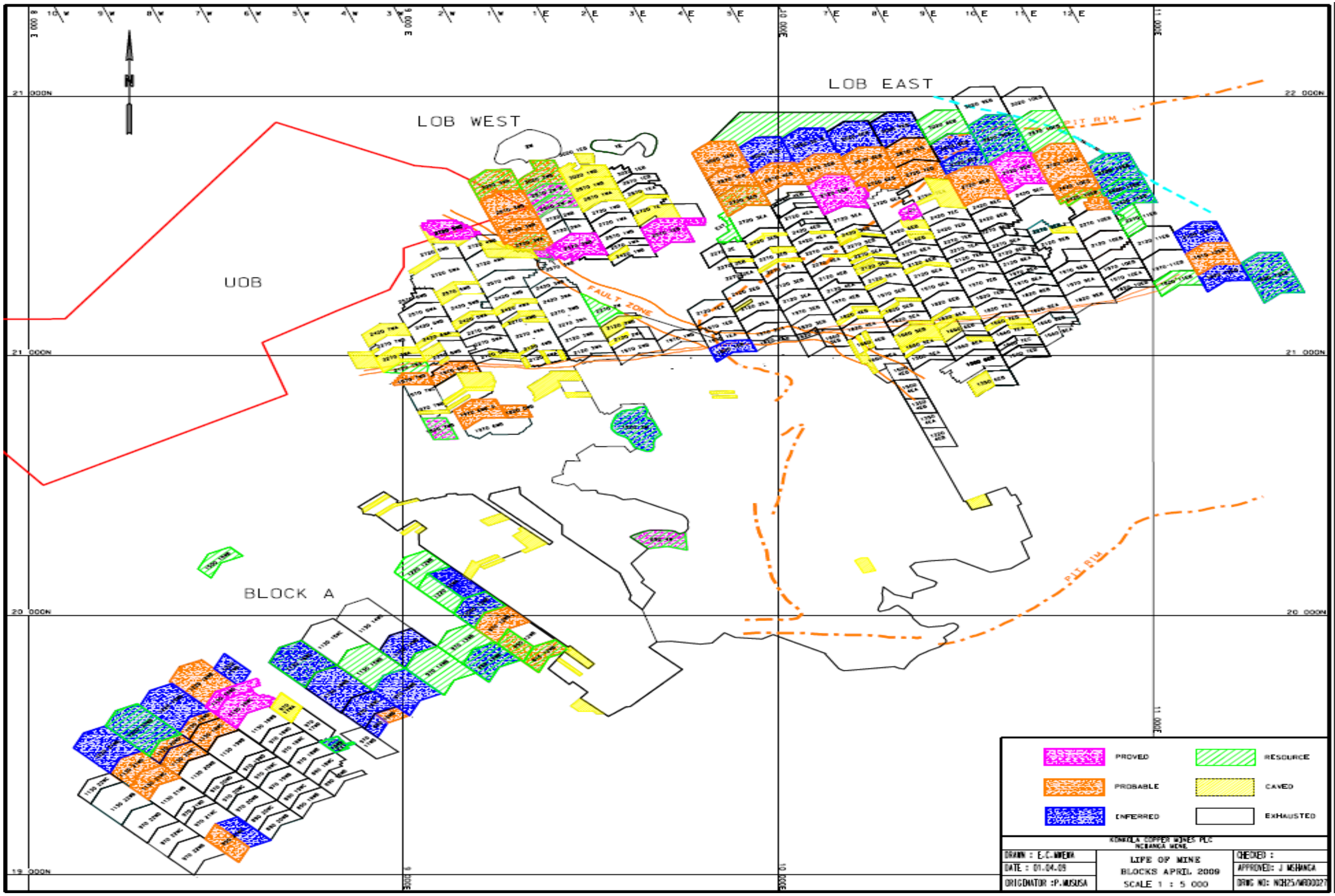
Underground Mine Training School, 2000. *History of Nchanga Mine*. (Unpublished Report). Chingola.

APPENDICES

Appendix 1: Determination of disturbance factor, D

Appearance of rock mass	Description of rock mass	Suggested value of D
	Excellent quality controlled blasting or excavation by Tunnel Boring Machine results in minimal disturbance to the confined rock mass surrounding a tunnel.	$D = 0$
	Mechanical or hand excavation in poor quality rock masses (no blasting) results in minimal disturbance to the surrounding rock mass. Where squeezing problems result in significant floor heave, disturbance can be severe unless a temporary invert, as shown in the photograph, is placed.	$D = 0$ $D = 0.5$ No invert
	Very poor quality blasting in a hard rock tunnel results in severe local damage, extending 2 or 3 m, in the surrounding rock mass.	$D = 0.8$
	Small scale blasting in civil engineering slopes results in modest rock mass damage, particularly if controlled blasting is used as shown on the left hand side of the photograph. However, stress relief results in some disturbance.	$D = 0.7$ Good blasting $D = 1.0$ Poor blasting
	Very large open pit mine slopes suffer significant disturbance due to heavy production blasting and also due to stress relief from overburden removal. In some softer rocks excavation can be carried out by ripping and dozing and the degree of damage to the slopes is less.	$D = 1.0$ Production blasting $D = 0.7$ Mechanical excavation

Appendix 2: Plan showing caved areas



APPENDIX 3A: TFQ ore - evaluated blocks

	RESOURCES			
		UNFACTORED		
0	BLOCKS	TONNES	TCU	ASCU
1	1820B1E	313830.7	2.389	1.345
2	1970A 1E	123608.5	2.171	1.269
3	1970B 1E	404670.4	2.554	1.53
4	2120A 1E	624454.2	4.488	2.295
5	2570B 1E	885770.7	0.819	0.801
6	2720 1E	946735.3	0.803	0.766
7	2870A 1E	334579.7	0.395	0.156
8	2870B 1E	236407.1	0.395	0.134
9	3020B 1E	214545	0.446	0.103
10	1500 2E	1070919	1.67	0.24
11	1660 2E	648386	1.49	0.46
12	1820 2E	546990	1.75	0.49
13	1970 2E	688203	1.99	0.83
14	2120 2E	1223194	2.24	1.28
15	2270 2E	377882	1.27	0.78
16	2420 2E	534152	1.14	0.67
17	2720 2EA	226944	1.14	0.796
18	1500 3E	822457	1.3	0.1
19	1660 3E	869356	1.48	0.57
20	1820 3E	919557	1.7	0.89
21	1970 3E	407440	1.81	0.92
22	2120 3E	1036457	1.81	1.06
23	2270 3E	799494	1.38	0.83
24	2420 3E	688692	1.06	1.0
25	2720 3EA	826516	0.99	0.9
26	2720 3EB	631127	1.1	0.8
27	1500 4E	837994	1.48	0.51
28	1660 4E	735760	1.3	0.37
29	1820 4E	696827	1.27	0.47
30	1970 4E	544544	1.4	0.56
31	2120 4E	832278	1.36	0.64
32	2270 4E	703369	1.35	0.48
33	2420 4E	815579	1.3	0.73

34	2720 4EA	469161	1.09	0.71
35	2720 4EB	653973	1.09	0.62
36	1500 5E	316796	2.26	0.89
37	1660 5E	656287	1.85	0.69
38	1820 5E	692795	1.49	0.48
39	1970 5E	526127	1.51	0.69
40	2120 5E	1110285	1.42	0.76
41	2270 5E	599247	1.89	0.78
42	2420 5E	495347	3.23	0.98
43	2720 5EA	620882	1.98	0.75
44	2720 5EB	932818	0.87	0.58
45	1500 6E	615739	1.86	0.52
46	1660 6E	971282	1.98	0.74
47	1820 6E	815864	1.75	0.41
48	1970 6E	412300	2.29	0.47
49	2120 6E	1171508	1.97	0.93
50	2270 6E	779045	1.74	0.86
51	2420 6E	522556	2.03	0.92
52	2720 6EA	713545	2.38	0.9
53	2720 6EB	361150	0.6	0.38
54	1500 7E	625046	2.8	0.65
55	1660 7E	755999	2.58	0.52
56	1820 7E	786480	1.87	0.32
57	1970 7E	684349	1.98	0.47
58	2120 7E	1117046	1.88	0.76
59	2270 7E	861323	1.83	0.93
60	2420 7E	914045	1.4	1.12
61	2720 7EA	1029247	1.37	0.83
62	2720 7EB	559061	0.43	0.16
63	1820 8E	752838	1.97	0.64
64	1970 8E	905250	2.93	0.9
65	2120 8E	617006	2.54	0.62
66	2270 8E	928493	1.99	0.36
67	2420 8E	771853	1.46	0.12
68	1970 9E	1029753	2.61	0.66
69	2120 9E	898383	1.72	0.35
70	2270 9E	347358	1.21	0.08
71	2420 9E	316567	1.23	0.07
72	1820 10E	308194	4.03	0.77

73	1970 10E	662173	2.27	0.51
74	2120 10E	987565	1.63	0.6
75	2270 10E	622078	1.33	0.5
76	2420 1WB	341237	1.028	0.661
77	2570 1WA	240508	0.893	0.745
78	2570 1WB	517443	0.708	0.674
79	2720 1WA	438122	0.727	0.4
80	2720 1WB	391438	1.06	0.157
81	2870 1WA	297696	1.078	0.1
82	2870 1WB	284863	0.96	0.101
83	2120 2WB	385224	3.023	1.066
84	2120 2WA	453857	4.245	1.095
85	1970 2WB	479034	1.728	1.028
86	2270 2WA	53009	1.8	1.15
87	2270 2WB	226690	1.98	1.161
88	2570 2WB	709435	0.602	0.535
89	2720 2WA	524850	0.613	0.489
90	2720 2WB	433583	1.42	0.494
91	2870 2WA	492242	1.943	0.506
92	2870 2WB	481237	1.659	0.41
93	3020 2WB	512429	0.878	0.287
94	1500 3WB	453816	3.799	1.529
95	1660 3WA	384157	3.441	1.356
96	2120 3WA	360203	1.515	1.152
97	2120 3WB	597715	1.947	1.343
98	2270 3WA	325556	2.18	1.421
99	2270 3WB	440248	1.449	1.047
100	2420 3WA	394093	1.436	1.004
101	2420 3WB	276515	1.863	1.26
102	2720 3WA	510385	0.811	0.299
103	2720 3WB	681819	2.421	0.998
104	2870 3WB	1179303	3.14	0.925
105	2120 4WA	303070	1.276	0.911
106	2120 4WB	403178	1.397	0.816
107	2270 4WA	421441	2.556	0.901
108	2270 4WB	406551	3.171	1.288
109	2420 4WA	358321	3.458	1.415
110	2420 4WB	459327	3.643	1.683
111	2570 4WB	507783	2.93	1.614

112	2720 4WA	861995	0.63	0.215
113	2720 4WB	490181	1.048	0.976
114	2120 5WB	309571.9	1.701	0.913
115	2270 5WA	394487.4	1.599	0.634
116	2270 5WB	353433.4	1.575	0.965
117	2420 5WA	482827.9	2.493	1.104
118	2420 5WB	420428.9	2.74	1.14
119	2570 5WB.	402881.8	2.84	0.922
120	2720 5WA	723775.8	2.619	0.56
121	2720 5WB	471525.4	1.99	0.827
122	1970 6WB	406041.6	1.548	1.028
123	1970 6WD	342078.7	1.682	0.86
124	2270 6W	585414.8	1.821	0.823
125	2420 6WA	299681	2.509	1.043
126	2420 6WB	308380.9	2.525	0.914
127	2570 6WA	369379.9	2.588	0.739
128	1820 7WB	134890.8	1.158	0.786
129	1970 7WD	131181.2	1.494	0.711
130	1970 7WB	180197.7	1.119	0.677
131	1970 7WC	158033.7	1.559	0.77
132	2120 7WA	172470.9	1.61	0.645
133	2270 7WA	181151	2.363	0.681
134	2270 7WB	186227.5	3.005	0.557
135	2420 7WA	165812	3.792	0.323
		75,743,960.80	**1.79**	**0.74**

APPENDIX 3B: Banded Sandstone (BSS) from the Model

TONNES	TCU	ASCU	TCO	ASCO	DENSITY
372	0.13	0.04	0	0	2.45
15,132	0.12	0.05	0	0	2.45
57,291	0.11	0.05	0	0	2.45
150,265	0.56	0.28	0.08	0.04	2.45
368,452	1	0.44	0.12	0.06	2.45
534,371	1.04	0.46	0.12	0.05	2.45
702,203	1.01	0.46	0.11	0.04	2.45
969,567	0.88	0.4	0.11	0.03	2.45
1,384,134	0.74	0.32	0.11	0.02	2.45
1,954,504	0.61	0.29	0.09	0.01	2.45
2,336,229	0.55	0.28	0.07	0.01	2.45
2,511,802	0.55	0.3	0.06	0.01	2.45
2,515,947	0.58	0.31	0.05	0.01	2.45
2,389,924	0.64	0.33	0.04	0.01	2.45
2,330,069	0.71	0.36	0.04	0.02	2.45
2,506,114	0.7	0.35	0.04	0.02	2.45
2,895,974	0.65	0.34	0.05	0.03	2.45
3,464,240	0.56	0.3	0.05	0.03	2.45
3,934,889	0.48	0.25	0.04	0.03	2.45
4,330,244	0.44	0.23	0.03	0.02	2.45
4,577,277	0.42	0.21	0.03	0.02	2.45
4,717,157	0.45	0.21	0.03	0.02	2.45
5,139,180	0.53	0.25	0.02	0.01	2.45
5,677,725	0.58	0.26	0.02	0.01	2.45
6,001,793	0.6	0.26	0.03	0.01	2.45
6,247,601	0.63	0.25	0.03	0.01	2.45
6,767,715	0.7	0.27	0.04	0.01	2.45
7,204,646	0.74	0.3	0.05	0.01	2.45
7,419,338	0.79	0.32	0.06	0.02	2.45
7,336,195	0.92	0.38	0.07	0.02	2.45
7,057,199	1.06	0.43	0.07	0.02	2.45
6,877,828	1.09	0.45	0.07	0.02	2.45
6,771,821	1.13	0.5	0.06	0.02	2.45
6,635,910	1.14	0.54	0.06	0.02	2.45
6,063,941	1.14	0.57	0.04	0.02	2.45
5,572,402	1.11	0.58	0.02	0.01	2.45

5,265,427	1.07	0.55	0.01	0.01	2.45
5,210,868	0.99	0.5	0.01	0.01	2.45
5,488,959	0.94	0.46	0.01	0.01	2.45
5,718,967	0.86	0.41	0.01	0.01	2.45
5,906,008	0.73	0.36	0.01	0.01	2.45
5,980,559	0.64	0.34	0.01	0.01	2.45
5,834,047	0.58	0.31	0.02	0.01	2.45
5,638,852	0.57	0.3	0.02	0.01	2.45
5,493,682	0.61	0.32	0.03	0.01	2.45
5,385,331	0.64	0.34	0.05	0.02	2.45
5,610,892	0.69	0.35	0.05	0.03	2.45
5,639,272	0.7	0.36	0.06	0.04	2.45
5,661,445	0.71	0.36	0.07	0.05	2.45
5,733,857	0.72	0.38	0.09	0.06	2.45
5,899,865	0.74	0.39	0.1	0.07	2.45
6,018,514	0.74	0.37	0.11	0.08	2.45
6,190,969	0.75	0.34	0.12	0.09	2.45
6,324,652	0.75	0.33	0.13	0.09	2.45
6,450,594	0.73	0.31	0.13	0.09	2.45
6,393,706	0.74	0.33	0.14	0.09	2.45
6,150,872	0.74	0.34	0.13	0.09	2.45
6,166,347	0.77	0.37	0.13	0.08	2.45
6,412,734	0.81	0.37	0.12	0.06	2.45
6,831,948	0.8	0.35	0.11	0.04	2.45
7,488,934	0.78	0.33	0.09	0.03	2.45
7,989,511	0.75	0.3	0.08	0.02	2.45
8,169,430	0.71	0.28	0.06	0.01	2.45
7,712,990	0.7	0.25	0.05	0.01	2.45
7,390,714	0.66	0.23	0.04	0.01	2.45
7,248,030	0.66	0.23	0.04	0.01	2.45
7,063,496	0.65	0.23	0.03	0.01	2.45
7,018,462	0.66	0.23	0.03	0.01	2.45
6,856,542	0.67	0.24	0.04	0.01	2.45
6,189,801	0.68	0.24	0.04	0.01	2.45
5,524,291	0.7	0.23	0.05	0.01	2.45
4,924,834	0.73	0.23	0.05	0.02	2.45
4,250,304	0.75	0.23	0.06	0.02	2.45
3,460,472	0.81	0.23	0.06	0.02	2.45
2,887,196	0.8	0.24	0.07	0.01	2.45

2,580,663	0.77	0.27	0.1	0.01	2.45
2,500,334	0.78	0.31	0.15	0.01	2.45
2,595,408	0.88	0.36	0.22	0.01	2.45
2,630,960	1.06	0.41	0.26	0.01	2.45
2,569,997	1.22	0.45	0.26	0.01	2.45
2,488,400	1.29	0.45	0.25	0.01	2.45
2,515,959	1.28	0.49	0.22	0.01	2.45
2,357,224	1.25	0.55	0.17	0.01	2.45
1,861,588	1.17	0.59	0.13	0.01	2.45
1,297,521	1.18	0.65	0.11	0.01	2.45
741,928	1.22	0.67	0.07	0.01	2.45
281,328	1.25	0.59	0.02	0.01	2.45
99,893	1.12	0.45	0.01	0.01	2.45
44,464	1.05	0.44	0.01	0.01	2.45
9,032	0.89	0.31	0.01	0.01	2.45
395,577,509	**0.77**	**0.34**	**0.07**	**0.03**	

Appendix 4: TFQ ore project economic analysis

TFQ ORE MINING PROJECT- ECONOMIC ANALYSIS (July 2011):					
UNIT COSTS					
extraction_cost	5.64	US $	Per tonne of ore hoisted		
Hoisting_Cost	3.09	US $	Per tonne of ore hoisted		
Tramming_Cost	3.15	US $	Per tonne of ore hoisted		
Overheads_cost	3.96	US $	Per tonne of ore hoisted	0.99	TFQ takes 25%
Support_cost	2500	US $	Per metre of support	5.77	$/t
Concentrator_cost	2.84	US $	Per tonne of ore milled		
TLP_Costs	5.12	US $	Per tonne of ore treated		
Refinery_cost	0.04	US $	Per pound of Refinery finished copper	88.18	$/t
Smelter_cost	0.21	US $	Per pound of Refinery finished copper	463.00	$/t
Realisation_Cost	0.071	US $	Per pound of copper sold	156.53	$/t
Div_cost	5.20	US $	Per tonne of ore hoisted	1.30	TFQ takes 25%
Corporate_cost	1.49	US $	Per tonne of ore hoisted	0.37	TFQ takes 25%
LTEP	1.540	US $	Per pound of copper sold		
Cathode_Rec	65.00	%	Per tonne of As Cu copper in ore hoisted		
AnodeRec	68.00	%			
NB: 1tonne = 2204.623lb					
Overall ore cost/tonne:			**Respective Costs:**		
1. Hoisted & Milled tonnes	28.27		1,713,025,417.45		
2. Finished Cu tonnes	707.68		515,358,643.67		
			2,228,384,061.12		
			-		
TFQ ORE					
TONNES	**%TCU**	**%ASCU**	**Recoverable Ore Tonnage (@ 80%)**	**TCu Tonnes**	**Finished Cu Tonnes**
75,743,960.80	**1.8**	**0.74**	**60,595,168.64**	**1,090,713.04**	**728,232.74**
				ASCU Tonnes:	291,462.76
WASTE -BSS				AICU Tonnes:	436,769.98
TONNES:	**151,487,921.60**				
WASTE - UNIT COSTS:					
extraction_cost	5.64	US $	Per tonne of ore hoisted		
Hoisting_Cost	3.09	US $	Per tonne of ore hoisted		
Tramming_Cost	3.15	US $	Per tonne of ore hoisted		
Overheads_cost	3.96	US $	Per tonne of ore hoisted	0.99	TFQ /BSS takes 25%
Overall BSS cost/tonne	12.87				
Total Ore Costs:	**2,228,384,061.12**				
Total Waste Costs:	**1,949,649,550.99**		**Projected Life Span to the Mine (Yrs):**		
OVERALL COST	**4,178,033,612.11**		Assuming current NUG annual output of 2,300,000t from recoverable ore tonnes:	26.35	
Break Even Price ($/t):	**5,737.22**				
Break Even Price ($/lb):	2.60				
Revenue ($) @ $8,500/t:	**6,189,978,262.08**				
Proceeds ($)	**2,011,944,649.97**				

Appendix 5: TFQ - BSS section Model