# Assessment of Geotechnical Considerations for Ground Control and Stability at Nkana Synclinorium Copper Mine Mopani Plc, Zambia

**Barnabas Mpaka**

A thesis submitted to the University of Zambia in partial fulfillment of the requirements for the degree of Master of Engineering in Rock Mechanics

The University of Zambia

Lusaka, Zambia

February, 2024.

## DECLARATION

I, Barnabas Mpaka, do hereby declare that this thesis has been written in accordance with the rules and regulations governing the award of a Master's degree from the University of Zambia.

I further declare that this thesis is the result of my own work. Where use has been made of other people's work, it has been acknowledged. This thesis has not been submitted for the award of degree, or diploma or other qualification at University of Zambia or any other university.

Signed: ………………………………. Date: ……………………………….

## APPROVAL

This thesis of Barnabas Mpaka has been approved as fulfilling the requirements for the award of the degree of Master of Engineering in Rock Mechanics by the University of Zambia.

**Supervisor**______________________________________________________________

**Name:** ……………………………. **Signature:** ………………….. **Date:** …………………

**Internal Examiner (1)** __________________________________________________________

**Name:** ……………………………. **Signature:** ………………….. **Date:** …………………

**Internal Examiner (2)** ________________________________________________________

**Name:** …………………………… **Signature:** …………………. **Date:** …………………..

**External Examiner** _________________________________________________________

**Name:** …………………………… **Signature:** ………………. **Date:** ……………………

**Thesis Chairperson** __________________________________________________________

**Name:** …………………………… **Signature:** …………………… **Date:** ………………..

## ABSTRACT

Nkana Synclinorium Mine is in the Southern part of the Nkana mining license area in Kitwe. The mine has been an underground operation since 1930 when the first shaft was sunk at Central Shaft. Vertical Crater Retreat (CVR) and Sub-Level Curving (SLC) are the main copper ore mining methods used at the mine. Due to increase in mine depth by 137m, from 3510ft to 3960ft (1.070 to 1.207km), mining operations have been encountering geotechnical challenges related to ground control and ground stability.

This study applied Empirical methods (*Barton's Q system, 1974*, *Bieniawski (1973) classification and* Hoek, 1994) for rock mass classification**.** Scanline mapping of geological structures was done to identify joints, bedding planes, schistosity and folds. The mapping was carried out along the crosscuts or tunnel walls at a 1.50m grade line elevation with geological features being picked along a stretched 100m tape. Borehole cores were examined for geological formations, grain size, colour, joints, and RQD. The assessment of rock reinforcement and surface support elements like cable bolts and shotcreting were done by using pull tests and uniaxial compressive strength (UCS) tests. Pull Testing was done in selected mine excavations to provide a quantitative measure of the relative performance of different anchor systems and compressive strength tests on concrete cubes were used to assess the quality and strength of the material used in shotcreting underground. Collection of survey data for over-break and under-break in tunnels mined underground was done by the author with the assistance of mine surveyors. The Theodolite was the major instrument used. Numerical modelling using MAP3D and Finite Element Analysis (FEA) softwares was used to assess ground stability around underground excavations. Evaluation of design systems for ground support and pillars was carried out with the reviewing of current design flow charts in comparison with design charts from literature for other mines around the world. Results of study indicate that the rock mass ratings (RMRs) for the samples of Basal Quartzite (BQ), Foot Wall Conglomerate (FCON), South Orebody Shale (SOBS) and Hanging Wall Argillite (HWA) rock formations compared well with Barton's Q system RMR ranges except for Foot Wall Sandstone (FSAN) which was out of range. All pull tests conducted at 3 sites had an average failure rate of 22.3% while compressive tests gave a high failure rate of 83.3%. Numerical analysis using Finite Element Analysis (FEA) software indicate more stresses and displacement around the excavation before rock bolt support, and less stresses and displacement after rock bolts are installed around the same excavation. Over-break and under-break data did not show any correlation with mined linear meters of the tunnels. Geological factors like jointing, weathering and shearing as well as drill-and-blast challenges were attributed to over-break and under-break. In conclusion, folds and joints are major factors affecting ground and stability at Synclinorium mine, and the geotechnical database has insufficient data. The confidence of the current geotechnical database at Synclinorium mine needs to be improved through more data collection. Design excavation and ground support designs need to be improved and matched with the real situation underground. The key issues affecting drill-and-blast like varying powder factors in the shots, lack of pragmatic consideration of geological and geotechnical information in the design, and diligent adherence to the blast designs need to be addressed.

## DEDICATION

This thesis is dedicated to my wife and siblings

This thesis is sincerely dedicated to my supportive wife, Alice, who encouraged and inspired me in conducting this study. She has never left my side throughout the process and gave me strength and hope when I thought of giving up. She provided me a great sense of enthusiasm and perseverance. Without her love and assistance, this research would not have been made possible. Moreover, I also dedicate this thesis to my young sister, Prisca, who sometimes helped me with accommodation while studying. I really appreciate your words of advice, and your continuously giving me moral, and emotional support. And lastly, I dedicate this thesis to the Almighty God who gives me strength, wisdom, guidance, power of thinking, security, competence, and who gives me good health.

## ACKNOWLEDGEMENT

I am very thankful for every effort that Dr. Victor Mutambo, who was my supervisor and Head for Department of Mining Engineering in, the School of Mines, put into guiding and supervising me during my research work. Words can barely express my gratitude for his invaluable patience and feedback. I also could not have undertaken this journey without Professor Emmanuel Chanda and Dr. Sam Kangwa who were very welcoming upon approach for any academic related facilitation and guidance at their offices. They would generously provide knowledge and expertise.

Additionally, this endeavor would not have been possible without the generous support from Dr. B. Besa and the entire School of Mines administration which he has overseen as the Dean. I have no doubt that the hospitality of staff at the school helped facilitate my registration for post graduate research.

All my workmates in the Mopani Technical department, namely Hugo Nicosia, Warren Mwenya, Moono Mwila, Kelvin Sichalwe and Denison Mwewa went to great lengths in one way or the other to help me in data collection underground for which I deeply thank them. Although the research was self-funded, Mopani Copper mines management provided a conducive environment for me to do my works and should be thanked for that.

I am also grateful to my family for their continuous support and encouragement during my research work at both the University of Zambia and Mopani Copper mines.

**TABLE OF CONTENTS**

## LIST OF FIGURES

## LIST OF TABLES

## ACRONYMS

| ACRONYMS | DESCRIPTION |
|---|---|
| RMR | Rock Mass Rating |
| RQD | Rock Quality Designation |
| GSI | Geological Strength Index |
| PLSI | Point Load Strength Index |
| SOB | South Ore Body |
| AHW | Assay Hanging Wall |
| AFW | Assay Foot Wall |
| ARG | Argillite |
| FWCONG | Foot Wall Conglomerate |
| CONG | Conglomerate |
| FWSAND | Foot Wall Sandstone |
| BQ | Basal Quartzite |
| UCS | Universal Compressive Strength |
| PPF | Perimeter Power factor |
| VCR | Vertical Crater Retreat |
| POF | Probability Of Failure |
| SLC | Sub Level Curving |
| FOS | Factor Of Safety |
| FOG | Fall Of Ground |
| MRMR | Mining Rock Mass Ratings |
| XC | Cross Cut |
| LDR | Loader Drive |
| RMS | Rock Mass Strength |
| SF | Safety Factor |
| RMS | Rock Mass Strength |

# CHAPTER 1-Introduction

## 1.1 Introduction

This chapter discusses the description of Synclinorium as a geological structure, its geographical and underground locations as well as how it impacts ground conditions. Some of the associated geotechnical considerations that need to be assessed and evaluated against application of standard geotechnical practices are also discussed. The chapter further discusses the background, problem statement, objective of the research, research questions and the significance of this study.

## 1.2 Background

The Synclinorium refers to the highly folded mineralized area at the keel of the Nkana Syncline currently extending from sections 1100mS at South Orebody (SOB) to zero at Central, then below 3140ft Level at SOB and 3360ft Level at Central (see Fig. 1.1). More recently, mine production at SOB Shaft has been largely concentrated in the Synclinorium. The Synclinorium Shaft will allow exploitation of the orebody below 3360ft Level and cover the area down to 3960ft Level.

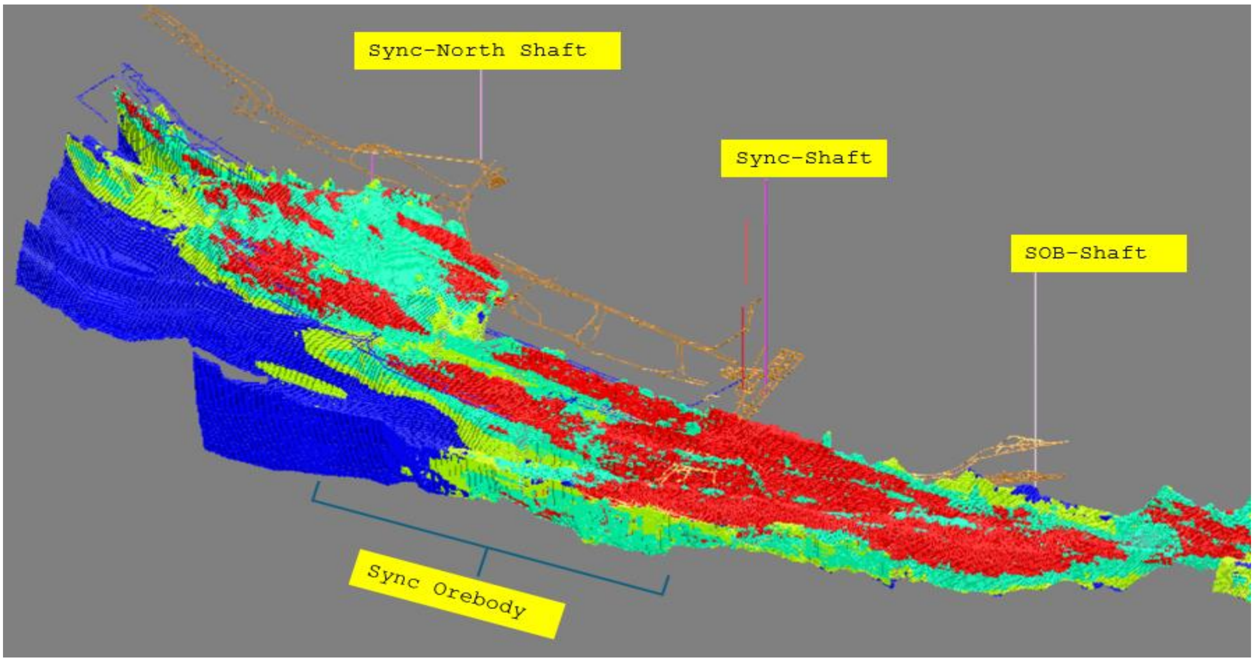



**Figure 1.1: Orebody model and Infrastructure for SOB, Synclinorium and Central**
(source-picture Mopani Geology manual – Nkana Synclinorium mine).

Due to the complex folded nature of the orebody at the Synclinorium mine different mining methods have been adopted to suit the orebody structure. Considering the complex nature of the orebody, ground control and stability has posed a serious geotechnical challenge at the mine. The potentially hazardous nature of underground mining requires the application of sound geotechnical engineering practices to determine the ground conditions, the ground support and reinforcement requirements, as well as the size, shape, and orientation of all the openings that can be safely and economically excavated in a particular rock mass.

At deeper levels of the current ore production (3960ft), mine development at Synclinorium mine has been facing geotechnical challenges involving ground control and ground stability. This has been exacerbated by the complexly folded, jointed, and sheared nature of the geological structure where the orebody is hosted. The importance of geological structure and its potential for adverse influence on rock stability cannot be over-emphasized.

Figure 1.2 shows an excavation driven in high stressed ground.

**Figure 1.2: Highly stressed and sheared ground on 3860 Level**

Rock falls have been the largest single cause of fatal accidents in the last nine years of operation. A total of 32 of fatal accidents were caused by rock falls at SOB/Synclinorium alone. This represents approximately 41% of the total (78) number of accidents. Therefore, a thorough understanding and more diligent application of geotechnical engineering practice in response to the challenges posed by the deeper underground environment of the Synclinorium mine is needed.

In addition, successful management of ground control at the deeper development levels of Synclinorium mine faces the following the geotechnical challenges:

(a) More water is encountered in the Footwall aquifers (other aquifers being Far Water, and Near Water aquifers) as new areas are opened up through mine-development.

(b) Deep level rock mass is characterized by high in-situ stress, high temperature, and high-water pressure. More discontinuities like Joints, Faults, Shears, bedding planes, Foliation and Schistosity influence rock stability underground; these are, in general, separations in a rock mass having zero or low tensile strength.

Other challenges faced at the mine include:

(a) Mine-induced Seismicity

This is primarily caused by the progressive build-up of high stress levels in the rock mass remaining around an excavation as it is enlarged by mining. With mine development approaching deeper levels (3960ft) the challenge associated with seismicity increases.

(b) High temperatures

The environment becomes hotter with higher temperatures ranging from 40 to 50$^{\circ}$ also influencing rock stability such that even geological and weathering processes impact more on the strength of naturally occurring geological materials at greater depth.

The prevailing situation of higher temperatures and highly stressed and sheared rock mass deeper underground calls for more attention and diligence in the application of all aspects of geotechnical engineering. This includes engineering geology, hydrogeology, and rock mechanics. Frequent or sustained assessment and inspection of all geotechnical work carried out underground needs to be done. Management at each underground mining operation should recognize, identify, and address the geotechnical issues that are unique to a particular mine, in an appropriate and professional manner, using current geotechnical knowledge, methodology, software and hardware. This requires that the techniques appropriate to a given set of conditions should be selected and applied.

### 1.2 Area of the study

The Nkana Mining Area (7625-HQ-LML) is 10,080 hectares in extent and lies immediately to the west of the city of Kitwe, which is situated approximately to the geographical center of the Zambian Copper belt and some 285 kilometers almost due north of the capital of Zambia, Lusaka.

Figure 1.3 shows Copperbelt towns and the location of study area in Kitwe

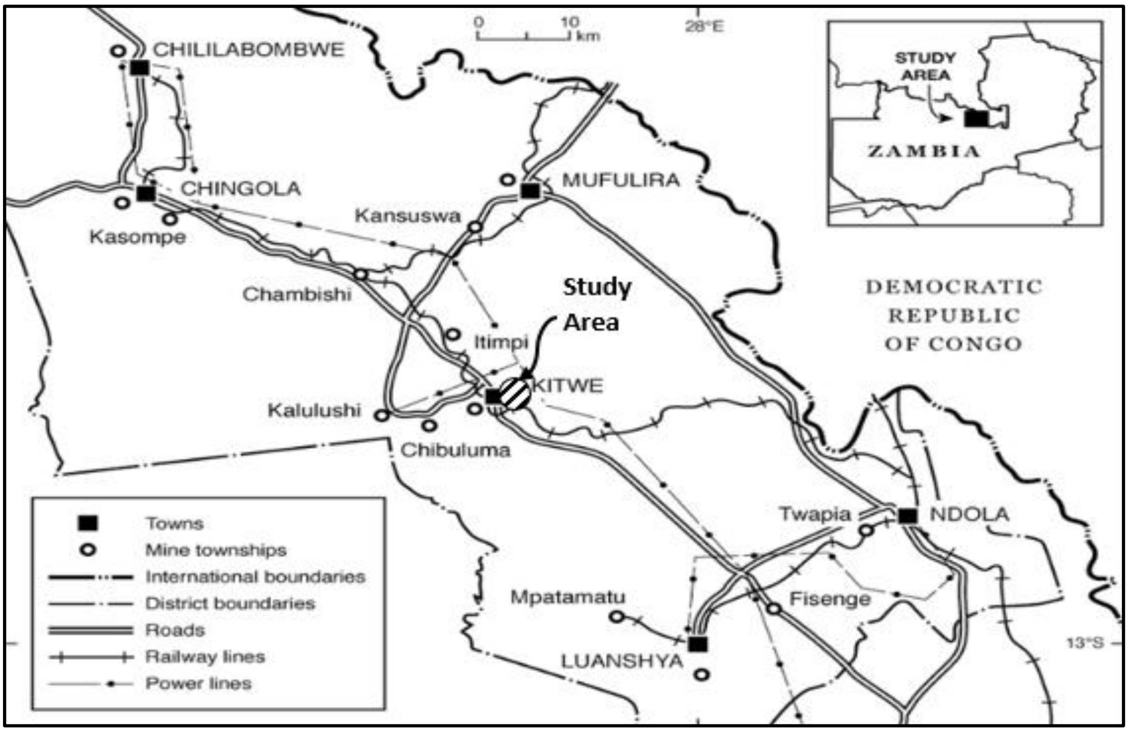



**Figure 1.3: Location Map of Study Area on the Copperbelt**

### 1.3 Geology of the Study Area

The deformation of the rocks of the Nkana Mining Area is related almost completely to the Lufilian Orogeny, which affected both the pre-Katanga and Katanga rocks. Within the axial zone of the Nkana syncline, shear folding is characteristic of the deformation style of the Lufilian Orogeny, with transposed bedding attenuated to sheared-out folds limbs, axial swelling and chevron structures being developed. The structures indicate thrusting directed towards the northeast, with decollement present at the Basement/Lower Roan and Footwall Formation/Ore formation boundaries.

The regional tectonic setting of Zambia and neigbouring countries is shown in Figure 1.4

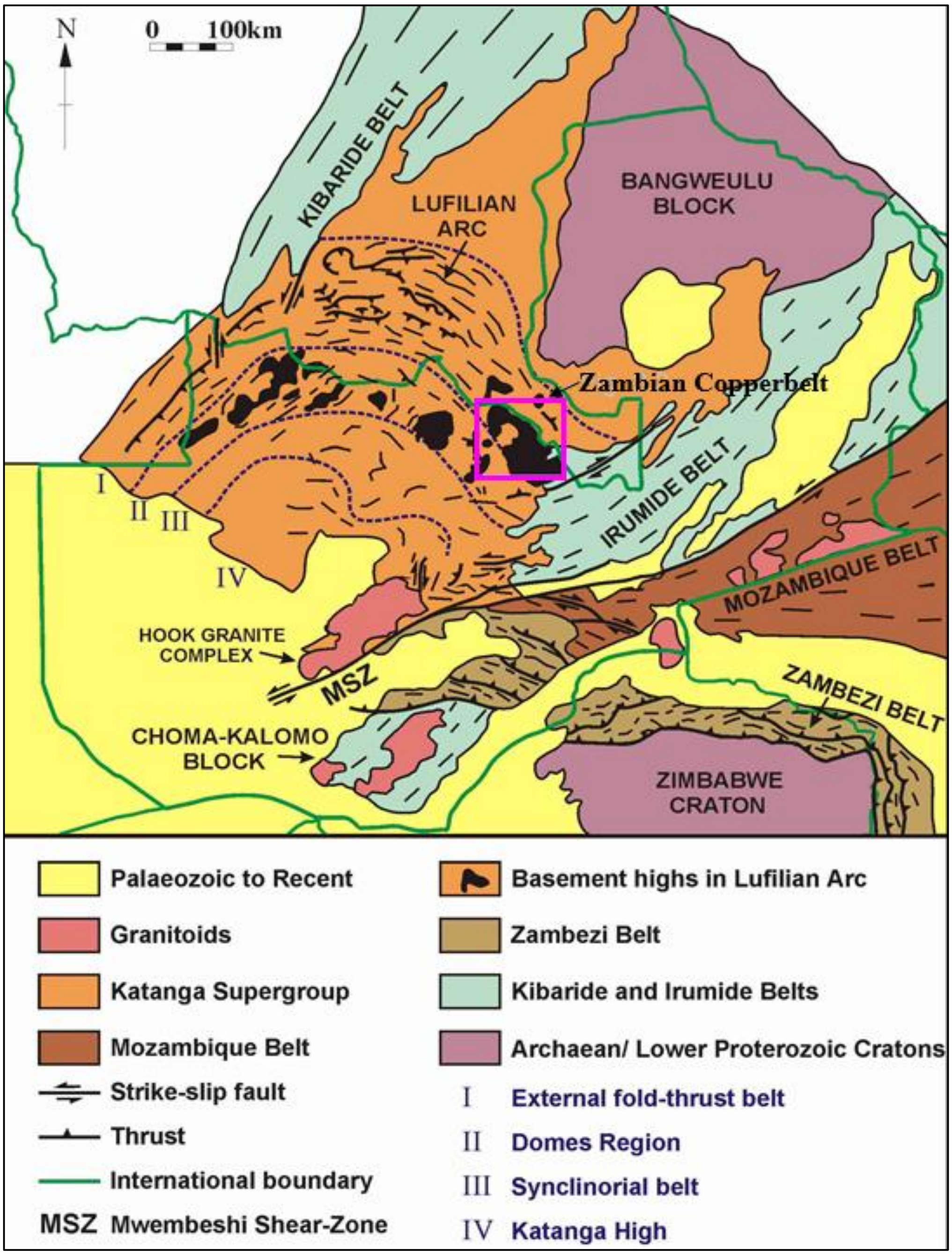



**Figure 1.4: Tectonic setting of Zambia and the Lufilian Arc**

(Porada, 1989; Binda & Porada, 1995).

Figure 1.5 shows the regional geology of the Zambian Copperbelt

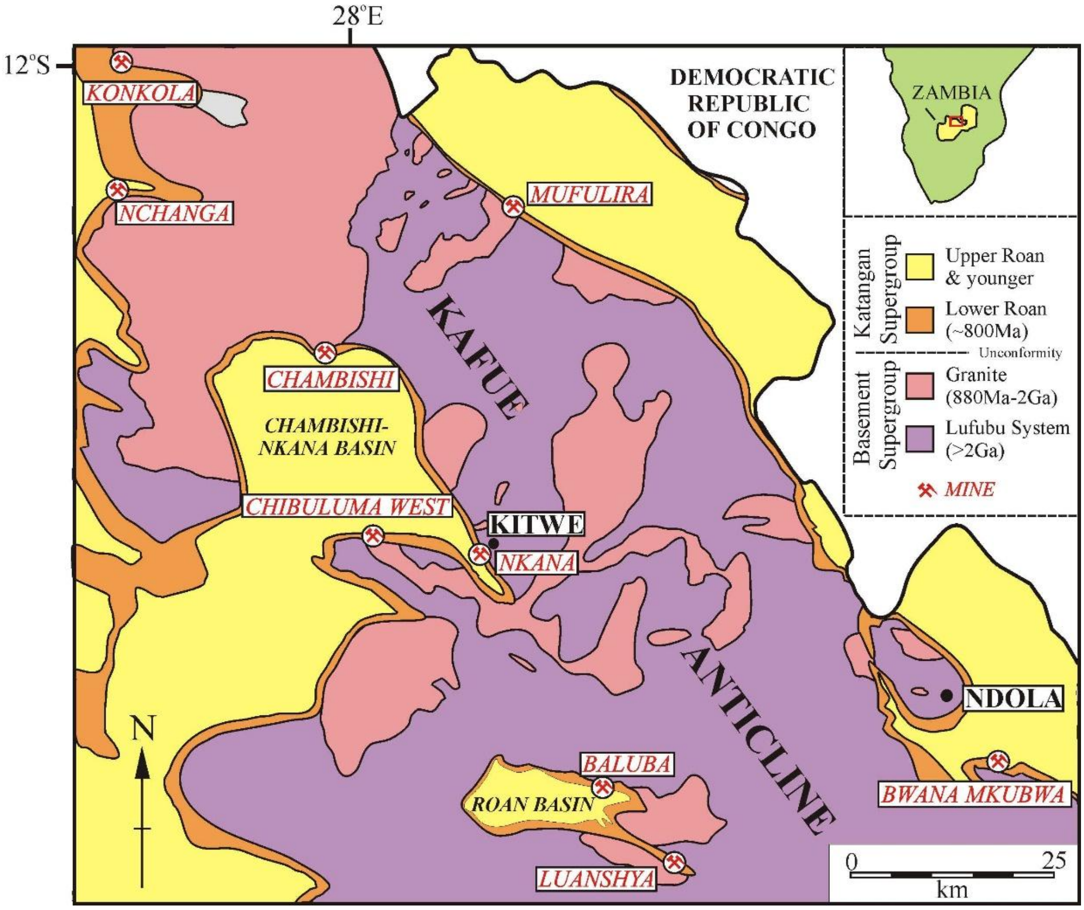



**Figure 1.5: Geological map of the Zambian Copperbelt showing the Katangan sedimentary rocks exposed on the flank of the Kafue Anticline (modified from Fleischer et al., 1976).**

### 1.3.1 Geological Description

The complex folded axial zone of the Nkana Syncline is best known at South Orebody Shaft. There are at least six major complex folds labelled A-F from east to west. The folds are arranged echelon and plunges undulate erratically. There is a marked flattening of plunge in barren gap areas, which are associated with basement 'highs', with a corresponding steepening of plunge off the flanks of the barren gaps. The folds are tightly compressed, with attenuation of limbs, bedding plane crumpling, axial plane cleavage and axial plane shearing well developed.

### 1.3.2 Deposit Type

The ore deposit type of Nkana south is part of the sediment hoisted nature of the Copperbelt orebodies. The typical South Orebody Ore Formation has at its base the Contact Shale Member which consists of contented laminated dolomite and argillite or massive dolomitic locally rich in tremolite. The member is up to four meters thick but may be absent. Overlying the Contact shale is the South Orebody Member, which is between 5 and 20 meters thick. A two-fold division is generally recognized. The lower beds consist of black carbonaceous shale with sulphide rick bands, laminae and veinlets of white dolomite.

### 1.3.3 Mineralization

In the underground workings, the principal copper ore minerals are chalcopyrite and bornite with subordinate chalcocite. There is a zoning in the geographical distribution of these minerals. Cobalt occurs as carrollite and cobaltiferious pyrite. "Oxide" minerals are the principal ore minerals malachite, pseudomalachite, chrysocolla, native cooper, cuprite and libethenite. In the open pit, malachite and chrysocolla are the principal ore minerals in the zone of oxidation closer to the surface. In some places however, vermiculite, malachite pseudomalachite and accessory wad are more important. At deeper levels chalcopyrite, bornite and chalcocite predominate.

## 1.4 Problem statement

In the past 9 years, Mopani Synclinorium Mine has been faced with geotechnical problems such as intense folding, jointing and stresses in its ground control and stability in the rock-mechanics section. The application of sound geotechnical engineering practice in the pursuit of safe, practical, cost-effective solutions to rock instability issues has been lacking, and this has seen a drop in the ground control standards. The following have been the consequences:

1. Rock falls resulting in fatalities (32 cases at synclinorium alone in the past 9 years)
2. Lack of effective ground control plan
3. Inadequate geotechnical data capture
4. Inconsistency in development and dewatering
5. Low standards in design, installation, quality control of rock support and reinforcement
6. Lack of adherence to design of underground excavations.

## 1.5 Main Objective

The general objective of the research study is to assess the related geotechnical considerations for ground control and stability at Nkana Synclinorium copper mine and make recommendations for improvement.

### 1.5.1 Sub-objectives

Specifically, the following objectives are outlined:

a) Assess ground control mechanisms and support standards

b) Evaluate the design and quality control of rock support and reinforcement.

c) Assess adherence of underground developments or excavations to design (size and geometry); recommend how to improve mining of development openings, stopes, and pillars according to design.

d) Recommend ways to improve ground control and stability at synclinorium mine.

## 1.6 Research Questions

The following questions must be answered to achieve the specified study objectives:

1. What geotechnical considerations should be examined in detail to improve ground control and stability at Synclinorium mine?

**2.** How could the current design, installation, and quality control of rock support and reinforcement be improved as well as made more cost-effective?

**3.** What can be done to sustain mining of underground developments or excavations to design (size and geometry) and avoid or minimize over-breaks/under-breaks. This includes development openings, stopes, and pillars.

**4.** What modern geotechnical software and seismic technology are lacking in addition to what is currently being used at the mined?

### 1.6.1 Significance of the study

Expected benefits of the study being carried out will be as follows:

1. Synclinorium mine would avoid violation of mine safety rules and regulations as well as alleviate the occurrence of unnecessary mine accidents from ground failure.
2. The cost of ore mining would be better managed as damage to equipment and infrastructure is minimized.
3. Synclinorium mine would improve the chances of passing an international audit if they achieved world class geotechnical standards.
4. With improve safety standards, copper ore production would increase.

# CHAPTER 2 – Literature Review

## 2.1 Introduction

This chapter presents an overview of the all previously published works related to the general objective of this study, which is to assess the related geotechnical considerations for ground control and stability at Nkana Synclinorium copper mine. The literature review includes full scholarly papers or sections of scholarly work such as books, or articles which are primary sources by the original researchers of a study and secondary sources by somebody other than the original researcher, e.g. a review article. The literature review also includes a critical evaluation of the study material.

There have been several other significant advances and prior studies in geotechnical engineering during the past that are of direct relevance to underground mining. However, there is no best approach or method arrived at because of the wide variety of and variability in the ground conditions and the mining methods in use.

## 2.1 Discussion of Findings

**Moshab, (1997) and Moshab, (2016)** have highlighted appropriate geotechnical considerations for ground control and stability listed below, though this has not been detailed enough:

1. Geological structure
2. Rock support and reinforcement
3. Excavation opening size and geometry
4. Timing of support

Kaiser et al., 1996 explains that mechanics of rock support is complex, and no models exist that can fully explain the interaction of various support components in a rock support system. Three key support functions summarized as:

(1) Reinforce the rock mass to strengthen it and to control bulking

(2) Retain broken rock to prevent fractured block failure and unraveling, and

(3) Hold fractured blocks and securely tie back the retaining element(s) to stable ground.

However, numerical modelling has been used by **Jessu and Spearing, 2019** to successfully estimate failure zones (failure depths) and also demonstrate stability of rock masses surrounding excavations after installing support systems.

**Selvasekaran et al.,** studied the effect of Joint Orientation in Tunneling. It is expected that the stability of surrounding rock is affected by the strike and dip of the joints and the direction of the tunnel axis, whether it is with the dip or against dip etc. Similarly, the spacing of joints will also affect the stability. The orientation of joints in different directions can form blocks liable to fall. The objective of this research project is to determine the degree of influence of joints' strike and dip orientation in tunneling. Field works related to this project was carried out at the Bogala Graphite Lanka Ltd. Tunnel mapping and other observations related to the project were made at 489.6m level in Bogala mine. Models shown in Figures 2.1 to 2.7 were made with joint spacing of 15mm with two joint sets (joint sets parallel to tunnel axis and joint sets perpendicular to tunnel axis). Tunnels were created with 90mm diameter with dip angles of joints are 0˚, 30˚, 60˚, and 90˚. The tunnel models are loaded using uniaxial compressive strength (UCS) machine and observed the behavior of rock mass around the tunnels during loading. From the results the most preferable dip angle for the joint strike perpendicular to the tunnel axis would be the 90˚ and for the joint strike parallel to the tunnel axis would be 0˚. However, **Selvasekaran et al.**, did not relate rock type and rock strength in his study of the effect of joint orientation to tunneling.

Figures 2.1 to 2.7 show conceptual models of 90mm diameter tunnels involving idealized underground tunnels with joint set planes truncating the tunnels at different dips from 0˚to 90˚, either parallel or perpendicular to the tunnel. Joint dips affect the underground tunnels differently depending on the joint inclination.

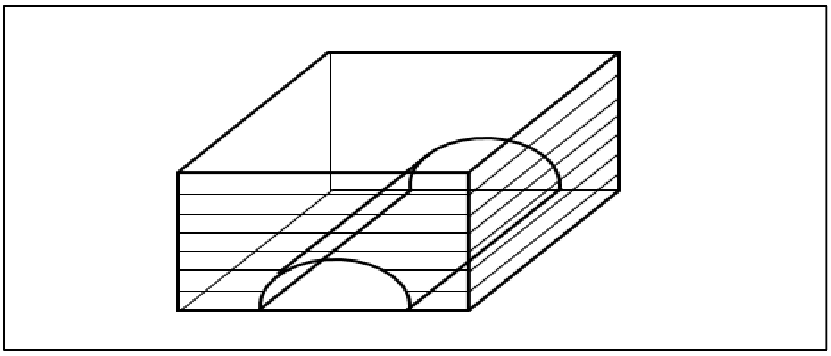


**Figure 2.1: Joint strike parallel to tunnel. Dip 0˚; Stress failure at 4.40 x $10^6$ N/m$^2$**

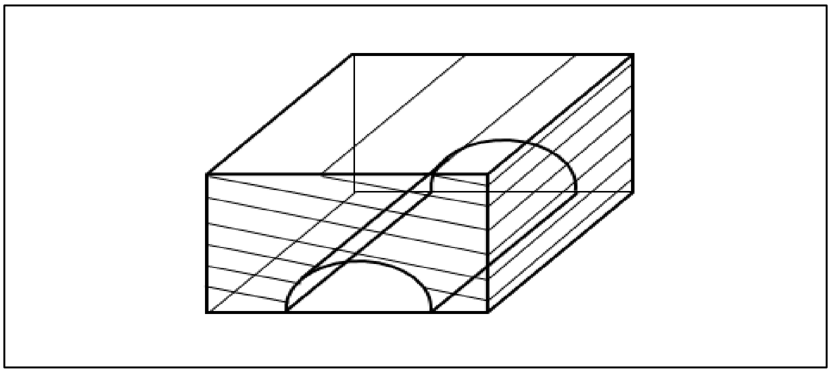


**Figure 2.2: Joint strike parallel to tunnel. Dip 30˚; Stress failure at 3.66 x $10^6$ N/m$^2$**

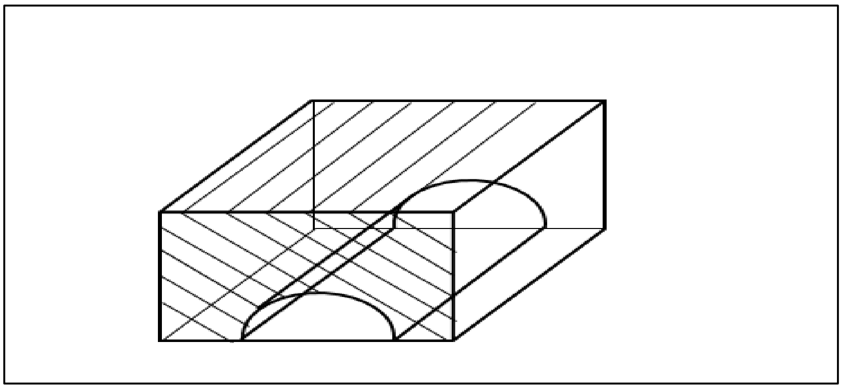


**Figure 2.3: Joint strike parallel to tunnel. Dip 60˚; Stress failure at 3.10 x $10^6$ N/m$^2$**

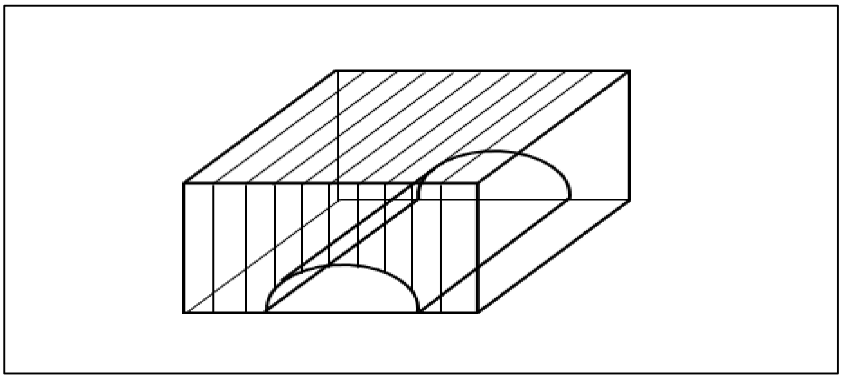


**Figure 2.4: Joint strike parallel to tunnel. Dip 90°; Stress failure at 2.68 x $10^6$ N/m$^2$**

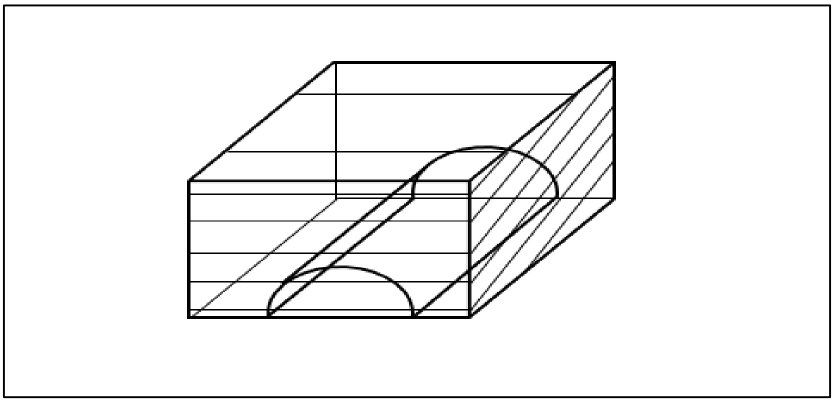


**Figure 2.5: Joint strike parallel to tunnel. Dip 30°; Stress failure at 3.75 x $10^6$ N/m$^2$**

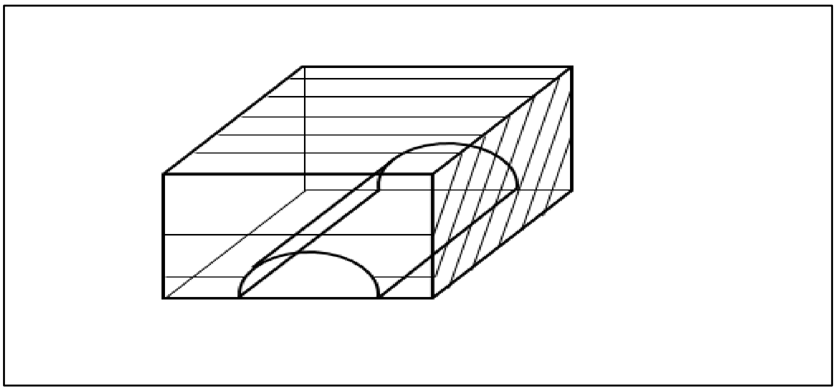


**Figure 2.6: Joint strike parallel to tunnel. Dip 60°; Stress failure at 4.80 x $10^6$ N/m$^2$**

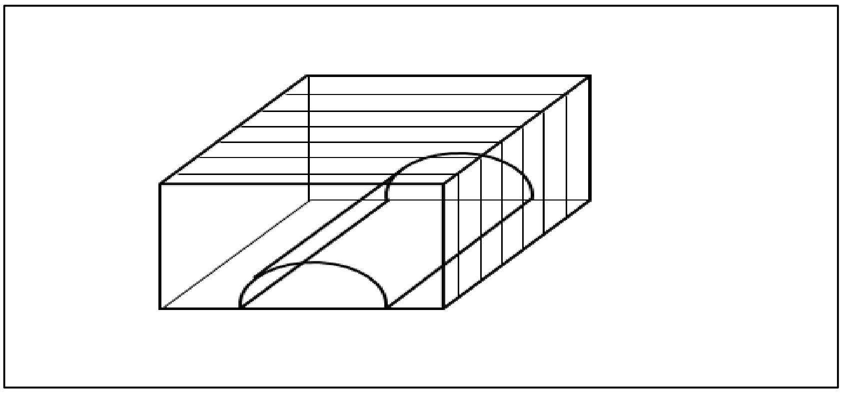


**Figure 2.7: Joint strike parallel to tunnel. Dip 90˚; Stress failure at 5.95 x $10^6$ N/m$^2$**

It was noted that joints with strikes perpendicular to the tunnel axis did not affect the ground conditions adversely compared to of similar dip, but parallel to the tunnel axis. The most preferable dip angle for the joint strike perpendicular to the tunnel axis would be the 90˚ and for the joints trike parallel to the tunnel axis would be 0˚.

**Stefan et al.,** carried out a study on factors influencing the Load-Bearing Capacity of Rock as Base Material for Post-installed anchors. In the case of fastenings on rock, because of the variability, it is quite difficult to make a preliminary assessment of the load-bearing capacity of rock as a base material.

However, this research has used Pull-tests conducted on post-installed anchors like split sets and cable bolts to assess the load-bearing of the rock mass surrounding an excavation.

According to Stefan, Zeman, and Klaus (2021) assessment of load-bearing capacity as it not only covers rock properties in different lithologies, but also associates weathering and quality of the rock mass. Hence the current research will draw some positive contributions in geotechnical assessment of synclinorium underground mine where rock support is concerned.

**Read et al., 1997**; **Volokhov**, **2003** wrote a paper on insitu studies and analysis of numerical modelling results which showed that the following groups of factors have the highest effect on the stability of a rock:

• the initial stress-strain state field and its change patterns;

• mechanical behaviour of the rock mass responding to stress fields changing;

• shape and size of excavations;

• type and characteristics of the support

Despite the studies and analyzes of Read et al., 1997 and Volokhov, 2003 being extensive, they focused more rock mechanical properties than the rock formation types and physical properties of rocks. The current research by this author needed balanced literature involving both mechanical and physical rock properties.

**Abdellah et. al, 2016** gives an insight on how the quality of rock mass significantly impacts on the stability of underground excavations. It also explores the influence of orientations and dips on the axis of excavations. The more data you have about joints from underground, the more confidence you build on the general understanding of the rock mass characteristics. However, consideration of the influence of rock formations affected by the joint sets is lacking.

**Protosenya & Vilner, 2021** assessed excavation intersections' stability in joint rock masses. In terms of ground control and stability, the research gives an insight of the rock mass characteristics and behaviour but lack sufficient geological structure and physical rock type property assessment. In addition, the research devoted only to assessing the stability and determining the size of the failure zone in excavations in the absence of support. However, introduction of support such as roof bolting, would affect the parameters of a marginal rock mass and change the patterns of deformation, especially in overstressed rock masses.

**Renjith & Venkat, 2015** carried out a Geological and Geotechnical Investigation of Some Rocks in Trivandrum Area, Kerala, India while **Arsyad et al., 2020** carried out an analysis of physical properties and mechanics of rocks in the karst region of Pangkep Regency, Indonesia.

Both studies above have not only shed some light on what needs to be done similarly for the rocks at Nkana Synclinorium Mine in Zambia to improve the database, but also revealed a gap in physical rock property tests like rock porosity. Porosity tests have never been done at Nkana Synclinorium and this represents a major gap in the rock mechanics database.

**Palmstrom & Stille 2015** established that Engineering geological mapping provides rock strength, location of weakness zones and discontinuity properties. Data collection techniques with geological and geotechnical mapping and core logging methods are applied to link information to describe the rock mass composition. Presence of shear zones and faults have potentially detrimental effects in mining excavations. Since discontinuities properties affect ground behaviour surrounding opening, mechanical joint properties such as spacing, orientation and joint wall conditions should be considered for this research.

Figure 2.8 shows the general description of the rock mass and related structures such as joints, faults, texture and weakness zones.

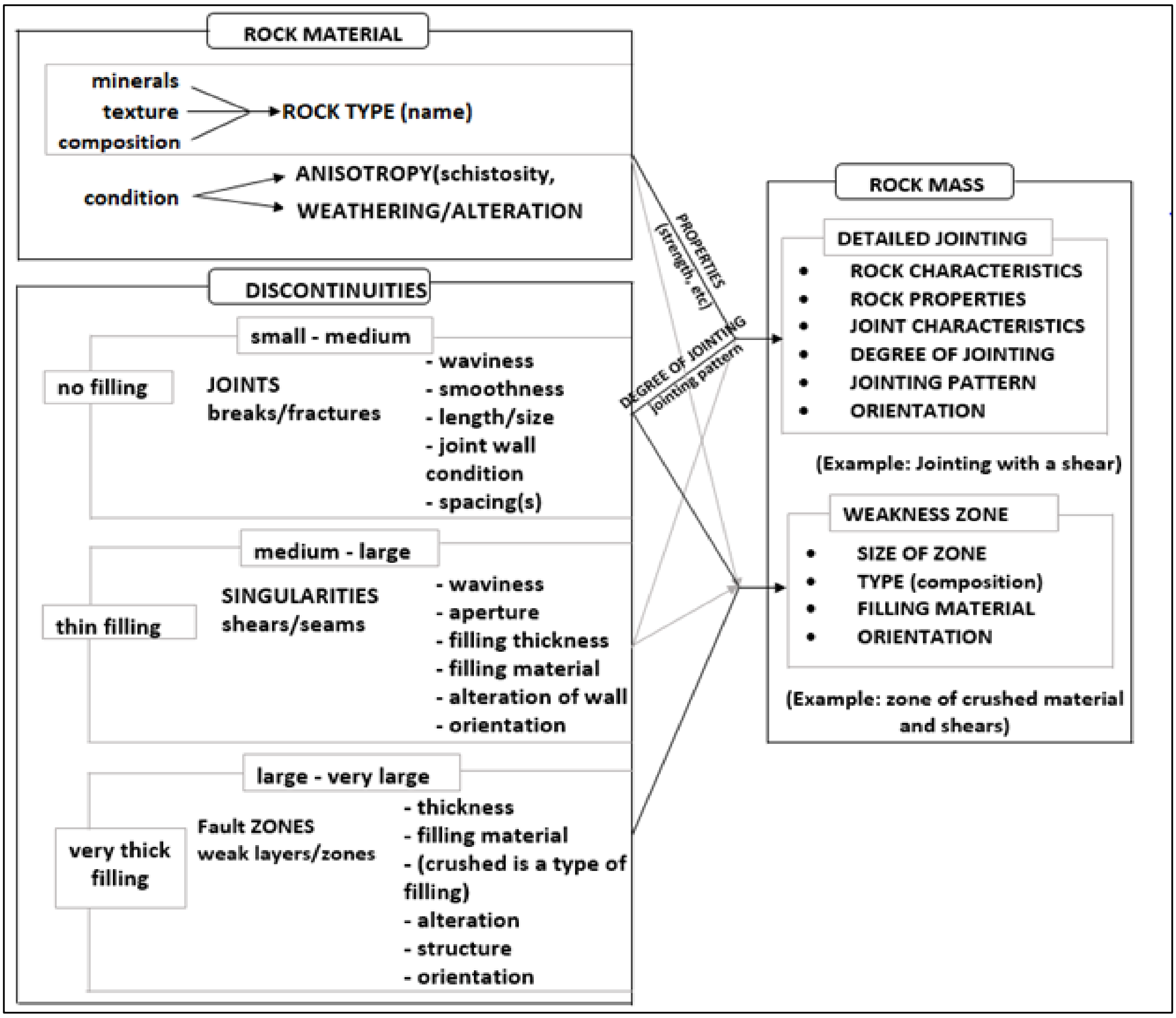



**Figure 2.8: The main features related to rock mass structure (Palmstrom 2015)**

Rock mass characterization is based on quantitative and qualitative site investigation and laboratory tests (**Potvin et al. 2015**). The type of project, topographical and geological condition, in situ stress and groundwater condition are some of the relevant parameters for selecting investigation principles. Geophysical techniques are used to estimate rock mass properties and geometry of orebodies.

An evaluation of Ground Management in Underground Excavation Design" by **Curtinet al., 2016** considers a ground management procedure in underground excavation design. In this paper rock mass structure is used to estimate the inherent properties of rock mass needed for predicting ground behaviour. Understanding the ground conditions makes it possible to assess instability conditions and failure mechanisms in the rock mass surrounding underground excavations. Based on the diagnosis of ground behaviour and failure mechanism, it is possible to select the appropriate tool(s) for design analysis. The results of design analysis determine the required ground support system, extraction rate and excavation sequence in underground practices.

**Ibarra et al.,1996** offers a review of factors influencing over-break and under-break. The newly developed method of measuring over-break and under-break is also highlighted. This provides an opportunity to measure and study the causes as well as the consequences such as significant reduction of tunnel stability and increases in the total cost of tunneling and post-construction maintenance in mines like the Nkana Synclinorium. Numerous studies have investigated the factors that contribute to over-break in underground tunnels (Ibarra et al., 1996; Mandal and Singh, 2009; Mottahedi et al., 2018b). These factors can be roughly classified into two geological and blasting factors. Geological factors include the physical and mechanical properties of rock mass, joint characteristics, water conditions and in-situ stress field (Chakraborty et al., 1994; Hagan, 1983; Oggeri and Ova, 2004), while blasting factors mainly consist of explosive types, powder factor, burden, spacing, delay time, charge structure, advance length and excavation area (Fodera et al., 2020; Jang and Topal, 2013; Singh and Xavier, 2005).

Many research projects have been conducted to understand the relationship between geological factors and over-break. For instance, Hagan (1992) emphasized the importance of pre-existing joints and beddings on in situ rock. According to his paper, fractures in the rock tend to dominate the nature of the blast-induced fracture pattern and it usually influences over-break more than the mechanical and physical properties of the rock. Among the geological factors, the orientation of discontinuity is one of the major factors influencing over-break phenomenon. Hoek and Brown (1980) reported that a discontinuity plane having strike parallel to the tunnel axis is considered to have an unfavorable effect on over-break. Generally, less over-break and under-breaks are observed where the strike of the discontinuity is nearly perpendicular to the tunnel axis and contrastively greater when they are nearly parallel. In view of other orientations, drives with dip are more favorable than drives against dip where the strike of discontinuity is perpendicular to tunnel axis and fair and very unfavorable for dip has angle of 20°- 45° and 45°- 90°, respectively, when the strike is parallel to tunnel axis (Bieniawski,1973).

The importance of both geological structures and mining induced stress fractures on the hanging wall stability in a deep level gold mine was studied by Quaye and Guler, 1998.

Among other things, this paper highlights the importance of characterization of geological conditions and the effects of orientation of major structures with underground openings. This borders on collection of sufficient geological and geotechnical data to optimize the database. As a result, confidence in the determination of rock mass characteristics is enhanced as we assign values to a set of rock parameters. An understanding of these processes can lead to the successful prediction of rock mass behaviour for different conditions. The evaluation of the Effect of Rock Joints on the Stability of Underground Tunnels by **Razougui et al** (**2016**) gives an insight on how the quality of rock mass significantly impacts on the stability of underground excavations. It also explores the influence of orientations and dips on the axis of excavations. The more data you have about joints from underground, the more confidence you build on the general understanding of the rock mass characteristics.

### 2.2 Evaluation of literature and how it applies to the topic.

All the literature reviewed for the current research is in line with geotechnical considerations for ground control and stability. Management at each underground mining operation should recognize, identify and address the geotechnical issues that are unique to a particular mine in an appropriate manner using current geotechnical knowledge, methodology, software and hardware.

This study and related research have unveiled some major issues related to ground control and stability at Synclinorium mine which must be well understood in underground operations; the most compelling ones relate to the complexity of rock support and lack of models that can fully explain the interaction of various support components in a rock support system (Kaiser et al., 1996). A brief comparison of Kaiser et al., 1996 and Lamplmair, Zeman, and Voit, 2021 offers a useful framework for assessing the design, installation, as well as quality control of rock support and reinforcement at Nkana Synclinorium Mine. Although enough work or research has been conducted concerning geological structure, not much has been done with regard to physical rock type property assessment and the associated rock formations. In light of this, there is need to strengthen geotechnical database at Nkana Synclinorium mine. Understanding of rock mass structure is key in estimation of inherent properties of rock mass to aid prediction of ground behaviour. Understanding the ground conditions makes it possible to assess instability conditions and failure mechanisms in the rock mass surrounding underground excavations. In line with the literature reviewed for this research, it must be ensured at every underground mine that the following things are done in relation to workplaces, travel ways and installations underground in the mine:

(a) Due consideration is given to local geological structure and its influence on rock stability;

(b) Rock damage at the excavation perimeter due to blasting is minimized by careful drilling and charging;

(c) Due consideration is given to the size and geometry of openings;

(d) Appropriate measures are taken to ensure the proper design, installation and quality control of rock support and reinforcement; and

(e) The installation of ground support is timed to take into account rock conditions. In relation to all development openings and stoping systems underground in the mine. -

(f) Geotechnical data (including monitoring of openings when appropriate) is systematically collected, analyzed and interpreted;

(g) Appropriate stope and pillar dimensions are determined;

(h) There is adequate design, control and monitoring of production blasts; and

(i) Rock support and reinforcement are adequately designed and installed.

In general, the literature reviewed seems to highlight most geotechnical aspects that are important for this study.

## 2.3 Significance of the study

This study on assessment of geotechnical considerations for ground control and stability at Nkana Synclinorium Mine in Zambia is an important addition to literature that may act as a guideline in the application of current geotechnical knowledge, methodology and hardware underground. It is emphasized that geotechnical information in this research is not totally inclusive of all factors concerning the application of geotechnical engineering in an underground metalliferous mine. It may not be totally suited to the specific requirements of every mine.

Table 2.1 shows literature review inconsistencies and Gaps

**Table 2.1: Highlights from the Literature Review and established gaps**

| Author | Study | Inconsistences or Gaps to Literature |
|---|---|---|
| Moshab (Mines Occupational Safety and Health Advisory Board) - Department of Industry and Resources, Australia, 1997. | Geotechnical Considerations in Underground Mines. | Though this Department of Industry and Resources seeks to encourage the application of current geotechnical knowledge, methodology, instrumentation and rock support and reinforcement hardware to the practical solution of geotechnical engineering issues in underground mining, it lacks guidance on standard ground control and pillar design procedures. |
| J.A. Ibarrh, N.H. Maerz, and J.A. Franklin, 1996. | Mining of underground openings (or excavations) according to mine planning design. | The paper offers a review of factors influencing over-break and under-break. The paper also offers an explanation and understanding of how over-break and under-break, including the related costs. However, the analysis of over-break and under-break in relation to geological formations in which tunnels or excavations are mined is lacking. |
| Renjith S Anand & D Venkat<br>M Arsyad, V A Tiwow, Sulistiawaty & I A Sahdian, 2020. | Geological and Geotechnical Investigation of Some Rocks in Trivandrum Area, Kerala, India; analysis of physical properties and mechanics of rocks in the karst region of Pangkep Regency, Indonesia. | Both studies above have not only shed some light on what needs to be done similarly for the rocks at Nkana Synclinorium Mine in Zambia, but also revealed some gaps in some physical rock property tests like rock porosity. Porosity tests have never been done at Nkana Synclinorium and this represents a major gap in the rock mechanics database. |

## 2.4 Literature Review Summary

Analysis of the material from the literature review, leads the author of this thesis to conclude that that most findings from the literature review describe the knowledge that is relevant for this research. This includes the following:

1. A review of factors influencing over-break and under-break as well as the costs associated with it by J.A. Ibarrh, N.H. Maerz, and J.A. Franklin, 1996.

2. Analysis of physical properties and mechanics of rocks in the karst region of Pangkep Regency, Indonesia by Renjith S Anand & D Venkat M Arsyad, V A Tiwow, Sulistiawaty & I A Sahdian, 2020.

3. Application of current geotechnical knowledge, methodology, instrumentation and rock support and reinforcement in underground mines by Moshab (Mines Occupational Safety and Health Advisory Board) - Department of Industry and Resources, Australia, 1997.

4. The information by Palmstrom & Stille 2015 about Engineering geological mapping providing rock strength, location of weakness zones and discontinuity properties gives knowledge on data collection techniques with geological and geotechnical mapping and core logging methods. This literature is relevant in terms of improving the geotechnical database at Nkana Synclinorium.

However, there are some gaps to note in some of the literature. Geotechnical data in terms of physical rock properties such as rock porosity and rock density have not been considered. Geological formations in the analysis of over-break and under-break by Renjith S Anand & D Venkat M Arsyad, V A Tiwow, Sulistiawaty & I A Sahdian, 2020. Abdellah et. al, 2016 gives information the quality of rock mass impacts on the stability of underground excavations. It involves the influence of orientations and dips on the axis of excavations, but does not consider the types of rock formations.

# CHAPTER 3 – Methodology

## 3.1 Introduction

This chapter describes methods which were used to address sub-objectives of study related geotechnical considerations for ground control and stability at Nkana Synclinorium copper mine. The mine software and equipment used are all reflected in this section.

## 3.2 Research design

This research will generally have a quantitative approach. Most of the data collected was numerical, through substantial use of measurements and quantitative analysis techniques.

The research mainly focused on geotechnical and geological data collection and analysis to gain an insight into the accuracy of parameters used in design of underground excavations as well as rock mass characterization and classification to explain possible ground behaviours that impact the design, planning and ground stability.

**Material and Methods**

This study used:

**(i) Empirical methods for rock mass classification**

The rock mass classification was done to assess the estimates of underground tunnel support Barton's Q system.

Barton et tal, 1974 rock mass classification was used because it is a more comprehensive classification system that accounts for the physical properties of discontinuities. This Q-system takes into account the Rock Quality Designation (RQD) developed by Deere in 1964 for quantifying the rock quality, the Jn (joint sets), Jr (joint roughness), Ja (joint alteration), Jw (water pressure) and SRF (stress reduction factor), (Kaya et al 2011). The relationship between the valuables are shown in equation 1.

$$Q = \frac{\mathrm{RQD}}{\mathrm{J_n}} \times \frac{\mathrm{J_r}}{\mathrm{J_a}} \times \frac{\mathrm{J_w}}{\mathrm{SRF}} \qquad (1)$$

Where; RQD is the Rock Quality Designation and is determined from equation 2.

$$\mathrm{RQD} = \frac{\sum OF\ CORE\ PICES \geq 100mm}{DRILL\ CORE\ RUN(TOTAL\ CORE\ LENGTH} \times 100\% \qquad (2)$$

$J_n$ is the joint number , $J_a$ is the joint alteration number, $J_r$ joint roughness number

$J_w$ joint water number, SRF is the Stress Reduction Factor

*Bieniawski (1973) classification*

Bieniawaski's classification gives a Rock Mass Rating (RMR) indicative of the quality of the rock mass. The RMR combines geologic characteristics of influence into one overall comprehensive index of rock mass quality. In this study, Bieniawski, 1989 Rock Mass Rating (RMR) values were obtained from equation 3.

$$RMR = 9 \cdot \ln(Q) + 44 \qquad (3)$$

*Geological Strength Index (GSI)*

This index was derived from evaluating the state of discontinuity surfaces, structure and the lithology of the rock mass using the Hoek-Brown failure criterion (Hoek, 1994), Hoek, Kaiser and Bawden (1995). It was estimated through visual inspection of mine excavation faces, as well as borehole cores.

Geological strength index is estimated as:

$$GSI \approx RMR76 \approx RMR89 - 5 \text{ for GSI} \geq 18 \text{ or RMR} \geq 23 \qquad (4)$$

$$\approx 9\text{In}Q' + 44 \text{ for GSI} < 18 \qquad (5)$$

$RMR_{76}$ represents the RMR value applied by Bieniawski in 1976 with maximum ground water rating of 10. Similarly, $RMR_{89}$ represents the RMR values applied by Bieniawski in 1989.

Where $Q'$ = modified rock mass quality

### (ii) Mapping of geological structures

Scanline mapping of geological structures was done to identify joints, bedding planes, schistosity and folds. The mapping was carried out along the crosscuts or tunnel walls at a 1.50m grade line elevation with geological features being picked along a stretched 100m tape. Known centre line survey pegs were used references. Brunton compass, and Crinol ruler were used to measure the strike and dip of the geological contacts and structures.

### Analysis of mapped joints

(a.) For the analysis of the mapped joints, Dips software was used. This is a program designed for the interactive analysis of orientation-based geological data with plotting features. Dips allows the user to analyze and visualize structural data following the same techniques used in manual stereonets.

To start with, the joints data collected from underground was entered in an excel spreadsheet file compatible with the Dips software. The input data in the Dips spreadsheet, also known as the Grid data view, is arranged such that it contains a maximum of 40 rows and the following columns:

- Two Orientation Columns
- A Quantity Column
- A Traverse Column
- A Distance Column
- Three X (Easting), Y (Northing), and Z (Elevation) Columns
- Three Extra Columns

Clicking on a Stereonet Plot View, the control panel at the side of the screen allows you to fully customize the data and display options for the Stereonet.

To display the planes (great circles) for all planar data in the Dips file the Sidebar options were used to select Planes and Grid data planes, and this finally displays the great circles.

Since each great circle is represented by dip vectors on the Stereonet, the vectors in turn represent the maximum dip orientation of a plane and is orthogonal to the pole vector of a plane, this author was able to interpret the general direction in which the joint planes were dipping on the Stereonet.

Selecting the Contour plot option generated a contour plot. The contour plot showed joint data concentrations or data clusters on the Stereonet.

(b.) In permanent excavations like haulages or Refuge cubbies, any major joint planes affecting ground stability must be mapped and analyzed diligently to understand the planes of failure. A practical example of such a case is presented in Figures 4.94 which illustrate a rock mass in which several geological discontinuities are known to exist along a haulage at Synclinorium mine. A cursory examination of Figure 4.94 shows that the two parallel major joints running diagonally to the left, towards the timber pack represent a threat to the stability of the system.

The forces acting on this potential failure plane are analyzed as follows:

**FW** – the total weight of the rock wedge

**FN** – the normal force acting across the plane

**FS** – the shear force acting along the plane

**FP** – the total force due to fluid [pressure (if present) within the rock mass. The condition for unstable equilibrium is

$$F_S = \mu(F_N - F_P)$$

where $\mu$ is the coefficient of friction which is effective along the potential failure plane. The requirement for stability of the slope can be expressed as follows:

$$\mu > \frac{Cos\theta}{Sin\theta - F_P / F_W}$$

From the limited amount of experimental data available, it is believed that the coefficient of friction, $\mu$ between two dry rock surfaces is in the region of 0.7; This value could be considered lower if the potential failure plane contains soft filling material or if the rock services had deteriorated.

In underground tunneling projects, the dip and dip orientation of joints greatly affect the total cost and cycle time of the tunnel. Therefore, it is very important to consider joints in designing tunnel supporting systems and tunnel direction through the rock

mass. The orientation of joints in excavation is concerned with the dip and dip direction of the rock formation and its relatively favourable or unfavourable effect on the rock mass. A commonsense check is to note the relative potential movement of the rock mass into or out of the excavation. These adjustments are important, since the way the rock mass is situated in relationship to the direction of the excavation will ultimately determine its stability.
The orientation of joints in different directions can result in a blocky rock mass thereby increasing the chances of ground failure in an excavation.

### (iii) Borehole logging

Geotechnical core logging involved detailed examination and description of rock samples obtained from boreholes onto log sheets. Borehole cores were examined for geological formations, colour, and grain size. Geological, structures like joints, bedding planes, folds and schistosity were also logged. Core angles and RQDs were also estimated using crinol ruler, measuring tape, and Brunton compass. Water and brush were used for washing the core.

**(iv) Pull Testing**

This was applied. Allowable Load Testing method was used. It involved fixing a hydraulic pull tester to an anchor and pulling it until it breaks. This test determines the stress levels required for the structure to collapse. Anchor Testing was repeated 5 to 8 times, with the average being the final result. The pull test was used to measure the working and ultimate capacities of a rock bolt anchor. To make sure the bolt response during the test was minimal and predictable, high strength, short-length (1.8 to 2.5 m) bolts were used. In this research the required anchorage ranged from 8 to 10 tons and the anchors used for pull testing were double-ring split sets.

**Criteria for site section of pull testing**

For underground pull tests, site selection can be the most critical aspect of the entire test program. Selection should be based on the following criteria:

1. The test program should be conducted in a rock mass which is representative of that in which the cable bolts are installed in normal operational practice. This may require tests at several different locations so as to access different rock types or rock masses of different quality. If the rock displays bedding or a foliation, test holes should be drilled at the same relative angle as operational cable bolt holes.
2. Only double-ring splits sets are subjected to pull testing. For every 100 split sets installed, 5% should be double-ring split sets, and every third split set installed should be a double ring split set.
3. The most preferred testing sites are those in active areas of operation, especially 10m away from a moving development end which is support compliant and also in permanent excavations like haulages which are frequently used by mobile equipment and mining personnel.

**Apparatus**

Loading System: The system for pulling the rock bolts consisted of a hollow-center hydraulic ram and a reaction frame. The hydraulic ram had a travel range of 50 mm. The loading system applied a force that deviates by no more than 5° from the long axis of the bolt during the test. Load Transducer: This was used to measure the load on the rock bolt. *Displacement Transducer*- A dial gauge was used to measure the displacement of the rock bolt head. It had an accuracy of 0.025 mm, a resolution of 0.013 mm, and a range of 50 mm.

**(v) Compressive Strength Testing of concrete cubes**

The concrete cube specimens were tested by compression testing machine after 7 days curing or 28 days curing. The load-bearing area of each specimen is calculated before testing. A compressive load was then applied gradually at the uniform rate of 13.7293 N/mm² (140 kg/cm² per minute) till the Specimens failed. Load at the failure divided by area of specimen gave the compressive strength of concrete.

**(vi) Evaluation of support design for underground excavations**

From geological data collection from underground mapping of joints, beddings and folds the rock mass was characterized and classified to assign rock mass properties for potential failure mode identification. Support design was assessed taking into account excavation sequences and availability of support materials.

Quality control of support installation regarding cable anchoring, tensioning and grouting was also assessed through pull tests and compressive strength testing of concrete cube testing for material used in shot-creting of permanent excavation walls and grouting of cable bolts.

**Sample Population and Size**

1. Mapping of Joints

33 joints were mapped on 3510 level in one limb and anticlinal area of 18 crosscuts in total, but only 12 were mapped. In the crosscuts of the synclinal area of 3960 level 25 joints were mapped in 8 crosscuts out of 10. The targeted population size was 70 joints if all crosscuts were accessible, but a total of 53 joints were mapped due inaccessibility representing **75%** of the data required for analysis.

2. Borehole logging

10 boreholes were logged out of 16 boreholes of NQ core size (standard outer diameter size of the drill rod, approximately 60.3 millimeters), representing 60% of the population needed for mechanical rock property testing in rock formations such as foot wall sandstone (FSAN), footwall conglomerate (FCON), basal sandstone (BQ), and south orebody shale (SOB).

3. Pull Testing

For every 100 double-ring cable bolts installed, 5% should be subjected to pull testing representing a target of 20 double-rings. However, in this research, 23 double-ring cable bolts were tested in the area of interest, representing 105%.

4. Compressive strength testing of concrete cubes

   The three companies contracted to do shotcreting at synclinorium underground are required to submitted 4 samples of concrete cubes each for compressive strength testing representing a total of 12 samples. This research tested 12 out of 12 samples of concrete cubes representing 100% of the required sample population.

**(vii) To assess ground control mechanisms and support standards, to improve ground stability.**

Geotechnical and geological data such as rock quality designation (RQD), geological strength index (GSI), rock mass rating (RMR), and uniaxial compressive strength (UCS) for understanding of the rock mass condition and behaviour was collected by the following means:

(i.) Geotechnical scanline mapping of geological structures.

(ii.) Rock mechanics laboratory/field testing.

(iii.) Borehole logging.

(iv.) Analysis of data and determination of quantitative geometry of rock mass.

(v.) Simulation of rock mass geometry by 3D statistical models.

Geological structures or discontinuities like joints, bedding planes, schistosity and folds were mapped as they cross the horizontal scanline in most cases. A 100m tape, Brunton campus, and Crinal ruler were the tools used. All the data collected would then be used to characterize the rock mass using the appropriate rock mass rating system (RMR) based on six rock parameters derived from the data. The analysis and evaluation of the geotechnical data collected results in the ability to predict and influence the behaviour of rock in a mining environment in line with the research objective. The geotechnical conditions that exist in the rock mass, together with the influence of mining activity, should be well understood in order to be able to predict or assess the ground conditions with any degree of reliability.

Rock behaviour under different stress conditions will be predicted using the following methods:

1. Rock types and properties

The first step in predicting rock behavior will be to identify the rock type and its physical and mechanical properties. Rocks in this research are classified into foot wall sandstone (FSAN), footwall conglomerate (FCON), basal quartzite (BQ), and south orebody shale (SOB).

Each rock type has different characteristics, such as porosity, permeability, density, strength, elasticity, and plasticity, that affect how it responds to stress. These properties will be measured in the field or in the laboratory, using various techniques and instruments, such as core samples, and rock tests.

2. Failure criteria and models

The second step in predicting rock behavior is to apply failure criteria and models that describe the relationship between stress and strain. Failure criteria are mathematical expressions that define the conditions under which a rock will fail or fracture under a given stress state. Some of the most common failure criteria are the Mohr-Coulomb, and Hoek-Brown criteria. Failure models are numerical or analytical methods that simulate the rock behavior under stress conditions.

**(viii) To evaluate the design, installation, and quality control of rock support and reinforce.**

(a.) The current rock support and reinforcement design process at Nkana Synclinorium was compared with other standard design processes from scholarly articles and books. Gaps and suggestions were made after comparisons. Nkana synclinorium mine has a standard rock support and reinforcement design process chart, and this chart lacked key steps that form part of the design process charts in world class underground mines cited in the literature of this research.

(b.) Empirical methods were used. This involved the use of objective, quantitative observation in a systematically controlled, replicable situation, in order to test how effective, the rock support and reinforcement was in underground excavations. Experienced-based application of known performance levels in tunnels, stope spans, installation inspections, and ground failures also formed part of the evaluation of the design process.

The evaluation of the design process will help to measure the effectiveness, usability, and desirability of the designed rock support and reinforcement. The evaluation was done by observing the actual standing time of some known, already supported and reinforced, excavations underground while identifying ground movement or failures, generating insights, and validating solutions.

Rock mass classification methods formed the formal part, and engineering judgment based on experience formed the informal part of empirical design.

(c.) Numerical modeling of underground excavations from credible world class base metal mines was considered and evaluated for possible application for ground support at the Nkana Synclinorium mine. In this study the elastic version of the computer program Map3D was used to determine the stress distribution around the underground excavations. The induced stresses were determined using the linear elastic numerical modelling. The required inputs are the insitu stress field with depth, and the estimated deformational properties such as Young modulous of elasticity. Typical output from numerical modelling includes stresses and displacements.

**(ix) To assess adherence of underground developments or excavations to design (size and geometry).**

(a.) Data from survey measurements for over-break and under-break was collected using a Theodolite. Data analysis and comparisons were made between original design of tunnel excavation size and geometry.

(b.) Laboratory tests rock specimen, and Numerical modeling simulations of crown pillars from research literature were reviewed and applied to situations at the research mine site during assessment. These were conducted to determine the effect of dip on the strength of pillars.

(c.) Empirical methods back-analysis were also used in the assessment of pillar design and strength using pillar stability charts. In back-analysis, rather than iterating to find the factor of safety for the pillar, the factor of safety is set to the user-specified value, and the force required to achieve the specified factor of safety is solved for.

### (ix) VCR Pillar Stability Assessment

Synclinorium mine rock mechanics section generally designs Rib pillars using analytical methods. Analytical methods are specific techniques or tools which are applied to collect, process, and interpret data. They can be quantitative, such as statistical analysis or calculations involving formulas related to the problem. Analytical methods help you to answer specific questions, test hypotheses, or evaluate outcomes.

The current pillar design process at Synclinorium mine is as outlined below:

**Pillar Design Process**

**1.** The first step is to obtain the RMR for the rock-mass forming pillar from joint mapping and core logging.

**2.** For unconfined stable pillar for entry rooms, first check for pillar stability w/h ratio => 1.5 (stable), for non-entry rooms, w/h ratio = 0.5 to 1.5 (unstable).

Safety factor, SF is then calculated as **SF =** Pillar strength/pillar load

The Pillar strength is estimated using the following empirical formula (Salamon and Munro)

$$\text{Pillar Strength} = \text{RMS}(\text{Wp}0.46/\text{Hp}0.66) \qquad (1)$$

Where:

**RMS =** Rock mass strength (36MPa) obtained from RMR.

**Wp =**Width of pillar

**Hp =** Height of pillar

This allows different pillar strength for pillars of different height and width

**4.** In the case of inclined orebodies, pillar stress is calculated based on vertical stress, σV, inclination of the orebody and the extraction ratio. The pillar stress can be estimated as follows:

$$\text{Pillar Stress} = (\sigma V\,\mathrm{Cos}2\alpha + \sigma h\,\sin 2\alpha)/1\text{-}\,\mathrm{e} = (83.6\mathrm{MPa}) \qquad (2)$$

Where:

**σV =** vertical stress field

**σh=** Horizontal stress field

**α =** Dip of ore body

**e =** Extraction ratio

An example of the design of a rib pillar at Synclinorium mine using the analytical approach is shown in Table 5.3

In the case of 790 VCR stope in Table 5.3, the pillar stress value is estimated to be 83 MPa at full pillar height of 50m, side. Whilst the pillar strength is estimated to be 7.2 MPa.

Giving Safety Factor: SF = 7.2/83.6 = 0.1, implying pillar failure.

This compares well with numerical modeling results which gives SF = 0.63 (with backfill on one side. Hence, the 8m pillar in VCR stopes with height in excess of 20m will fail as these are considered to be slender pillars and are only meant to separate the broken ore-producing stope from waste in the preceding backfilled stope. When yield, the pillar will still have some level of load carrying capacity under confinement.

**5.** The fifth check is by using numerical modeling using input from RMR, where the induced stress level in the pillar is compared with pillar strength. Where safety factor SF is < 1.0, it is assumed the pillar will fail.

In VCR method of bulk mining the rib pillar of 8m width and more than 40m height, the rib pillar at full stope height is basically a slender pillar and will not be strong enough to support the walls without confinement from the sides. In order to maintain the integrity of the rib pillar and to reduce the occurrence of sloughing it is essential that the preceding stope remains filled with backfilled and shrinkage ore in producing stope. As blasting progresses upward, the stope should be drawn under shrinkage.

# CHAPTER 4 – Results and Analysis

## 4.0 Introduction

Results and analyses or evaluations of research work relating to all the sub-objectives in the study area and in consistence with the research topic have been presented in this chapter. These are presented both in tabular form and graphical form.

## 4.1 Assessment of ground control mechanisms and support standards for stability.

### 4.1.1 Rock mass properties

Rock mass parameters based on Barton's Q system have been officially adopted by Nkana Synclinorium mine for underground evaluation. These were used in empirical stability graphs for underground excavation design (Table 4.1). Bieniawski RMR (1989) values have been converted to Q values using the formula. Table 4.2 shows summarized rock mass parameters and ratings by rock formation as determined by this research while Table 4.3 shows how the rock mass ratings from Table 4.2 compare with Barton's Q system RMR ranges

**RMR = 9 lnQ+44**

**Table 4.1: Nkana South Mine RMR values (Bieniawski, 1989)**

| ROCK UNIT | RQD Range | IRS Range | RMR Range | RMR Description | Q Range | Q Description |
|---|---|---|---|---|---|---|
| Basal Quartzite | 80-100 | 100-250 | 70-80 | Good | 18 - 55 | Good - very Good |
| Lower Conglomerate | 60-80 | 50-100 | 38 - 58 | Poor - Fair | 1 - 5 | Poor - Fair |
| Foot wall Quartzite | 40-80 | 50-100 | 60 - 70 | Fair - Good | 6 - 18 | Fair - Good |
| SOB Shale | 40-68 | 50-100 | 40 - 60 | Poor - Fair | 1 - 6 | Poor - Fair |
| Hanging wall Argillite | 40-70 | 50-100 | 40 - 60 | Poor - Fair | 1 - 6 | Poor - Fair |

Table 4.2 shows summarized data used in the study of the rock samples. The raw data is in Appendix A

**Summary of data used in the study-raw data is in the Appendix A**

**Table 4.2: Rock Mass Ratings (RMRs) for rock samples at Synclinorium mine**

| ROCK UNIT | PLS Test (MPa) | RQD (%) | JOINT SPACING (cm) | RMR | GSI | Q |
|---|---|---|---|---|---|---|
| SOBS | 47 | 62 | 14 | 58 | 59 | 25 |
| HWA | 65 | 75 | 26 | 70 | 62 | 43 |
| FSAN | 116 | 76 | 20 | 78 | 64 | 46 |
| FCON | 105 | 70 | 16 | 65 | 67 | 40 |
| BQ | 178 | 83 | 33 | 82 | 75 | 53 |

Five rock types SOBS, HWA, FSAN, FCON and BQ had their rock samples subjected to rock mass rating. The RMRs for each rock type are derived from Tables 4.4 to 4.8 and the composite data is tabulated in Table 4.2 for each rock type.

Table 4.3 shows a comparison between RMRs used in this study against the Barton's Q system RMR ranges.

**Table 4.3: RMRs used in this study versus Barton's Q system RMR ranges.**

| FORMATION | LOGGED SAMPLES RMR | Barton's Q system RMR | RMR Description |
|---|---|---|---|
| BQ | 82 | 70 - 80 | Good |
| FCON | 65 | 38 - 58 | Poor - Fair |
| FSAN | 78 | 60 - 70 | Fair - Good |
| SOB | 58 | 40 - 68 | Poor - Fair |
| HWA | 70 | 40 - 70 | Poor - Fair |

The RMRs shown for rock samples in Table 4.2 (column 5) are now subjected to comparison with the Barton's Q system RMR range, in Table 4.3 to check how they fair.

The detailed ratings for individual rock formations studied and analyzed for RMRs are outlined in the Tables 4.4 - 4.8 for each rock type. The explanation of the values in the named tables is given in Chapter 5 on discussion of results.

**Table 4.4: Detailed Rock Mass Rating for Shale (SOB)**

| LITHOLOGY | ITEM | VALUE | RATING |
|---|---|---|---|
| Shale (SOBS) | Point Load Index | 47 MPa | 12 |
| | RQD | 62% | 11 |
| | Spacing of discontinuities | 20cm | 13 |
| | Condition of discontinuities | - | 15 |
| | Ground water | - | 12 |
| | Adjustment for joint orientation | - | -5 |
| TOTAL | | | 58 |

**Table 4.5: Detailed Rock Mass Rating for Hanging Wall Argillite (HWA)**

| LITHOLOGY | ITEM | VALUE | RATING |
|---|---|---|---|
| Hanging Wall Argillite (HWA) | Point Load Index | 65 MPa | 18 |
| | RQD | 75% | 15 |
| | Spacing of discontinuities | 170 cm | 20 |
| | Condition of discontinuities | - | 12 |
| | Ground water | - | 10 |
| | Adjustment for joint orientation | - | -5 |
| TOTAL | | | 70 |

**Table 4.6: Detailed Rock Mass Rating for Foot Wall sandstone (FSAN)**

| LITHOLOGY | ITEM | VALUE | RATING |
|---|---|---|---|
| Foot Wall Sandstone (FSAN) | Point Load Index | 116 MPa | 14 |
| | RQD | 76% | 15 |
| | Spacing of discontinuities | 35cm | 18 |
| | Condition of discontinuities | - | 21 |
| | Ground water | - | 15 |
| | Adjustment for joint orientation | - | -5 |
| TOTAL | | | 78 |

**Table 4.7: Detailed Rock Mass Rating for Foot Wall Conglomerate (FCON)**

| LITHOLOGY | ITEM | VALUE | RATING |
|---|---|---|---|
| Foot Wall Conglomerate (FCON) | Point Load Index | 105 MPa | 15 |
| | RQD | 70% | 14 |
| | Spacing of discontinuities | 32 cm | 13 |
| | Condition of discontinuities | - | 16 |
| | Ground water | - | 12 |
| | Adjustment for joint orientation | - | -5 |
| TOTAL | | | 65 |

**Table 4.8: Detailed Rock Mass Rating for Basal Quartzite (BQ)**

| LITHOLOGY | ITEM | VALUE | RATING |
|---|---|---|---|
| Basal Quartzite (BQ) | Point Load Index | 178 MPa | 20 |
| | RQD | 83% | 17 |
| | Spacing of discontinuities | 245cm | 21 |
| | Condition of discontinuities | - | 16 |
| | Ground water | - | 13 |
| | Adjustment for joint orientation | - | -5 |
| TOTAL | | | 82 |

The rock types of the various samples evaluated in the research were described based on their UCS values using the Point Load Index test method classifying their strength as shown in Table 4.9

**Table 4.9: Description of Rock Strength based on the Point Load Index test**

| ROCK TYPE | DESCRIPTION | EQUIVALENT UCS (MPa) |
|---|---|---|
| Basal Quartzite (BQ) | Extra strong | 178 |
| FW Sandstone | Strong rock | 116 |
| FW Conglomerate | Very strong rock | 105 |
| HWA (Argillite) | Strong rock | 65 |
| SOB (Shale) | Medium Strong rock | 47 |

The rock types of the various samples evaluated in the research were described based on their UCS values using the Point Load Index test method classifying their strength as shown in Table 4.9

The Point Load (PL) is a test that aims at characterizing rock materials in terms of strength. The UCS equivalent strengths of the rock types shown were derived from Point Load Strength Index and are now characterized by a standard rock strength description.

### 4.1.2 Correlation Analysis

In Figure 4.1 a relationship was drawn between Unconfirmed Compressive Strength (UCS) and Geological Strength Index (GSI), both of which border on rock strength, for the samples of the rock types tested at the laboratory.

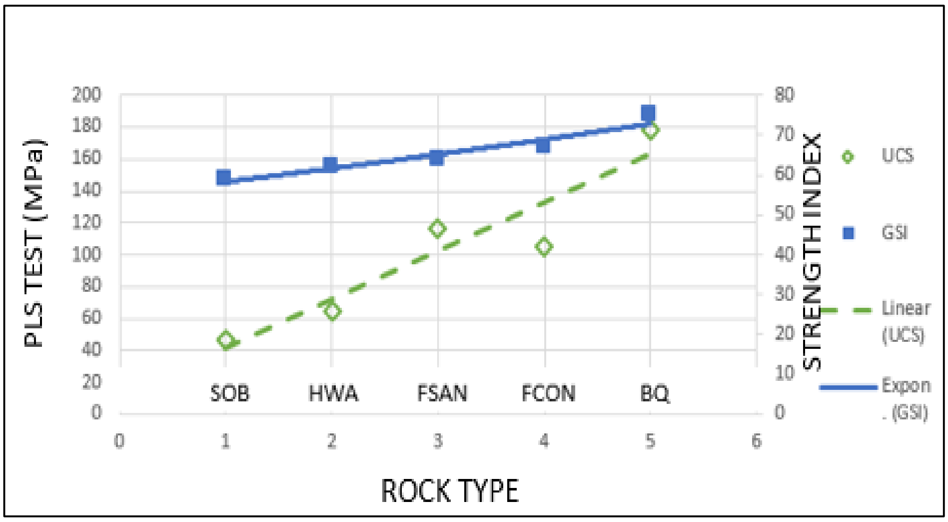



**Figure 4.1: The dependence of Rock Strength (UCS and GSI) on Rock Type**

Unconfirmed Compressive Strength (UCS), measured in MPa, was plotted against the Geological Strength Index (GSI). This was aimed at checking how the rock types of the samples compare with regard to the two rock parameters. Therefore, as observed in the Figure 4.1 SOB proves to be the weakest rock type in terms of UCS at 40MPa, with the lowest GSI. BQ (Basal Quartzite) is the strongest rock with UCS at 70 MPa and GSI at 72MPa.

Figure 4.2 shows a comparison between UCS and the two other parameters which are closely associated (RQD and Joint spacing).

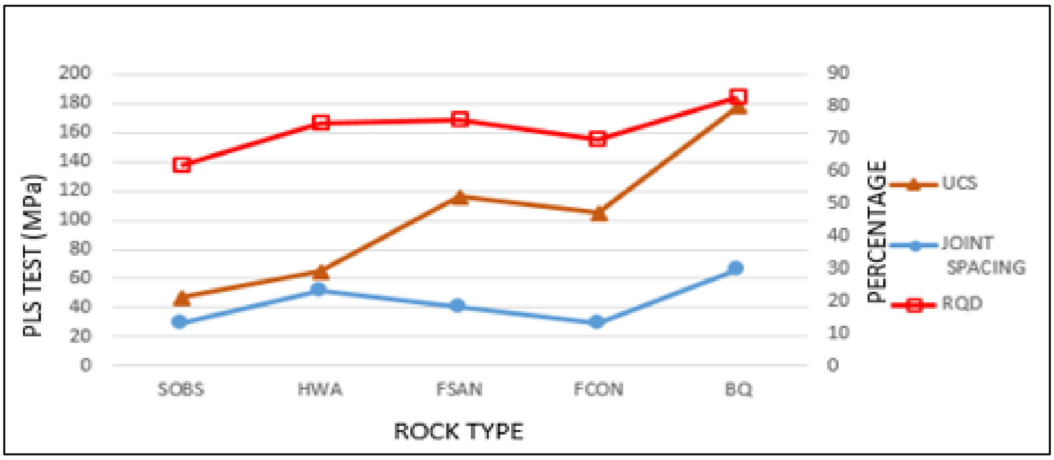



**Figure 4.2: Potential Correlation-Rock Strength (UCS) and RQD/Joint Spacing**

Rock Quality Designation (RQD) and Joint Spacing are some of six input parameters used in rock mass classifications, and both affect the strength of the rock. Hence their correlation in the graph in Figure 4.2, but in terms of UCS and RQD the graph shows there is also a tendency for RQD to increase as UCS increases. Higher values of RQD indicate better rock quality and can only endure failure at higher uniaxial compressive strength.

Figure 4.3 shows comparison between UCS and Density of the rocks to see how the latter varies with rock type.

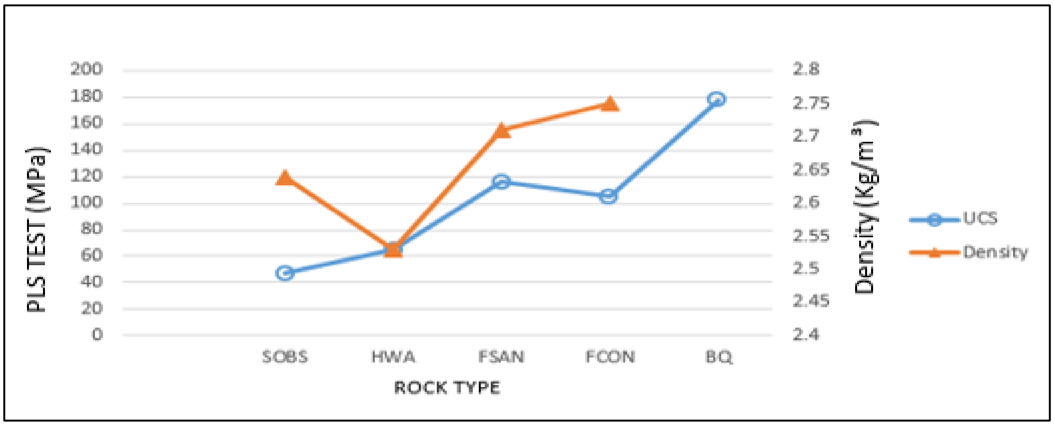



**Figure 4.3: Potential Correlation-Rock Strength (UCS) and Rock Density**

Denser rock types (FSAN, FCON, BQ) are more likely to be stronger with more strength than the highly weathered SOBS and HWA, as shown in Figure 4.3

### 4.1.3 Orientation Analysis of mapped joints

The observations made mapping on 3510 and 3960 levels of underground are presented separately under discussion of results in Chapter. Figure 4.4 shows the complex folding of the orebody in section with some mined-out areas in grey hatches while Figure 4.5 shows the mapped limbs and anticline.

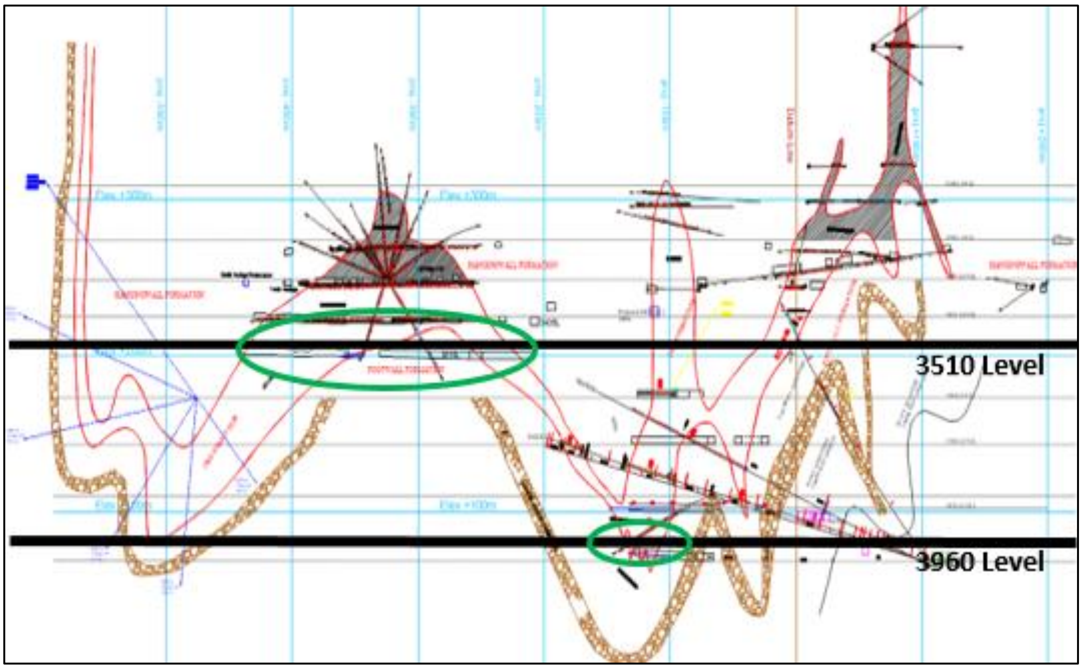



**Figure 4.4: 1038mN Section showing areas complex folded mapped in the Synclinorium**

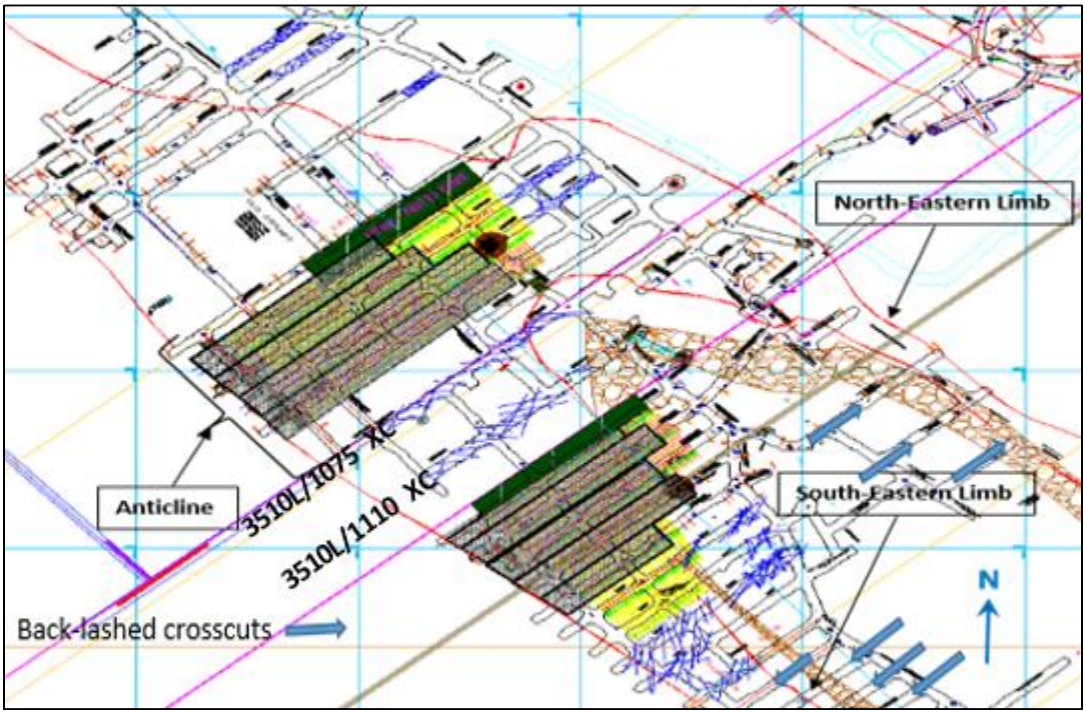
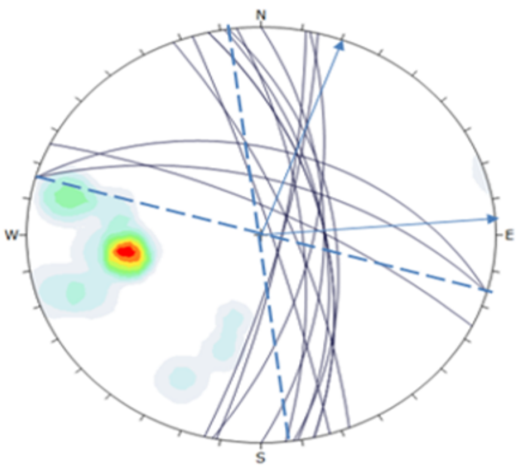



**Figure 4.5: 3510 Level Plan showing areas mapped (Grey areas indicate ore stoped out).**

Scale 1:1000; Source-Mopani Geology Dept.

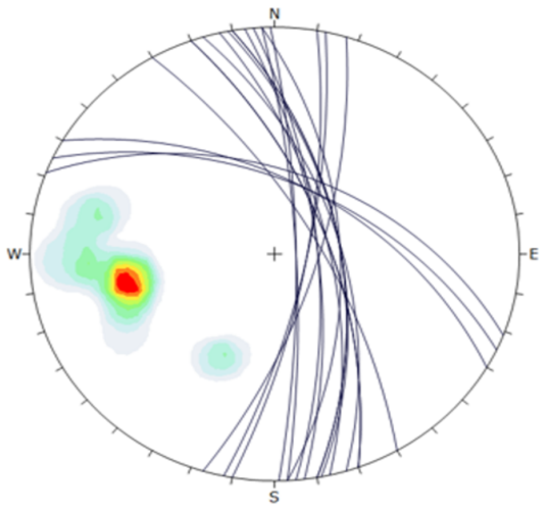
Figures 4.6 and 4.7 show contoured Stereonets of cross cuts (3510L/1075 and 3510/1110) in the anticline zone in Figure 4.5




| Colour | Density Concentrations |
|---|---|
| | 0.00 - 3.30 |
| | 3.30 - 6.60 |
| | 6.60 - 9.90 |
| | 9.90 - 13.20 |
| | 13.20 - 16.50 |
| | 16.50 - 19.80 |
| | 19.80 - 23.10 |
| | 23.10 - 26.40 |
| | 26.40 - 29.70 |
| | 29.70 - 33.30 |
| Maximum Density | 32.69% |
| Contour Data | Pole Vectors |
| Contour Distortion | Fisher |
| Contour Circle Size | 10% |
| Maximum Mode | Pole Vectors |
| Vector Count | 24 (24 Entries) |
| Hemisphere | Lower |
| Projection | Equal Angle |

**Figure 4.6: Stereonet for 3510 Level-1075 Cross Cut**

The joints sets indicated by black half-circles dip East and North-east. The joint density concentration ranges from 26 to 33 (Brown/Red) in the SW.




| Colour | Density Concentrations |
|---|---|
| | 0.00 - 3.60 |
| | 3.60 - 7.20 |
| | 7.20 - 10.80 |
| | 10.80 - 14.40 |
| | 14.40 - 18.00 |
| | 18.00 - 21.60 |
| | 21.60 - 25.20 |
| | 25.20 - 28.80 |
| | 28.80 - 32.40 |
| | 32.40 - 36.00 |
| Maximum Density | 35.99% |
| Contour Data | Pole Vectors |
| Contour Distortion | Fisher |
| Contour Circle Size | 10% |
| Maximum Mode | Pole Vectors |
| Vector Count | 19 (19 Entries) |
| Hemisphere | Lower |
| Projection | Equal Angle |

**Figure 4.7: Stereonet for 3510 Level-1110N Cross Cut**

(Joint sets dip East and North East, with concentration in SW).

The joint sets in Table 4.10 dip steeply, with an average dip of 66˚, to the East and North-East in the anticlinal structure of the Synclinorium orebody on 3510 level.

**Table 4.10: Joint set for mapped cross cuts on 3510 Level**

| ID | Dip | Dip Direction |
|---|---|---|
| 1 | 60 | 77 |
| 2 | 60 | 77 |
| 3 | 45 | 16 |
| 4 | 75 | 68 |
| 5 | 80 | 104 |
| 6 | 75 | 26 |
| 7 | 60 | 90 |
| 8 | 65 | 102 |
| 9 | 70 | 84 |
| 10 | 60 | 81 |
| 11 | 80 | 101 |
| 12 | 55 | 82 |
| 13 | 85 | 73 |
| 14 | 60 | 16 |

| ID | Dip | Dip Direction |
|---|---|---|
| 1 | 60 | 87 |
| 2 | 65 | 70 |
| 3 | 80 | 89 |
| 4 | 55 | 25 |
| 5 | 60 | 79 |
| 6 | 60 | 80 |
| 7 | 70 | 107 |
| 8 | 60 | 80 |
| 9 | 65 | 70 |
| 10 | 70 | 83 |
| 11 | 70 | 60 |
| 12 | 75 | 100 |
| 13 | 80 | 87 |
| 14 | 65 | 77 |
| 15 | 50 | 21 |
| 16 | 60 | 73 |
| 17 | 70 | 85 |
| 18 | 75 | 102 |
| 19 | 55 | 30 |

(3510 Level/1075 and 3510 Level 1110).

As shown in Figure 4.5, lack of accessibility to both limbs of the anticline resulted in only one limb being mapped extensively on 3510 level.

Figure 4.8 is an idealized stereonet illustrating what the scenario would be if both limbs in the structure were equally accessed and mapped.

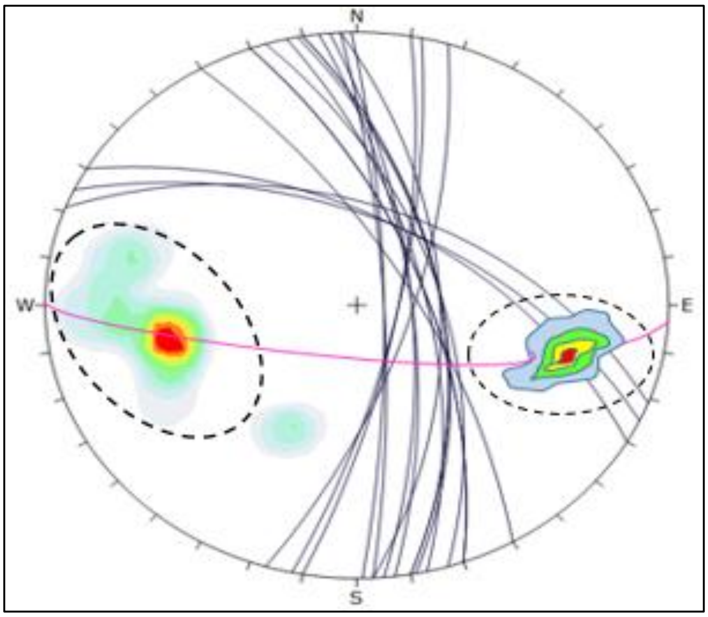



**Figure 4.8: Idealized Stereonet depicting well-covered and mapped limbs of Figure 4.5**

Figure 4.9 shows 3960 level which marks the base of the synclinorium structure and no limbs are expected to be mapped.

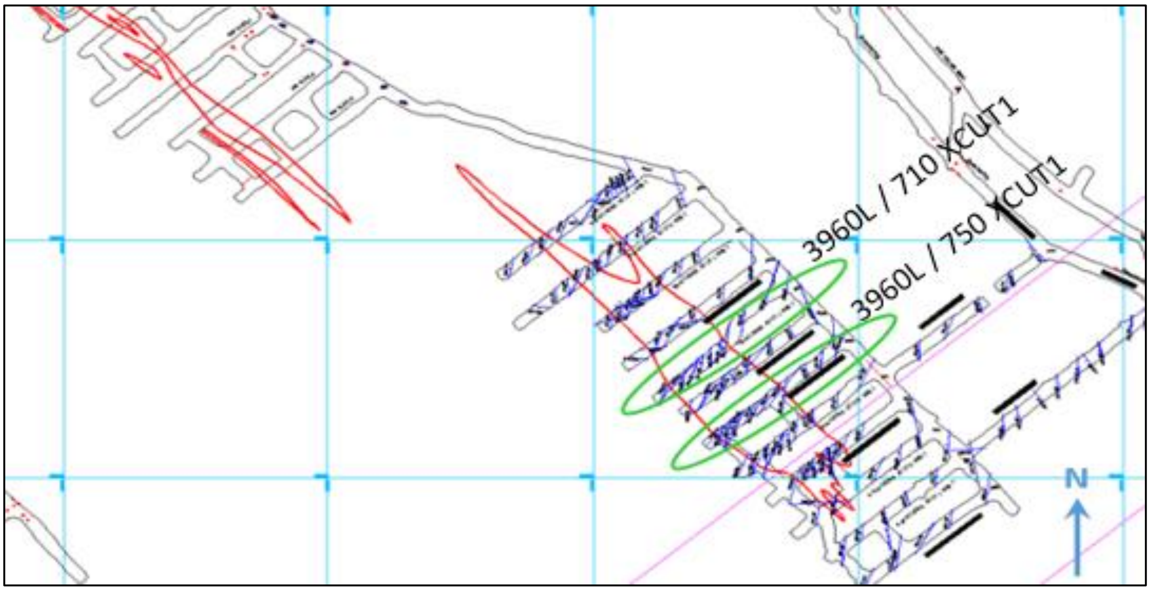
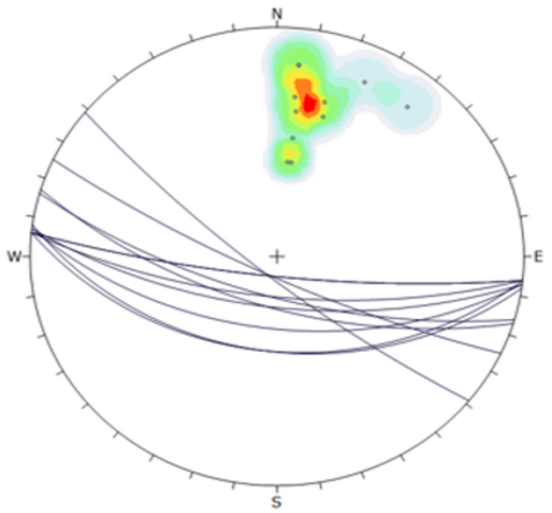



**Figure 4.9: 3960 Level Plan showing the areas mapped**

(The ore body is shown in red outline. Scale 1:1000; Source-Mopani Geology Dept.)

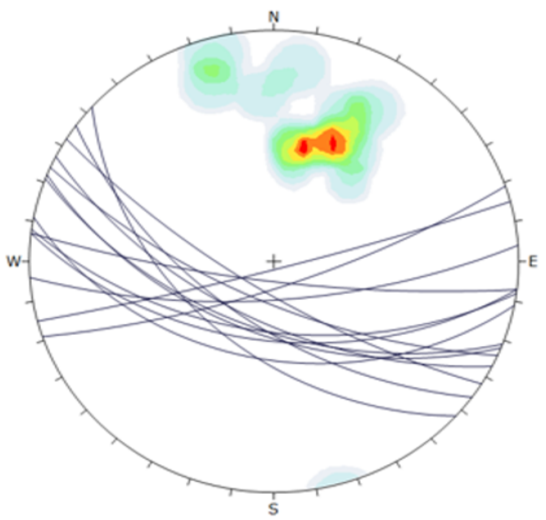
Figures 4.10 and 4.11 show contoured Stereonets of cross cuts in the SE limb of the Synclinorium base, on section in figure 4.5




| Colour | Density Concentrations |
|---|---|
| | 0.00 - 3.20 |
| | 3.20 - 6.40 |
| | 6.40 - 9.60 |
| | 9.60 - 12.80 |
| | 12.80 - 16.00 |
| | 16.00 - 19.20 |
| | 19.20 - 22.40 |
| | 22.40 - 25.60 |
| | 25.60 - 28.80 |
| | 28.80 - 32.00 |
| Maximum Density | 31.24% |
| Contour Data | Pole Vectors |
| Contour Distortion | Fisher |
| Contour Circle | 10% |
| Maximum Mode | Pole Vectors |
| Vector Count | 11(11 Entries) |
| Hemisphere | Lower |
| Projection | Equal Angle |

**Figure 4.10: Stereonet for 3960 Level - 750 Cross Cut**

(Joint density concentrations are in the North and are generally dipping southwards).




| Colour | Density Concentrations |
|---|---|
| | 0.00 - 2.40 |
| | 2.40 - 4.80 |
| | 4.80 - 7.20 |
| | 7.20 - 9.60 |
| | 9.60 - 12.00 |
| | 12.00 - 14.40 |
| | 14.40 - 16.80 |
| | 16.80 - 19.20 |
| | 19.20 - 21.60 |
| | 21.60 - 24.00 |
| Maximum Density | 23.64% |
| Contour Data | Pole Vectors |
| Contour Distortion | Fisher |
| Contour Circle | 10% |
| Maximum Mode | Pole Vectors |
| Vector Count | 11(11 Entries) |
| Hemisphere | Lower |
| Projection | Equal Angle |

**Figure 4.11: Stereonet for 3960 Level - 710 Cross Cut**

(The Joint density concentration lies in the North-East and generally dips in the South and SW).

The dip of the joints mapped at the base of the syncline on 3960 level in Table 4.11 are also steeply dipping averaging 65˚. As expected, folded areas of the structure are associated with steep joint planes.

**Table 4.11: Joint set orientation data for 3960L/750 and 710 Cross Cuts**

| ID | Dip | Dip Direction |
|---|---|---|
| 1 | 80 | 186 |
| 2 | 65 | 197 |
| 3 | 45 | 186 |
| 4 | 70 | 186 |
| 5 | 55 | 187 |
| 6 | 45 | 188 |
| 7 | 65 | 187 |
| 8 | 70 | 196 |
| 9 | 80 | 186 |
| 10 | 80 | 219 |
| 11 | 80 | 205 |

| ID | Dip | Dip Direction |
|---|---|---|
| 1 | 57 | 201 |
| 2 | 71 | 176 |
| 3 | 80 | 187 |
| 4 | 86 | 165 |
| 5 | 52 | 187 |
| 6 | 60 | 216 |
| 7 | 75 | 212 |
| 8 | 70 | 204 |
| 9 | 61 | 207 |
| 10 | 55 | 202 |
| 11 | 77 | 161 |
| 12 | 44 | 191 |
| 13 | 56 | 188 |
| 14 | 49 | 222 |

A haulage inspection-walk along 3360 level of synclinorium mine reveled a sidewall excavation meant for a waiting place or refuge bay. The hanging wall had two joint sets criss-crossing along strike and across strike as shown in Figure 4.12. A highly stressed timber set installed for support is tilted leaving major joint sets unsecured over a large span of hanging wall.


**Figure 4.12: Timber pack under a jointed and stressed rock mass**

(Source - Picture taken at Nkana Synclinorium underground, 3360 Level Haulage).

The joint orientation and different stresses acting on the surrounding rock mass are illustrated and evaluated in Figure 4.13

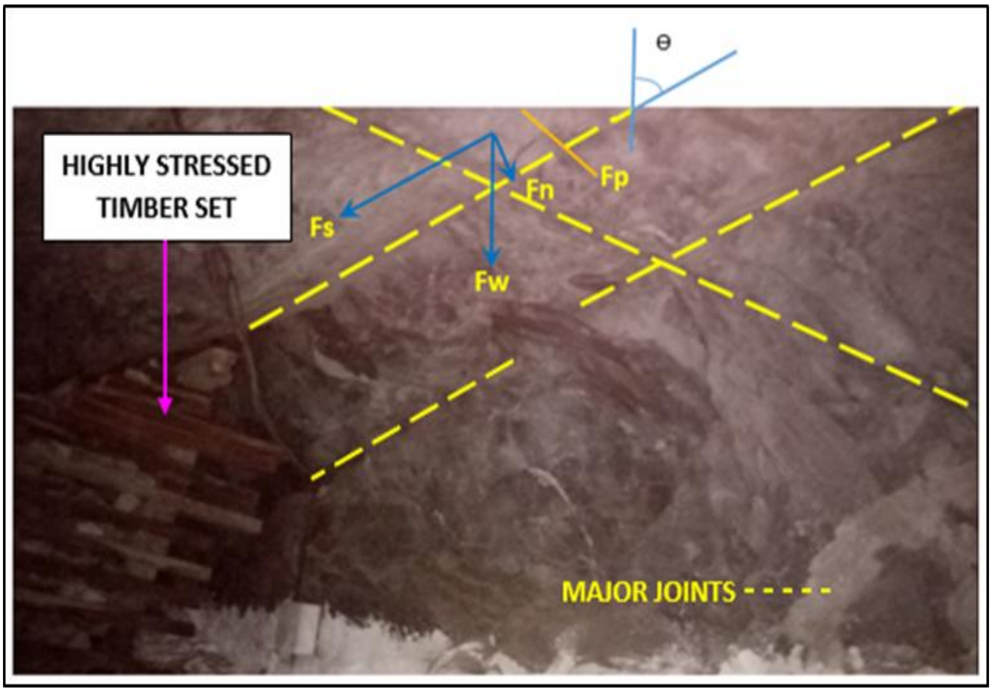



**Figure 4.13: Analysis of joint patterns and joint plane inclinations**

(Source - Picture taken at Nkana Synclinorium underground, 3360 Level Haulage).

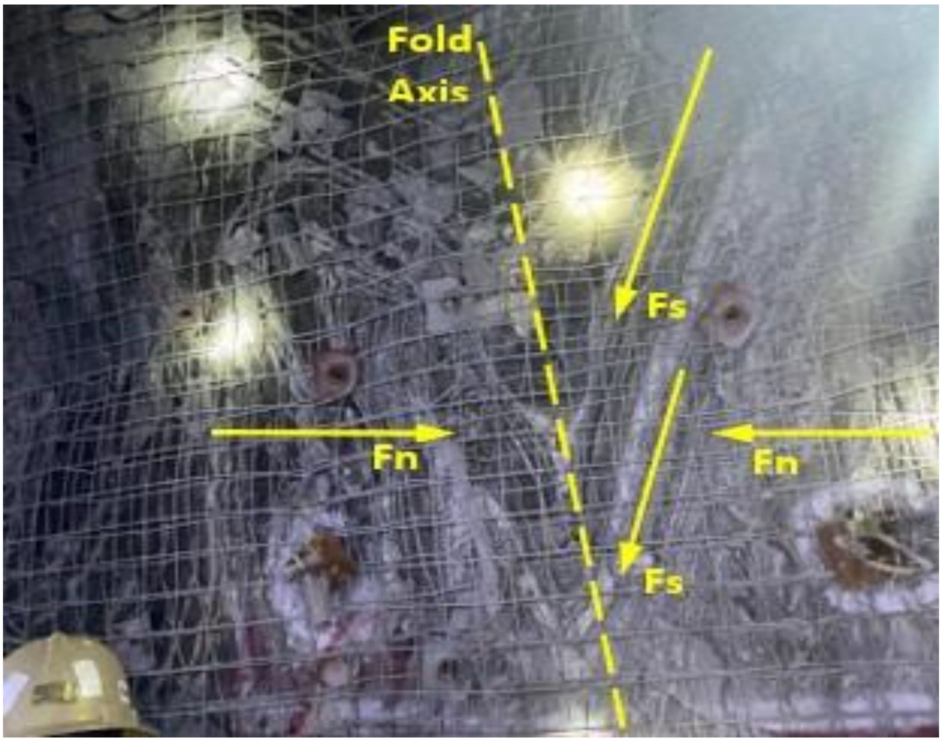
This helps in understanding the distribution of forces acting in a stressed rock. Hence any passive support like Props or Timber packs will be effectively applied. The scenario in Figure 4.13 has been discussed in more detail in chapter 4, under methodology.

Figure 4.14 shows intense folding and shearing in a cross cut mined through the ore shale formation at 3435 level of synclinorium underground.


**Figure 4.14: Highly folded and sheared mineralized Shale**

(Source - Picture taken by author at Nkana Synclinorium underground, 3435 Level).

Shear displacement and stress forces acting on a synclinal structure in 810 cross cut on 3860 level of synclinorium mine are outlined in Figure 4.15




Forces acting on the fold limbs:

**Fs** = Shear Displacements

**Fn** = Stress forces perpendicular to limbs

**Figure 4.15: Highly folded and mineralized sheared Shale supported**

(Source - Picture taken by author at Nkana Synclinorium underground, 3860 Level).

At the Synclinorium underground, site observations, diamond drilling and geological mapping in this research have revealed large scale folds and small localized folds. It is evident that the folds are not only characterized by joints, but also by shear displacements along the limbs towards the fold axis and stress forces perpendicular to the limb (Figure 4.16), and this affects the stability of the folded rock mass.

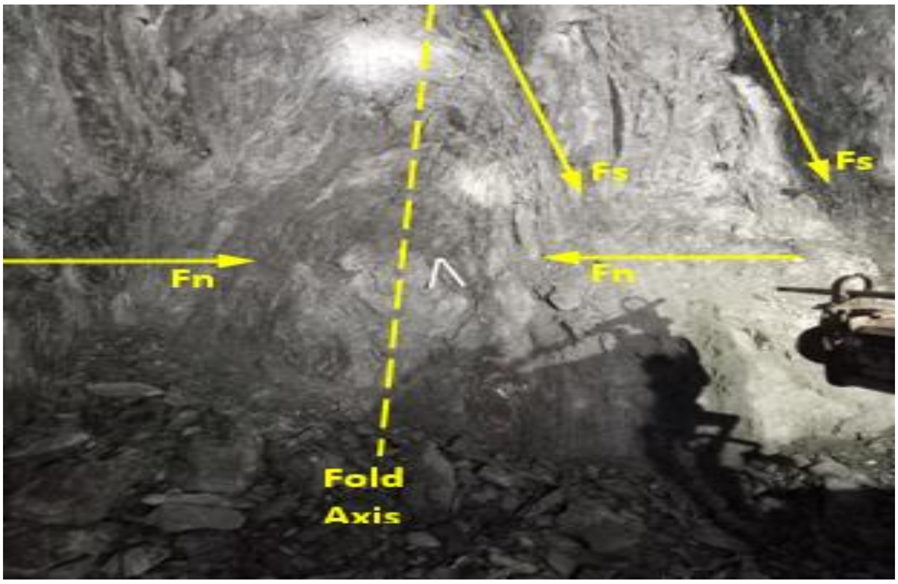
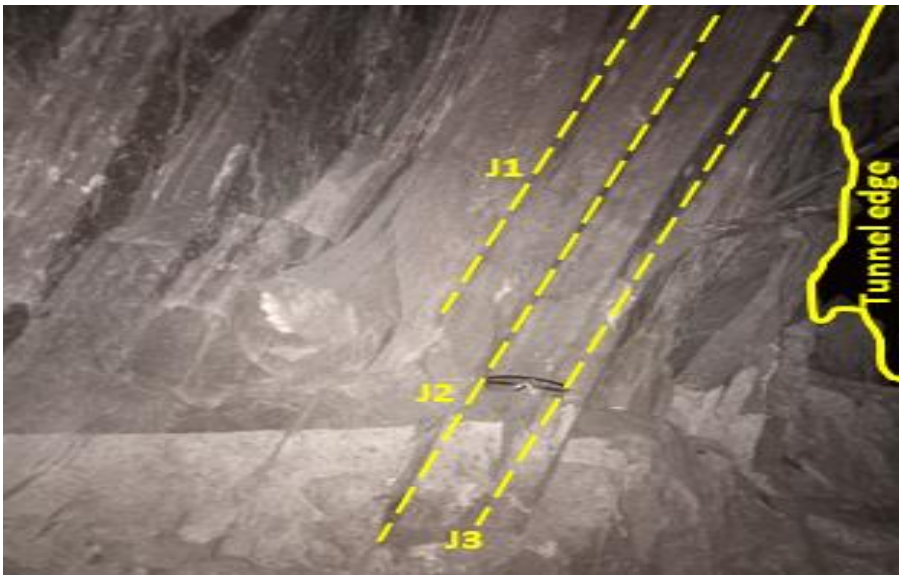



**Figure 4.16: Highly folded and weathered mineralized Shale.**

(Source-Picture taken at Nkana Synclinorium underground, 3585 Level).

White 1m 4 Folds Plastic Ruler - folded half in picture is 25cm long.

Figure 4.17 shows fresh black shale with joint planes parallel to the fine laminations on 3760 level.




Strike of joint planes (**J1**, **J2**, **J3**) parallel to tunnel axis but dip **70°**;

**Figure 4.17: Fresh laminated and jointed barren Shale**

(Source-Picture taken by author at Nkana Synclinorium underground, 3760 Level).

Length of safety goggles is about 20cm)

### 4.1.4 Evaluation of Ground Support Standards at Synclinorium mine

All the reinforcement support elements commonly used at synclinoriun underground such as split sets, cable bolts and groutable rock bolts as well as surface elements like shotcrete are tracked for quality test on a weekly basis at synclinorium mine. The ground support installation quality and effectiveness of these support elements, as they arrest the displacement of a rock underground, are monitored through site observations, pull tests and laboratory tests like compressive uniaxial tests. In the case of rock bolts, they transfer the load from the unstable exterior to the confined (and much stronger) interior of the rock mass. If a rock bolt is subjected to pull testing and fails below a threshold strength of 8 tons, for example then the quality of installation was of poor standard its ground support performance is not effective.

For shotcreting, concrete cube samples of the shotcrete material have to be subjected to compressive testing in which case compressive load would be applied gradually at the compressive strength of 13.7293 N/mm². Failure of concrete cube below this compressive strength shows that the shotcrete material used for shotcreting is of poor quality.

**Table 4.12: Recommended ground support testing frequency and specifications**

| Element | Testing method | % of element to be tested | Frequency | Required anchorage (tons) |
|---|---|---|---|---|
| Groutable Rock bolt | Pull test – full column | 5% of installed bolts | Weekly | 10 |
| Split set | Pull test – full anchor | 5 per batch | Weekly | 8 |
| Shotcrete | Cube tests | | | |
| | Flow test | | | |
| | Depth probe | | Per application | |
| | Panel test | | Per 50m³placed | |
| | Insitu cores | | | |
| Cable bolt | Load in cable bolt – SMART cable | 5% of installed bolts | | |
| Timber support | Load in packs | | | |
| Backfill | Load | | | |

Figure 4.18 shows a plot of results for Pull Test1 conducted on 3360 level haulage.

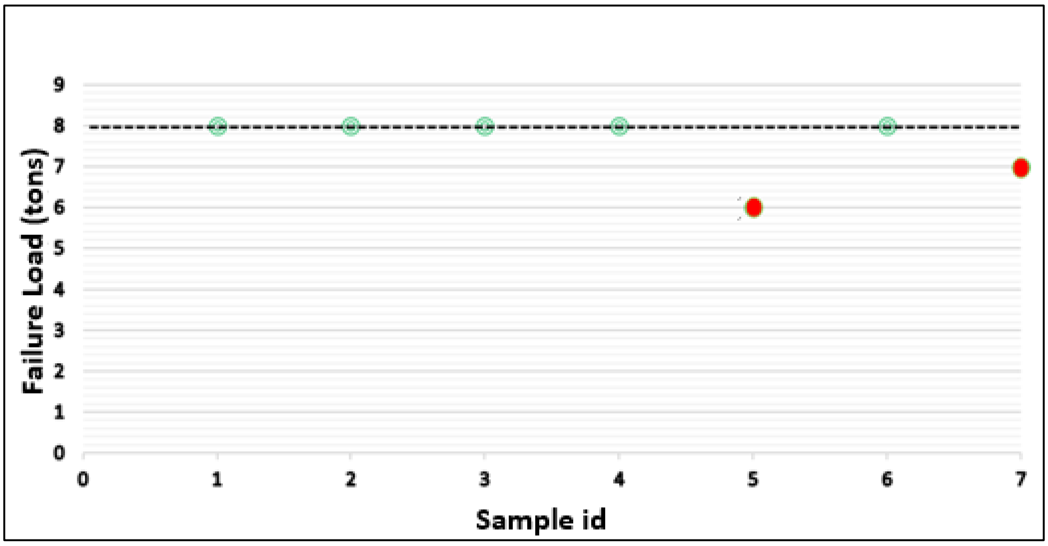



**Figure 4.18: Pull Test1 Graph (3360 Level Haulage)**

Eight Rock bolts (split sets) were subjected to pull testing with a threshold strength set at 8 tons. Five border-line cases in light blue passed at load of 8 tons, and the two red ones failed at the loads of 6 and 7 tons respectively.

The results of Pull Test1 carried out on 3360 level Main Haulage were captured and tabulated as shown in Table 4.13

**Table 4.13: Pull Test1 Results**

| Test No. | Distance(m) Ref. Peg No. (C1583) | Wall | Bolt Length | Bond Type | Displacement (mm) | Test to 8 Tons | | Remarks |
|---|---|---|---|---|---|---|---|---|
| | | | | | | PASS | FAIL | |
| SS1 | 65.7 | East | 2.4m | Friction | None | 8 | | Pass no displacement |
| SS2 | 65.7 | West | 2.4m | Friction | None | 8 | | Pass no displacement |
| SS3 | 68.3 | East | 2.4m | Friction | None | 8 | | Pass no displacement |
| SS4 | 90.4 | West | 2.4m | Friction | None | 8 | | Pass no displacement |
| SS5 | 117.3 | West | 2.4m | Friction | >50mm | | 6 | Fail with noticeable displacement (Installed in strongly foliated zone) |
| SS6 | 121.8 | West | 2.4m | Friction | None | 8 | | Pass no displacement |
| SS7 | 123.4 | East | 2.4m | Friction | >50mm | | 7 | Fail with noticeable displacement (Installed in strongly foliated zone) |

Table 4.14 shows a layout for the pull test carried out on 3360 Main Haulage, including recommendations for split set size and rock mass characteristics.

**Table 4.14: Pull Test1 Preparation**

| | |
|---|---|
| **Project:** | 3360L _MAIN HAULAGE |
| **Pull test no:** | **1** |
| **Anchor type:** | Split set; 2.4m long |
| **Rock bolt diameter:** | 46mm |
| **Hole diameter:** | 43mm |
| **Tunnel size:** | 4.5m x 4.5m |
| **Rock Mass Classification:** | - Basement Schistose: Fair to good rock mass with little persistent joints<br>- Bedded rock mass parallel to excavation axis<br>- Foliations exposed on the western side wall intersected by moderate joints<br>- RMR range of (41- 60) fair rock mass.<br>- UCS = 50 - 100 MPa |
| **Date of Installation** | 15/08/2022 – 16/09/2022 |
| **Date of test:** | 16.08.2023 |

Preparation for Pull Test1 included knowing the project site and rock types, rock bolt type and diameter, as well as Tunnel size.

Figure 4.19 shows the plan indicating the sites of pull tests on 3360L Main Haulage and the associated results. The plot of results for a pull test conducted on Seven (07) double ring split sets installed along the grade line of the 3360 level Main Haulage, two (02) failed and five (05) passed the test. The Pull Test results represent a 71% pass rate and 29% failed bolts.

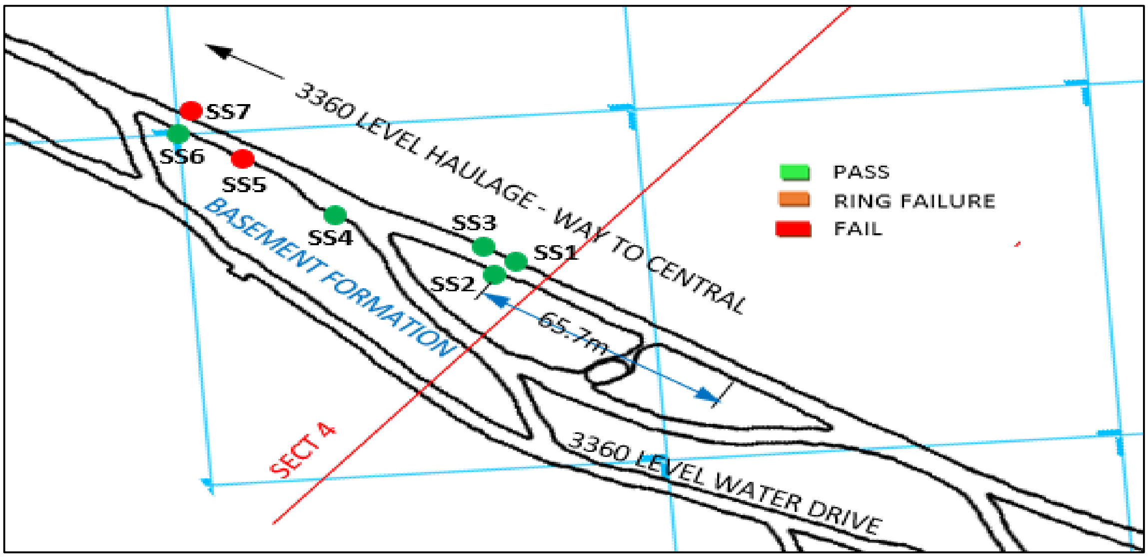



**Figure 4.19: Pull Test1 - Double Ring Split Sets**

In Figure 4.20, the graph indicates 3 failed pull tests for test 2 (in Red for a threshold set at 10 tons). Fives tests passed at the load of 10 tons.

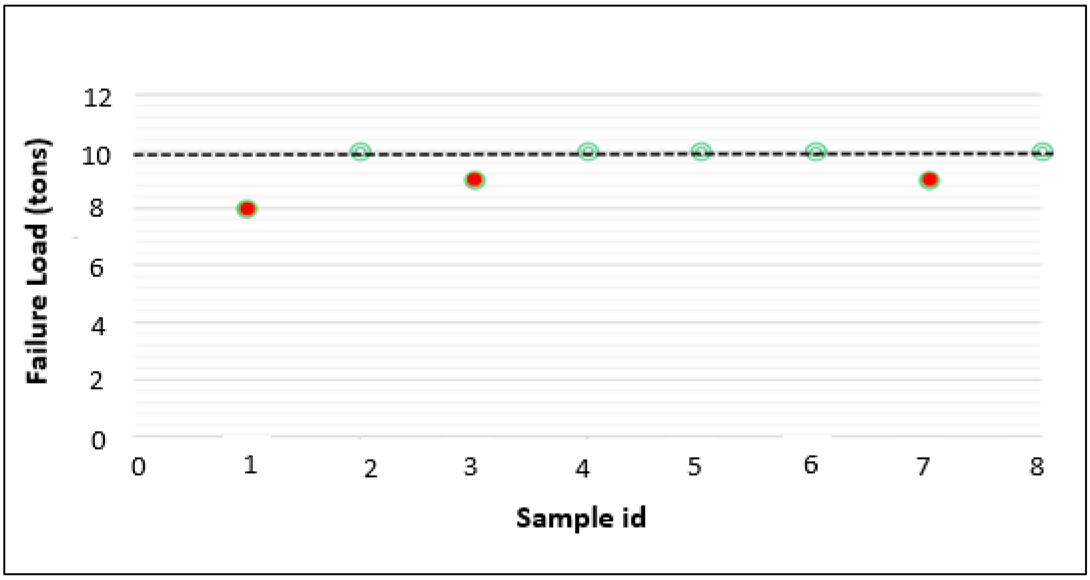



**Figure 4.20: Pull Test2 Graph (3960Level-1150mN Loader Drive North)**

Table 4.15 shows the Pull Test 2 results as captured at the site. The 3 failed tests recorded were attributed to the split set rings.

**Table 4.15: Pull Test 2 Results**

| Test No. | Distance(m) Ref. Peg No. (T5999) | Wall | Bolt Length | Bond Type | Displacement (mm) | Test to 10 Tons | | Remarks |
|---|---|---|---|---|---|---|---|---|
| | | | | | | PASS | FAIL | |
| SS1 | 1.4m | North | 2.4m | Friction | None | 10 | 8 | Pass – Ring failure no displacement |
| SS2 | 6.9m | North | 2.4m | Friction | None | 10 | | Pass - no displacement |
| SS3 | 6.9m | South | 2.4m | Friction | None | 9 | 9 | Pass – Ring failure no displacement |
| SS4 | 9.9m | South | 2.4m | Friction | None | 10 | | Pass - no displacement |
| SS5 | 12.9m | North | 2.4m | Friction | None | 10 | | Pass - no displacement |
| SS6 | 12.9m | South | 2.4m | Friction | None | 10 | | Pass - no displacement |
| SS7 | 24.4m | South | 2.4m | Friction | None | 10 | 9 | Pass – Ring failure no displacement |
| SS8 | 25.6m | South | 2.4m | Friction | None | 10 | | Pass - no displacement |

Table 4.16 shows a layout for the pull test carried out on 3960 level 1150N Loop Loader Drive, including recommendation for split set size and rock mass characteristics.

**Table 4.16: Pull Test 2 Preparation**

| | |
|---|---|
| **Project:** | 3960L 1150mN LOOP LOADER DRIVE NORTH |
| **Pull test no:** | **2** |
| **Anchor type:** | Split set; 2.4m long |
| **Rock bolt diameter:** | 46mm |
| **Hole diameter:** | 43mm |
| **Tunnel size:** | 4.5m x 4.5m |
| **Rock Mass Classification:** | (i.) Basal Sandstone – Lower Conglomerate rock unit<br>(ii.) Fairly competent rock and dry rock mass<br>(iii.) RMR fair rock unit.<br>(iv.) Estimated UCS of 150Mpa and 45Mpa respectively<br>(v.) Unfavorable joint orientation with respect to the tunnel |
| **Date of Installation** | 01/05/2021 – 29/06/2021 |
| **Date of test:** | 30.05.2023 |

Before carrying out pull test 2 preparations were put in place, and these included project area, Rock bolt type and diameter as well as Tunnel size. Rock types in the area of the project were also known.

Pull test 2 was carried out in 3960 level/1150N Loop Loader Drive shown on plan in Figure 4.21

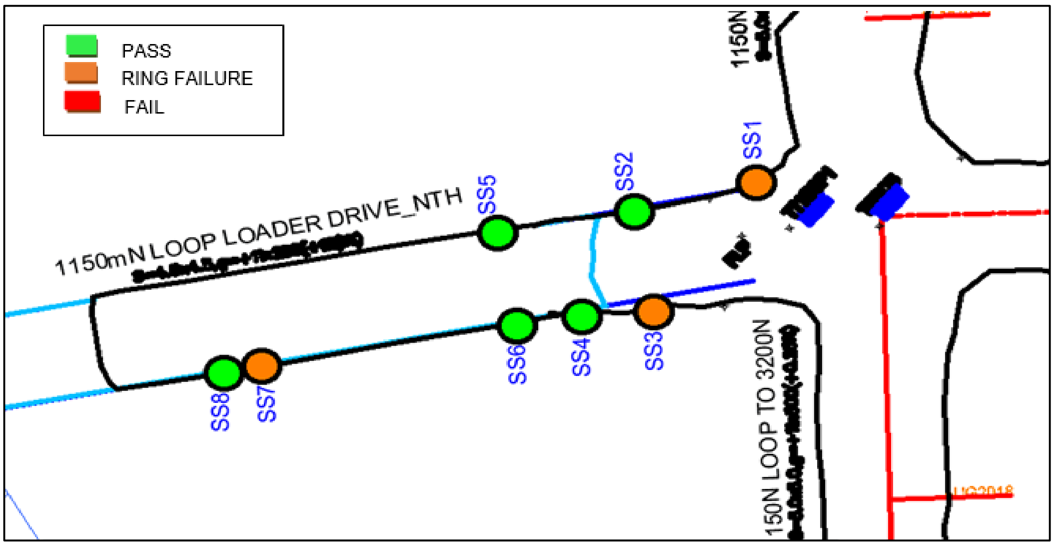



**Figure 4.21: Pull Test 2 Location-3960/1150mN Loop Loader Drive North**

Figure 4.22 shows a graph of Pull Test 3 results conducted at 3510 L/820 XC

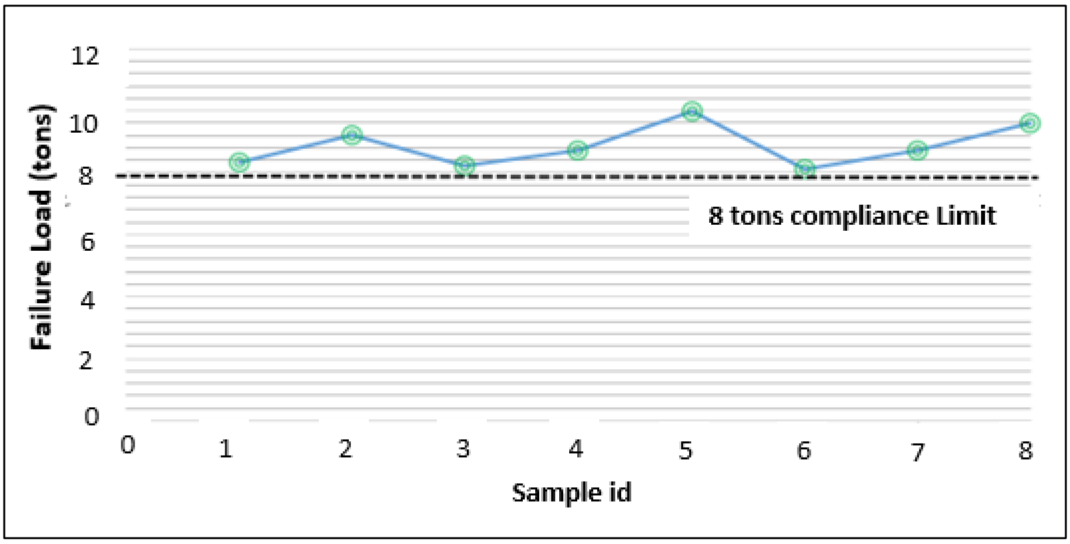



**Figure 4.22: Pull Test 3 Graph**

All the eight pull tests passed, with 3 tests at loads of 8 tons, just on the compliance limit. The rest of the tests returned results above limit (see pull test location plan in Figure 4.23)

Table 4.17 shows pull Test 3 results for the tests conducted on 3510 level - C Interlimb

**Table 4.17: Pull Test 3 Preparation**

| Test No. | Distance from Ref. Peg M6665 | Wall | Bolt Length | Bond Type | Displacement (mm) | *Test to 8 Tons* | | *Remarks* |
|---|---|---|---|---|---|---|---|---|
| | | | | | | PASS | FAIL | |
| PT1 | 41 | South | 2.4m | Friction | None | 8.3 Tons | | Pass |
| PT2 | 37 | South | 2.4m | Friction | None | 9.2 Tons | | Pass |
| PT3 | 34 | South | 2.4m | Friction | None | 8.2 Tons | | Pass |
| PT4 | 31 | South | 2.4m | Friction | None | 8.7 Tons | | Pass |
| PT5 | 23 | South | 2.4m | Friction | None | 10 Tons | | Pass |
| PT6 | 22 | South | 2.4m | Friction | None | 8.1Tons | | Pass |
| PT7 | 15 | North | 2.4m | Friction | None | 8.7 Tons | | Pass |
| PT8 | 14 | South | 2.4m | Friction | None | 9.6 Tons | | Pass |

Pull test 3 was carried out in 3510 level/820XC shown on plan in Figure 4.23

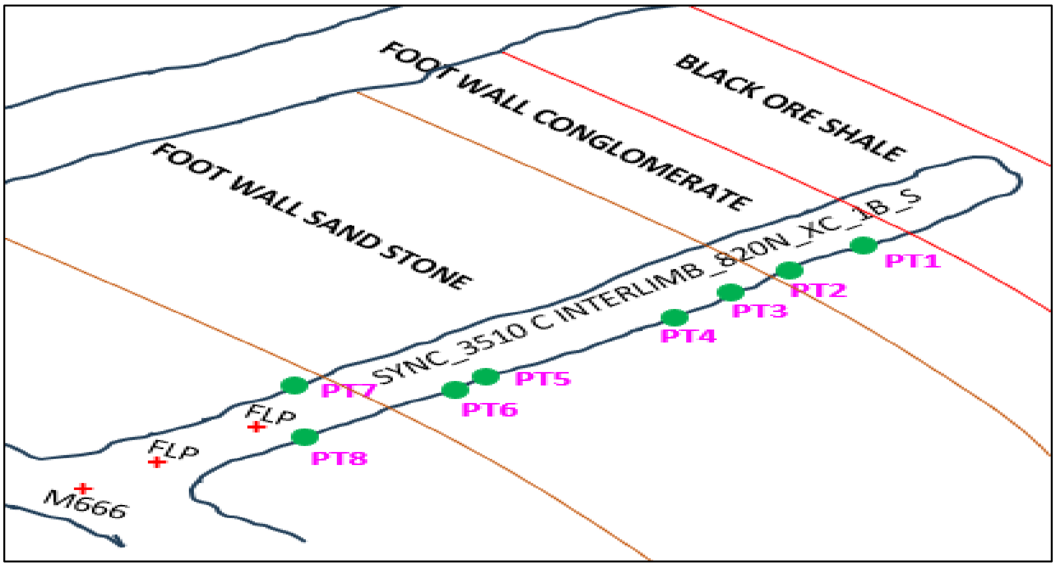



**Figure 4.23: Pull Test3 Location - 3510/820XC – C Interlimb**

In Figure 4.24 a Pull test rig is shown mounted on a ring split set slightly above the white grade line and connected to the gauge which gives the readings of the applied force.

**Figure 4.24: Group Four Minerals Ltd specialists mounting the Pull Test Rig**

### 4.1.5 Compressive Strength Test - Shotcrete Cube

The results in Table 4.18 are a summary of the analysis of 8 concrete cube samples done by the Alfred Knight laboratory. The results show that only two cubes (cube 3 and 4), for Brittatec, have met the required compressive strength for underground structures within the 25 – 40 MPa range. They are all the analyzed concrete cubes for (numbered 1 to 4) Carmine and Frekam fell below 25 MPa. This suggests that the material used for Carmine and Frekam contractors has poor quality and unsuitable for shotcreting underground excavation walls.

**Table 4.18: Results by Standard Compressive Strength**

| CUBE No. | Frekam | Carmine | Brittatec | Required Strength 25 - 40 MPa |
|---|---|---|---|---|
| 1 | 12.2 MPa | 14.0 MPa | 14.4 MPa | - |
| 2 | 16.1 MPa | 14.8 MPa | 14.3 MPa | - |
| 3 | 15.4 MPa | 12.2 MPa | **26.5 MPa** | **Yes** |
| 4 | 19.3 MPa | 14.4 MPa | **25.1 MPa** | **Yes** |

### 4.1.6 Evaluation of the design, installation, and quality control of rock support and reinforcement.

The chart in Figure 4.25 represents the official ground support design process for Synclinorium underground. This has been used for evaluation against the author's suggested design process chart in Figure 4.26

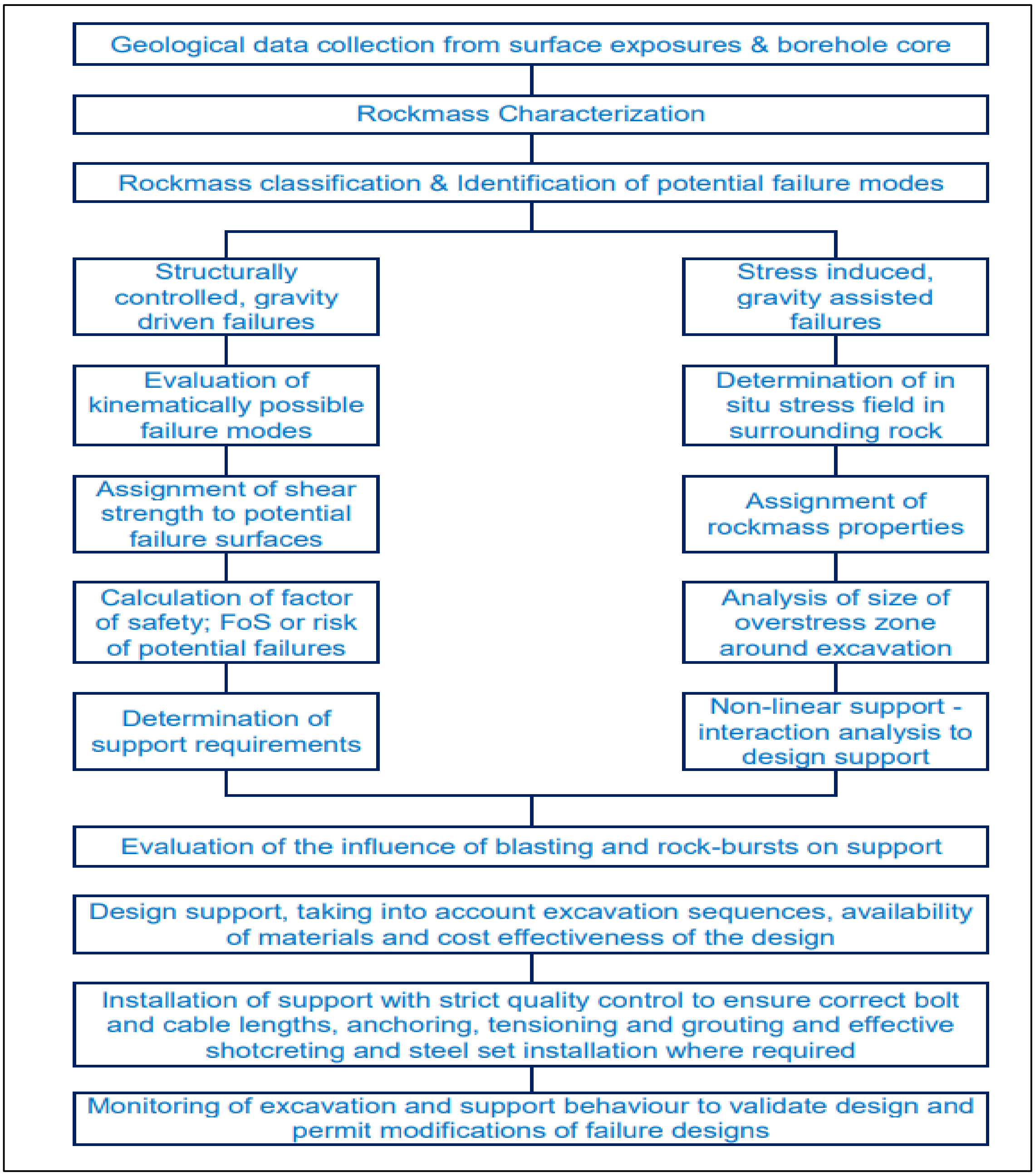



**Figure 4.25: Steps in support design for underground excavations in hard rock**

Figure 4.26 showing the Design process chart selected by the author from literature reviewed after comparisons and evaluations were done with the Synclinorium rock support design chart.

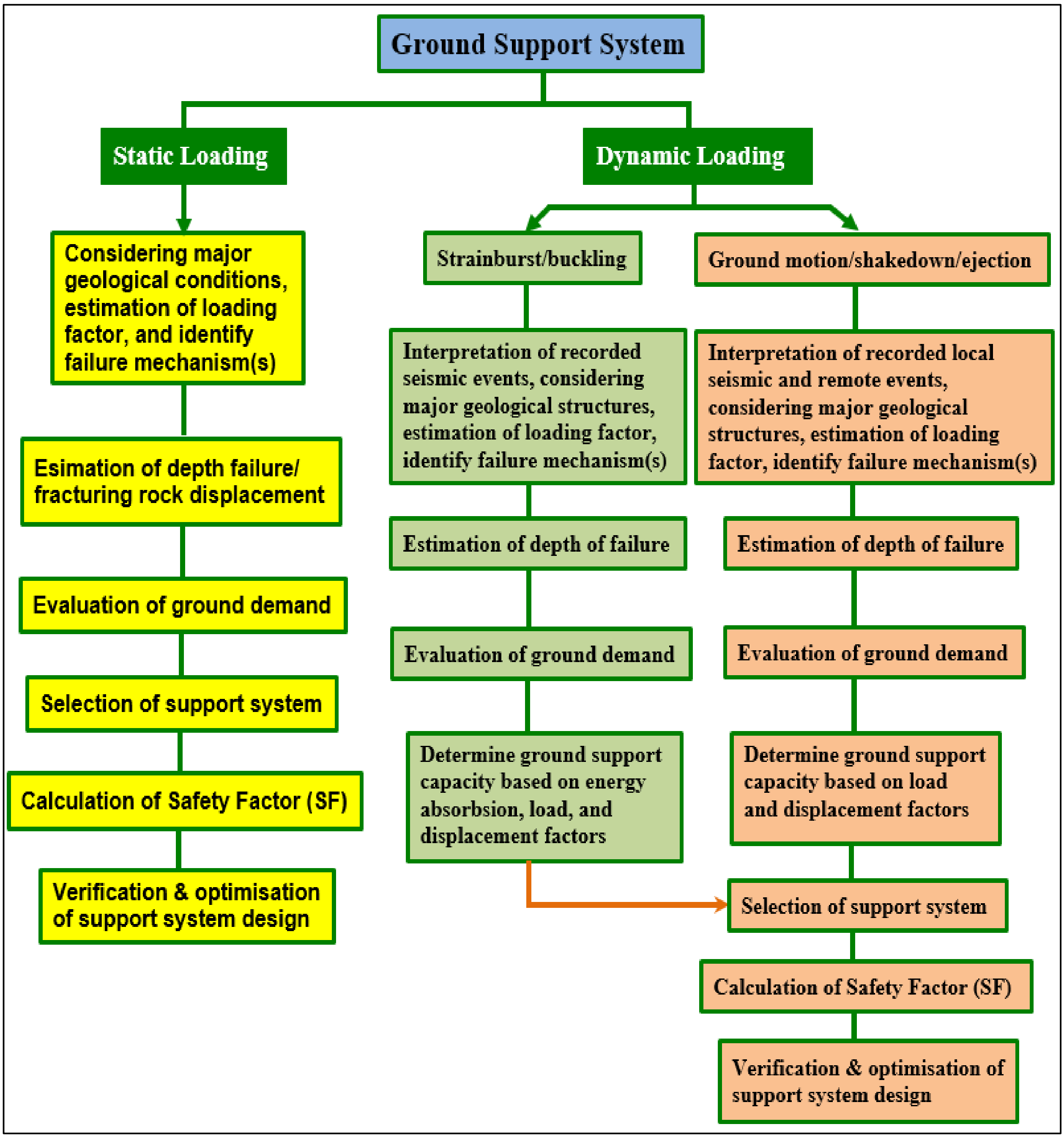



**Figure 4.26: Ground support design in deep underground mines**

Source: B. Rahimi et al. / Journal of Rock Mechanics and Geotechnical Engineering 12 (2020)1-20

### 4.1.7 Evaluation of Rock Support Installations

Figure 4.27 shows the Finite Element Analysis software (FEA) numerical modelling results of an Access Decline in a Gold mine in Australia. The access is of 5.20m width and 5.7m height; Rahim et. Al, 2020.

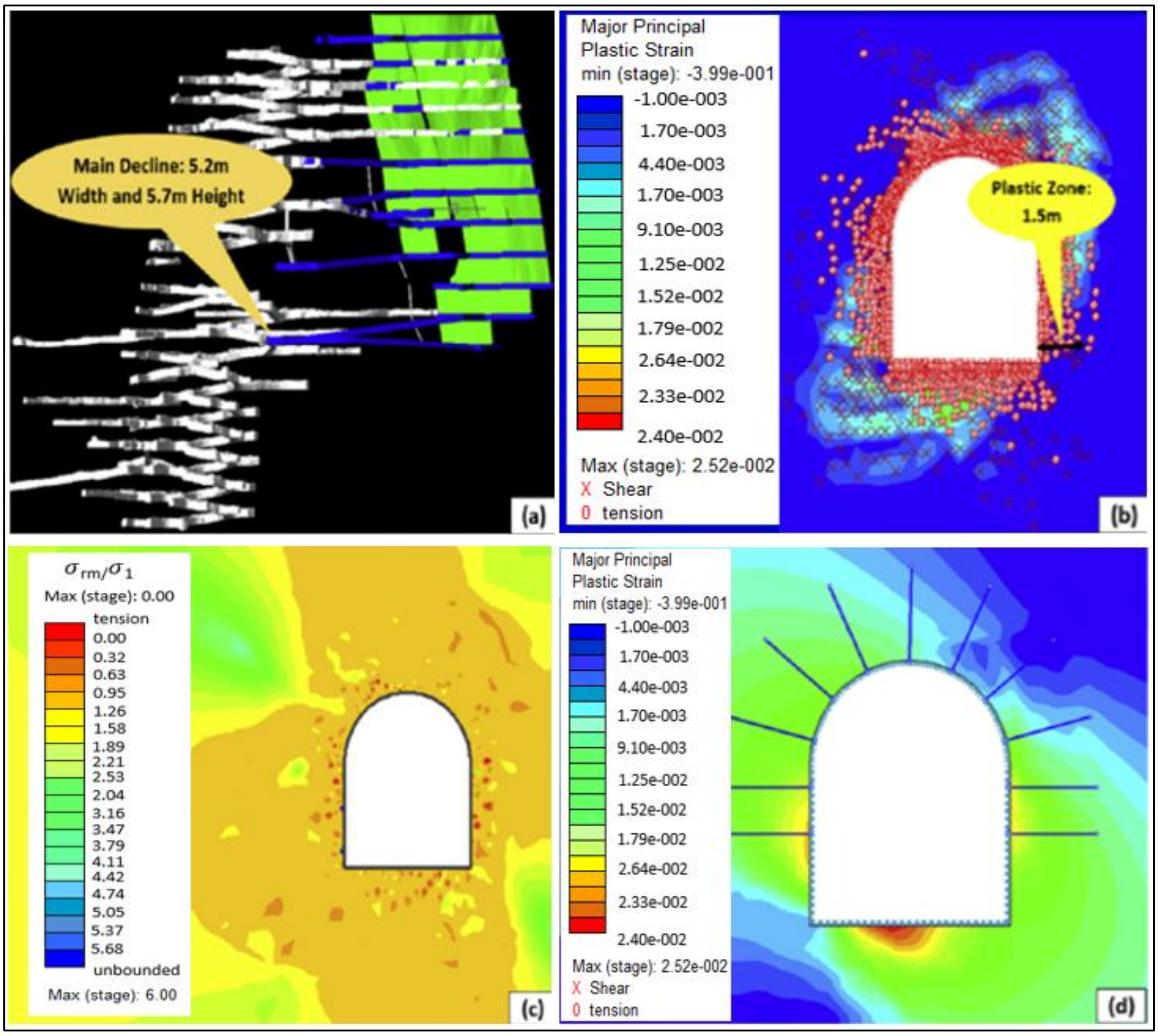



**Figure 4.27**: **Results of the Numerical Modelling of a Main Decline Access**

In Figure 4.27, the modeling involves the solid of the excavation (Decline Access) in (a). Parts (b), and (c) show the stages of numerical modeling before rockbolts are installed. Simulated rock bolt installation has been done in the rockbolt pattern shown in part (d), followed by numerical modeling simulation to compare the results before and after installation of the rockbolts in the rock mass.

In relation to Figure 4.27, simulated rock bolt installation was carried out for a key underground tunnel at Synclinorium mine as depicted by the results in Figure 4.28. To evaluate the effectiveness of rock bolt support on the rock mass surrounding the excavation, pull testing was also done on two of the bolts installed in the excavation A and B at 30˚and 60˚respectively (Figure 4.29). The result for pull test on the rock bolt at 60˚was positive at 8 tons which was the threshold while the 30˚installed rock bolt failed at 6 tons below the threshold of 8 tons.

Figure 4.28 shows contours of vertical displacement (m) for the modeling of 1100N Foot wall drive on 3510 level of Synclinorium mine. The model shows the scenario without rockbolts (left) and with rockbolts (right). The rock bolts have a bolt length of 1.5m and are 0.8m apart. Rock bolts A and B were subjected to pull testing. Finite Element Analysis (FEA) software was also used in Figure 4.28 for simulation in the numerical modeling of 1085 level drive.

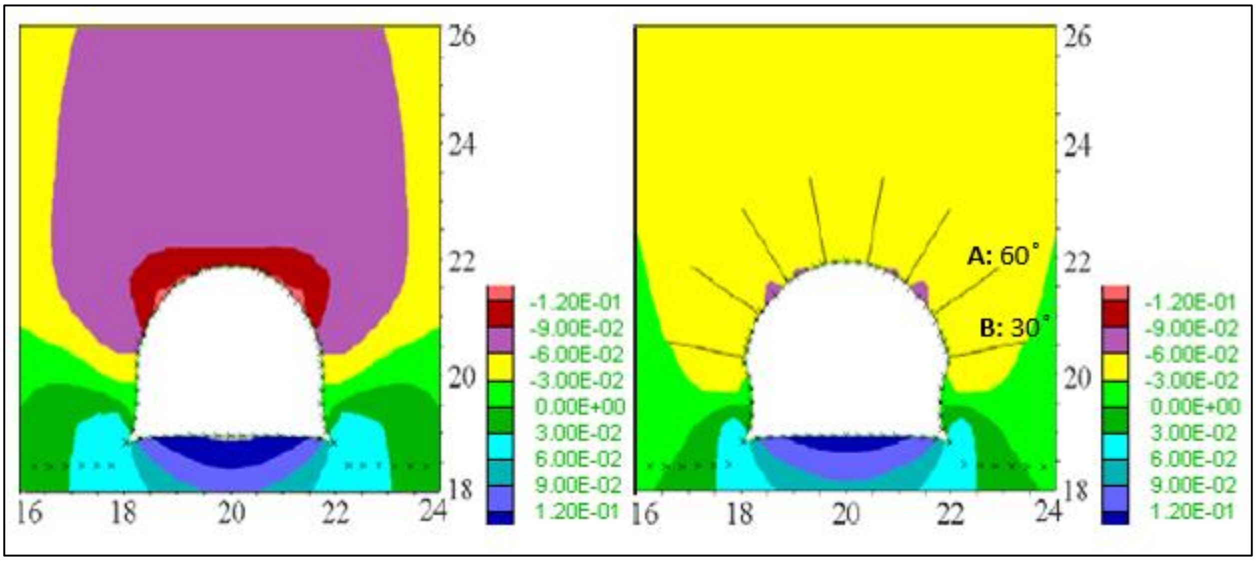



**Figure 4.28: Numerical modeling of 1085L Drive (Synclinorium underground mine)**

When the opening is unsupported (left), the maximum displacements reach 12 cm or 0.12m (1.20E-01 i.e. pink zone). Bolting, when installed (right), reduces the maximum displacements to 9 cm or 0.09m (9.00E-02 i.e. purple zone).

### 4.1.8 Assessment of the adherence of underground developments or excavations to design (size and geometry).

#### 4.1.8.1 Over-break and Under-break

Table 4.19 and 4.20 show mine survey data from Theodolite measurements underground. From the two tables some tunnels or drives like 3585/1005N, 3585/990N, 3585/1100N-LDR, and

N510/1010N-LDR have been picked for comparison of linear meters mined and over-breaks recorded in the two consecutive months of May and June 2023.

The data presented in Tables 4.19 and 4.20 in terms of linear meters measured for development face advance and over-break was also expressed in percentage over-break and under-break respectively.

For the month of May 2023, the percentage over-break averaged 23% while for June 2023 the percentage over-break was 31% on average.

**Table 4.19: Measured meters for May 2023**

MOPANI
An IRH Managed Company

Mopani Copper Mines PLC

SOB SHAFT - NKANA MINE SITE
MINE TECHNICAL SERVICES

OVER-BREAK AND UNDER-BREAK FOR THE MONTH OF MAY 2023

| SHAFT | END MEASURED | DIST MEASURED (m) | LAYOUT SIZE | PLANNED (m) | MINED ($m^3$) | OVERBREAK ($m^3$) | % OVERBREAK | UNDERBREAK ($m^3$) | % UNDERBREAK | TOTAL BREAK (m) | TOTAL BREAK (%) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| SOB | 3585L_1005N_X/C | 11.1 | 4.5X4.5 | 213.1 | 229.9 | 29.4 | 14% | 12.6 | 6% | 42.0 | 20% |
| SOB | 3585L_990N_X/C | 5.8 | 4.5X4.5 | 112.4 | 144.1 | 32.6 | 29% | 0.9 | 1% | 33.5 | 30% |
| SOB | 3585L_970N_X/C | 25.1 | 4.5X4.5 | 483.1 | 611.3 | 132.2 | 27% | 4.0 | 1% | 136.2 | 28% |
| SOB | 3760L_400N_LDR | 8.8 | 4.5X4.5 | 169.0 | 206.8 | 42.9 | 25% | 5.1 | 3% | 48.0 | 28% |
| SOB | 3300L_1210S_X/C | 29.0 | 4.5X4.5 | 558.3 | 650.8 | 119.1 | 21% | 26.7 | 5% | 145.8 | 26% |
| SOB | 3300L_1010S_LDR | 16.0 | 4.5X4.5 | 307.9 | 339.7 | 43.4 | 14% | 11.5 | 4% | 54.9 | 18% |
| SOB | 3510L_1010N_LDR | 38.3 | 4.5X4.5 | 738.9 | 932.3 | 205.8 | 28% | 12.3 | 2% | 218.1 | 30% |
| SOB | 3510L_790N_X/C | 17.6 | 4.5X4.5 | 336.9 | 418.5 | 91.2 | 27% | 9.7 | 3% | 100.9 | 30% |
| SOB | 3585L_1100N_LDR | 8.8 | 4.5X4.5 | 168.9 | 192.8 | 26.2 | 16% | 2.4 | 1% | 28.6 | 17% |
| TOTAL | | 160.5 | | 3088.5 | | 722.8 | 23% | 85.2 | 3% | 808.0 | 26% |

**Table 4.20: Measured meters for June 2023**

MOPANI

Mopani Copper Mines PLC

SOB SHAFT - NKANA MINE SITE
MINE TECHNICAL SERVICES

**OVER-BREAK AND UNDER-BREAK FOR THE MONTH OF JUNE 2023**

| SHAFT | END MEASURED | DIST MEASURED (m) | LAYOUT SIZE | PLANNED (m) | MINED ($m^3$) | OVERBREAK ($m^3$) | % OVERBREAK | UNDERBREAK ($m^3$) | % UNDERBREAK | TOTAL BREAK (m) | TOTAL BREAK (%) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| SOB | 3585L_1005N_X/C | 26.0 | 4.5X4.5 | 501.2 | 622.5 | 142.8 | 28% | 21.5 | 4% | 164.3 | 33% |
| SOB | 3585L_990N_X/C | 14.4 | 4.5X4.5 | 283.6 | 320.7 | 65.8 | 23% | 28.8 | 10% | 94.6 | 33% |
| SOB | 3585L_970N_X/C | 11.7 | 4.5X4.5 | 225.8 | 308.8 | 83.8 | 37% | 0.7 | 0% | 84.5 | 37% |
| SOB | 3760L_400N_LDR | 18.4 | 4.5X4.5 | 355.6 | 493.2 | 140.4 | 39% | 2.8 | 1% | 143.2 | 40% |
| SOB | 3300L_1210S_X/C | 18.8 | 4.5X4.5 | 360.8 | 423.3 | 83.0 | 23% | 20.4 | 6% | 103.4 | 29% |
| SOB | 3300L_1010S_LDR | 31.9 | 4.5X4.5 | 615.3 | 805.7 | 208.3 | 34% | 17.8 | 3% | 226.1 | 37% |
| SOB | 3510L_1010N_LDR | 28.0 | 4.5X4.5 | 540.6 | 693.9 | 155.7 | 29% | 2.4 | 0% | 158.1 | 29% |
| SOB | 3510L_790N_X/C | 10.0 | 4.5X4.5 | 194.6 | 258.1 | 67.0 | 34% | 7.5 | 4% | 74.5 | 38% |
| SOB | 3585L_1100S_LDR | 31.0 | 4.5X4.5 | 596.0 | 799.9 | 207.0 | 35% | 3.2 | 1% | 210.2 | 35% |
| SOB | 3585L_1100N_LDR | 19.9 | 4.5X4.5 | 383.5 | 483.8 | 115.7 | 30% | 15.4 | 4% | 131.1 | 34% |
| TOTAL | | 210.1 | | 4057.0 | | 1269.5 | 31% | 120.5 | 3% | 1390.0 | 26% |

The over-break linear meters for the months of May and June 2023 are presented graphically in Figures 4.29 and 4.30 respectively.

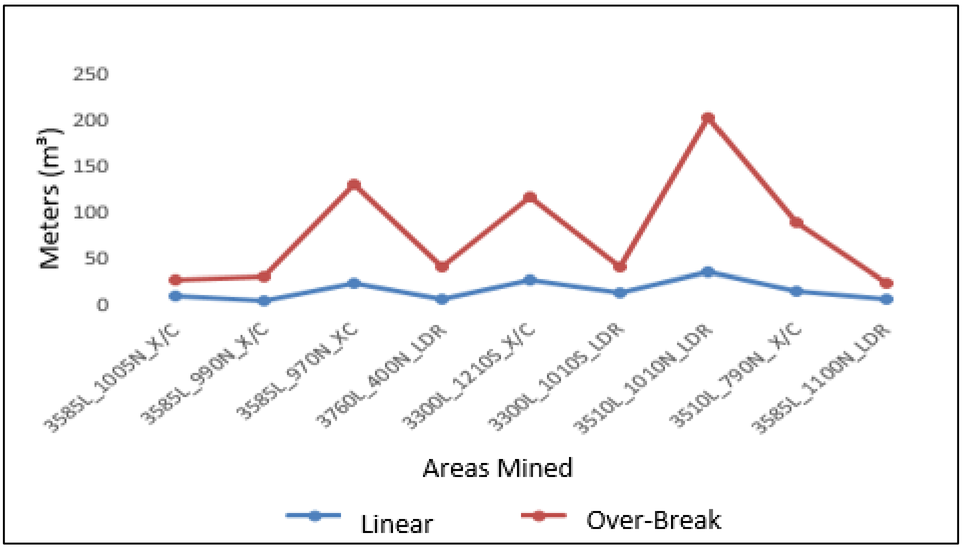



**Figure 4.29: Over-Break and Linear meters in areas mined (May 2023)**

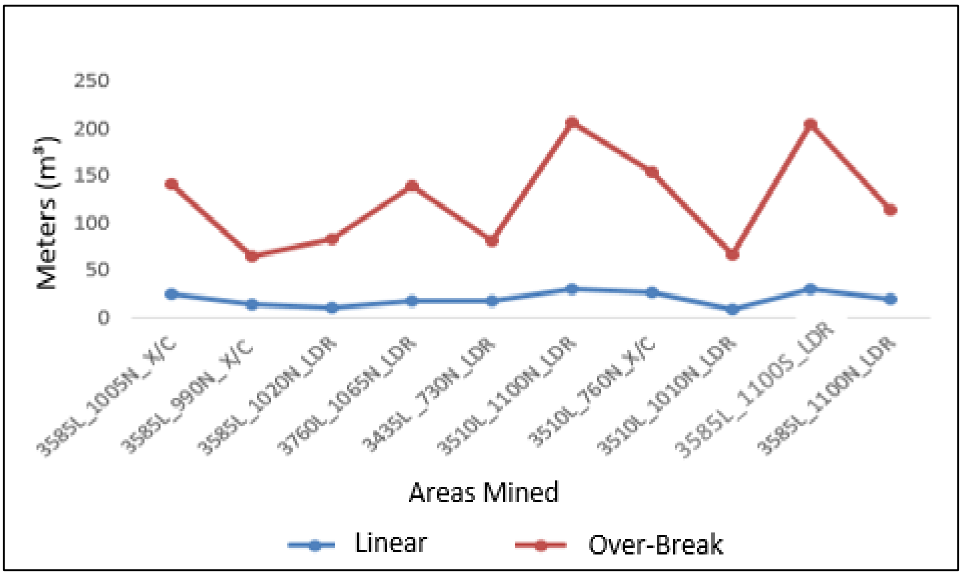



**Figure 4.30: Over-Break and Linear meters in areas mined (June 2023)**

In the two months of development, the following comparisons have been made between the tunnels mined:

3. For excavations mined in May 2023 as shown in Table 4.19 and Figure 4.29, the highest over-break of 205.8m³ was recorded against 3510L_1010N_LDR in May with more linear meters mined of 38.3m, while the lowest over-break of 26.2m was recorded against 3585L_1100N_LDR in the same month with less linear meters of 8.8m mined.

4. In Table 4.20 for May 2023, 3760L_400N_XC which had an over-break of 42.9m³ in 8.8m advance was compared with 3585L_1100N_LDR which had 26.2m³ in 8.8m advance. The linear advances in meters are the same, but there is a huge difference in over-break of 16.5m³.

5. Looking at Table 4.20 and Figure 4.30 for June 2023, the highest over-break of 208.3m³ was recorded against 3300L_1010S_LDR, with mined linear meters of 31.9m in June. In May 2023, the same development drive recorded an over-break of 43.4m³ with mined linear meters of 16.0m.

6. Again for June 2023 in Table 4.30, 3585L_990N_XC had an over-break of 65.8m³ in a linear advance of 14.4m while 3510L_790N_LDR had 67.0m³ over-break in only 10.0m advance.

From the analysis above there is no correlation between over-break volume and the linear meters mined in each drive.

The high over break is attributed:

1. More exposure to geological factors such weaker ground characterized by fissure zones, joints and less weathering over the longer stretch of linear development of the drive.

2. Drill-and blast challenges such as:

   - Hole deviation, spacing and diameter
   - Poor hole cleaning
   - Explosive characteristics and consumption

Table 4.21 and Figure 4.31 showing over-break and Perimeter Power Factor (PPF) in correlation: As PPF increases, the over-break also increases and vice versa.

**Table 4.21: Over-Break and Perimeter Power Factors (Jan 2023 – August 2023)**

MOPANI

Mopani Copper Mines PLC

SOB SHAFT - NKANA MINE SITE
MINE TECHNICAL SERVICES

Over-Break and Perimeter Power Factor for 2023

| MONTH | JAN | FEB | MAR | APR | MAY | JUN | JUL | AUG |
|---|---|---|---|---|---|---|---|---|
| Kg/TON | 1.80 | 1.66 | 1.69 | 1.59 | 1.55 | 1.68 | 1.72 | 1.75 |
| % O/ BREAK | 33 | 22 | 25 | 23 | 23 | 31 | 30 | 31 |

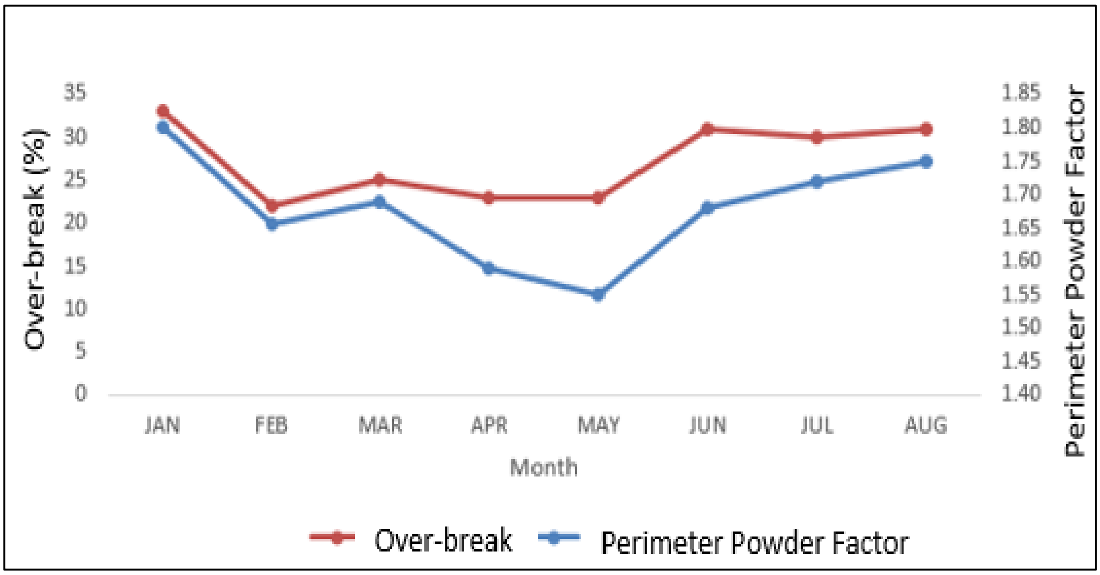



**Figure 4.31: Over-Break and Perimeter Power Factor Comparison**

### 4.1.9 VCR Pillar Stability Assessment

A template in Table 4.22 is used to input parameters such rock mass rating, rock strength, and pillar dimensions to calculate pillar strength, pillar stress and safety factors.

The input parameters will be entered in white back ground and the results will appear in light blue back ground.

In Table 4.22 an example given is for 790 VCR stope on 3960 level of synclinorium mine.

**Table 4.22: Design of Rib Pillar in inclined Orebody-Analytical Method**

| | | | | |
|---|---|---|---|---|
| A | 62 | A = Total Rock Mass Rating | | |
| B | 13 | B = I.R.S. Rating | | |
| C | 75 | C = Intact Rock Strength | | |
| W | 8 | Pillar Width | | |
| H | 50 | Pillar Height | | |
| | | | | |
| RMS | 36.75 | RMS = ((A - B)/80) x C x (80/100) | | |
| | | | | |
| $W^{0.46}$ | 2.602684 | | | |
| | | | | |
| $H^{0.66}$ | 13.2227 | | | |
| | | | | |
| Pillar Strength | 7.233667 | Pillar Strength (Capacity), $\sigma_c$ = RMS x ($W^{0.46}/H^{0.66}$) | | |
| | | | | |
| h | 988 | Depth in meters (m). | | |
| k | 0.7 | k ratio, ($k=\sigma_h/\sigma_v$) | | |
| $w_o$ | 25 | Width of Opening | | |
| $w_p$ | 8 | Width of Pillar | | |
| $\alpha$ | 90 | Stope Inclination or Orebody Dip | Alpha | |
| | | | | |
| $\sigma_v$ | 26.676 | Virgin stress at depth ( use stress gradient 0.027MPa/m of depth) | sigv | |
| $\sigma_h$ | 18.6732 | Hor. Stress ( $k=\sigma_h/\sigma_v$) | sigh | Hor. Stress ( k= sigh/sigv) |
| e | 0.757576 | Extraction ratio e = ( $w_o/(w_o+w_p)$) | | |
| | | | | |
| Pillar Stress | 83.65468 | Pillar Stress (Demand), $\sigma_d = (\sigma v Cos^2\alpha + \sigma h Sin^2\alpha)/1-e$ | | |
| | | | | |
| Safety Factor | 0.086471 | Safety Factor, SF = Pillar Strength $\sigma_c$ / Pillar Stress $\sigma_d$ | | |
| | | | | |
| NOTE: | INPUTS TO BE IN WHITE BACKGROUND | | | |

The explanation of all the steps taken in this design is given in the discussion of results in chapter 2 under methodology.

σv = 26.676, is the virgin stress at depth in MPa (using stress gradient of 0.027MPa/m) σh = 18.6732, is the horizontal stress.

The relation between Virgin stress and Horizontal stress is expressed as K = σh/σv As a function of the poisson's ratio, K = σh/σv= V/(1-V), where V is the poisson's ratio. For a typical poisson's ratio of 0.3, the K ratio used in calculation of horizontal stress is 0.43 Therefore, σh= K σv= 0.43 xσv

Selected VCR stopes and associated pillars in the mine were evaluated. Rock samples for key rock formations of the pillars were analyzed for rock parameters. The solids of the pillars designed in Surpac software were subjected to modeling using Map3D software (Figure 6.4 and 6.5).

Figures 4.32 and 4.33 showing selected VCR pillars on 3360 level of Synclinorium mine on plan and section which were subjected to stability assessment:

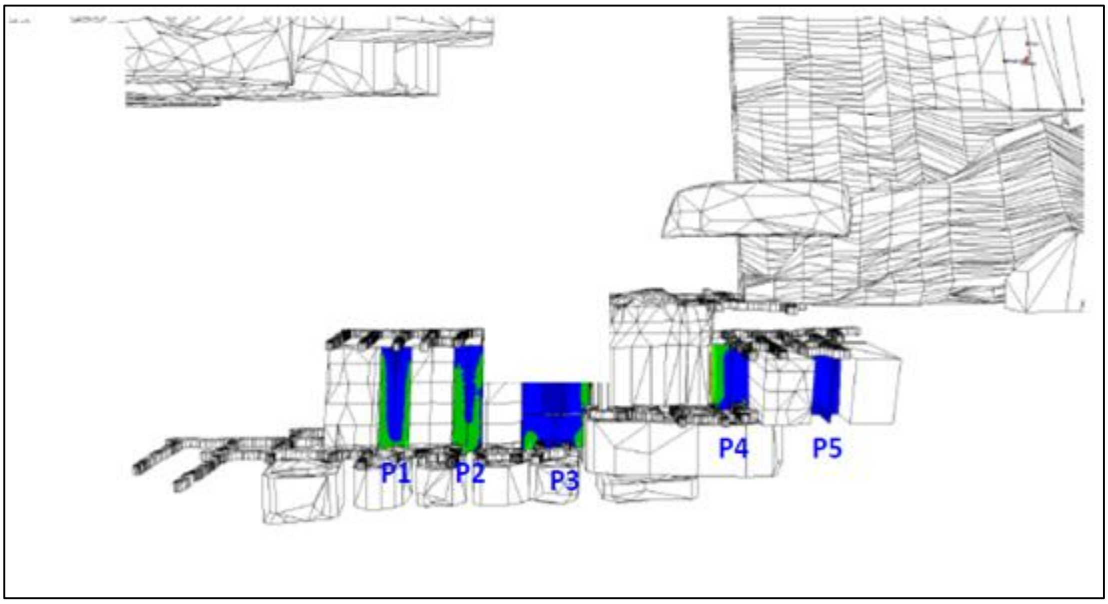



**Figure 4.32: VCR Model Geometry in 3D**

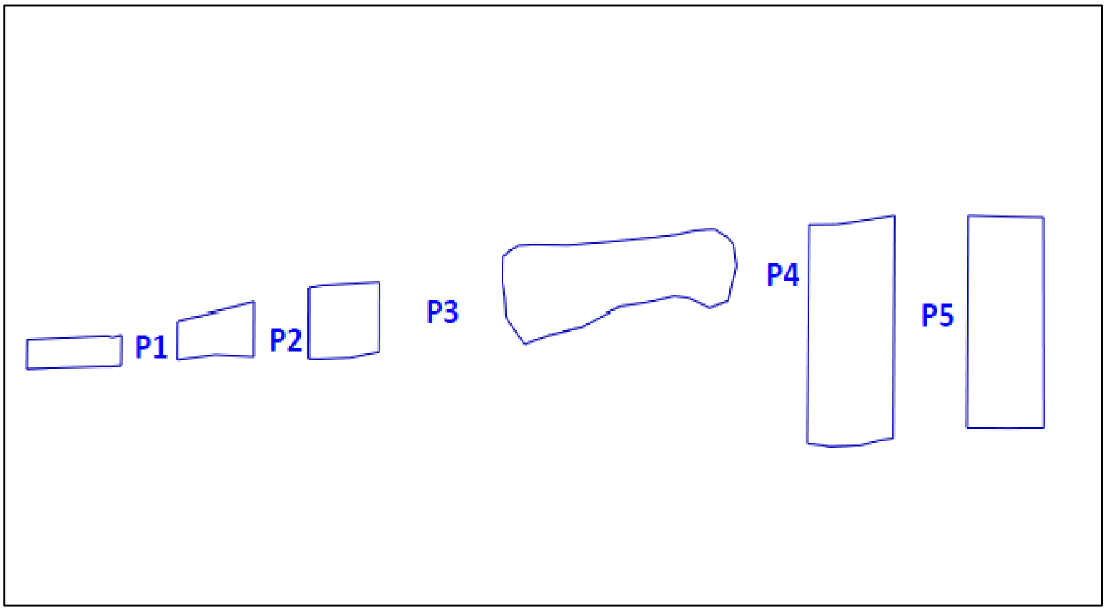
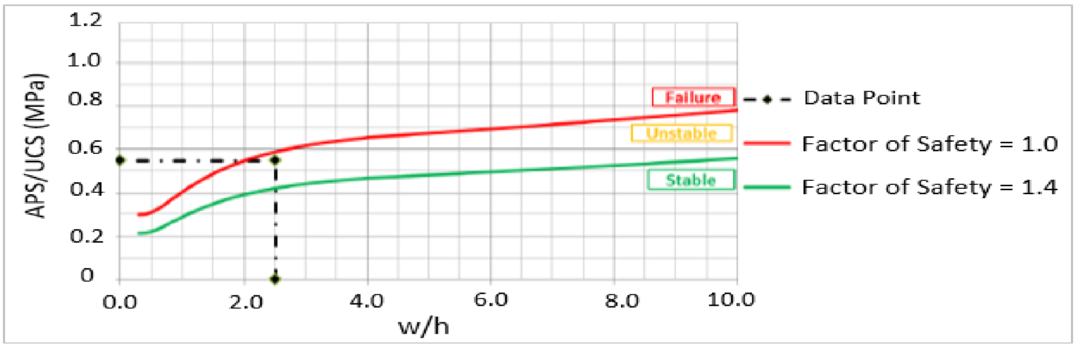



**Figure 4.33: VCR Model Geometry in Plan**

Figure 4.35 shows the Confinement Formular Stability Graph used to assess the stability of pillar1. The green line indicates the stability limit below which a pillar is deemed stable, and above that is a zone of instability. Over and above the red line a pillar is considered to fail. This was applicable to the assessment of rest of the pillars as shown in appendix D:




| | | | |
|---|---|---|---|
| INPUT | h | Pillar Height (m) | 10.20 |
| | w | Pillar Width (m) | 16.60 |
| | l | Pillar Length (m) | 56.70 |
| | APS | Average Pillar Stress (Mpa) | 42.20 |
| | UCS | Intact Uniaxial Compressive | 77.00 |
| OUTPUT | $W_{eff}$ | Effective Pillar Width (m) | 25.68 |
| | w/h | Width to Height ratio | 2.52 |
| | APS/UCS | Average Pillar Stress/UCS | 0.55 |
| | $CP_{av}$ | Average Pillar Confinement | 0.23 |

**Lunder & Pakalnis Confinement Formula**

**Figure 4.35: Pillar 1 Stability Assessment-Confined Formula Graph**

The Confinement Formula Stability graph uses input of geotechnical parameters shown in Figure 4.35 to estimate the factors of safety. The factor of safety for pillar1falls between 1 and 4. The different rock types which generally make up a typical pillar at synclinorium mine had samples tested at the laboratory and results of the associated parameters are shown in Table 4.23

**Table 4.23: Pillar Strength-Values used in confinement formula**.

**Laboratory UCS Results and 25% and 75% Percentile Values of UCS**

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

Notes: * correspond to assumed values based on the trend observed in the graphs.

Note: VCR Pillars (both crown and Rib) form part of the SOBS ore formation. Having determined the Pillar1 range of factors of safety from Figure 4.35, they are plotted on the Frequency Distribution Graph as shown in Figure 4.36

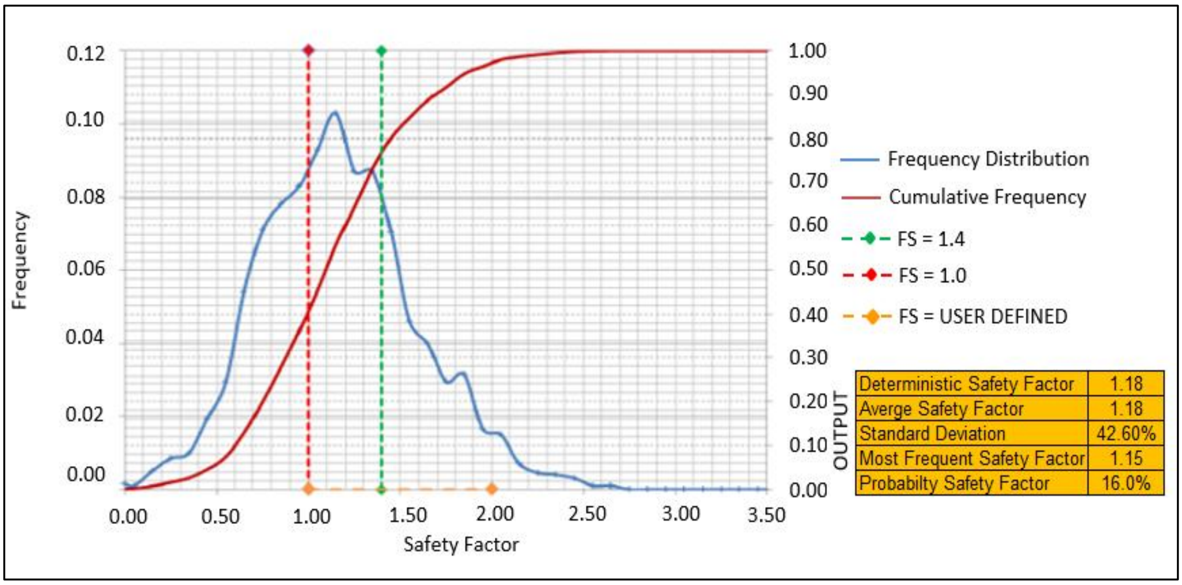



**Figure 4.36: Pillar 1 Factors of Safety Frequency Distribution**

The distribution of Factors of Safety is relatively normal, with the peak falling within the 1.0 and 1.4 range.

## 4.2 In-situ Pillar monitoring to improve the empirical equations

(Pillar Design Methodology)

Figure 4.37 shows what is currently in use as a design flow chart or guideline for pillar design at Synclinorium mine.

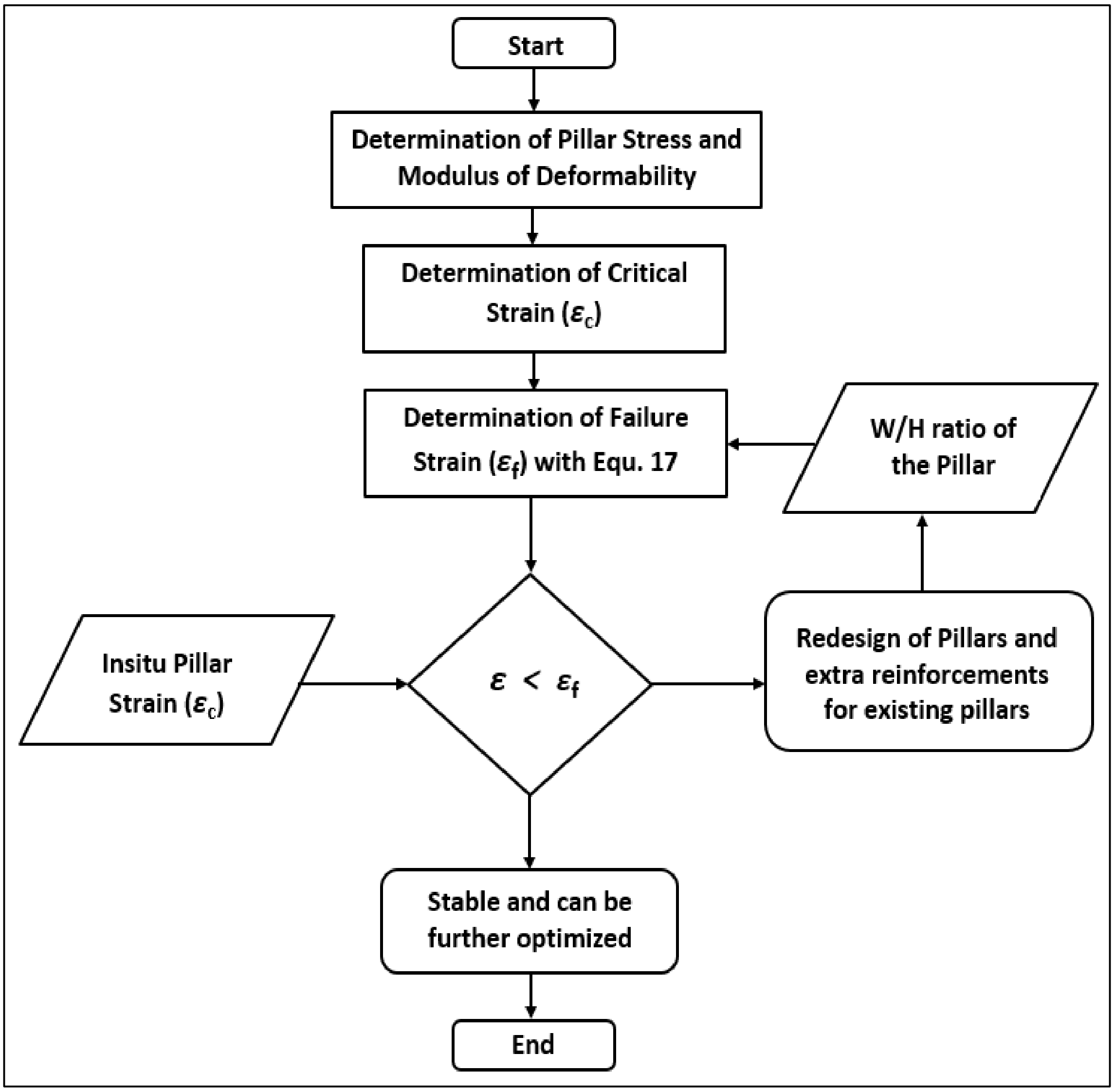



**Figure 4.37: Flow Chart for applicability of the critical strain for the pillar design**

This is what is currently in use as a design flow chart or guideline for pillar design at Synclinorium mine.

### 4.3 Proposal for the improved pillar design methodology

The design methodology in current use at Synclinorium (Figure 4.37) does not incorporate the depth at which the pillar is being designed and also the inclination of the orebody compared to improved design methodology in Figure 4.38. Depth and orebody inclination are also critical parameters to consider in the design because they have an impact on the stability of pillars underground. Mining at Synclinorium has moved from upper levels above 3360 feet to lower levels up to 3960 feet with increasing mining induced stresses, and the same methodology being applied in the design of pillars.

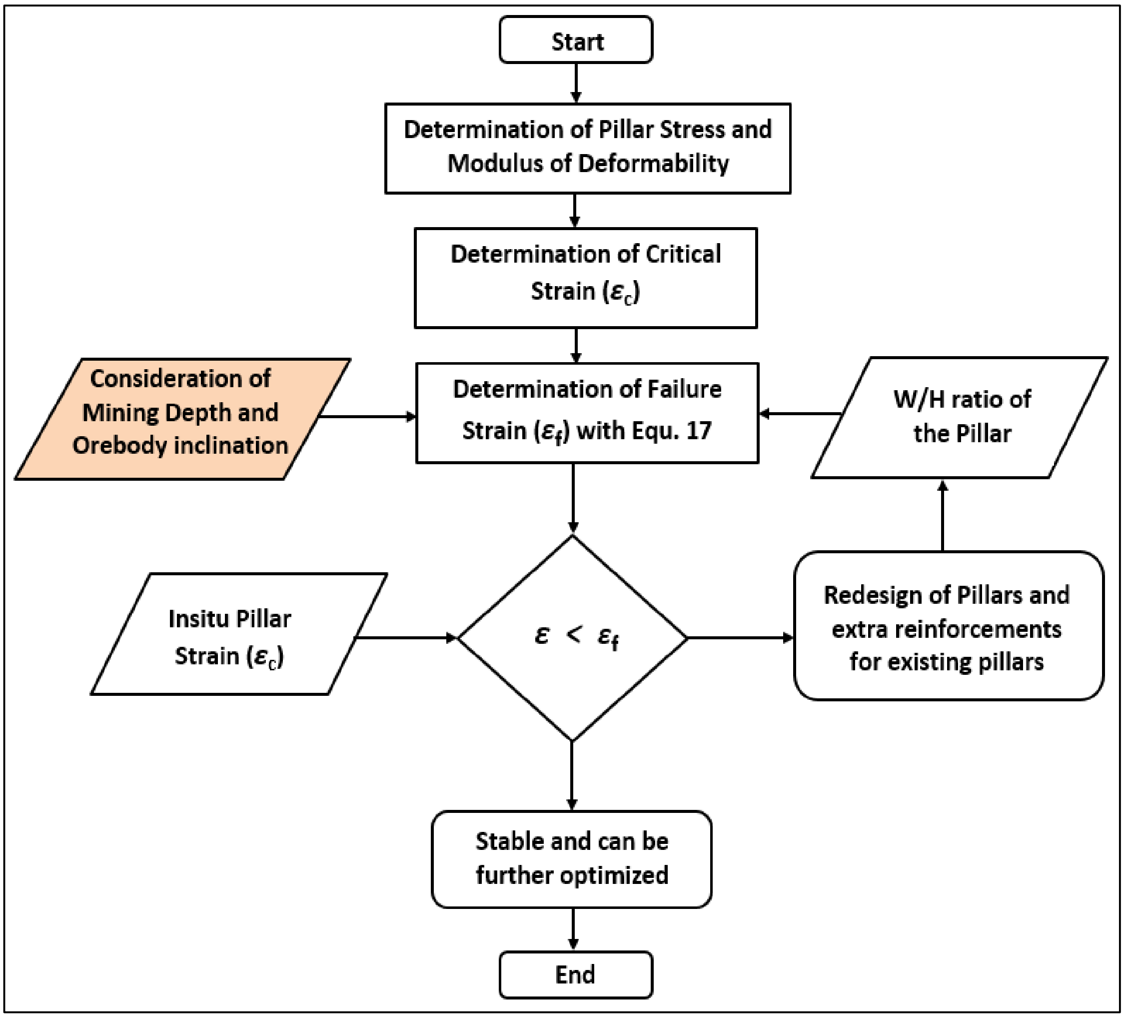



**Figure 4.38: Improved pillar design methodology**

In Figure 4.38, this author has suggested incorporation of mining depth and orebody inclination (represented by yellow part) to improve the Pillar design methodology currently being used at Synclinorium mine at Mopani Nkana.

## 4.4 Evaluation of Stability for 16 VCRs on other levels

The stability of sixteen VCR stopes across synclinorium was evaluated by this research in terms of pillar-height ratio and ore extraction in Table 4.24

**Table 4.24: Pillar parameters used for evaluation**

| Pillar | *l* (m) | *w* (m) | Hp (m) | Weff (m) | Weff:H | *h* (m) | $W_O$ | $W_O + W_P$ | $\sigma_v$ | $\sigma_h$ | e Ratio |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 3435L/1300 VCR | 35.0 | 8.0 | 27 | 13.02 | 0.48 | 1047 | 20 | 33 | 28 | 12 | 0.6 |
| 3435L/1330 VCR | 32.4 | 7.3 | 34 | 11.92 | 0.35 | 1047 | 20 | 32 | 28 | 12 | 0.6 |
| 3585L/445 VCR | 32.3 | 7.6 | 55 | 12.30 | 0.22 | 1093 | 15 | 27 | 30 | 13 | 0.5 |
| 3585L/410 VCR | 32.6 | 9.2 | 56 | 14.35 | 0.26 | 1093 | 18 | 32 | 30 | 13 | 0.6 |
| 3585L/400 VCR | 48.0 | 8.0 | 48 | 13.71 | 0.29 | 1093 | 13 | 27 | 30 | 13 | 0.5 |
| 3585L/473 VCR | 31.0 | 8.4 | 53 | 13.22 | 0.25 | 1093 | 15 | 28 | 30 | 13 | 0.5 |
| 3585L/1015 VCR | 44.0 | 7.0 | 47 | 12.08 | 0.26 | 1093 | 12 | 24 | 30 | 13 | 0.5 |
| 3760L/900 VCR | 40.0 | 7.5 | 43 | 12.63 | 0.29 | 1146 | 10 | 23 | 31 | 13 | 0.4 |
| 3760L/940 VCR | 55.0 | 8.0 | 43 | 13.97 | 0.33 | 1146 | 13 | 27 | 31 | 13 | 0.5 |
| 3760L/975 VCR | 52.0 | 8.0 | 38 | 13.87 | 0.36 | 1146 | 25 | 39 | 31 | 13 | 0.6 |
| 3760L/1000 VCR | 60.0 | 7.3 | 47 | 13.02 | 0.28 | 1146 | 42 | 55 | 31 | 13 | 0.8 |
| 3760L/640 VCR | 54.0 | 7.7 | 31 | 13.48 | 0.44 | 1146 | 40 | 53 | 31 | 13 | 0.7 |
| 3760L/675 VCR | 52.0 | 8.4 | 30 | 14.46 | 0.48 | 1146 | 32 | 46 | 31 | 13 | 0.7 |
| 3760L/710 VCR | 50.0 | 9.3 | 29 | 15.68 | 0.54 | 1146 | 31 | 47 | 31 | 13 | 0.7 |
| 3760L/810 VCR | 56.0 | 9.0 | 29 | 15.51 | 0.53 | 1146 | 26 | 42 | 31 | 13 | 0.6 |
| 3760L/790 VCR | 49.0 | 8.3 | 27 | 14.20 | 0.52 | 1146 | 33 | 47 | 31 | 13 | 0.7 |

Extraction ratio:

$$e = \frac{W_O}{W_O + W_{eff}}$$

The above equation is related to Figure 4.39

Where **$W_o$** Stope opening width in meters, and **Weff** is the pillar with in meters;

The various parameters used in the assessment of the VCRs are explained as follows:

**l** is the Pillar length in meters;

**w** is the initial pillar width in meters;

**Weff** is the effective width in meters i.e. Weff = 4A/P

Where A is the area of the pillars in square meters, and P is the perimeter of the pillar.

**Hp** is the pillar height is meters;

**h** is depth in the mine at which the VCR is situated;

**Weff:** H is the effective width-to-pillar height ratio in meters;

In plan:

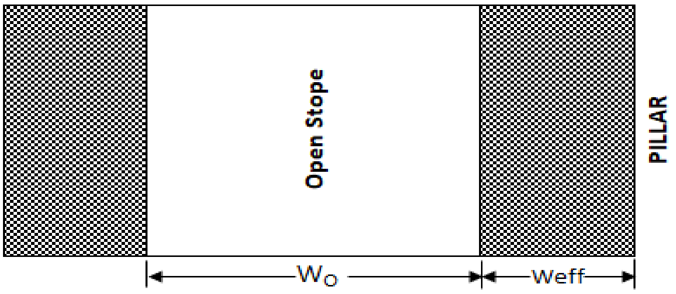



**Figure 4.39: Illustration of Extraction ratio calculation in a VCR**

The relation between Virgin stress and Horizontal stress is expressed as $K = \sigma_h/\sigma_v$

As a function of the poisson's ratio, $K = \sigma_h/\sigma_v = V/(1\text{-}V)$, where V is the poisson's ratio.

For a typical poisson's ratio of 0.3, the K ratio used in calculation of horizontal stress is 0.43

Therefore, $\boldsymbol{\sigma_h} = K\ \sigma_v = 0.43 \times \sigma_v$

$\boldsymbol{\sigma_v}$ is the virgin stress at depth in **MPa** (using stress gradient of 0.027MPa/m)

$\boldsymbol{\sigma_h}$ is the horizontal stress.

Based on data in Table 4.24, evaluation of the VCR stopes in terms of the relationship between extraction ratio and the width-to-height ratio was done and plotted in Figure 4.40:

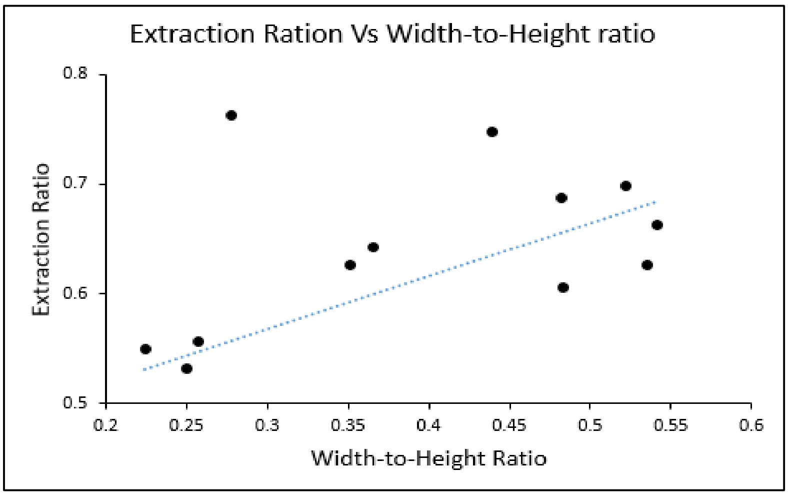



**Figure 4.40: Relationship between Extraction Ratio and Width-to-height Ratio**

It can be noted from the graph in Figure 4.40 that ore extraction increases with the width-to-height ratio. This means that extraction increases with stope height.

# CHAPTER 5 – Discussion of Results

## 5.1 Introduction

This chapter focuses on the results and analysis of the research work carried out in line with the main objective, and the sub-objectives in detail. All the results presented in chapter 4 are discussed under each sub-objective of this thesis.

## 5.2 Assessment of Ground Control Mechanisms and Support Standards for stability

### 5.2.1 Rock mass properties

Rock mass parameters based on Barton's Q system have been officially adopted by Nkana Synclinorium mine for underground evaluation. These were used in empirical stability graphs for underground excavation design (Table 4.1).

Ideally, sound geotechnical practice requires that the geotechnical database is updated regularly with new geological/geotechnical information for characterization of the rock mass in new areas being opened by mine-development. Relying on engineering parameters which are not updated may make the data analysis and determination of quantitative geometry of the rock mass inaccurate, and even affect support design.

In light of the inadequate and irregular update of the geotechnical database, this research selected some recently drilled holes for geotechnical core logging (see Appendix A) to test the continued application of the rock mass parameters based on Barton's Q system for use in empirical stability graphs in excavation design.

Six parameters below were used to classify the rock mass using the RMR system:

1. Uniaxial compressive strength of rock material.
2. Rock Quality Designation (RQD).
3. Spacing of discontinuities.
4. Condition of discontinuities.
5. Groundwater conditions.
6. Orientation of discontinuities.

The rock mass classification was done to assess the estimates of underground tunnel support (raw data in Appendix A).

A summarized version of the rock mass ratings for the rock types with selected key parameters is given in Table 4.2 of Chapter 4 after geotechnical logging and rock strength determination. Detailed ratings for individual rock formations studied and analyzed for RMRs are outlined in the Tables 4.4 - 4.8. From the Tables, the following points have been observed:

3. Hanging Wall Argillite is a well laminated and generally slightly weathered rock type, but has a better rock strength than Shale at 65 MPa and total RMR of 70. Since it overlies the orebody on the hanging wall side it helps stabilize the roof of most ore stopes and contribute to a longer standing time with its characteristics.

4. The Sandstone underlies most of the host rock for copper mineralization on the foot wall side. It has a fine grain size and slightly weathered in general. With the rock strength at 116 MPa and RMR of 78, sandstone has a competent rock mass. However, being part of a highly stressed and folded zone, sandstone is also characterized by discontinuities like joints and bedding planes. Excavations mined through sandstone are mostly part of secondary development near the orebody. These require urgent and intensive roof support to stabilize the surrounding rock mass for improved standing time.

5. Another rock unit which meets the parameters of rock stability is foot wall conglomerate with strength value of 105 MPa, and RMR of 65.

6. Basal Quartzite (BQ) is the strongest rock type of all the listed rocks with 178 MPa as indicated in Table 4.8. It is rated the highest at RMR 82 and is almost able to stand on its own for very long periods with minimum support. 90% of the BQ rock mass is found in primary developments like haulages and water drain drives which are permanent excavations.

7. The samples in this study were classified according to the standard UCS (ISRM, 1979) as presented in Table 4.9. According to this classification, the rock types of the area of study are classified into three classes of Medium Strong (MS), Strong (S), and Very Strong strength (VS). In assessing the effect of each rock type and associated discontinuities on ground stability underground, the description of rock strength based on the point load index test and it's equivalent UCS value is given.

A comparison subsequently done between RMRs (Table 4.2) for the core samples used in this study and Barton's Q system RMR ranges is demonstrated in Table 4.3

From the comparison in Table 4.3 it is clear that for the logged samples, only the RMRs of SOB and HWA are within range of Barton's Q system. For BQ, FCON, and FSAN the RMRs are out of range. This calls for the need to continually update Synclinorium mine rock mass parameters through regular data collection from underground mapping, laboratory tests, and core-logging.

### 5.2.2 Correlation Analysis of Rock parameters

The factors, which mainly influence the stability of a typical underground excavation are the rock mechanical properties and characteristics of structural discontinuities present in the surrounding rock mass.

(i.) In Figure 4.1 an almost linear relationship was discovered between UCS and GSI. In general, the strength of the rock (UCS or GSI) increased with the physical quality of the rock type.

(ii.) There are also potential correlations in the following pairs of comparisons:

1. Rock Strength (UCS) and RQD/Joint Spacing in Figure 4.2; and
2. Rock Strength (UCS) and Rock Density in Figure 4.3

### 5.2.3 Orientation Analysis of Mapped joints

The observations were made in each of the two mapped levels of Synclinorium mine 3510 and 3960 Levels are presented separately in Figures 4.6 and 4.7 respectively. Both of them are also shown on section in Figure 4.4

In Figure 4.7, the resulting contours show data density patterns generally representing eastern and north-eastern dipping joint sets. The mapping carried out has shown biased data which produce clusters or patterns that do not reflect reality due to inaccessibility in other key areas.

A well accessed, covered, and mapped folded area of Figure 4.7 could have resulted in contoured stereonets with more accurate about joint set orientation aligned with the limbs and nature of the folds (whether cylindrical or non-cylindrical fold form, or symmetrical or asymmetrical fold form) as demonstrated in an idealized stereonet shown in Figure 4.8

Note the great circle girdle pattern in pink is evident in Figure 4.8. The pink line is a hand-picked estimate of the equivalent girdle and fold axis. These are relatively upright and tight folds, with sub-horizontal axes which would match 3510 level folds under study if the whole study area was equally covered without any bias.

The pink great circle in Figure 4.8 is a computer best-fit solution generated by the Dips program. It can be noted that two limbs are evident in the two bullseye, and that the one limb is better represented and thus the folds may be asymmetric. This is what would be expected in the 3510 level folded area in Figure 4.5

In Figure 4.9, the two cross cuts 750 XCUT1 and 710 XCUT1 were preferred for analysis on the basis that they have a higher density of joints and fall in the middle of the study area thereby representing the base of the Synclinorium structure. Stereonets in Figure 4.6 and 4.7 show density patterns representing joint sets which generally have a south-west dip direction as most of the great circles of the joint planes face in the named direction. Looking at the joints raw data for 750 XCUT1 and 710XCUT1 in Table 4.10, the average dip of the joints these mapped cross cuts is 67˚and 65˚respectively. Note that:

1. poles shown in the density patterns are close to the stereonet perimeter and further away from the center;
2. conversely the great circles representing the joint planes are closer to the center of the stereonet where dip is about 90˚.

In underground tunneling projects it is very important to consider the dip and dip orientation in designing tunnel supporting systems and tunnel direction through the rock mass. The orientation of joints in excavation is affected by the dip and dip direction of the rock formation and its relatively favourable or unfavourable effect on the rock mass. A commonsense check is to note the relative potential movement of the rock mass into or out of the excavation because of the planes of weakness created by joints. These adjustments are important, since the way the rock mass is situated in relationship to the direction of the excavation will ultimately determine its stability.

The orientation of joints in different directions can result in a blocky rock mass thereby increasing the chances of ground failure in an excavation.

### 5.2.4 Evaluation of Ground Support Standards at Synclinorium mine

Ground support standards for the development and production headings have been created for developments in waste and in ore formation of ground. Each development is placed into a category based on the prevailing ground conditions. Detailed ground support design templates are provided in Appendix B and F respectively;

Nearly all types of ground support are included in a weekly support schedule designed by the rock mechanics department at the mine (see Table 4.13). Three (03) tests in total were conducted in selected locations underground. It is mandatory to subject cable bolts like split sets to random tests.

The results obtained following the tests were plotted on graphs and analyzed for each test in Figures 4.18, 4.20 and 4.22. Three companies contracted by Mopani copper mines to do routine shotcreting on permanent excavation walls at Synclinorium underground were asked to submit four (04) shotcrete cubes each. Representing material assumed to be ready and suitable for shotcreting, the cubes were sent to a private laboratory for compressive strength tests and the results were returned. From the results only two concrete cube samples from one contractor gave results of 26.5 and 25.1 MPa which were within the required strength range 25-40MPa.

The common problems during the shotcreting process at Synclinorium underground mine are difficulty in achieving correct consistency (especially water/cement ratio, W/C), sprayability, and utilization of admixtures and use of correct nozzle distance by operators (Talbot and Burke, 2013).

### 5.2.5 Evaluation of the design, installation, and quality control of rock support and reinforcement.

In this chapter scrutiny of the current design process for Nkana Synclinorium mine was carried out with reference to design flow charts gathered from the literature review data. Numerical analysis to evaluate interaction of the installed support and reinforcement surrounding the underground excavations has also been assessed and analyzed.

Figure 4.25 shows the guideline in support design for underground excavations currently being mined at Synclinorium mine. Figure 4.26 shows the author’s suggested support design guideline based on literature review data.

There are a number of ground support and reinforcement design methods that can be used. All these methods rely on having a good understanding of the prevailing ground conditions before undertaking the design.

The design methods that are used at Synclinorium include the following:

1. Empirical methods (e.g. method proposed by the US Army Corps of Engineers);
2. Observational methods (e.g. New Austrian Tunneling Method);
3. Rock mass classification methods (e.g. RMR, Q, MRMR);
4. Stability graph method;
5. Numerical stress analysis methods;

### 5.2.6 Evaluation of the design Process

Compared to the 'Ground support design in deep underground mines' flow chart in Figure 4.26, the 'Nkana Synclinorium mine design process' chart Figure 4.25 lacks clarity in regard to the type and origin of loading (stress). The chart in Figure 4.26 which clearly presents the design principles and a procedure for ground support and reinforcement in deep and hard rock conditions has been proposed by the author.

The most effective steps in the design of ground support system for Synclinorium mine should clearly categorize loading are as follows:

1. Identification of the loading types
**2.** Static loading, and
**3.** Dynamic loading
**4.** Determination of the main source of loading
    - Origin of static loading: Gravity, in situ/induced stress, tectonic activities, groundwater, residual stresses and temperature; and
    - Origin of dynamic loading: Seismic events, strain burst, fault slip, pillar burst, gravity collapse, loading/unloading rate, and blasting.

When it comes to the design stage, the chart in Figure 4.26 looks at dynamic loading in two different ways:

1. Strain burst or pillar burst; and
2. Ground motion/shakedown/ejection.

The latter incorporates not only load and displacement factors, but also energy absorption in the determination of support capacity. Dynamic loading sources at Synclinorium mine should make a distinction between the strain burst/pillar burst type and that involving ground movement, shake-down, and ejection. The support design capacity for the later should consider energy accumulation in rock masses.

### 5.2.7 Evaluation of Rock Support Installations

In this subsection, rock stress fields around excavations at Synclinorium mine are analyzed. These provide the driving forces that can cause considerable rock instability. Also, examples of actual Rock Support Installations presented in Appendix E are compared with some Rock Support designs in Appendix B. Figure 4.27 Shows how the numerical modelling simulation of the main decline access of 5.20m width and 5.7m height in a Gold Mine in Australia (B. Rahim et. Al/Journal of Rock mechanics and Geotechnical Engineering 12 (2020) can be used to evaluate the interaction of rock bolts with the rock mass surrounding the excavation.

Relating the numerical modelling of Figure 4.27 to the modelling of a Synclinorium mine excavation in Figure 4.28, the numerical results demonstrate the reliability of the failure depth estimation through numerical modeling simulation compared with empirical methods and observational methods. Also the ratio of safety factor/loading factor ($\sigma cm/\sigma 1$) is presented at the top of the legend in (c). The maximum displacement of the rocks surrounding the excavation could be estimated by supplementing numerical modeling simulation with empirical methods and observational methods after installing ground support system as shown. The numerical results as explained in Figure 4.27 and 4.28 demonstrate stability of the rock masses surrounding the excavation after installing the support systems.

### 5.2.8 Assessment of the adherence of underground developments or excavations to design (size and geometry).

In this chapter 4, all underground tunnels mined e.g. cross cuts, orebody drives, haulages and loader drives were assessed based on over-breaks and under-breaks in cross section. In large open stopes like VCRs, pillar sizes and strength were assessed.

#### 5.2.8.1 Over-break and Under-break

It should be noted that in the absence of state-of-the art instruments, over-break or under-break volumes indicated in Tables 4.20 and 4.21 is not calculated by simply subtracting ‘Planned ($m^3$)’ from ‘Mined ($m^3$)’ figures. Mine survey takes into account deviations from the 1.6m grade line above the floor of the drive as well as the drive center line to penalize the mining contractor. Other factors are also applied to arrive at the figures for over-break and under-break volumes.

A comparison of selected development advances like 3585L_990N_XC in Table 4.20, and 3510L_1010N_LDR and Table 4.21 which were mined in both May and June 2023 has shown that there is no correlation between over-break volume and the linear meters mined in each drive. However, Powder Factor applied around the perimeter of an excavation during blasting has a strong correlation with over-break and under-break (Table 4.22 and Figure 4.32 respectively). Therefore, it is evident that a mixture of factors that can influence over-break and under-break is at play here, and include the following:

(i.) Unfavourable geological and geotechnical environments, with varied rock mass features.

(ii.) Drill-and-blast design and execution challenges. Difficult areas may not be blasted. This could be due to a range of factors, including:

1. Holes not drilled to design, missing holes or short holes.
2. Damaged or destroyed drill holes.
3. Drill holes not loaded.
4. Areas within a shot may be over- or under-blasted.
5. Powder factors may vary throughout the shot due to inconsistent hole depth and dipping strata.

In the months of May and June 2023 under review, most of the mining development was planned and carried out around the highly folded, jointed, sheared, and stressed areas of 3510 and 3585 levels as depicted on section and plan in Figures 4.4 and 4.5. In light of this, inconsistencies in over-break incidents and ground control can be expected in the areas concerned.

#### 5.2.8.2 In-situ monitoring to improve the empirical equations

A flowchart was developed to continuously monitor the in-situ pillars and make the necessary changes to the empirical equations (Jessu and Spearing, 2019). As shown in Figure 4.38 Synclinorium mine is currently using this methodology. However, this author has suggested an improvement involving consideration for mining depth and orebody dip and orientation (Figure 4.39). Continuous geotechnical monitoring of the pillars needs to be conducted via inspections to improve the performance of the pillars in situ and the pillar design methodology.

For example, the monitoring could involve measuring room and intersection width, the actual pillar dimensions, the pillar fracturing using a borescope, and pillar stress changes using flat-jacks. This would guide back-analysis of the empirical equations and can be reformed to improve the performance of the pillars.

#### 5.2.8.3 Proposal for the improved pillar design methodology

Based on the authors' experience and knowledge, a new insight into the pillar design process is proposed. For this purpose, a step-by-step procedure, as illustrated in Figure 4.39, is required to develop a safe and optimized pillar design. As indicated in the pillar design methodology in Figure 4.39, a number of factors are involved and contribute to the strength characteristics of a pillar. The first step consists of choosing the most appropriate pillar formula for the specific situation, such as the Salmon and Munro, (1967) equation for coal mines and Lunder and Pakalnis, 1997 equation for hard-rock mines. The second step comprises determining the pillar and room dimensions appropriate for the depth, rock strength, and required extraction ratio with an initial acceptable FOS. Then the various factors, such as the orientation of the orebody, depth of mining, excavation method and presence of geological features are incorporated to design an appropriately sized pillar with final desirable FOS.

All of the 16 VCR stopes surveyed employed regular pillar patterns (hereafter referred to as rib pillars) whose length is greater than their width. None of the current VCRs use square pillars, i.e. whose width is equal to the length. Nine VCRs use rectangular pillars (hereafter referred to as rib pillars) whose length is greater than their width. Because all the VCRs surveyed use rectangular pillars in a highly folded orebody, and contribute 70% of ore production, the present study also focusses on the analysis of this design method.

## 5.3 Chapter Summary

The results and analysis of the data collected in the research to assess the geotechnical considerations for ground control and stability at Nkana synclinorium mine is outlined in chapter 4 in line with the research objectives.

The discussion of the results and analysis is summarized as follows:

1. Assessment of ground control mechanisms and support standards for stability.

    - Rockmass properties such as UCS, GSI, Density, Joint spacing and RQD of the various rock formations including Shale (SOB), Argillite (HWA), Conglomerate (FCON), Sandstone (FSAN), and Quarzite (BQ) and analyzed using graphs. Shale is the weakest with 47 MPa and Quartzite the strongest with 178 MPa.
    - Geological structures such major joints have an influence on underground excavations in terms of dip and orientation was. High concentration of joints affect the ground stability especially when they are closely spaced and cross-crossing. Major folds characterized by shear displacements along the limbs towards the fold axis also affect stability of the folded rock mass.

2. Evaluation of Ground Support Standards at Synclinorium mine.

To assess the effectiveness of ground support, pull tests, compressing strength tests involving concrete cubes (shotcreting material) and numerical modelling data have been used in the evaluation of support standards.

3. Evaluation of the design processes

Existing design process charts have either been reviewed by this author to include critical steps which are missing in the design process or suggestions have been made to place the existing chart design process with alternative charts selected through literature review.

4. Assessment of the adherence of developments or excavations to design

Results of under-break and over-break have been analyzed and factors influencing over-break and under-break discussed such as unfavourable geological and geotechnical environments, with varied rock mass features. This includes drill-and-blast design and execution challenges like holes not drilled according to design and varying powder factors.

# CHAPTER 6 - Conclusion and Recommendations

## 6.1 Introduction

This research assessed the key geotechnical considerations for ground control and stability at Synclinorium mine in line with the main objective and sub-objectives of the research. Conclusions are based on the data collected, analysis and the results. Recommendations are in turn made in line with the conclusions.

## 6.2 Conclusion

In terms of the data analysis and the results, conclusions have been made. These conclusions are aligned with the main objective and sub-objective of the research, and are outlined as follows:

After the This research assessed the key geotechnical considerations for ground control and stability at Synclinorium mine in line with the main objective and sub-objectives of the research.

(a) In line with ground control mechanisms and support standards, the findings were as follows:

1. The collection of geotechnical data from underground has not been satisfactory over the years and has been restricted due to lack of mine development and accessibility as conditions in more areas become unfavourable. Although the analysis may indicate correlations in some rock mass data such as UCS and GIS, UCS and RQD/Joint spacing, including UCS and Density used in the research, this does not justify irregular update of the geotechnical database. Thus the geotechnical database confidence is low, at 60%, and may lead to inaccuracies in the determination of physical and mechanical rock properties for use in rock mass characterization as well as stope and rock support designs.

2. Geological structures such folds (anticlines/synclines) and joints are major contributors to rock instability around underground excavations at Synclinorium mine. The study has shown that rocks characterized by discontinuities and folding have weak mechanical rock properties, with rock strength of 47 MPa and 65 MPa for Shale and Argillite respectively, and are associated with ground instability. Using the RMR

system, all six parameters have indicated that rock types of sedimentary nature, like Shale, are associated with most of the ground instability at Synclinorium mine.

3. Pull Tests and Compressive Strength Tests on Shotcrete Cubes did not give satisfactory results about the integrity of installed cable bolts and quality of the shotcrete mix. For all the tests carried out at 3 sites, the average pass rate was 77.7%. The requirement is 100% pass rate pull tests carried out underground. In the case of compressive tests carried out on concrete cubes the pass rate was just 16.7%, which is well below the requirement of 100%. Physical inspection and observations made underground on support installations also revealed poor quality and inconsistent rock support standards (Appendix E).

(b) In line with the evaluation of design, installation, and quality control of rock support and reinforcement, which involved use of numerical modelling (Figure 4.29) and review of ground support process chart (Figure 4.26 and 4.27) the findings were as follows:

(i.) The design process used currently at Synclinorium mine as a deep underground mine needs to be robust in areas like rock stresses (loading). Investigation and understanding of loading types and their sources is an integral part of ground control and stability.

(ii.) Empirical and observational design methods should be practical and result in design changes where it becomes necessary. Comparisons between actual and design should lead to adjustments in the field.

(iii.) The design process remains rigid despite the mine development becoming deeper and deeper. Depth is barely a factor in the design methods.

(c) In line with assessment of adherence of underground developments (excavations) to design, most excavations hardly conform to the designs. This includes tunnels and VCR open stopes/pillars.

The following contributing factors have been reviewed:

(i.) Unfavourable geological and geotechnical environments, with varied rock mass features. Highly weathered, folded and jointed rock mass alternating with competent rock mass.
(ii.) Drill-and-blast design and execution challenges.

In general, based on the analysis of the data and the results, this research concludes that the application of sound geotechnical practices does not meet expected standards at Nkana Synclinorium mine.

## 6.3 Recommendation

In view of the current research the following recommendations have been suggested:

(a.) Improve the collection of geotechnical data from underground by making modern equipment such as more stress monitors, pull testing machines, and point load testing machines available. Development excavations should be well secured and accessible in terms of the unfavorable geological and geotechnical environment to facilitate geotechnical data collection.

(b.) In terms of design process evaluation, the design methods should be always aligned with the situation underground. Design methods should be flexible to include and take into account factors could impact ground conditions such as mining depth, geological structure of the orebody and its dip and orientation, especially where Pillar design is concerned.

(c.) Geotechnical issues that are unique to Synclinorium underground mine can only be recognized, identified, and addressed in an appropriate and professional manner, using current geotechnical knowledge, methodology, software and hardware. This entails that there is need to modernize the geotechnical engineering approach at the mine with modern geotechnical equipment.

(d.) Improve supervision of companies contracted to carry out underground support installations in the areas of shotcreting, rock bolting, and wire-meshing.

## 7.0 REFERENCES


1. Anthony T. Innacchione Ph.D. Civil Engineering, University of Pitttsburg,1999. Analysis of Pillar Design Practices and Techniques for USA Limestone.

2. Anatolii Protosenya & Maria Vilner, 2021 – St Petersburg Mining University. Assessment of excavation intersections' stability in jointed rock masses.

3. Al-Azhar University; Gamal Y. Boghdady, Assiut University; Mohamed E. I. abdelrasoul, Assiut University - 2016). How the quality of rock mass significantly impacts on the stability of underground excavations.

4. M Arsyad, V A Tiwow, Sulistiawaty & I A Sahdian, 2020. An analysis of physical properties and mechanics of rocks in the karst region of Pangkep Regency, Indonesia.

5. Bieniawski, Z T, 1973. Engineering classification of jointed rock masses, in Trans S Afr Inst Civ. Eng., 15:335-344.

6. Barton, N, Lien, R and Lunde, J, 1974. Engineering classification of rock masses for the design of tunnel support, in Rock Mech, 6:183-236.

7. B. Rahim et. Al/Journal of Rock mechanics and Geotechnical Engineering 12 (2020). Results of the numerical modelling of a main decline access of 5.20m width and 5.7m height in a Gold Mine in Australia.

8. Cai M, Kaiser PK (2012). Rock burst phenomenon and support characteristics, MIRARCO Mining innovation. In: Rock burst support reference book. Sudbury, Canada; 2018.

9. Dunnicliff, J, 1993. Geotechnical Instrumentation for Monitoring Field Performance, 577 p (John Wiley & Sons: New York).

10. Fodera et al., 2020; Jang and Topal, 2013; Singh and Xavier, 2005. Factors influencing over-break volumes in drill-and-blast tunnel excavation.

11. Hoek. E, Kaiser, P K and Bawden, W F, 1995. Support of Underground Excavations in Hard Rock, 215 p (Balkema: Rotterdam).

12. Hagan (1992). Importance of beddings and pre-existing joints on in-situ rock.

13. Hutchinson, D Jean and Diederichs, Mark S, 1996. Cable-bolting in Underground Mines, 406 p (BiTech Publishers: Richmond, British Columbia).

14. J.A Ibarra, N.H Maerz and JA Franklin. Over-break and Under-break in underground openings; 1996.

15. Jessu and Spearing, 2019. Understanding effect of ore body Inclination on the strength of rectangular pillars.

16. Jager and Ryder, 1999. designing layouts or support for multi-reef mining should always be approached as a unique problem.

17. Kaiser et al., 1996. Design of rock support system under rockburst condition.

18. Lunder & Pakalnis Confinement Formula (1997). Consolidated the various pillar classification systems from the combined database to compare the stability conditions of the pillars.

19. Laubscher, D H, 1990. A geomechanics classification system for the rating of rock mass in mine design, in J S Afr Inst Min Metall, 90(10):257-273.

20. Mandal and Singh [2009], Mottahedi et al. [2018]. Comparing measured and predicted over-breaks for Support vector machine (SVM)

21. Moshab (1997) Geotechnical Considerations in Underground Mines Geotechnical Considerations in Underground Mines. Department of Industry and Resources, Australia. Report,Pp.66Source[https://www.dmp.wa.gov.au/Documents/Safety/MSH_G_GeotechnicalConsiderationsUG Mines.pdf](https://www.dmp.wa.gov.au/Documents/Safety/MSH_G_GeotechnicalConsiderationsUG Mines.pdf)

22. Moshab (2016) Geotechnical considerations underground audit. – guide. Approved: 10 February 2016. 1 Adelaide Terrace, East Perth WA 6004. Postal address: Mineral House, 100.

23. Mopani Geology Department, Nkana Synclinorium, 2022. Working Plans and Sections

24. Ortlepp, W D, 1992. Invited lecture: The design of support for the containment of rock burst damage in tunnels - An engineering approach, in Rock Support in Mining and Underground Construction, (Ed: P K Kaiser and D R McCreath), pp 593-609 (Balkema: Rotterdam).

25. Open House Management Solutions (OHMS), 2017. SOB VCR Pillar Assessment.

26. G.B. Quaye and G. Guler, 1998. The importance of both geological structures and mining induced stress fractures on the hanging wall stability in a deep level gold mine.

27. Palmstrom & Stille 2015. The main features related to rock mass structure; factors influencing mechanical properties of rock.

28. Abdelrasoul, Assiut (2016). The evaluation of the Effect of Rock Joints on the Stability of Underground Tunnels. Journal of Rock Mechanics, Vol. 3, No. 11, 2016, pp. 1790-1799.

29. P. Kaiser et al., 2022 and A. Moss Deformation-based support design for highly stressed ground with a focus on rock burst damage mitigation (P56-66).

30. P. Kaiser and Bo Hyun Kim (2008) Rock mechanics advances of underground construction and mining.

31. Potvin Y, Wesseloo J, Heal D. An interpretation of ground support capacity submitted to dynamic loading. Mining Technology 2010;119(4):233-245.

32. Priest, S D, 1993. Discontinuity Analysis for Rock Engineering, 473 p (Chapman & Hall: London).

33. Persson, P-A, Holmberg, R and Lee, J, 1994. Rock Blasting and Explosives Engineering, 540 p (CRC Press: Boca Raton).

34. Rahimi B, Sharifzadeh M. Evaluation of ground management in underground excavation design. In: Proceedings of the 8th international conference on deep and high stress mining. Perth: Australian Centre for Geomechanics; 2017. p. 813-826.

35. Razougui Abdellah, University Ahmed Draia – Adrar; Mahrous Ali Mohamed Ali, Al-Azhar University; Gamal Y. Boghdady, Assiut University; and Mohamed E. I.

36. Read et al., 1997; Volokhov, 2003. Insitu studies and analysis of numerical modelling results.

37. Renjith S Anand & D Venkat Reddy, 2015. A Geological and Geotechnical Investigation of Some Rocks in Trivandrum Area, Kerala, India.

38. Stillborg, B, 1994. Professional Users Handbook for Rock Bolting, second edition, 164 p (Trans Tech Publications: Clausthal-Zellerfeld).

39. Stefan Lamplmair, Oliver Zeman, and Klaus Voit, 2021. Factors Influencing the Load-Bearing Capacity of Rock as Base Material for Post-installed anchors.

40. Selvasekaran R, Appuhami KAIS, Kumarasiri SAI, Fernandopulle 2009. Effects of Joint Orientation in Tunneling.

41. S.N., Lakmal, H.M.A.G., & Dharmaratne, P.G.R. (2009). Thompson, A G and Windsor, C R, 1996. SAFEX - Stability Assessment for Excavations in Rock, 142 p (Rock Technology Pty Ltd: Perth, Western Australia).

## **Appendix A**: Rock Mass Classifications for rock quality

**ROCK MASS CLASSIFICATION**

*Total A4 factor comprises these factors:*
*persistence/Length (E1)*
*- Separation/Aperture (E2)*
*- Surface Roughness (E3)*
*- Infilling/Gouge (E4)*

-

*FVRB = FAVOURABLE*
*VFVRB = FAVOURABLE*
*UNFVRB = UNFAVOURABLE*

| | | A1 | | A2 | | A3 | | WEATHERING OF JOINT WALL | | | A5 | | | | Bieniawski's 1989 | Bieniawski's 1979 | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Rock Name | Weathering Code | UCS (MPa) PLS.TEST | A1 FACTOR | RQD (%) | A2 FACTOR | SPACING OF DISCONTINUITIES (CM) | A3 FACTOR | E5 | E5 FACTOR | TOTAL A4 FACTOR | GROUND WATER | A5 FACTOR | JOINT ORIENTATION ADJ. | B FACTOR | RMR | GSI | Q | Description |

## Rock Mass Classification: SOUTH OREBODY SHALE

| | | | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| SOBS | FR | 40 | 7 | 70 | 10 | 14 | FALSE | UNWEATHERED | 6 | 16 | DRY | 14 | FAIR | -5 | 50 | 66 | 21 | Fair rock |
| SOBS | FR | 40 | 7 | 40 | 8 | 43 | 13 | UNWEATHERED | 6 | 19 | DRY | 14 | FAIR | -5 | 45 | 61 | 12 | Fair rock |
| SOBS | FR | 40 | 7 | 45 | 9 | 20 | 13 | UNWEATHERED | 6 | 17 | DRY | 14 | FAIR | -5 | 48 | 62 | 13 | Fair rock |
| SOBS | FR | 40 | 10 | 46 | 9 | 20 | 13 | UNWEATHERED | 6 | 23 | DRY | 12 | FAIR | -5 | 60 | 62 | 13 | Fair rock |
| SOBS | FR | 40 | 10 | 77 | 11 | 21 | FALSE | UNWEATHERED | 6 | 26 | DRY | 12 | FAIR | -5 | 60 | 73 | 46 | Fair rock |
| SOBS | SW | 40 | 10 | 60 | 12 | 17 | 12 | UNWEATHERED | 6 | 21 | DRY | 12 | FAIR | -5 | 55 | 73 | 46 | Fair rock |
| SOBS | SW | 40 | 10 | 70 | 10 | 10 | 12 | UNWEATHERED | 6 | 19 | DRY | 12 | FAIR | -5 | 42 | 55 | 31 | Fair rock |
| SOBS | SW | 40 | 10 | 65 | 10 | 19 | 12 | UNWEATHERED | 6 | 21 | DRY | 12 | FAIR | -5 | 65 | 60 | 39 | Good rock |
| SOBS | SW | 40 | 10 | 55 | 8 | 20 | 12 | UNWEATHERED | 6 | 13 | DRY | 12 | FAIR | -5 | 48 | 40 | 31 | Fair rock |
| SOBS | SW | 40 | 10 | 70 | 8 | 17 | 12 | UNWEATHERED | 6 | 22 | DRY | 12 | FAIR | -5 | 37 | 53 | 31 | Poor rock |
| SOBS | SW | 45 | 10 | 69 | 6 | 14 | 12 | UNWEATHERED | 6 | 10 | DRY | 12 | FAIR | -5 | 49 | 45 | 35 | Fair rock |
| SOBS | SW | 45 | 10 | 75 | 9 | 15 | 12 | UNWEATHERED | 6 | 17 | DRY | 15 | FAIR | -5 | 30 | 56 | 31 | Poor rock |
| SOBS | SW | 65 | 10 | 50 | 10 | 20 | FALSE | UNWEATHERED | 6 | 21 | DAMP | 10 | FVRB | FALSE | 60 | 55 | 13 | Fair rock |
| SOBS | SW | 50 | 13 | 63 | 8 | 43 | FALSE | UNWEATHERED | 6 | 16 | DAMP | 10 | FAIR | -5 | 44 | 62 | 12 | Fair rock |
| SOBS | SW | 50 | 12 | 57 | 11 | 18 | FALSE | UNWEATHERED | 6 | 20 | DAMP | 10 | FAIR | -5 | 50 | 60 | 11 | Fair rock |
| SOBS | SW | 50 | 12 | 80 | 9 | 21 | FALSE | UNWEATHERED | 6 | 20 | DRY | 15 | FAIR | -5 | 45 | 70 | 31 | Fair rock |
| SOBS | SW | 50 | 12 | 57 | 9 | 20 | FALSE | UNWEATHERED | 6 | 23 | DAMP | 10 | FAIR | -5 | 40 | 60 | 11 | Poor rock |
| SOBS | SW | 50 | 12 | 60 | 12 | 43 | 13 | UNWEATHERED | 6 | 19 | DAMP | 10 | FAIR | -5 | 53 | 58 | 12 | Fair rock |
| SOBS | SW | 50 | 12 | 33 | 7 | 23 | 13 | UNWEATHERED | 6 | 20 | DRY | 12 | FAIR | -5 | 66 | 50 | 11 | Good rock |
| SOBS | SW | 50 | 12 | 67 | 10 | 17 | 13 | UNWEATHERED | 6 | 26 | DRY | 12 | FAIR | -5 | 50 | 67 | 23 | Fair rock |
| SOBS | SW | 50 | 12 | 90 | 12 | 37 | 13 | UNWEATHERED | 6 | 22 | DRY | 12 | FAIR | -5 | 47 | 69 | 39 | Fair rock |
| SOBS | SW | 50 | 12 | 98 | 12 | 18 | 13 | UNWEATHERED | 6 | 19 | DRY | 13 | FAIR | -5 | 65 | 70 | 47 | Good rock |
| SOBS | SW | 42 | 12 | 87 | 12 | 20 | 13 | UNWEATHERED | 6 | 23 | DRY | 13 | FAIR | -5 | 68 | 70 | 37 | Good rock |
| SOBS | SW | 42 | 12 | 48 | 10 | 18 | 13 | UNWEATHERED | 6 | 19 | DRY | 13 | FAIR | -5 | 59 | 75 | 49 | Fair rock |
| SOBS | SW | 42 | 12 | 60 | 12 | 20 | 13 | UNWEATHERED | 6 | 21 | DRY | 13 | FAIR | -5 | 60 | 66 | 49 | Fair rock |
| SOBS | SW | 42 | 7 | 55 | 11 | 20 | 13 | UNWEATHERED | 6 | 21 | DRY | 13 | FAIR | -5 | 58 | 60 | 49 | Fair rock |
| SOBS | SW | 42 | 7 | 65 | 13 | 18 | 13 | UNWEATHERED | 6 | 21 | DRY | 13 | FAIR | -5 | 60 | 68 | 49 | Fair rock |
| SOBS | SW | 42 | 7 | 70 | 14 | 20 | 13 | UNWEATHERED | 6 | 21 | DRY | 13 | FAIR | -5 | 65 | 60 | 49 | Good rock |
| SOBS | SW | 42 | 7 | 50 | 10 | 17 | 13 | UNWEATHERED | 6 | 21 | DRY | 13 | FAIR | -5 | 49 | 58 | 49 | Fair rock |
| SOBS | FR | 42 | 7 | 70 | 14 | 23 | 13 | UNWEATHERED | 6 | 19 | DRY | 13 | FAIR | -5 | 62 | 58 | 25 | Good rock |
| SOBS | FR | 42 | 7 | 45 | 9 | 21 | 13 | UNWEATHERED | 6 | 19 | DRY | 15 | FAIR | -5 | 55 | 60 | 31 | Fair rock |
| SOBS | FR | 42 | 7 | 50 | 10 | 18 | 13 | UNWEATHERED | 6 | 20 | DAMP | 10 | FAIR | -5 | 63 | 55 | 30 | Good rock |
| SOBS | FR | 46 | 7 | 51 | 10 | 20 | 13 | UNWEATHERED | 6 | 20 | DRY | 14 | FAIR | -5 | 58 | 62 | 15 | Fair rock |
| SOBS | FR | 46 | 7 | 80 | 16 | 37 | 13 | UNWEATHERED | 6 | 21 | DRY | 14 | FAIR | -5 | 60 | 67 | 31 | Fair rock |
| SOBS | FR | 46 | 7 | 25 | 5 | 20 | FALSE | UNWEATHERED | 6 | 19 | DRY | 14 | FAIR | -5 | 52 | 58 | 27 | Fair rock |
| SOBS | SW | 46 | 7 | 50 | 10 | 20 | FALSE | UNWEATHERED | 6 | 19 | DRY | 14 | FAIR | -5 | 49 | 62 | 23 | Fair rock |
| SOBS | SW | 46 | 7 | 50 | 10 | 18 | FALSE | UNWEATHERED | 6 | 19 | DRY | 14 | FAIR | -5 | 50 | 62 | 13 | Fair rock |
| SOBS | SW | 46 | 7 | 70 | 14 | 13 | FALSE | UNWEATHERED | 6 | 19 | DRY | 14 | FAIR | -5 | 61 | 66 | 20 | Good rock |
| SOBS | SW | 46 | 7 | 60 | 12 | 22 | FALSE | UNWEATHERED | 6 | 19 | DRY | 12 | FAIR | -5 | 69 | 62 | 16 | Good rock |
| SOBS | SW | 46 | 7 | 60 | 12 | 20 | FALSE | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 65 | 59 | 49 | Good rock |
| SOBS | SW | 40 | 11 | 20 | 4 | 37 | FALSE | UNWEATHERED | 6 | 18 | WET | 7 | FAIR | -5 | 53 | 48 | 3 | Fair rock |
| SOBS | SW | 40 | 11 | 60 | 12 | 20 | FALSE | UNWEATHERED | 6 | 24 | DAMP | 10 | FAIR | -5 | 64 | 55 | 9 | Good rock |
| SOBS | SW | 40 | 11 | 60 | 12 | 19 | 12 | UNWEATHERED | 6 | 24 | DAMP | 10 | FAIR | -5 | 66 | 54 | 9 | Good rock |
| SOBS | SW | 40 | 11 | 50 | 10 | 15 | 12 | UNWEATHERED | 6 | 24 | DAMP | 10 | FAIR | -5 | 62 | 57 | 7 | Good rock |
| SOBS | SW | 40 | 11 | 60 | 12 | 15 | 12 | UNWEATHERED | 6 | 24 | DAMP | 10 | FAIR | -5 | 64 | 59 | 9 | Good rock |
| SOBS | SW | 40 | 11 | 50 | 10 | 16 | 12 | UNWEATHERED | 6 | 20 | DRY | 12 | FAIR | -5 | 67 | 62 | 13 | Good rock |
| SOBS | SW | 40 | 11 | 50 | 10 | 19 | 12 | UNWEATHERED | 6 | 20 | DRY | 12 | FAIR | -5 | 50 | 65 | 13 | Fair rock |
| SOBS | SW | 40 | 11 | 60 | 12 | 19 | 12 | UNWEATHERED | 6 | 20 | DRY | 12 | FAIR | -5 | 49 | 60 | 16 | Fair rock |
| SOBS | SW | 48 | 11 | 60 | 12 | 17 | 12 | UNWEATHERED | 6 | 20 | DRY | 12 | FAIR | -5 | 57 | 59 | 16 | Fair rock |
| SOBS | FR | 48 | 11 | 90 | 11 | 23 | 12 | UNWEATHERED | 6 | 20 | DRY | 12 | FAIR | -5 | 45 | 55 | 31 | Fair rock |
| SOBS | FR | 48 | 11 | 80 | 11 | 16 | 12 | UNWEATHERED | 6 | 20 | DRY | 14 | FAIR | -5 | 60 | 54 | 39 | Fair rock |
| SOBS | FR | 48 | 11 | 90 | 11 | 19 | 12 | UNWEATHERED | 6 | 24 | DRY | 14 | FAIR | -5 | 65 | 60 | 31 | Good rock |
| SOBS | FR | 48 | 7 | 86 | 11 | 37 | 12 | UNWEATHERED | 6 | 24 | DRIP | FALSE | FAIR | -5 | 68 | 55 | 6 | Good rock |
| SOBS | FR | 48 | 7 | 50 | 10 | 24 | 12 | UNWEATHERED | 6 | 24 | DRY | 13 | FAIR | -5 | 54 | 58 | 9 | Fair rock |
| SOBS | FR | 50 | 13 | 50 | 10 | 20 | FALSE | UNWEATHERED | 6 | 24 | DRY | 12 | FAIR | -5 | 64 | 59 | 9 | Good rock |
| SOBS | FR | 47 | 7 | 50 | 10 | 21 | FALSE | UNWEATHERED | 6 | 24 | DRY | 13 | FAIR | -5 | 40 | 56 | 13 | Poor rock |
| SOBS | FR | 47 | 7 | 24 | 5 | 17 | FALSE | UNWEATHERED | 6 | 19 | DRY | 14 | FAIR | -5 | 60 | 51 | 4 | Fair rock |
| SOBS | FR | 47 | 7 | 24 | 5 | 16 | FALSE | UNWEATHERED | 6 | 19 | DRY | 14 | FAIR | -5 | 56 | 51 | 4 | Fair rock |

| | | | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| SOBS | SW | 55 | 13 | 75 | 15 | 16 | FALSE | UNWEATHERED | 6 | 21 | DRY | 13 | FVRB | FALSE | 60 | 52 | 28 | Fair rock |
| SOBS | SW | 50 | 13 | 80 | 16 | 19 | FALSE | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 70 | 64 | 31 | Good rock |
| SOBS | SW | 42 | 7 | 60 | 12 | 20 | FALSE | UNWEATHERED | 6 | 21 | DRY | 11 | FVRB | FALSE | 58 | 49 | 49 | Fair rock |
| SOBS | SW | 42 | 7 | 90 | 18 | 13 | 14 | UNWEATHERED | 6 | 21 | DRY | 11 | FAIR | -5 | 67 | 57 | 39 | Good rock |
| SOBS | SW | 42 | 7 | 75 | 15 | 16 | 14 | UNWEATHERED | 6 | 21 | DRY | 11 | FAIR | -5 | 55 | 50 | 49 | Fair rock |
| SOBS | SW | 42 | 7 | 70 | 14 | 19 | 14 | UNWEATHERED | 6 | 21 | DRY | 11 | FAIR | -5 | 59 | 47 | 49 | Fair rock |
| SOBS | SW | 42 | 7 | 44 | 9 | 20 | 14 | UNWEATHERED | 6 | 16 | DRY | 11 | | | 63 | 55 | 8 | Good rock |
| SOBS | SW | 42 | 7 | 80 | 16 | 13 | 14 | UNWEATHERED | 6 | 22 | DRY | 11 | FAIR | -5 | 60 | 52 | 18 | Fair rock |
| SOBS | SW | 42 | 7 | 60 | 12 | 17 | 14 | UNWEATHERED | 6 | 22 | DRY | 11 | FAIR | -5 | 69 | 60 | 16 | Good rock |
| SOBS | SW | 42 | 7 | 70 | 14 | 13 | 14 | UNWEATHERED | 6 | 22 | DRY | 11 | FAIR | -5 | 71 | 60 | 20 | Good rock |
| SOBS | SW | 42 | 7 | 60 | 12 | 20 | 12 | UNWEATHERED | 6 | 22 | DRY | 13 | FAIR | -5 | 59 | 45 | 16 | Fair rock |
| SOBS | MW | 60 | 13 | 50 | 10 | 24 | 12 | UNWEATHERED | 6 | 19 | DRY | 11 | FAIR | -5 | 64 | 58 | 9 | Good rock |
| SOBS | MW | 60 | 13 | 60 | 12 | 21 | FALSE | UNWEATHERED | 6 | 17 | DRY | 14 | FAIR | -5 | 70 | 65 | 18 | Good rock |
| SOBS | MW | 60 | 13 | 60 | 12 | 17 | 13 | UNWEATHERED | 6 | 17 | DRY | 12 | FAIR | -5 | 67 | 62 | 13 | Good rock |
| SOBS | MW | 55 | 13 | 60 | 12 | 22 | 13 | UNWEATHERED | 6 | 14 | DRY | 12 | | | 59 | 54 | 35 | Fair rock |
| SOBS | MW | 55 | 13 | 78 | 16 | 13 | 13 | UNWEATHERED | 6 | 17 | DRY | 11 | FAIR | -5 | 71 | 66 | 19 | Good rock |
| SOBS | MW | 60 | 13 | 70 | 14 | 21 | 13 | UNWEATHERED | 6 | 18 | DRY | 13 | FAIR | -5 | 66 | 61 | 12 | Good rock |
| SOBS | MW | 60 | 13 | 52 | 10 | 24 | 13 | UNWEATHERED | 6 | 18 | DRY | 10 | | | 62 | 57 | 38 | Good rock |
| SOBS | MW | 60 | 13 | 60 | 12 | 24 | 13 | UNWEATHERED | 6 | 22 | DRY | 12 | FAIR | -5 | 67 | 62 | 13 | Good rock |
| SOBS | MW | 60 | 13 | 55 | 11 | 20 | FALSE | UNWEATHERED | 6 | 22 | DRY | 14 | FAIR | -5 | 66 | 61 | 12 | Good rock |
| SOBS | MW | 60 | 13 | 70 | 14 | 20 | 10 | UNWEATHERED | 6 | 19 | DRY | 12 | FAIR | -5 | 71 | 66 | 20 | Good rock |
| SOBS | MW | 60 | 13 | 90 | 18 | 22 | 10 | UNWEATHERED | 6 | 19 | DRY | 11 | FAIR | -5 | 75 | 70 | 31 | Good rock |
| SOBS | MW | 60 | 13 | 86 | 17 | 13 | 10 | UNWEATHERED | 6 | 19 | DRIP | FALSE | FAIR | -5 | 63 | 58 | 25 | Good rock |
| SOBS | MW | 65 | 13 | 50 | 10 | 17 | FALSE | SLIGHT | 5 | 18 | DRIP | FALSE | FAIR | -5 | 50 | 45 | 32 | Fair rock |

## Rock Mass Classification: FW SANDSTONE

| | | | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| FSAN | SW | 75 | 13 | 97 | 19 | 36 | 19 | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 68 | 60 | 46 | Good rock |
| FSAN | SW | 75 | 13 | 95 | 19 | 30 | 19 | UNWEATHERED | 6 | 24 | DRY | 15 | FAIR | -5 | 85 | 70 | 31 | Very good rock |
| FSAN | SW | 75 | 13 | 95 | 19 | 41 | 19 | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 85 | 62 | 39 | Very good rock |
| FSAN | SW | 75 | 13 | 95 | 19 | 37 | 19 | UNWEATHERED | 6 | 24 | DRY | 15 | FAIR | -5 | 90 | 60 | 31 | Very good rock |
| FSAN | SW | 75 | 13 | 80 | 16 | 39 | 19 | UNWEATHERED | 6 | 24 | DRY | 15 | FAIR | -5 | 95 | 70 | 31 | Very good rock |
| FSAN | SW | 75 | 13 | 80 | 16 | 40 | 16 | UNWEATHERED | 6 | 24 | DRY | 15 | FAIR | -5 | 90 | 71 | 35 | Very good rock |
| FSAN | SW | 75 | 13 | 80 | 16 | 42 | 16 | UNWEATHERED | 6 | 24 | DRY | 15 | FAIR | -5 | 90 | 70 | 31 | Very good rock |
| FSAN | SW | 80 | 13 | 0 | 0 | 33 | 16 | UNWEATHERED | 6 | 24 | DRY | 15 | FAIR | -5 | 95 | 44 | 2 | Very good rock |
| FSAN | SW | 165 | 12 | 95 | 19 | 46 | 16 | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 85 | 78 | 76 | Very good rock |
| FSAN | SW | 160 | 12 | 70 | 14 | 30 | 16 | UNWEATHERED | 6 | 18 | DRY | 15 | FAIR | -5 | 80 | 76 | 61 | Good rock |
| FSAN | SW | 160 | 12 | 65 | 13 | 32 | 16 | UNWEATHERED | 6 | 18 | DRY | 15 | FAIR | -5 | 95 | 76 | 61 | Very good rock |
| FSAN | SW | 160 | 12 | 95 | 19 | 35 | 16 | UNWEATHERED | 6 | 18 | DRY | 15 | FAIR | -5 | 90 | 75 | 55 | Very good rock |
| FSAN | SW | 160 | 12 | 0 | 0 | 36 | 20 | UNWEATHERED | 6 | 18 | DRY | 15 | FAIR | -5 | 90 | 41 | 1 | Very good rock |
| FSAN | SW | 120 | 12 | 55 | 11 | 30 | 21 | UNWEATHERED | 6 | 18 | DRY | 15 | FAIR | -5 | 95 | 76 | 61 | Very good rock |
| FSAN | SW | 120 | 12 | 84 | 17 | 37 | 22 | UNWEATHERED | 6 | 18 | DRY | 15 | FAIR | -5 | 90 | 71 | 34 | Very good rock |
| FSAN | FR | 75 | 13 | 84 | 17 | 28 | 23 | UNWEATHERED | 6 | 18 | DRY | 15 | FAIR | -5 | 85 | 67 | 22 | Very good rock |
| FSAN | SW | 75 | 13 | 60 | 12 | 48 | 21 | UNWEATHERED | 6 | 22 | DRY | 15 | FAIR | -5 | 90 | 70 | 39 | Very good rock |
| FSAN | SW | 75 | 12 | 85 | 17 | 42 | 24 | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 85 | 70 | 49 | Very good rock |
| FSAN | SW | 75 | 12 | 65 | 13 | 30 | 24 | UNWEATHERED | 6 | 23 | DRY | 15 | FAIR | -5 | 80 | 69 | 28 | Good rock |
| FSAN | SW | 75 | 12 | 88 | 18 | 32 | 24 | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 90 | 65 | 49 | Very good rock |
| FSAN | SW | 75 | 12 | 70 | 14 | 35 | 24 | UNWEATHERED | 6 | 24 | DRY | 15 | FAIR | -5 | 75 | 66 | 39 | Good rock |
| FSAN | SW | 75 | 12 | 80 | 16 | 49 | 24 | UNWEATHERED | 6 | 23 | DRY | 15 | FAIR | -5 | 80 | 60 | 61 | Good rock |
| FSAN | SW | 75 | 12 | 85 | 17 | 30 | 24 | UNWEATHERED | 6 | 24 | DRY | 15 | FAIR | -5 | 95 | 72 | 39 | Very good rock |
| FSAN | SW | 75 | 12 | 68 | 14 | 42 | 19 | UNWEATHERED | 6 | 24 | DRY | 15 | FAIR | -5 | 98 | 67 | 22 | Very good rock |
| FSAN | SW | 75 | 12 | 90 | 18 | 40 | 19 | UNWEATHERED | 6 | 20 | DRY | 15 | FAIR | -5 | 92 | 73 | 44 | Very good rock |
| FSAN | SW | 75 | 12 | 70 | 14 | 36 | 19 | UNWEATHERED | 6 | 21 | DRY | 15 | FVRB | FALSE | 75 | 60 | 68 | Good rock |
| FSAN | SW | 75 | 12 | 95 | 19 | 30 | 19 | UNWEATHERED | 6 | 24 | DRY | 15 | FAIR | -5 | 77 | 65 | 39 | Good rock |
| FSAN | SW | 75 | 12 | 80 | 16 | 41 | 19 | UNWEATHERED | 6 | 21 | DRY | 15 | FVRB | FALSE | 68 | 60 | 68 | Good rock |
| FSAN | SW | 75 | 12 | 65 | 13 | 37 | 19 | UNWEATHERED | 6 | 21 | DRY | 15 | UNFVRB | FALSE | 70 | 60 | 85 | Good rock |
| FSAN | SW | 75 | 12 | 97 | 19 | 39 | 19 | UNWEATHERED | 6 | 24 | DRY | 15 | FAIR | -5 | 77 | 70 | 39 | Good rock |
| FSAN | SW | 75 | 13 | 70 | 14 | 40 | 21 | UNWEATHERED | 6 | 23 | DRY | 15 | FAIR | -5 | 70 | 65 | 61 | Good rock |
| FSAN | MW | 75 | 13 | 85 | 17 | 42 | 21 | UNWEATHERED | 6 | 23 | DRY | 15 | FVRB | FALSE | 72 | 60 | 61 | Good rock |
| FSAN | SW | 75 | 13 | 96 | 19 | 35 | 21 | UNWEATHERED | 6 | 24 | DRY | 15 | FAIR | -5 | 67 | 50 | 39 | Good rock |
| FSAN | SW | 75 | 13 | 80 | 16 | 38 | 21 | UNWEATHERED | 6 | 24 | DRY | 15 | FAIR | -5 | 70 | 64 | 39 | Good rock |
| FSAN | MW | 75 | 13 | 95 | 19 | 29 | 21 | UNWEATHERED | 6 | 23 | DRY | 15 | FVRB | FALSE | 74 | 66 | 85 | Good rock |
| FSAN | MW | 75 | 13 | 90 | 18 | 44 | 21 | UNWEATHERED | 6 | 23 | DRY | 15 | FVRB | FALSE | 65 | 60 | 85 | Good rock |
| FSAN | MW | 75 | 13 | 90 | 18 | 33 | 21 | UNWEATHERED | 6 | 23 | DRY | 15 | FVRB | FALSE | 75 | 60 | 85 | Good rock |
| FSAN | MW | 75 | 13 | 88 | 18 | 40 | 18 | UNWEATHERED | 6 | 24 | DRY | 15 | UNFVRB | FALSE | 70 | 65 | 18 | Good rock |
| FSAN | FR | 160 | 12 | 60 | 12 | 33 | 18 | UNWEATHERED | 6 | 20 | DRY | 15 | FAIR | -5 | 72 | 75 | 68 | Good rock |
| FSAN | FR | 160 | 12 | 60 | 12 | 26 | 20 | UNWEATHERED | 6 | 28 | DRY | 15 | FAIR | -5 | 75 | 60 | 68 | Good rock |
| FSAN | FR | 160 | 12 | 60 | 12 | 30 | 20 | UNWEATHERED | 6 | 28 | DRY | 15 | FAIR | -5 | 73 | 69 | 76 | Good rock |
| FSAN | FR | 160 | 12 | 77 | 15 | 32 | 20 | UNWEATHERED | 6 | 28 | DRY | 15 | FAIR | -5 | 73 | 64 | 76 | Good rock |
| FSAN | FR | 160 | 12 | 90 | 18 | 37 | 20 | UNWEATHERED | 6 | 28 | DRY | 15 | FAIR | -5 | 72 | 70 | 68 | Good rock |
| FSAN | FR | 160 | 12 | 86 | 17 | 40 | 20 | UNWEATHERED | 6 | 28 | DRY | 15 | FAIR | -5 | 85 | 59 | 68 | Very good rock |
| FSAN | FR | 160 | 12 | 90 | 18 | 35 | 20 | UNWEATHERED | 6 | 28 | DRY | 15 | FAIR | -5 | 89 | 60 | 76 | Very good rock |

| | | | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| FSAN | MW | 70 | 13 | 60 | 12 | 36 | 20 | UNWEATHERED | 6 | 28 | DRY | 15 | FAIR | -5 | 90 | 60 | 16 | Very good rock |
| FSAN | MW | 70 | 13 | 90 | 18 | 30 | 20 | UNWEATHERED | 6 | 28 | DRY | 15 | FAIR | -5 | 96 | 71 | 35 | Very good rock |
| FSAN | MW | 70 | 13 | 50 | 10 | 43 | 20 | UNWEATHERED | 6 | 28 | DRY | 15 | FAIR | -5 | 88 | 60 | 14 | Very good rock |
| FSAN | MW | 70 | 13 | 50 | 10 | 45 | 20 | UNWEATHERED | 6 | 20 | DRY | 15 | FAIR | -5 | 90 | 55 | 14 | Very good rock |
| FSAN | MW | 70 | 13 | 55 | 11 | 24 | 19 | UNWEATHERED | 6 | 22 | DRY | 15 | FAIR | -5 | 95 | 69 | 39 | Very good rock |
| FSAN | MW | 70 | 13 | 80 | 16 | 32 | 19 | UNWEATHERED | 6 | 22 | DRY | 15 | FAIR | -5 | 70 | 62 | 39 | Good rock |
| FSAN | MW | 70 | 13 | 75 | 15 | 36 | 19 | UNWEATHERED | 6 | 22 | DRY | 15 | FAIR | -5 | 76 | 70 | 39 | Good rock |
| FSAN | MW | 70 | 13 | 60 | 12 | 32 | 19 | UNWEATHERED | 6 | 22 | DRY | 15 | FAIR | -5 | 78 | 72 | 39 | Good rock |
| FSAN | FR | 175 | 12 | 85 | 17 | 24 | 22 | UNWEATHERED | 6 | 22 | DRY | 15 | FAIR | -5 | 80 | 75 | 55 | Good rock |
| FSAN | FR | 175 | 12 | 60 | 12 | 32 | 22 | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 70 | 65 | 31 | Good rock |
| FSAN | FR | 175 | 12 | 97 | 19 | 40 | 22 | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 70 | 58 | 76 | Good rock |
| FSAN | FR | 175 | 12 | 90 | 18 | 32 | 22 | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 75 | 60 | 61 | Good rock |
| FSAN | FR | 175 | 12 | 75 | 15 | 24 | 22 | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 70 | 56 | 76 | Good rock |
| FSAN | MW | 75 | 13 | 76 | 15 | 35 | 22 | UNWEATHERED | 6 | 24 | DRY | 15 | | | 72 | 67 | 23 | Good rock |
| FSAN | MW | 75 | 13 | 84 | 17 | 42 | 22 | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 89 | 74 | 49 | Very good rock |
| FSAN | MW | 75 | 13 | 90 | 18 | 48 | 17 | UNWEATHERED | 6 | 24 | DRY | 15 | FAIR | -5 | 97 | 72 | 39 | Very good rock |
| FSAN | MW | 75 | 13 | 83 | 17 | 28 | 17 | UNWEATHERED | 6 | 26 | DRY | 15 | FAIR | -5 | 81 | 66 | 19 | Very good rock |
| FSAN | SW | 175 | 17 | 80 | 16 | 33 | 17 | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 80 | 61 | 106 | Good rock |
| FSAN | SW | 175 | 17 | 95 | 19 | 42 | 16 | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 75 | 55 | 106 | Good rock |
| FSAN | SW | 175 | 17 | 90 | 18 | 35 | 21 | UNWEATHERED | 6 | 24 | DRY | 15 | FAIR | -5 | 80 | 55 | 85 | Good rock |
| FSAN | SW | 175 | 17 | 70 | 14 | 35 | 21 | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 76 | 68 | 106 | Good rock |
| FSAN | MW | 75 | 13 | 75 | 15 | 40 | 21 | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 77 | 65 | 39 | Good rock |
| FSAN | MW | 75 | 13 | 75 | 15 | 42 | FALSE | UNWEATHERED | 6 | 21 | DRIP | FALSE | FAIR | -5 | 76 | 61 | 12 | Good rock |
| FSAN | MW | 75 | 13 | 75 | 15 | 29 | FALSE | UNWEATHERED | 6 | 21 | DRIP | FALSE | FAIR | -5 | 97 | 61 | 12 | Very good rock |
| FSAN | MW | 75 | 13 | 75 | 15 | 35 | FALSE | UNWEATHERED | 6 | 23 | DRIP | FALSE | FAIR | -5 | 88 | 63 | 14 | Very good rock |
| FSAN | MW | 75 | 13 | 75 | 15 | 42 | FALSE | UNWEATHERED | 6 | 23 | DRY | 15 | FAIR | -5 | 79 | 74 | 49 | Good rock |
| FSAN | MW | 75 | 13 | 75 | 15 | 35 | FALSE | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 70 | 66 | 39 | Good rock |
| FSAN | MW | 75 | 13 | 75 | 15 | 26 | FALSE | UNWEATHERED | 6 | 21 | DRY | 15 | | | 68 | 55 | 39 | Good rock |
| FSAN | MW | 75 | 13 | 75 | 15 | 42 | FALSE | UNWEATHERED | 6 | 21 | DRY | 15 | | | 75 | 60 | 39 | Good rock |
| FSAN | MW | 75 | 13 | 75 | 15 | 35 | FALSE | UNWEATHERED | 6 | 21 | DRY | 15 | FVRB | FALSE | 77 | 65 | 39 | Good rock |
| FSAN | MW | 75 | 13 | 70 | 14 | 45 | FALSE | UNWEATHERED | 6 | 23 | DRY | 15 | FVRB | FALSE | 78 | 63 | 44 | Good rock |
| FSAN | MW | 75 | 13 | 70 | 14 | 40 | FALSE | UNWEATHERED | 6 | 23 | DRY | 15 | FVRB | FALSE | 70 | 65 | 44 | Good rock |
| FSAN | MW | 75 | 13 | 70 | 14 | 35 | FALSE | UNWEATHERED | 6 | 23 | DRY | 15 | FVRB | FALSE | 70 | 69 | 44 | Good rock |
| FSAN | MW | 75 | 13 | 70 | 14 | 46 | FALSE | UNWEATHERED | 6 | 23 | DRY | 15 | FVRB | FALSE | 75 | 65 | 44 | Good rock |
| FSAN | MW | 75 | 13 | 80 | 16 | 26 | FALSE | UNWEATHERED | 6 | 23 | DRY | 15 | FVRB | FALSE | 65 | 59 | 55 | Good rock |
| FSAN | MW | 75 | 13 | 75 | 15 | 46 | FALSE | UNWEATHERED | 6 | 23 | DRY | 15 | FVRB | FALSE | 70 | 60 | 49 | Good rock |
| FSAN | MW | 75 | 13 | 70 | 14 | 28 | FALSE | UNWEATHERED | 6 | 23 | DRY | 15 | FVRB | FALSE | 75 | 64 | 44 | Good rock |
| FSAN | MW | 75 | 13 | 75 | 15 | 48 | FALSE | UNWEATHERED | 6 | 21 | DRY | 15 | FVRB | FALSE | 77 | 69 | 39 | Good rock |
| FSAN | MW | 75 | 13 | 71 | 14 | 42 | FALSE | UNWEATHERED | 6 | 21 | DRY | 15 | FVRB | FALSE | 76 | 65 | 36 | Good rock |
| FSAN | FR | 175 | 17 | 70 | 14 | 35 | FALSE | UNWEATHERED | 6 | 21 | DRY | 15 | FVRB | FALSE | 86 | 81 | 106 | Very good rock |
| FSAN | FR | 175 | 17 | 70 | 14 | 40 | FALSE | UNWEATHERED | 6 | 21 | DRY | 15 | FVRB | FALSE | 86 | 81 | 106 | Very good rock |
| FSAN | FR | 175 | 17 | 70 | 14 | 42 | FALSE | UNWEATHERED | 6 | 21 | DRY | 15 | FVRB | FALSE | 86 | 81 | 106 | Very good rock |
| FSAN | MW | 75 | 13 | 95 | 19 | 35 | FALSE | UNWEATHERED | 6 | 21 | DRY | 15 | FVRB | FALSE | 69 | 60 | 49 | Good rock |
| FSAN | MW | 75 | 13 | 80 | 16 | 28 | FALSE | UNWEATHERED | 6 | 21 | DRY | 15 | FVRB | FALSE | 79 | 65 | 49 | Good rock |
| FSAN | MW | 75 | 13 | 65 | 13 | 46 | FALSE | UNWEATHERED | 6 | 21 | DRY | 15 | FVRB | FALSE | 70 | 66 | 49 | Good rock |
| FSAN | MW | 75 | 13 | 60 | 12 | 40 | FALSE | UNWEATHERED | 6 | 25 | DRY | 15 | FVRB | FALSE | 69 | 60 | 16 | Good rock |
| FSAN | MW | 75 | 13 | 80 | 16 | 46 | FALSE | UNWEATHERED | 6 | 25 | DRY | 15 | FVRB | FALSE | 73 | 60 | 25 | Good rock |
| FSAN | MW | 75 | 13 | 72 | 14 | 28 | FALSE | UNWEATHERED | 6 | 25 | DRY | 15 | FVRB | FALSE | 77 | 70 | 39 | Good rock |
| FSAN | FR | 75 | 13 | 70 | 14 | 35 | FALSE | UNWEATHERED | 6 | 25 | DRY | 15 | FVRB | FALSE | 79 | 65 | 39 | Good rock |
| FSAN | MW | 175 | 17 | 93 | 19 | 32 | FALSE | UNWEATHERED | 6 | 25 | DRY | 15 | FAIR | -5 | 80 | 48 | 61 | Good rock |
| FSAN | MW | 175 | 17 | 90 | 18 | 32 | FALSE | UNWEATHERED | 6 | 25 | DRY | 15 | FAIR | -5 | 61 | 55 | 61 | Good rock |
| FSAN | MW | 175 | 17 | 75 | 15 | 36 | FALSE | UNWEATHERED | 6 | 25 | DRY | 15 | | | 72 | 60 | 61 | Good rock |
| FSAN | MW | 175 | 17 | 60 | 12 | 32 | FALSE | UNWEATHERED | 6 | 24 | DRY | 15 | FAIR | -5 | 59 | 46 | 61 | Fair rock |
| FSAN | FR | 175 | 17 | 86 | 17 | 32 | 13 | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 70 | 63 | 76 | Good rock |
| FSAN | FR | 175 | 17 | 70 | 14 | 24 | 13 | UNWEATHERED | 6 | 23 | DRY | 15 | FAIR | -5 | 75 | 68 | 95 | Good rock |
| FSAN | FR | 175 | 17 | 65 | 13 | 32 | 13 | UNWEATHERED | 6 | 23 | DRY | 15 | FAIR | -5 | 68 | 55 | 95 | Good rock |
| FSAN | FR | 175 | 17 | 60 | 12 | 40 | 13 | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 65 | 52 | 76 | Good rock |
| FSAN | FR | 175 | 17 | 70 | 14 | 32 | 13 | UNWEATHERED | 6 | 26 | DRY | 15 | | | 70 | 67 | 44 | Good rock |
| FSAN | FR | 175 | 17 | 69 | 14 | 32 | 13 | UNWEATHERED | 6 | 24 | DRY | 15 | FAIR | -5 | 95 | 50 | 31 | Very good rock |
| FSAN | FR | 175 | 17 | 62 | 12 | 24 | 13 | UNWEATHERED | 6 | 25 | DRY | 15 | FAIR | -5 | 70 | 65 | 19 | Good rock |
| FSAN | FR | 175 | 17 | 66 | 13 | 32 | 13 | UNWEATHERED | 6 | 25 | DRY | 15 | FAIR | -5 | 71 | 66 | 21 | Good rock |
| FSAN | FR | 175 | 17 | 38 | 8 | 36 | 13 | UNWEATHERED | 6 | 25 | DRY | 15 | FAIR | -5 | 66 | 61 | 11 | Good rock |
| FSAN | FR | 175 | 17 | 80 | 16 | 32 | 13 | UNWEATHERED | 6 | 25 | DRY | 15 | FAIR | -5 | 74 | 69 | 28 | Good rock |
| FSAN | MW | 175 | 17 | 66 | 13 | 24 | 13 | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 70 | 61 | 36 | Good rock |
| FSAN | MW | 175 | 17 | 90 | 18 | 32 | FALSE | UNWEATHERED | 6 | 20 | DRY | 15 | FAIR | -5 | 68 | 60 | 55 | Good rock |
| FSAN | MW | 175 | 17 | 98 | 20 | 40 | FALSE | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 73 | 68 | 76 | Good rock |
| FSAN | FR | 175 | 17 | 95 | 19 | 32 | FALSE | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 64 | 55 | 68 | Good rock |
| FSAN | FR | 175 | 17 | 97 | 19 | 24 | FALSE | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 75 | 60 | 76 | Good rock |
| FSAN | SW | 75 | 13 | 90 | 18 | 32 | FALSE | UNWEATHERED | 6 | 22 | DRY | 15 | FAIR | -5 | 69 | 56 | 31 | Good rock |
| FSAN | SW | 75 | 13 | 60 | 12 | 36 | FALSE | UNWEATHERED | 6 | 19 | DRY | 15 | FAIR | -5 | 84 | 44 | 16 | Very good rock |
| FSAN | SW | 75 | 13 | 50 | 10 | 32 | FALSE | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 65 | 57 | 9 | Good rock |
| FSAN | SW | 75 | 13 | 50 | 10 | 24 | FALSE | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 76 | 59 | 9 | Good rock |
| FSAN | SW | 75 | 13 | 70 | 14 | 32 | FALSE | UNWEATHERED | 6 | 17 | DRY | 15 | FAIR | -5 | 73 | 68 | 25 | Good rock |
| FSAN | MW | 75 | 13 | 50 | 10 | 32 | FALSE | UNWEATHERED | 6 | 20 | DRY | 15 | FAIR | -5 | 69 | 64 | 16 | Good rock |

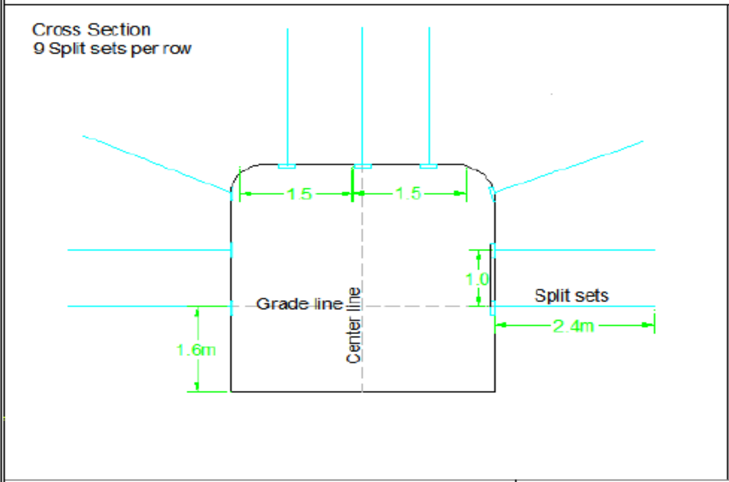
| | | | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| FSAN | FR | 175 | 17 | 80 | 16 | 32 | FALSE | UNWEATHERED | 6 | 22 | DRY | 15 | FAIR | -5 | 79 | 74 | 49 | Good rock |
| FSAN | FR | 175 | 17 | 90 | 18 | 36 | FALSE | UNWEATHERED | 6 | 23 | DRY | 15 | FAIR | -5 | 61 | 56 | 61 | Good rock |
| FSAN | FR | 175 | 17 | 100 | 20 | 32 | FALSE | UNWEATHERED | 6 | 23 | DRY | 15 | FAIR | -5 | 60 | 54 | 76 | Fair rock |
| FSAN | MW | 75 | 13 | 80 | 16 | 24 | FALSE | UNWEATHERED | 6 | 17 | DRY | 15 | FAIR | -5 | 73 | 68 | 25 | Good rock |
| FSAN | MW | 75 | 13 | 90 | 18 | 32 | 14 | UNWEATHERED | 6 | 17 | DRY | 15 | FAIR | -5 | 74 | 69 | 28 | Good rock |
| FSAN | MW | 75 | 13 | 90 | 18 | 40 | 14 | UNWEATHERED | 6 | 17 | DRY | 15 | FAIR | -5 | 75 | 65 | 28 | Good rock |
| FSAN | MW | 75 | 13 | 68 | 14 | 32 | 14 | UNWEATHERED | 6 | 17 | DRY | 15 | FAIR | -5 | 77 | 69 | 28 | Good rock |
| FSAN | FR | 175 | 17 | 60 | 12 | 24 | 14 | UNWEATHERED | 6 | 17 | DRY | 15 | FAIR | -5 | 71 | 65 | 21 | Good rock |
| FSAN | FR | 175 | 17 | 95 | 19 | 32 | 14 | UNWEATHERED | 6 | 17 | DRY | 15 | | | 60 | 52 | 66 | Fair rock |
| FSAN | FR | 175 | 17 | 80 | 16 | 36 | 14 | UNWEATHERED | 6 | 17 | DRY | 15 | FAIR | -5 | 70 | 57 | 68 | Good rock |
| FSAN | FR | 175 | 17 | 76 | 15 | 32 | 14 | UNWEATHERED | 6 | 19 | DRY | 15 | FAIR | -5 | 68 | 55 | 68 | Good rock |
| FSAN | FR | 175 | 17 | 70 | 14 | 32 | 14 | UNWEATHERED | 6 | 23 | DRY | 15 | FAIR | -5 | 80 | 75 | 61 | Good rock |
| FSAN | FR | 175 | 17 | 40 | 8 | 32 | 14 | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 96 | 61 | 12 | Very good rock |
| FSAN | FR | 175 | 17 | 95 | 19 | 40 | 14 | UNWEATHERED | 6 | 22 | DRY | 15 | FAIR | -5 | 90 | 69 | 76 | Very good rock |
| FSAN | FR | 175 | 17 | 80 | 16 | 32 | 14 | UNWEATHERED | 6 | 22 | DRY | 15 | FAIR | -5 | 81 | 56 | 61 | Very good rock |
| FSAN | FR | 175 | 17 | 88 | 18 | 24 | 18 | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 88 | 57 | 76 | Very good rock |
| FSAN | FR | 175 | 17 | 97 | 19 | 32 | 18 | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 75 | 53 | 76 | Good rock |
| FSAN | FR | 175 | 17 | 82 | 16 | 36 | 18 | UNWEATHERED | 6 | 25 | DRY | 15 | FAIR | -5 | 85 | 68 | 41 | Very good rock |
| FSAN | FR | 175 | 17 | 29 | 6 | 32 | 18 | UNWEATHERED | 6 | 25 | DRY | 15 | FAIR | -5 | 94 | 59 | 9 | Very good rock |
| FSAN | FR | 175 | 17 | 0 | 0 | 32 | 18 | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 88 | 53 | 5 | Very good rock |
| FSAN | FR | 175 | 17 | 42 | 8 | 32 | 18 | UNWEATHERED | 6 | 20 | DRY | 15 | FAIR | -5 | 80 | 61 | 12 | Good rock |
| FSAN | FR | 175 | 17 | 34 | 7 | 40 | 18 | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 95 | 60 | 10 | Very good rock |
| FSAN | MW | 75 | 13 | 74 | 15 | 32 | 18 | UNWEATHERED | 6 | 23 | DRY | 15 | FAIR | -5 | 80 | 62 | 13 | Good rock |
| FSAN | MW | 75 | 13 | 92 | 18 | 36 | 18 | UNWEATHERED | 6 | 20 | DRY | 15 | FAIR | -5 | 93 | 68 | 26 | Very good rock |
| FSAN | MW | 75 | 13 | 80 | 16 | 32 | 18 | UNWEATHERED | 6 | 16 | DRY | 15 | FAIR | -5 | 81 | 66 | 20 | Very good rock |
| FSAN | SW | 125 | 17 | 80 | 16 | 30 | 16 | SLIGHT | 5 | 16 | WET | 7 | FAIR | -5 | 90 | 55 | 6 | Very good rock |
| FSAN | SW | 120 | 17 | 65 | 13 | 43 | FALSE | SLIGHT | 5 | 16 | DRY | 15 | FAIR | -5 | 95 | 60 | 10 | Very good rock |
| FSAN | SW | 100 | 17 | 70 | 14 | 40 | FALSE | UNWEATHERED | 6 | 21 | DRY | 15 | FAIR | -5 | 80 | 65 | 18 | Good rock |

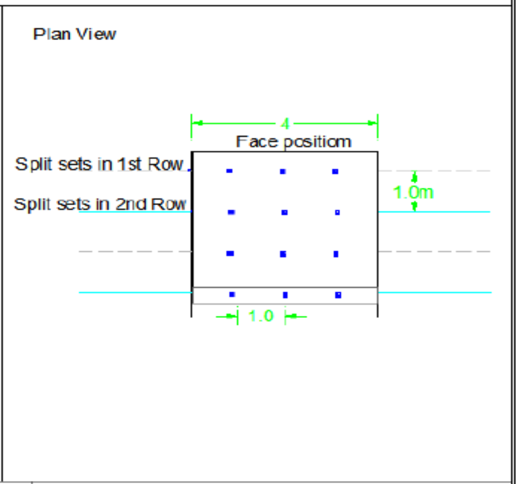
## Rock Mass Classification: HW ARGILLITE

| | | | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| HWA | FR | 75 | 13 | 65 | 13 | 188 | 11 | UNWEATHERED | 6 | 19 | DAMP | 10 | FAIR | -5 | 75 | 50 | 50 | Good rock |
| HWA | FR | 60 | 13 | 70 | 14 | 195 | 22 | UNWEATHERED | 6 | 17 | DAMP | 10 | FAIR | -5 | 70 | 61 | 42 | Good rock |
| HWA | FR | 75 | 13 | 75 | 15 | 220 | 30 | UNWEATHERED | 6 | 19 | DAMP | 10 | FAIR | -5 | 66 | 51 | 40 | Good rock |
| HWA | FR | 60 | 13 | 70 | 14 | 180 | 14 | UNWEATHERED | 6 | 19 | DAMP | 10 | FAIR | -5 | 62 | 57 | 57 | Good rock |
| HWA | FR | 55 | 13 | 70 | 14 | 270 | 22 | UNWEATHERED | 6 | 15 | DAMP | 10 | FAIR | -5 | 65 | 59 | 39 | Good rock |
| HWA | FR | 70 | 13 | 60 | 12 | 190 | 30 | UNWEATHERED | 6 | 15 | DRY | 13 | FAIR | -5 | 72 | 67 | 33 | Good rock |
| HWA | SW | 70 | 13 | 80 | 16 | 32 | 22 | UNWEATHERED | 6 | 19 | DRY | 13 | FAIR | -5 | 62 | 54 | 50 | Good rock |
| HWA | FR | 60 | 22 | 80 | 16 | 180 | 30 | UNWEATHERED | 6 | 20 | DRY | 13 | FAIR | -5 | 77 | 72 | 39 | Good rock |
| HWA | FR | 60 | 22 | 85 | 17 | 32 | 13 | UNWEATHERED | 6 | 19 | DRY | 13 | FAIR | -5 | 75 | 70 | 31 | Good rock |
| HWA | FR | 60 | 19 | 80 | 16 | 190 | 30 | UNWEATHERED | 6 | 20 | DRY | 13 | FAIR | -5 | 62 | 57 | 37 | Good rock |
| HWA | FR | 60 | 23 | 70 | 14 | 32 | 19 | UNWEATHERED | 6 | 16 | DRY | 13 | FAIR | -5 | 67 | 62 | 33 | Good rock |
| HWA | FR | 75 | 18 | 70 | 14 | 180 | 19 | UNWEATHERED | 6 | 14 | DRY | 13 | FAIR | -5 | 67 | 62 | 53 | Good rock |
| HWA | SW | 60 | 15 | 60 | 12 | 195 | 17 | UNWEATHERED | 6 | 17 | DRY | 11 | | | 69 | 64 | 36 | Good rock |
| HWA | FR | 55 | 19 | 90 | 18 | 180 | 30 | UNWEATHERED | 6 | 19 | DRY | 11 | FAIR | -5 | 77 | 72 | 59 | Good rock |
| HWA | FR | 75 | 15 | 90 | 18 | 270 | 30 | UNWEATHERED | 6 | 12 | DRY | 15 | FAIR | -5 | 78 | 70 | 50 | Good rock |
| HWA | FR | 70 | 16 | 80 | 16 | 190 | 30 | UNWEATHERED | 6 | 13 | DRY | 15 | FAIR | -5 | 75 | 60 | 39 | Good rock |

## Appendix B: Specification of support types

# SOB-GSRS1: 4m x 4m FSAN AND BASEMENT - REINFORCEMENT/SUPPORT LAYOUT- 2017




MOPANI

| Designation | Name | Signature | Date |
|---|---|---|---|
| Asst. Rock Eng | | | |
| Rock Engineer | | | |
| Mine Captain | | | |
| U/G Manager | | | |

| Ground Condition Details | |
|---|---|
| Rock Type | FSAN,BQ |
| RMR/Classification | 61-80/ Good |
| Stress Condition | Moderate |
| Risk of FoG | Low |

| Excavation Details | |
|---|---|
| Type | Temporal |
| Size(m) | W=4m H= 4m |
| Purpose | Development opening |

**PRIMARY-SUPPORT**

| Split sets Specification | |
|---|---|
| Name | Splitset |
| Type & quantity | Galvanized /XXX |
| Diameter (mm) | 46mm |
| Length(m) | 2.4m |

| Drill hole Details | |
|---|---|
| Diameter (mm) | 43mm |
| Length(m) | 2.5m |
| Inclination (degrees) | 90 |
| Row Spacing (m) | 1.0m |
| Borehole Spacing (m) | 1.0m |
| Pattern | Square |

**SECONDARY-SUPPORT**

**Cable Bolts Specifications**

N/A

**Reinforcement/ Support Details**

N/A

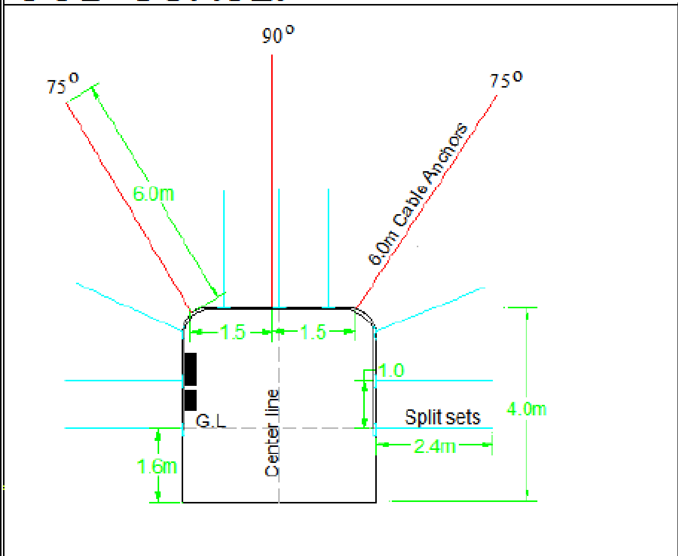
**Welded Mesh Support**

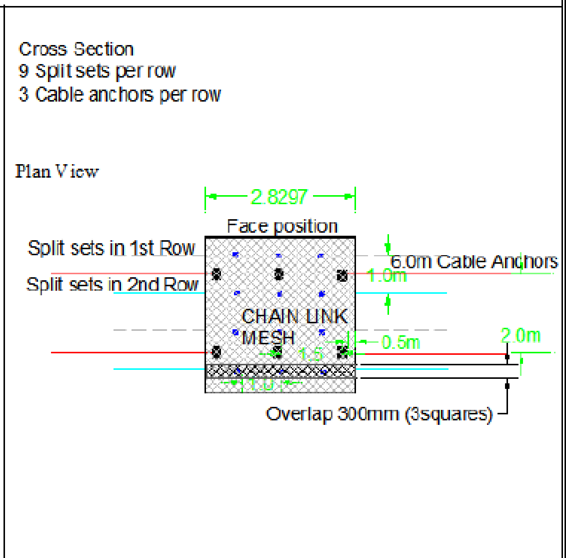
N/A

SECONDARY-SUPPORT WILL BE RECOMMENDED BY ROCK MECHANICS IN THESE FORMATIONS

*Payment for cable anchors is only made upon complete installation*

# SOB-GSRS2: 3510L 1125 XC WEST -REINFORCEMENT/SUPPORT LAYOUT-JULY 2017




MOPANI

| Designation | Name | Signature | Date |
|---|---|---|---|
| Asst. Rock Eng | | | |
| Rock Engineer | | | |
| Mine Captain | | | |
| U/G Manager | | | |

| Ground Condition Details | |
|---|---|
| Rock Type | ORE ZONE |
| RMR/Classification | RMR(61-80) |
| Stress Condition | Moderate |
| Risk of FoG | High |

| Excavation Details | |
|---|---|
| Type | Temporal |
| Size(m) | W=4m H= 4m |
| Purpose | Production opening |

**PRIMARY-SUPPORT**

| Split sets Specification | |
|---|---|
| Name | Splitsets |
| Type & quantity | Galvanized/ 360 |
| Diameter (mm) | 46mm |
| Length(m) | 2.4m |

| Drill hole Details | |
|---|---|
| Diameter (mm) | 43mm |
| Length(m) | 2.5m |
| Inclination (degrees) | 90 |
| Row Spacing (m) | 1.0m |
| Borehole Spacing (m) | 1.0m |
| Pattern | Square |

**SECONDARY-SUPPORT**

| Cable Bolts Specifications | |
|---|---|
| Type | Tensionable |
| Diameter (mm) | 18mm |
| Length (m) | 6.0m |
| Capacity | 38 tons |

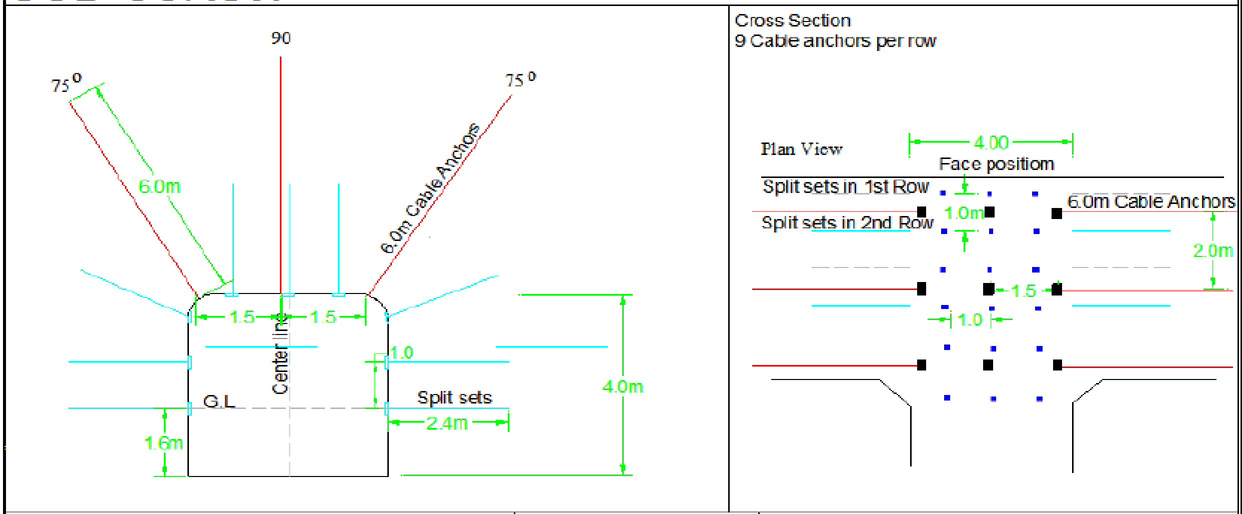
**Welded Mesh Support**

(1)Install mesh over the roof and down to grade line.
(2)Overlap to be 300mm
(3)Pining of Mesh to be done with 0.9m Split sets
(4) No mesh should be installed over a stretch of 10m without pining

| Reinforcement/ Support Details | | | |
|---|---|---|---|
| Borehole Dia. (mm) | 51mm | | |
| Borehole Length(m) | 6.5m | | |
| Borehole ID | 1 | 2 | 3 |
| Inclination (degrees) | 1.5 | Centerl | 1.5 |
| Row Spacing (m) | 2.0 | | |
| Borehole Spacing (m) | 1.5 | | |
| Pattern | Rectangular | | |
| No. of rows | 20 | | |
| Total No. of Anchors | 60 | | |
| Length of Cable Anchors | 6m | | |
| B5 Cement | 120(10Kg bags) | | |

*Payment for cable anchors is only made upon complete installation*

# SOB-GSRS3: 4m x 4m ALL JUNCTIONS - REINFORCEMENT/SUPPORT LAYOUT- 2017




MOPANI

| Designation | Name | Signature | Date |
|---|---|---|---|
| Asst. Rock Eng. | | | |
| Rock Engineer | | | |
| Mine Captain | | | |
| U/G Manager | | | |

| Ground Condition Details | |
|---|---|
| Rock Type | ORE ZONE |
| RMR/Classification | VARIES |
| Stress Condition | Moderate |
| Risk of FoG | High |

| Excavation Details | |
|---|---|
| Type | Temporal |
| Size(m) | W=4m, H= 4m |
| Purpose | Production opening |

**PRIMARY-SUPPORT**

| Split sets Specification | |
|---|---|
| Name | Splitsets |
| Type & quantity | Galvanized / XXX |
| Diameter (mm) | 46mm |
| Length(m) | 2.4m |

| Drill hole Details | |
|---|---|
| Diameter (mm) | 43mm |
| Length(m) | 2.5m |
| Inclination (degrees) | 90 |
| Row Spacing (m) | 1.0m |
| Borehole Spacing (m) | 1.0m |
| Pattern | Square |

**SECONDARY-SUPPORT**

| Cable Bolts Specifications | |
|---|---|
| Type | Tensionable |
| Diameter (mm) | 18mm |
| Length (m) | 6.0m |
| Capacity | 38 tons |

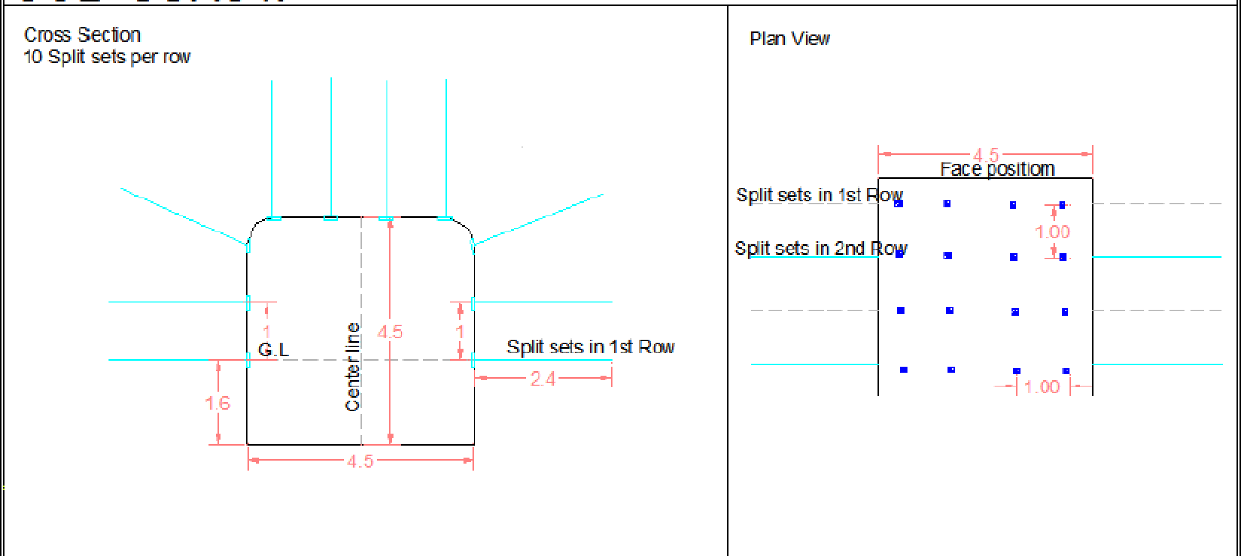
Welded Mesh Support

N/A

| Reinforcement/ Support Details | | | |
|---|---|---|---|
| Borehole dia. (mm) | 51mm | | |
| Borehole Length(m) | 6.5m | | |
| Borehole ID | 1 | 2 | 3 |
| Inclination (degrees) | 1.5 | Center | 1.5 |
| Row Spacing (m) | 2.0 | | |
| Borehole Spacing (m) | 1.5 | | |
| Pattern | Rectangular | | |
| No. of rows | 3 | | |
| Total No. of Anchors | 9 | | |
| Length of Cable Anchor | 6m | | |
| B5 Cement | 18(10Kg bags) | | |

Payment for cable anchors is only made upon complete installation

# SOB-GSRS4: 4.5m x 4.5m FSAN AND BASEMENT - REINFORCEMENT/SUPPORT LAYOUT- 2017




MOPANI

| Designation | Name | Signature | Date |
|---|---|---|---|
| Asst. Rock Eng. | | | |
| Rock Engineer | | | |
| Mine Captain | | | |
| U/G Manager | | | |

| Ground Condition Details | |
|---|---|
| Rock Type | FSAN,BQ |
| RMR/Classification | 61-81 Good |
| Stress Condition | Moderate |
| Risk of FoG | Low |

| Excavation Details | |
|---|---|
| Type | Temporal |
| Size(m) | W=4.5m, H= 4.5m |
| Purpose | Development opening |

**PRIMARY-SUPPORT**

| Split sets Specification | |
|---|---|
| Name | Splitsets |
| Type & quantity | Galvanized / XXX |
| Diameter (mm) | 46mm |
| Length(m) | 2.4m |

| Drillhole Details | |
|---|---|
| Diameter (mm) | 43mm |
| Length(m) | 2.5m |
| Inclination (degrees) | 90 |
| Row Spacing (m) | 1.0m |
| Borehole Spacing (m) | 1.0m |
| Pattern | Square |

**SECONDARY-SUPPORT**

Cable Bolts Specifications

N/A

Welded Mesh Support

N/A

Reinforcement/ Support Details

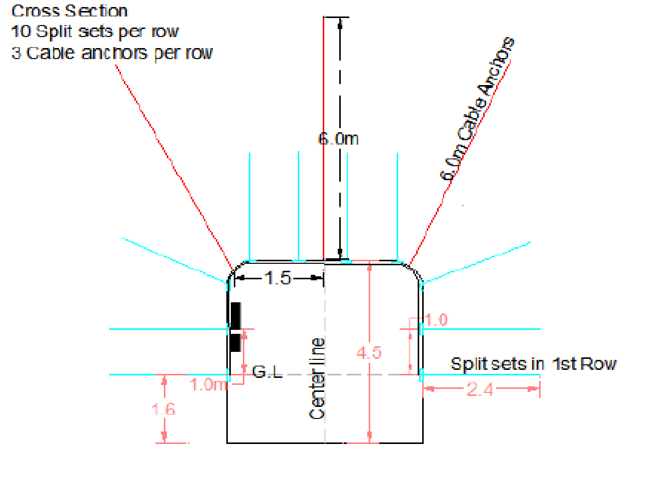
N/A

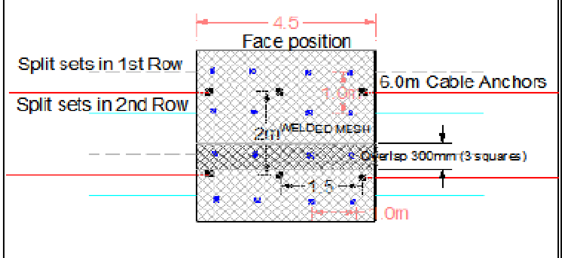
SECONDARY-SUPPORT WILL BE RECOMMENDED BY ROCK MECHANICS IN THESE FORMATIONS

Payment for cable anchors is only made upon complete installation

# SOB-GSRS5: 4.5m x 4.5m - ORE ZONE & LCONG -REINFORCEMENT/SUPPORT LAYOUT-JUNE 2017

Cross Section
10 Split sets per row
3 Cable anchors per row




| Designation | Name | Signature | Date |
|---|---|---|---|
| Asst. Rock Eng. | | | |
| Rock Engineer | | | |
| Mine Captain | | | |
| U/G Manager | | | |

MOPANI

| Ground Condition Details | |
|---|---|
| Rock Type | ORE ZONE,LCONG |
| RMR/Classification | RMR(61-80) |
| Stress Condition | Moderate |
| Risk of FoG | High |

| Excavation Details | |
|---|---|
| Type | Temporal |
| Size(m) | W=4m, H= 4m |
| Purpose | Production opening |

**PRIMARY-SUPPORT**

| Split sets Specification | |
|---|---|
| Name | Splitsets |
| Type & quantity | Galvanized/ XXX |
| Diameter (mm) | 46mm |
| Length(m) | 2.4m |

| Drill hole Details | |
|---|---|
| Diameter (mm) | 43mm |
| Length(m) | 2.5m |
| Inclination (degrees) | 90 |
| Row Spacing (m) | 1.0m |
| Borehole Spacing (m) | 1.0m |
| Pattern | Sqaure |

**SECONDARY-SUPPORT**

| Cable Bolts Specifications | |
|---|---|
| Type | Tensionable |
| Diameter (mm) | 18mm |
| Length (m) | 6.0m |
| Capacity | 38 tons |

**Welded Mesh Support**

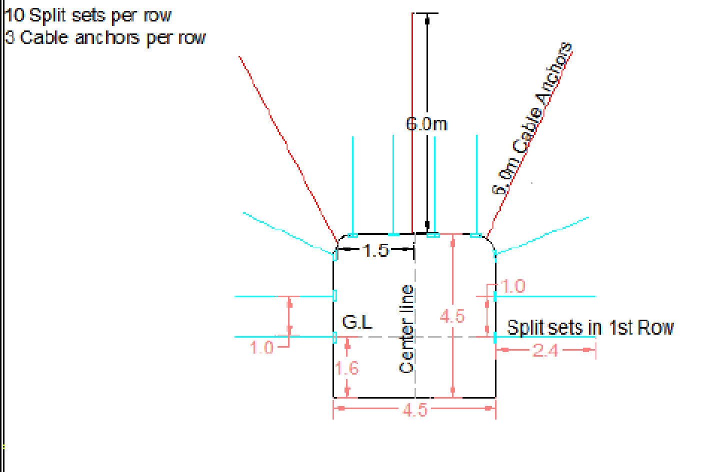
(1)Install welded mesh over the roof and down to grade line.
(2)overlap to be 300mm
(3)Pining of Welded Mesh to be done with 0.9m Split sets
(4) No welded mesh should be installed over a stretch of 1 0m without pining

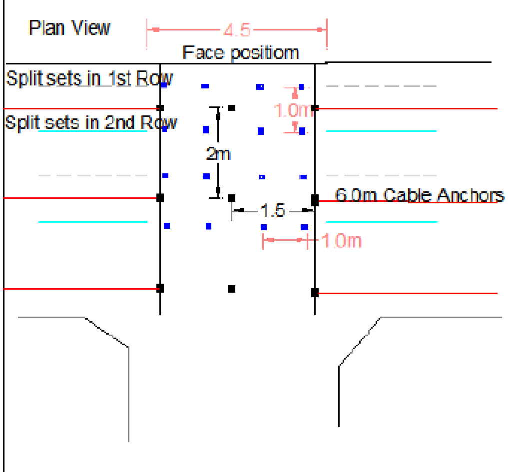
| Reinforcement/ Support Details | | | |
|---|---|---|---|
| Borehole Diameter (mm) | 51mm | | |
| Borehole Length(m) | 6.5m | | |
| Borehole ID | 1 | 2 | 3 |
| Inclination (degrees) | 1.5m | Center | 1.5m |
| Row Spacing (m) | 1.0 | | |
| Borehole Spacing (m) | 1.5 | | |
| Pattern | Rectangular | | |
| No. of rows | XXX | | |
| Total No. of Anchors | XXX | | |
| Length of Cable Anchors | 6m | | |
| B5 Cement | XXX(10Kg bags) | | |

*Payment for cable anchors is only made upon complete installation*

# SOB-GSRS6: 4.5m x 4.5m ALL JUNCTIONS - REINFORCEMENT/SUPPORT LAYOUT- 2017

Cross Section
10 Split sets per row
3 Cable anchors per row




| Designation | Name | Signature | Date |
|---|---|---|---|
| Asst. Rock Eng. | | | |
| Rock Engineer | | | |
| Mine Captain | | | |
| U/G Manager | | | |

MOPANI

| Ground Condition Details | |
|---|---|
| Rock Type | ORE ZONE |
| RMR/Classification | VARIES |
| Stress Condition | Moderate |
| Risk of FoG | High |

| Excavation Details | |
|---|---|
| Type | Temporal |
| Size(m) | W=4m, H= 4m |
| Purpose | Production opening |

**PRIMARY-SUPPORT**

| Split sets Specification | |
|---|---|
| Name | Splitsets |
| Type & quantity | Galvanized / XXX |
| Diameter (mm) | 46mm |
| Length(m) | 2.4m |

| Drill hole Details | |
|---|---|
| Diameter (mm) | 45mm |
| Length(m) | 2.5m |
| Inclination (degrees) | 90 |
| Row Spacing (m) | 1.0m |
| Borehole Spacing (m) | 1.0m |
| Pattern | Sqaure |

**SECONDARY-SUPPORT**

| Cable Bolts Specifications | |
|---|---|
| Type | Tensionable |
| Diameter (mm) | 18mm |
| Length (m) | 6.0m |
| Capacity. | 58 tons |

**Welded Mesh Support**

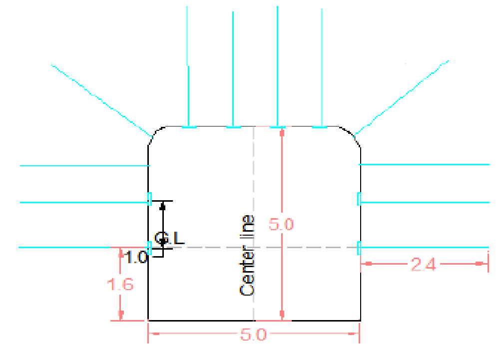
N/A

| Reinforcement/ Support Details | | | |
|---|---|---|---|
| Borehole dia. (mm) | 51mm | | |
| Borehole Length(m) | 6.5m | | |
| Borehole ID | 1 | 2 | 3 |
| Inclination (degrees) | 1.5 | Center | 1.5 |
| Row Spacing (m) | 2.0 | | |
| Borehole Spacing (m) | 1.5 | | |
| Pattern | Rectangular | | |
| No. of rows | 3 | | |
| Total No. of Anchors | 9 | | |
| Length of cable anchor | 6m | | |
| B5 Cement | 18(10Kg bags) | | |

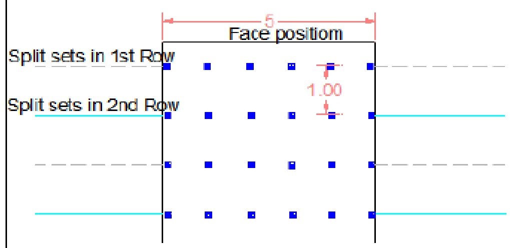
*Payment for cable anchors is only made upon complete installation*

# SOB-GSRS7: 5m x 5m FSAN AND BASEMENT - REINFORCEMENT/SUPPORT LAYOUT - 2017

Cross Section
12 Split sets per row




Plan View




MOPANI

| Designation | Name | Signature | Date |
|---|---|---|---|
| Asst. Rock Eng. | | | |
| Rock Engineer | | | |
| Mine Captain | | | |
| U/G Manager | | | |

| Ground Condition Details | |
|---|---|
| Rock Type | FSAN,BQ |
| RMR/Classification | 61-81 Good |
| Stress Condition | Moderate |
| Risk of FoG | Low |

| Excavation Details | |
|---|---|
| Type | Temporal |
| Size(m) | W=5m, H= 5m |
| Purpose | Development opening |

**PRIMARY-SUPPORT**

| Split sets Specification | |
|---|---|
| Name | Splitsets |
| Type & quantity | Galvanized / XXX |
| Diameter (mm) | 46mm |
| Length(m) | 2.4m |

| Drillhole Details | |
|---|---|
| Diameter (mm) | 43mm |
| Length(m) | 2.5m |
| Inclination (degrees) | 90 |
| Row Spacing (m) | 1.0m |
| Borehole Spacing (m) | 1.0m |
| Pattern | Square |

**SECONDARY-SUPPORT**

Cable Bolts Specifications

N/A

Welded Mesh Support

N/A

Reinforcement/ Support Details

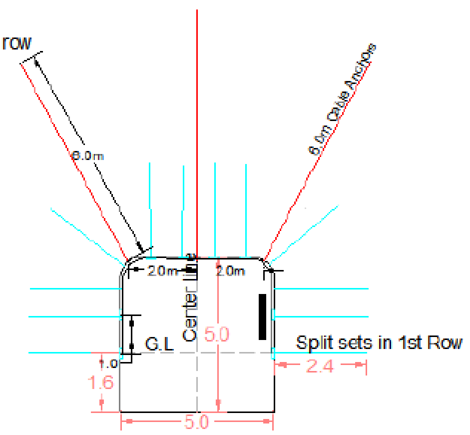
N/A

SECONDARY-SUPPORT WILL BE RECOMMENDED BY ROCK MECHANICS IN THESE FORMATIONS

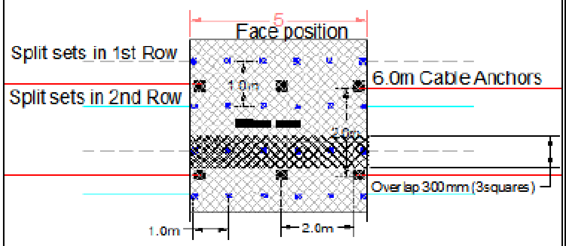
*Payment for cable anchors is only made upon complete installation*

# SOB-GSRS8: 4.5m x 4.5m - ORE ZONE & LCONG -REINFORCEMENT/SUPPORT LAYOUT - 2017

Cross Section
12 Split sets per row
3 Cable anchors per row




Plan View




MOPANI

| Designation | Name | Signature | Date |
|---|---|---|---|
| Asst. Rock Eng. | | | |
| Rock Engineer | | | |
| Mine Captain | | | |
| U/G Manager | | | |

| Ground Condition Details | |
|---|---|
| Rock Type | ORE ZONE,LCONG |
| RMR/Classification | RMR(61-80) |
| Stress Condition | Moderate |
| Risk of FoG | High |

| Excavation Details | |
|---|---|
| Type | Temporal |
| Size(m) | W=5m, H= 5m |
| Purpose | Production opening |

**PRIMARY-SUPPORT**

| Split sets Specification | |
|---|---|
| Name | Splitsets |
| Type & quantity | Galvanized/XXX |
| Diameter (mm) | 46mm |
| Length(m) | 2.4m |

| Drill hole Details | |
|---|---|
| Diameter (mm) | 43mm |
| Length(m) | 2.5m |
| Inclination (degrees) | 90 |
| Row Spacing (m) | 1.0m |
| Borehole Spacing (m) | 1.0m |
| Pattern | Square |

**SECONDARY-SUPPORT**

| Cable Bolts Specifications | |
|---|---|
| | Tensionable |
| | 18mm |
| | 6.0m |
| | 38 tons |

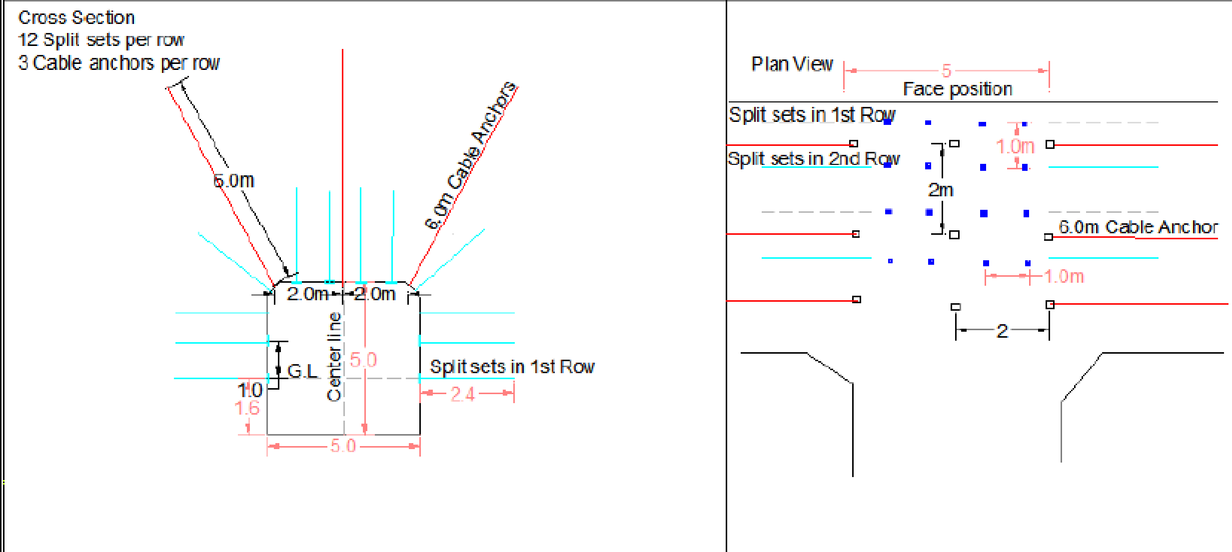
Welded Mesh Support

(1) Install welded mesh over the roof and down to grade line.
(2) overlap to be 300mm
(3) Pining of Welded Mesh to be done with 0.9m Split sets
(4) No welded mesh should be installed over a stretch of 10m without pining

| Reinforcement/ Support Details | | | |
|---|---|---|---|
| Borehole dia (mm) | 51mm | | |
| Borehole Length(m) | 6.5m | | |
| Borehole ID | 1 | 2 | 3 |
| Inclination (degrees) | 2m | Center | 2m |
| Row Spacing (m) | 2.0 | | |
| Borehole Spacing (m) | 2.0 | | |
| Pattern | Rectangular | | |
| No. of rows | xxx | | |
| Total No. of anchors | xxx | | |
| Length of cable anchor | 6m | | |
| E5 Cement | XX (10Kg bag) | | |

*Payment for cable anchors is only made upon complete installation*

# SOB-GSRS9: 5m x 5m ALL JUNCTIONS - REINFORCEMENT/SUPPORT LAYOUT- 2017




| Designation | Name | Signature | Date |
|---|---|---|---|
| Asst. Rock Eng | | | |
| Rock Engineer | | | |
| Mine Captain | | | |
| U/G Manager | | | |

MOPANI

| Ground Condition Details | |
|---|---|
| Rock Type | ORE ZONE |
| RMR/Classification | VARIES |
| Stress Condition | Moderate |
| Risk of FoG | High |

| Excavation Details | |
|---|---|
| Type | Tempord |
| Size(m) | W=5m<br>H= 5m |
| Purpose | ALL |

## PRIMARY-SUPPORT

| Split sets Specification | |
|---|---|
| Name | Splitsets |
| Type & quantity | Galvanized / XXX |
| Diameter (mm) | 46mm |
| Length(m) | 2.4m |

| Drill hole Details | |
|---|---|
| Diameter (mm) | 43mm |
| Length(m) | 2.5m |
| Inclination (degrees) | 90 |
| Row Spacing (m) | 1.0m |
| Borehole Spacing (m) | 1.0m |
| Pattern | Square |

## SECONDARY-SUPPORT

| Cable Bolts Specifications | |
|---|---|
| Type | Tensionable |
| Diameter (mm) | 18mm |
| Length (m) | 6.0m |
| Capacity | 38 tons |

**Welded Mesh Support**

N/A

| Reinforcement/ Support Details | | | |
|---|---|---|---|
| Borehole dia (mm) | 51mm | | |
| Borehole Length(m) | 6.5m | | |
| Borehole ID | 1 | 2 | 3 |
| Inclination (degrees) | 2m | Center | 2m |
| Row Spacing (m) | 2.0 | | |
| Borehole Spacing (m) | 2.0 | | |
| Pattern | Rectangular | | |
| No. of rows | 3 | | |
| Total No. of anchors | 9 | | |
| Length of cable anchor | 6m | | |
| B5 Cement | 18(10Kg bags) | | |

*Payment for cable anchors is only made upon completa installation*

## Appendix C: Nkana Rock Mechanics Database

Structural Mapping Database – Excavations mapped up to June 2020

MOPANI COPPER MINES PLC-NKANA SOUTH SOB
ROCK MECHANICS SECTION
ROCK MASS SCANLINE UNDERGROUND MAPPING-DATA BASE

| Date | Level | Ref peg | Stretch from Peg | Excavation Name | Sidewall | Fracture Frequency | ROCK UCS | Rock Type | RMR (Tunnel) (Bieniawski'89) | Q (Tunnel) | GSI (Hoek'95) | RQD | Mapped by |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 7/31/2017 | 3510L | | 0-10m | 1140 OBD NRTH | West | 9.667 | R3 | SOBS | 65 | | 62 | 67.0 | PL/DM |
| 7/31/2017 | 3510L | | 0-10m | 1155 OBD NRTH | West | 9.167 | R4 | SOBS | 69 | | 66 | 70.0 | PL/DM |
| 7/31/2017 | 3510L | | 0-10m | 1170 OBD NRTH | West | 9.333 | R4 | SOBS | 69 | | 66 | 69.0 | PL/DM |
| 7/31/2017 | 3510L | | 0-10m | 1185 OBD NRTH | West | 8.833 | R4 | SOBS | 71 | | 68 | 71.0 | PL/DM |
| 10/5/2018 | 3435L | T3744 | 0-10m | 1070 acc to B FOLD | | | R4 | SOBS | 58 | | 55 | 56.0 | DM/FS |
| 10/4/2019 | 3960L | T5835 | 0-30 | 660 Acces to 660 LDR | West | 7 | R4 | BSAN | 63 | | 58 | 69.0 | DM/ES |
| 10/4/2019 | 3960L | T5835 | 30-60 | 660 Acces to 660 LDR | West | 12 | R4 | BSAN | 60 | | 55 | 50.0 | DM/ES |
| 10/4/2019 | 3960L | T5835 | 60-90 | 660 Acces to 660 LDR | West | 22 | R3 | BSAN | 16 | | 0 | 0.0 | DM/ES |
| 22/6/19 | 3510L | T6854 | 0-15 | 1280 X/Cut | West | 11 | R3 | SOBS | 55 | | 50 | 64.0 | DM/FS |
| 22/6/19 | 3510L | T6854 | 15-30 | 1281 X/Cut | East | 7 | R3 | SOBS | 56 | | 51 | 52.0 | DM/FS |
| 22/6/19 | 3510L | T6854 | 30-45 | 1282 X/Cut | West | 6 | R3 | SOBS | 58 | | 53 | 48.0 | DM/FS |
| 22/6/19 | 3510L | T6854 | 45-60 | 1282 X/Cut | East | 8 | R3 | SOBS | 57 | | 52 | 55.0 | DM/FS |
| 20/06/20 | 3325L | | 0-25 | 1330 X/Cut | West | 7 | R3 | H/W | 54 | | 49 | 67.0 | KS |

UCS from Point Load database - last updated in December 2022

UCS From Point Load Test_Data Base - Excel

Mopani Copper Mines - 3360L 1125mN Probing Hole Project - Point Load Test Results

Point Load Test Machine Type: Model: Version:

| Bore Hole ID | Rock Formation | Weathering Grade | From | To | Sample Length L (mm) | Loading Direction (Axial/Diam) | Breaking Load P (kN) | De mm | $De^2$ $mm^2$ | Is (KN) | F | K | Is(50) Mpa | UCS MPa | Average UCS MPa | Remarks |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| NSP048 | SOBS | FR | | | 0.340 | Diametral | 13.2 | 48 | 2304 | 0.0057 | 0.9818 | 0.0004 | 5.62 | 126 | | Good break |
| NSP048 | SOBS | FR | | | 0.105 | Diametral | 3.4 | 48 | 2304 | 0.0015 | 0.9818 | 0.0004 | 1.45 | 32 | | Very strong. |
| NSP048 | SOBS | FR | | | 0.140 | Diametral | 17.2 | 48 | 2304 | 0.0075 | 0.9818 | 0.0004 | 7.33 | 164 | | Good break |
| NSP048 | SOBS | FR | | | 0.100 | Diametral | 15.2 | 48 | 2304 | 0.0066 | 0.9818 | 0.0004 | 6.48 | 145 | | Good break |
| NSP048 | SOBS | FR | | | 0.105 | Diametral | 18.1 | 48 | 2304 | 0.0079 | 0.9818 | 0.0004 | 7.71 | 173 | | Good break |
| NSP048 | SOBS | FR | | | 0.100 | Diametral | 14.2 | 48 | 2304 | 0.0062 | 0.9818 | 0.0004 | 6.05 | 136 | | Good break |
| NSP048 | SOBS | FR | | | 0.140 | Diametral | 8.2 | 48 | 2304 | 0.0036 | 0.9818 | 0.0004 | 3.49 | 78 | | Good break |
| NSP048 | SOBS | FR | | | 0.125 | Diametral | 12.3 | 48 | 2304 | 0.0053 | 0.9818 | 0.0004 | 5.24 | 117 | | Good break |
| NSP048 | SOBS | FR | | | 0.120 | Diametral | 9.8 | 48 | 2304 | 0.0043 | 0.9818 | 0.0004 | 4.18 | 94 | | Good break |
| NSP048 | SOBS | FR | | | 0.140 | Diametral | 12.3 | 48 | 2304 | 0.0053 | 0.9818 | 0.0004 | 5.24 | 117 | | Good break |
| NSP048 | SOBS | FR | | | 0.100 | Diametral | 9.3 | 48 | 2304 | 0.0040 | 0.9818 | 0.0004 | 3.96 | 89 | | Good break |
| NSP048 | SOBS | FR | | | 0.150 | Diametral | 11.5 | 48 | 2304 | 0.0050 | 0.9818 | 0.0004 | 4.90 | 110 | 115 | Good break |
| NSP048 | SOBS | FR | | | 0.125 | Diametral | 15.3 | 48 | 2304 | 0.0066 | 0.9818 | 0.0004 | 6.52 | 146 | | Very strong. |
| NSP048 | SOBS | FR | | | 0.135 | Diametral | 13.8 | 48 | 2304 | 0.0060 | 0.9818 | 0.0004 | 5.88 | 132 | | Very strong. |
| NSP048 | SOBS | FR | | | 0.140 | Diametral | 6.3 | 48 | 2304 | 0.0027 | 0.9818 | 0.0004 | 2.68 | 60 | | Very strong. |
| NSP048 | SOBS | FR | | | 0.110 | Diametral | 6.1 | 48 | 2304 | 0.0026 | 0.9818 | 0.0004 | 2.60 | 58 | | Good break |
| NSP048 | SOBS | FR | | | 0.100 | Diametral | 11.8 | 48 | 2304 | 0.0051 | 0.9818 | 0.0004 | 5.03 | 113 | | Good break |
| NSP048 | SOBS | FR | | | 0.120 | Diametral | 6.3 | 48 | 2304 | 0.0027 | 0.9818 | 0.0004 | 2.68 | 60 | | Good break |
| NSP048 | COS | FR | | | 0.100 | Diametral | 15.3 | 48 | 2304 | 0.0066 | 0.9818 | 0.0004 | 6.52 | 146 | | Good break |
| NSP048 | COS | FR | | | 0.115 | Diametral | 10.4 | 48 | 2304 | 0.0045 | 0.9818 | 0.0004 | 4.43 | 99 | | Good break |
| NSP048 | SOBS | FR | | | 0.130 | Diametral | 5.60 | 48 | 2304 | 0.0024 | 0.9818 | 0.0004 | 2.39 | 53 | | Good break |
| NSP048 | SOBS | FR | | | 0.080 | Diametral | 10.20 | 48 | 2304 | 0.0044 | 0.9818 | 0.0004 | 4.35 | 97 | | Good break |
| NSP048 | SOBS | FR | | | 0.090 | Diametral | 4.30 | 48 | 2304 | 0.0019 | 0.9818 | 0.0004 | 1.83 | 41 | | Good break |

Template | NSP0039_38 | NSP0526 | NS0540 | NSP047 | NSP048 | NSP055 | NSP056 | NSP057 | NSX0093 | Sheet2

Point Load Test Results for samples up to 2008

MOPANI COPPER MINES PLC- SOB SHAFT-ROCK MECHANICS POINT LOAD TEST-[ISRM 1985:ASTM 2008]

| Sample ID | Lithology | Weathering | Plan Sample Diameter (mm) | Plan Sample Length (mm) | Actual Sample Diameter (mm) | Actual Sample Length (mm) | % Sample preparation acuracy | Test Type | Load at Failure (kN) | Strength Index (MPa) | UCS (MPa) | Discription |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 10 | LCONG | Fresh | 47 | 70 | 47.70 | 69 | 99.97% | Diametric | 4.2 | 1.84 | 44.24 | |
| 11 | LCONG | Fresh | 47 | 70 | 48.00 | 70.6 | 98.51% | Diametric | 3.9 | 1.71 | 41.08 | |
| 12 | LCONG | Fresh | 47 | 70 | 47.70 | 69 | 99.97% | Diametric | 1.6 | 0.70 | 16.85 | |
| 13 | LCONG | Fresh | 47 | 70 | 47.70 | 72 | 97.83% | Diametric | 2.2 | 0.97 | 23.17 | |
| 14 | LCONG | Fresh | 47 | 70 | 48.00 | 72 | 97.51% | Diametric | 1.2 | 0.53 | 12.64 | |
| 15 | LCONG | Fresh | 47 | 70 | 48.00 | 70 | 98.94% | Diametric | 3.4 | 1.49 | 35.81 | |
| 16 | LCONG | Fresh | 47 | 70 | 48.00 | 69 | 99.65% | Diametric | 7.5 | 3.29 | 79.00 | |
| 17 | SOB SHALE | Fresh | 47 | 70 | 48.00 | 70 | 98.94% | Diametric | 2.1 | 0.92 | 22.12 | |
| 18 | SOB SHALE | Fresh | 47 | 70 | 48.00 | 69.8 | 99.08% | Diametric | 9.4 | 4.13 | 99.02 | |
| 19 | SOB SHALE | Fresh | 47 | 70 | 47.00 | 71 | 99.29% | Diametric | 8.9 | 3.91 | 93.75 | |
| 20 | SOB SHALE | Fresh | 47 | 70 | 47.70 | 70.2 | 99.11% | Diametric | 5.7 | 2.50 | 60.04 | |
| 21 | SOB SHALE | Fresh | 47 | 70 | 47.00 | 73 | 97.86% | Diametric | 9.2 | 4.04 | 96.91 | |
| 22 | SOB SHALE | Fresh | 47 | 70 | 47.70 | 70.6 | 98.83% | Diametric | 9.5 | 4.17 | 100.07 | |
| 23 | SOB SHALE | Fresh | 47 | 70 | 47.70 | 70 | 99.26% | Diametric | 5.8 | 2.55 | 61.10 | |
| 24 | SOB SHALE | Fresh | 47 | 70 | 48.00 | 70 | 98.94% | Diametric | 5.3 | 2.33 | 55.83 | |
| 25 | SOB SHALE | Fresh | 47 | 70 | 47.70 | 71 | 98.54% | Diametric | 7.3 | 3.20 | 76.90 | |
| 26 | SOB SHALE | Fresh | 47 | 70 | 48.00 | 70 | 98.94% | Diametric | 8.4 | 3.69 | 88.48 | |
| 27 | SOB SHALE | Fresh | 47 | 70 | 47.70 | 70.3 | 99.04% | Diametric | 6.2 | 2.72 | 65.31 | |
| 28 | SOB SHALE | Fresh | 47 | 70 | 48.00 | 70.8 | 98.36% | Diametric | 5.9 | 2.59 | 62.15 | |
| 29 | SOB SHALE | Fresh | 47 | 70 | 47.70 | 70 | 99.26% | Diametric | 4.5 | 1.98 | 47.40 | |
| 30 | SOB SHALE | Fresh | 47 | 70 | 47.70 | 70 | 99.26% | Diametric | 4.3 | 1.89 | 45.29 | |
| 31 | SOB SHALE | Fresh | 47 | 70 | 47.70 | 70 | 99.26% | Diametric | 8.3 | 3.64 | 87.43 | |
| 32 | SOB SHALE | Fresh | 47 | 70 | 47.70 | 70 | 99.26% | Diametric | 9.1 | 3.99 | 95.86 | |
| 33 | LCONG | Fresh | 47 | 70 | 47.7 | 70 | 99.26% | Diametric | 5.16 | 2.26 | 54.35 | |
| 34 | LCONG | Fresh | 47 | 70 | 47.6 | 70 | 99.36% | Diametric | 5.06 | 2.22 | 53.30 | |
| 35 | FSAN | Fresh | 47 | 70 | 47.5 | 70 | 99.47% | Diametric | 2.96 | 1.30 | 31.18 | |
| 36 | FSAN | Fresh | 47 | 70 | 47.6 | 95 | 81.50% | Diametric | 7.76 | 3.41 | 81.74 | |
| 37 | FSAN | Fresh | 47 | 70 | 48 | 95 | 81.08% | Diametric | 6.06 | 2.66 | 63.83 | |
| 38 | FSAN | Fresh | 47 | 70 | 47.6 | 95 | 81.50% | Diametric | 7.06 | 3.10 | 74.37 | |
| 39 | FSAN | Fresh | 47 | 70 | 46.8 | 95 | 82.36% | Diametric | 6.06 | 2.66 | 63.83 | |
| 40 | FSAN | Fresh | 47 | 70 | 47.6 | 95 | 81.50% | Diametric | 7.76 | 3.41 | 81.74 | |
| 41 | FSAN | Fresh | 47 | 70 | 47.4 | 95 | 81.72% | Diametric | 7.36 | 3.23 | 77.53 | |

UCS FIELD Analysis Sheet3

## Appendix D: Compressive Strength testing of concrete cubes

ALFRED KNIGHT

ALFRED H KNIGHT
MATERIALS EVALUATION DEPARTMENT
COMPRESSIVE STRENGTH OF CONCRETE CUBES

| | | | | | |
|---|---|---|---|---|---|
| Client Name & Address | : | CENTRAL SHAFT, KITWE | Issue Date | : | 5th March, 2023 |
| Report No. | : | ME- | | | |
| Order Ref. No. | : | Customer Reference No. S50842 | Page | : | 1 of 1 |

Report Title : COMPRESSIVE STRENGTH TESTING OF 08 CONCRETE CUBES

| | | | | | |
|---|---|---|---|---|---|
| Date Received | : | 27th February, 2023 | Concrete Mix Design | : | - |
| Received from | : | Henry Lundu | Required Grade | : | - |
| | | | Cement Type | : | - |
| Received By | : | Widan Kapata | Additional information | : | Mopani Copper Mines Plc |

| Sample No. | Date Cast | Test Date | Age (Days) | Checked Nominal Size of Cube (mm x mm) | Saturated | | | Compressive Strength (N/mm²) | Failure Type |
|---|---|---|---|---|---|---|---|---|---|
| | | | | | Wt of Cube (g) In air | Density (kg/m³) | Ultimate Load (kN) | | |
| FREKAM CUBE 1 | 26/01/23 | 26/01/23 | 32 | 150 x 150 | 7424 | 2199 | 276.5 | 12.2 | A |

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| **FREKAM CUBE 2** | 26/01/23 | 27/02/23 | 32 | 150 x 150 | 7322 | 2169 | 363.2 | 16.1 | A |
| **FREKAM CUBE 3** | 02/02/23 | 04/03/23 | 30 | 150 x 150 | 7432 | 2202 | 348.5 | 15.4 | A |
| **FREKAM CUBE 4** | 02/02/23 | 04/03/23 | 30 | 150 x 150 | 7244 | 2146 | 434.3 | 19.3 | A |
| **CARMINE CUBE 1** | 03/02/23 | 04/03/23 | 29 | 150 x 150 | 7784 | 2306 | 326.0 | 14.0 | A |
| **CARMINE CUBE 2** | 04/02/23 | 04/03/23 | 29 | 150 x 150 | 8383 | 2484 | 333.2 | 14.8 | A |
| **CARMINE CUBE 3** | 20/02/23 | 04/03/23 | 12 | 150 x 150 | 7641 | 2264 | 274.9 | 12.2 | A |
| **CARMINE CUBE 4** | 20/02/23 | 04/03/23 | 12 | 150 x 150 | 7866 | 2331 | 324.9 | 14.4 | A |

**Type of Failure:**

Normal: Abnormal:


Remarks on Receipt : All samples had fibre reinforcement

Remarks on Test : These results conclude the report

**A. Chandawe**
**Test Engineer**

**V. Chongo**
**Head – Materials Evaluation**

**For and on behalf of Alfred H. Knight (Zambia) Limited**



**Alfred H. Knight (Zambia) Limited**
Corner Mindola/Golf Club Roads, Nkana West,
P O Box 20303, Kitwe, Copperbelt, Zambia.
Tel: +260 212 226433/4 Fax: +260 212 226306/226424
Email: ahk.zambia@ahkgroup.com

Company Reg No. 41960 VAT No. 10126160-33

BRITTATEC

Report Title : **COMPRESSIVE STRENGTH TESTING OF 04 CONCRETE CUBES**

| | | |
|---|---|---|
| Date Received : **14th July, 2023** | Concrete Mix Design | : - |
| Received from : Brittatec | Required Grade | : - |
| | Cement Type | : - |
| Received By : Widan Kapata | Additional information | : MCM Central |

| Sample No. | Date Cast | Test Date | Age (Days) | Checked Nominal Size of Cube (mm x mm) | Saturated | | | Compressive Strength (N/mm²) | Failure Type |
|---|---|---|---|---|---|---|---|---|---|
| | | | | | Wt of Cube (g) In air | Density (kg/m³) | Ultimate Load (kN) | | |
| **CONCRETE CUBE 1** | 16/07/23 | 16/07/23 | 28 | 150x150 | 7454 | 2209 | 324.0 | 14.4 | A |
| **CONCRETE CUBE 2** | 16/07/23 | 16/08/23 | 28 | 150x150 | 7314 | 2167 | 323.9 | 14.3 | A |
| **CONCRETE CUBE 3** | 16/07/23 | 16/07/23 | 28 | 150x150 | 7454 | 2209 | 552.8 | 26.5 | A |
| **CONCRETE CUBE 4** | 16/07/23 | 16/07/23 | 28 | 150x150 | 7366 | 2183 | 563.8 | 25.1 | A |

**Type of Failure:**

Normal: Abnormal:


Remarks on Receipt : -

Remarks on Test : These results conclude the report

A. Chandawe
Test Engineer

V. Chongo
Head – Materials Evaluation

For and on behalf of Alfred H. Knight (Zambia) Limited

*ALL WORK IS UNDERTAKEN SUBJECT TO OUR STANDARD TRADING TERMS AND CONDITIONS OF BUSINESS*

**Alfred H. Knight (Zambia) Limited**
Corner Mindola/Golf Club Roads, Nkana West,
P O Box 20303, Kitwe, Copperbelt, Zambia.
Tel: +260 212 226433/4 Fax: +260 212 226306/226424
Email: ahk.zambia@ahkgroup.com

Company Reg No. 41960 VAT No. 10126160-33

## Appendix E: Support standards photographed from Synclinorium underground

Smooth Tunnel Profile on 3760 level awaits support

Poor Mesh Support on 3510 level

Poor Cable Bolt Support on 3510 level - No face plate

Good Cable Bolt Support on 3510 level - Load indicator showing yellow

Poor Cable Bolt Support on 3360 level - protruding Bolts

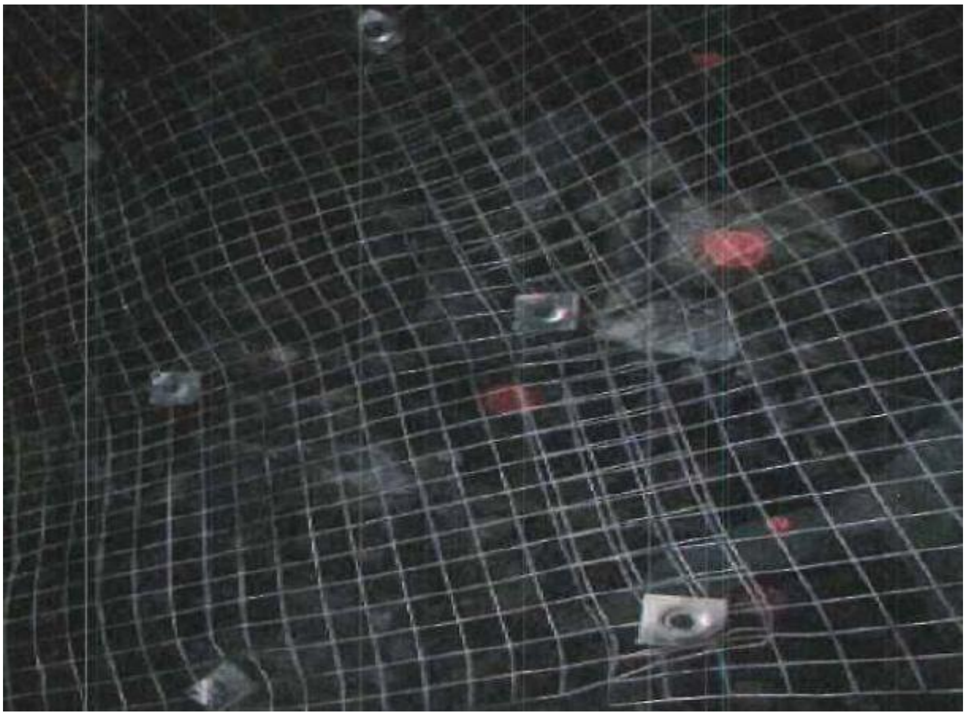
Good Mesh Support on 3760 level - Tight against rock walls

Poor ground conditions on 3510 level - buckled sidewalls under stress

Good Rock Bolt support (100mm stick-out); 3240 LEVEL

Poor shotcrete support- Thin and cracked (3360 HLGE)

Poor shotcrete support- Thin and cracked (3360 HLGE)



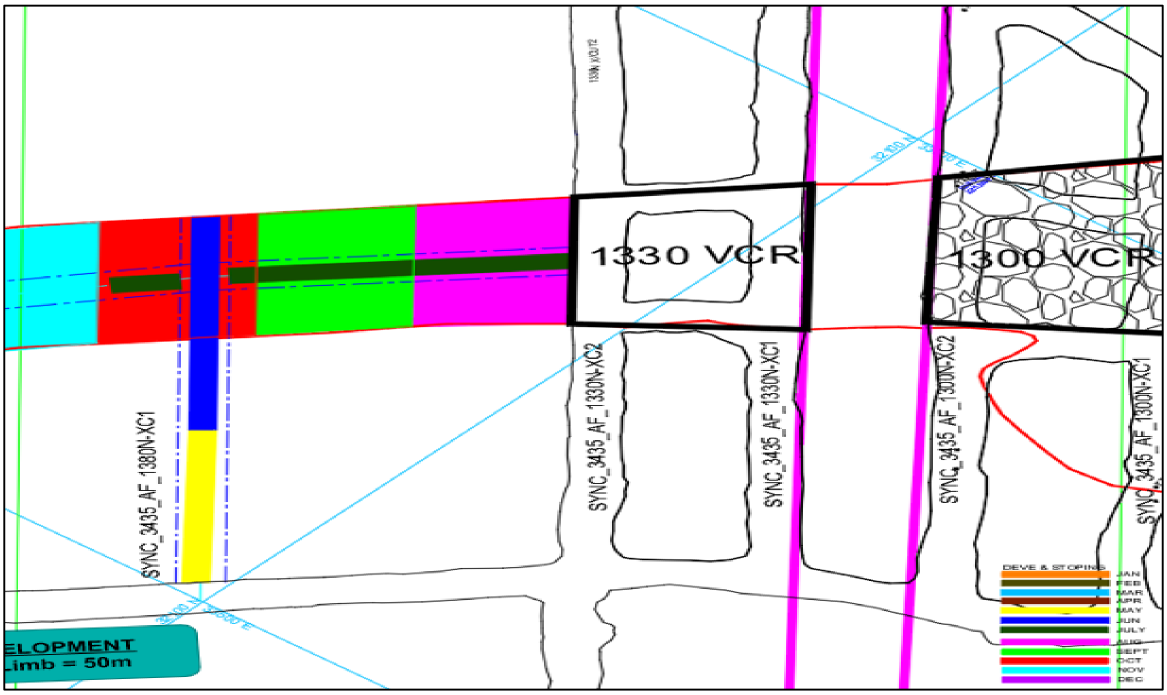
Good Split Set Support– Tight against the rock (3760 Level)

Poor Timber support on Central Shaft 3360 Level (Tilted and One-sided)

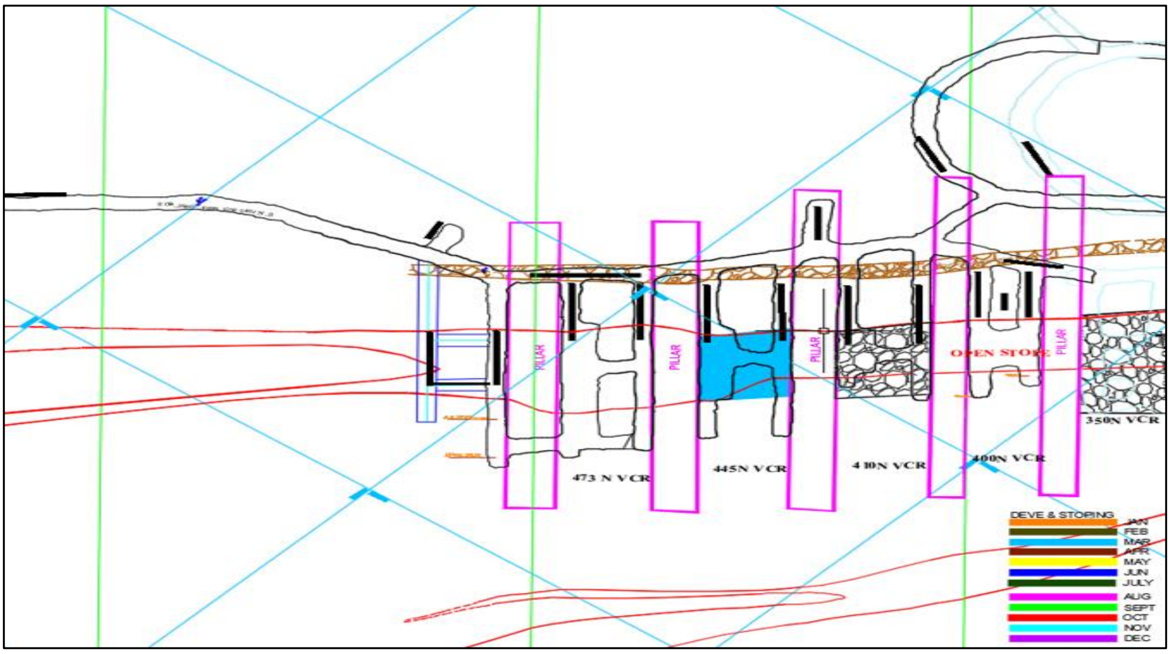
Poor Timber support on Synclinorium/SOB Shaft 3360 Level (One-sided)

### Appendix D: VCRs and Pillars studied by this research

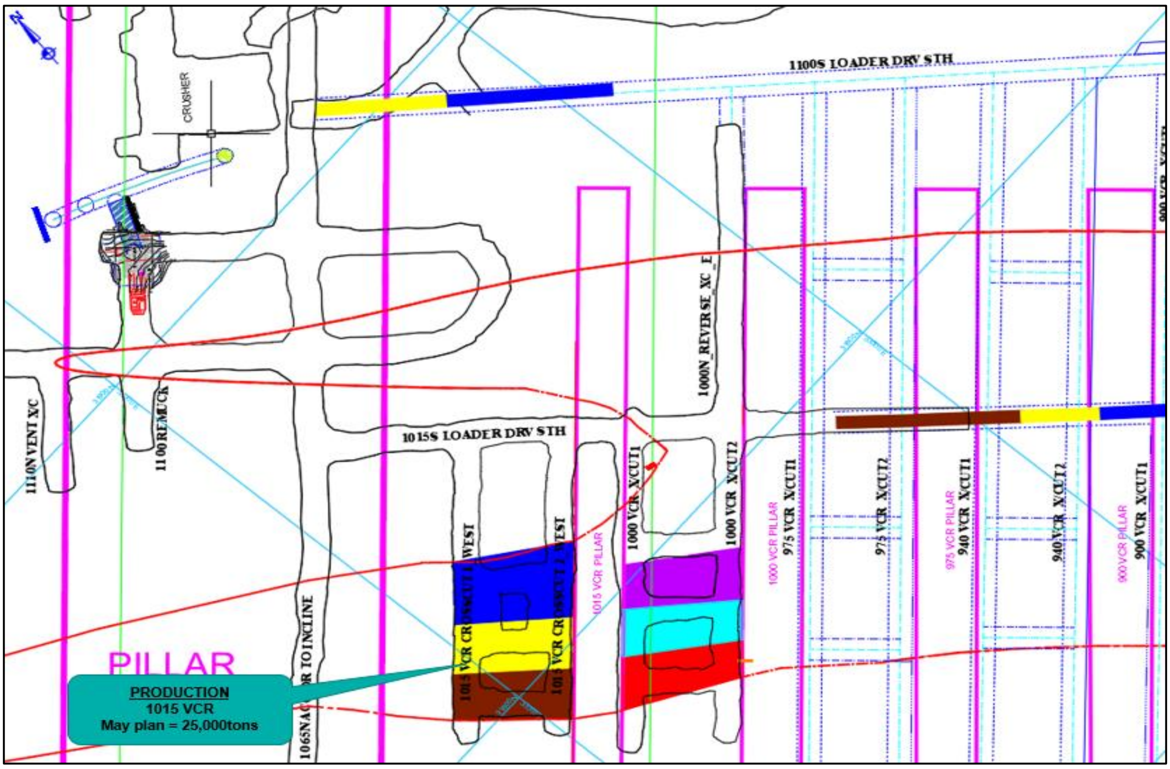
Nkana South - 3435L_A-FOLD_LIMB




Nkana South – 3585L 700_BC




Nkana South – 3760L 4800 Sync (Development Only)



1100S LOADER DRV STH
CRUSHER
1000N_REVERSE_XC_E
1110N VENT X/C
1100 REMUCK
1015S LOADER DRV STH
1000 VCR X/CUT1
1000 VCR X/CUT2
1015 VCR PILLAR
1000 VCR PILLAR
975 VCR X/CUT1
975 VCR X/CUT2
975 VCR PILLAR
940 VCR X/CUT1
940 VCR X/CUT2
900 VCR PILLAR
900 VCR X/CUT1
PILLAR
PRODUCTION
1015 VCR
May plan = 25,000tons