Quantification of the impacts of rock mass quality on stope width control and pillar stability in a hard rock narrow reef mine

Omberai Mandingaisa
Student Number: 699038

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DECLARATION

I declare that I am familiar with the School of Mining Engineering's policy on plagiarism, that the work contained in this research report is my own unaided work, that it is written in my own words and that all sources of material contained within this report have been suitably acknowledged.

Signed:

Date: 29 July 2018

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ABSTRACT

In bord and pillar mining, pillar stability is a key element of the mining process. This is usually underpinned by successful adherence to planned mining stope width. Stope width control is the backbone to the grade control process in platinum mines on the great Dyke of Zimbabwe. Poor rock mass has always been used to explain the failures by mining personnel to meet the requisite stoping width. A quantification process for this risk in monetary terms is tested and proves that geotechnical risk at times does less damage to the business value stream than malpractices. A review process followed in this research shows the vital path to value preservation and reduction of unnecessary dilution of the ore.

A robust pillar support system is critical in a bord and pillar setup in shallow mines. These pillars are designed not to yield nor crush. Despite meeting design criteria, however, pillars are still found to fail. A tool to quantify this risk in monetary terms is an unparalleled advantage. A classical case is presented in this research illustrating the critical steps that can be followed to scientifically provide management with the financial information on which to base decisions.

Poor rock mass conditions will always require to be adequately supported for sustainability of the mining business. This normally requires the installation of longer tendons, a time-consuming process. A slightly more expensive support product (the Flexibolt) was tested in this research to optimise the support process resulting in great value addition to the business. A case study is presented in this research report. Proposals for inclusion of a geotechnical risk quantification process to assist management to make value-based mining layout and operational decisions are also presented in this report.

ACKNOWLEDGEMENTS

This work is dedicated to my late parents Pauline and Titus Mandingaisa, your guidance for me as a child taught me to always soldier on even if the going gets tough, today it has paid dividend.

I acknowledge assistance and guidance given by Prof TR Stacey, words fail me to express my gratitude for the timeous responses and reading of my document. I still remember you asking if my vocabulary had commas. To me that was quite a highlight of how we miss small things in ground investigations that cause major incidents.

To my wife Nyari, thank you for being patient with me, while I spent time working on this research report. Zoe and Joel my dear princess and prince you rock my world and all this hard work is meant to be a challenge to you. Please work hard and surpass what your dad has done. You have the brains and the intellect.

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Abbreviations & Symbols

4E – ore grade declared as total of four elements (platinum, palladium, rhodium and gold)

A – Area

APS – Average pillar strength

BMSZ – Base of Base Metal Subzone

C80 – Represents the upper limit to the lowest values achieved 20% of the time

C – Circumference or perimeter

CBE – Capital Budget Estimate

DRMS – Design Rock Mass Strength

FoS – Factor of Safety

FOG – Fall of Ground

g – Acceleration due to gravity

g/t – grams per tonne (4E for purposes of this report)

h – Height

IMS – Institute of Mine Seismology

J_a – Joint alteration rating

J_n – Joint set Number rating

J_r – Joint Roughness rating

J_v – Joint Volume

J_w – Joint Water condition

K – Constant equal to 1/3 of UCS or equal to DRMS

km – kilometres

kN – kilo Newtons

ktpm – kilo tonnes per month

LHD – Load Haul and Dump

m – Metres

MPa – Mega Pascals

MRMR – Mining Rock Mass Rating

MSZ – Main Sulphide Zone

N – Newtons

NGI –Norwegian Geotechnical Institute

P75 – represents the higher quartile achievement. i.e. the lower limit to the highest values achieved 25% of the time

P1 – Pyroxenite Layer Number 1

Pa – Pascals

PGM – Platinum Group Metals

PS - Pillar strength

Q – A rock mass rating system developed by Barton et al

ρ – Density

RMC – Rock Mass Classification

RMR – Rock Mass Rating

RQD –Rock Quality Designation

RMS – Rock Mass Strength

s - Seconds

SML – Special Mining Lease

SRF – Stress Retention Factor

SW – Stope Width

t – Tonnes

TAT – Tributary Area Theory

UCS - Uniaxial Compressive Strength

w_e – Effective pillar width

w/h – width to height ratio

1 INTRODUCTION

In narrow reef, bord and pillar, platinum mining on the Great Dyke of Zimbabwe strict stope width control is critical to attain the required grade for the viability of mining projects. The grade distribution of platinum group metals (PGMs) in the vertical profile declines from 7.6 grams per tonne to 0.4 grams per tonne of four elements grade normally referred to as 4E grade (i.e. total of platinum, palladium, rhodium and gold) over 40-60 cm into the hanging wall from the arbitrary reference marker horizon, referred to as the Base of the Base Metal Sub-Zone (BMSZ). This marker horizon is identified by a sharp decrease in the amount of base metal sulphides, leaving the zone above the marker as the base metal subzone and that below the BMSZ as the platinum group metal subzone. The quick transition to very low grades in the hanging wall calls for good practices to reduce the amount of waste ingress into the ore stream from hanging wall over-breaks (Figure 1).

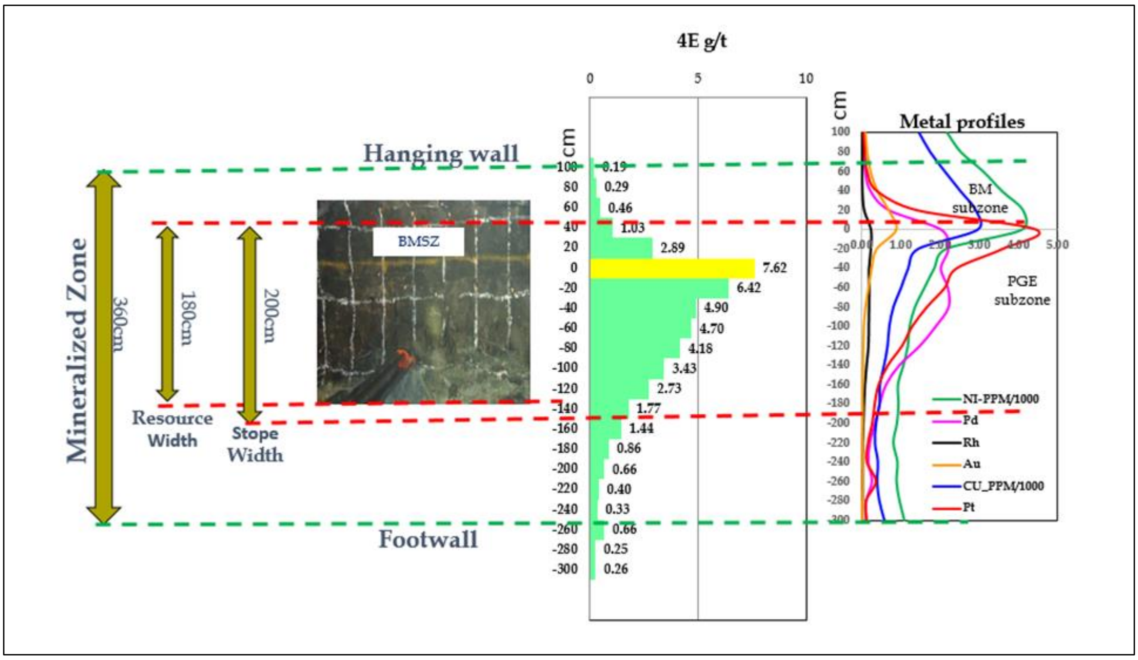

Figure 1: Grade vs. stope width plot for the Unki Resource (Mwatahwa et al, 2017)

A good understanding of the influence of parameters/factors that result in hanging wall over-break is key if the operation is to manage the hanging wall dilution challenges effectively. It is also key that the risk associated with the influence of the geotechnical parameters affecting hanging wall stability be understood. As such it is imperative that there is need to have a mechanism or ability to quantify the impacts in monetary terms, which would enable management to make informed decisions in the business planning process for both short term and long-term viability.

The design of shallow, narrow reef, bord and pillar mechanized mines is based on high throughput and a solid pillar system, where the pillars are designed to carry all the overburden weight and avoid subsidence. The strength of these pillars is mainly dependent on the material making up the pillar, the pillar foundation material, the number of discontinuities cutting through the pillar, pillar width to height ratio, the method of excavation and the pillar load to strength ratio. In order to achieve the correct mix of good grade ore, high tonnages and a stable pillar system, the mining cut or stope width (SW) becomes a key ingredient. Excessive stope widths lead to high ore dilution and failure to meet the design mining height, leading to lower width to height (w/h) ratio values in pillars thus compromising their integrity.

The mining cut is always topical at all bord and pillar mines on the Great Dyke that fail to meet requisite ounces or develop instabilities in pillars. Poor stope width control is always largely considered to result from poor rock mass quality, a position usually accepted from literature without scientific interrogation, leading to reduced targets and efficiencies for the mining teams. However, very few studies, if any, have been conducted on the Great Dyke to scientifically prove this widely adopted explanation in production meetings, though it is well known that dilution results in low recoveries and high processing and tramming costs, as waste is trammed with ore.

There is no system or mechanism available to enable management to determine the contribution of poor rock mass conditions to challenges in poor stope width control and pillar instabilities. Thus, this geotechnical risk's impact on the business plan is always skated around in business planning meetings and becomes applicable as an explanation for failure to meet targets. This lack of tools to quantify the risk, creates loopholes in decision making and association of huge losses of revenue with poor rock mass quality, which could possibly have resulted from other easy to manage factors influencing the poor stope widths. It is against this background that this proposed research has been formulated.

1.1 Unki Mine Stope width

Unki mine was designed using the Hedley and Grant (1972) formula as modified by Noble (2001). This requires the attainment of a pillar width/height ratio of 3 to be applicable. However, the mine has never achieved the w/h ratio of 3, nor the required stope width of 190 cm (180 cm mining cut plus 10 cm planned dilution). Current stope width is at 204 cm which has been a significant improvement from the 223 cm achieved in the past. Several factors have contributed to the poor achievement but, one key factor worth mentioning was the decision by management to purchase a fleet of larger capacity LHDs. These machines could not fit easily in a 200 cm excavation, leading to a deliberate move to increase the stope width to 210 cm. This saw the stope width increase to 223 cm. The process to bring the stope width down to 204 cm while phasing out the large LHDs has not been an easy route.

Given the poor achievements in stope width control, the mine was beginning to over mine its design. The design pillar strength calculations were reverted back to the original Hedley and Grant (1972) formula, leading to a loss of reserve, as this has the net effect of lowering pillar strength, leading to extraction adjustment to meet the design criterion of a safety factor greater than 1.6 for the pillars, as reported by Mandingaisa and Musa (2017). This impact is often not amplified, but is a significant loss to the business, and finding a solution to get the stope width to desired level, so that the originally applicable design that benefits the business is readopted, is important. The failure to meet the mining height was largely attributed to the poor rock mass conditions, a state that has not been scientifically validated as previously discussed.

Some pillars have been noted as deteriorating and these have been rehabilitated or are on a rehabilitation schedule which brings about another cost dimension to the failure to mine to correct dimensions. These pillar failures have been ascribed to poor rock mass quality. A scientific investigation is required to test this theory, ascribe monetary value to the impact of poor rock mass, as well as the other controllable factors, to aid management in decision making.

1.2 Problem Definition

It is widely accepted at the mine that failure to achieve the mining cut is largely due to poor rock mass conditions. Mechanisms and methods of assessing the rock mass quality that ascribe values to quality of the rock mass have been developed over time and two methods are being applied at the mine i.e. the Barton et al (1974) NGI Q system and Bieniawski's (1989) RMR system. This process has been conducted in approximately 80% of the mine creating a good database, excluding the mining conducted during the research phase, i.e. before the inception of the Rock Engineering Department. Stope width measurements are collected on each face before the face is marked and drilled for the next blast. The mine was designed to be optimally mined at 180 cm (190 cm including 10 cm over-break) stope width which has not been achieved to date. This failure has been largely ascribed to the poor rock mass conditions encountered during mining, though it is acknowledged to a lesser extent that the blasting practices and management decisions have also played a role in the failure to achieve the requisite and optimum mining cut.

No work has been done to scientifically examine the correlation between the rock mass quality and the stope width, though this appears to be a widely acceptable explanation for the failure to meet design stope width. The mine could thus be losing a lot of money from easily correctable inefficiency issues hidden behind the poor rock mass explanation. If there is a low correlation between the rock mass quality and the stope width, it means that there are other factors at play. This research therefore seeks to fill the gap in scientific knowledge on this relationship. The cost implication related to the contribution of rock mass quality to the failure to meet the 180 cm stope width is currently unknown, and the contribution of the other factors has also not been quantified, though the impacts could be significant inputs into business planning.

These are normally just covered by modifying factors, thereby masking critical information for decision making.

1.3 Objectives

The main objectives of this research are:

a. To determine the contribution of rock mass quality to the poor stope width control and pillar stability.
b. To determine and quantify the impact of rock mass quality on pillar stability
c. To determine and quantify in monetary terms the impacts of rock mass quality on mine profitability and the business plan.

Achieving these objectives will enable good decisions to be made whenever challenging ground is encountered. High stope widths are associated with higher demands for ground support, and it is also important to quantify this in monetary terms, and to be in a position to give management information that supports business planning and viability assessment.

1.4 Research Hypotheses

i. Contribution of rock mass quality to poor stope widths

H_0: Rock mass quality does not contribute to poor stope width control

H_1: Rock mass quality contributes to poor stope width control

ii. Impact of rock mass quality on pillar stability

H_0: Rock mass quality does not have a quantifiable impact on pillar stability

H_1: Rock mass quality has a quantifiable impact on pillar stability

iii. Monetary value of the impact of rock mass quality on the business plan

H_0: There is no quantifiable monetary value of the impact of rock mass quality on the business plan

H_1: There is a quantifiable monetary value of the impact of rock mass quality on the business plan

1.5 Content of the research report

In this research report, Chapter 2 gives brief look at the geotechnical setting of the research area (Unki mine) and a review of the literature on this site to set a solid base for the research. Geotechnical risk is reviewed to conclude this chapter. Chapter 3 reviews rock mass quality as applicable in the mining industry, focusing on the systems applicable in the area where the research was carried out. A review of drilling and blasting impacts, ore dilution and pillar stability with respect to bord and pillar mining is also taken into consideration. Chapter 4 documents the applicable methodology and tools applied in this study in the collection of data. Results and analysis are presented in Chapter 5. Chapter 6 focuses on the discussion of the results, conclusions and recommendations of a risk quantification process to assist management to make value-based mining layout and operational decisions.

2 GEOTECHNICAL SETTING

2.1 Unki Mine geotechnical environment

Mandingaisa and Musa (2017) state that *"Unki Mine is located within the Selukwe Sub-chamber of the South Chamber of the Great Dyke in Zimbabwe. The Great Dyke is a layered igneous complex comprising of mafic and ultramafic rocks. The Unki ore body forms a syncline which outcrops on both the eastern and western flanks of the Great Dyke. At the outcrops, the ore body dips at approximately 15 degrees. From the eastern outcrop this shallows towards the centre of the dyke, where the dip reaches zero degrees, before increasing again in the opposite direction towards the western side of the dyke. The ore body at Unki is bounded to the north by a major fault (Paarl Fault approx. 100 m displacement). The location of Unki relative to the other platinum group element mines on the great Dyke is shown in Figure 2***Error! Reference source not found.***. It must be noted that Unki is part of a Special Mining Lease extending over 30 km along strike."* Refer to Figure 2**Error! Reference source not found.** and Figure 3**Error! Reference source not found.**.

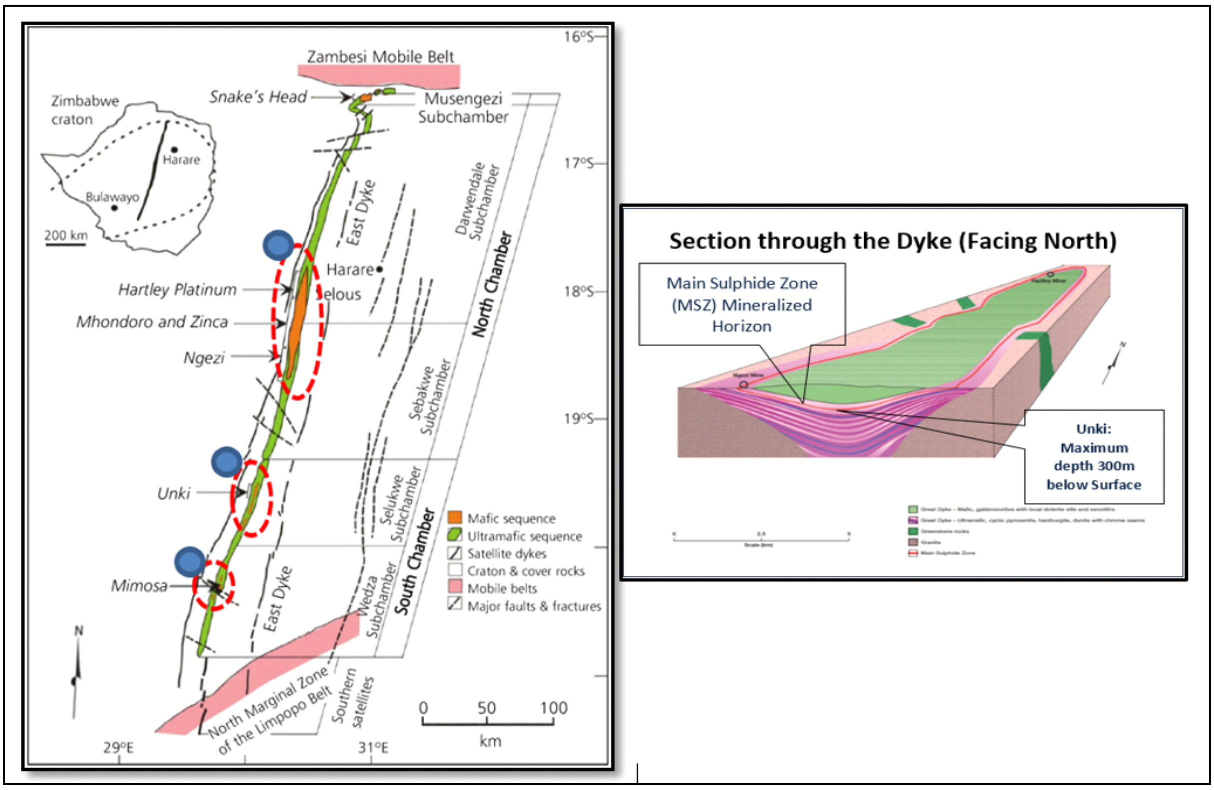

Figure 2: Location of Unki Mine relative to major towns and a schematic section across the Great Dyke (adopted from Mandingaisa and Musa, 2017)

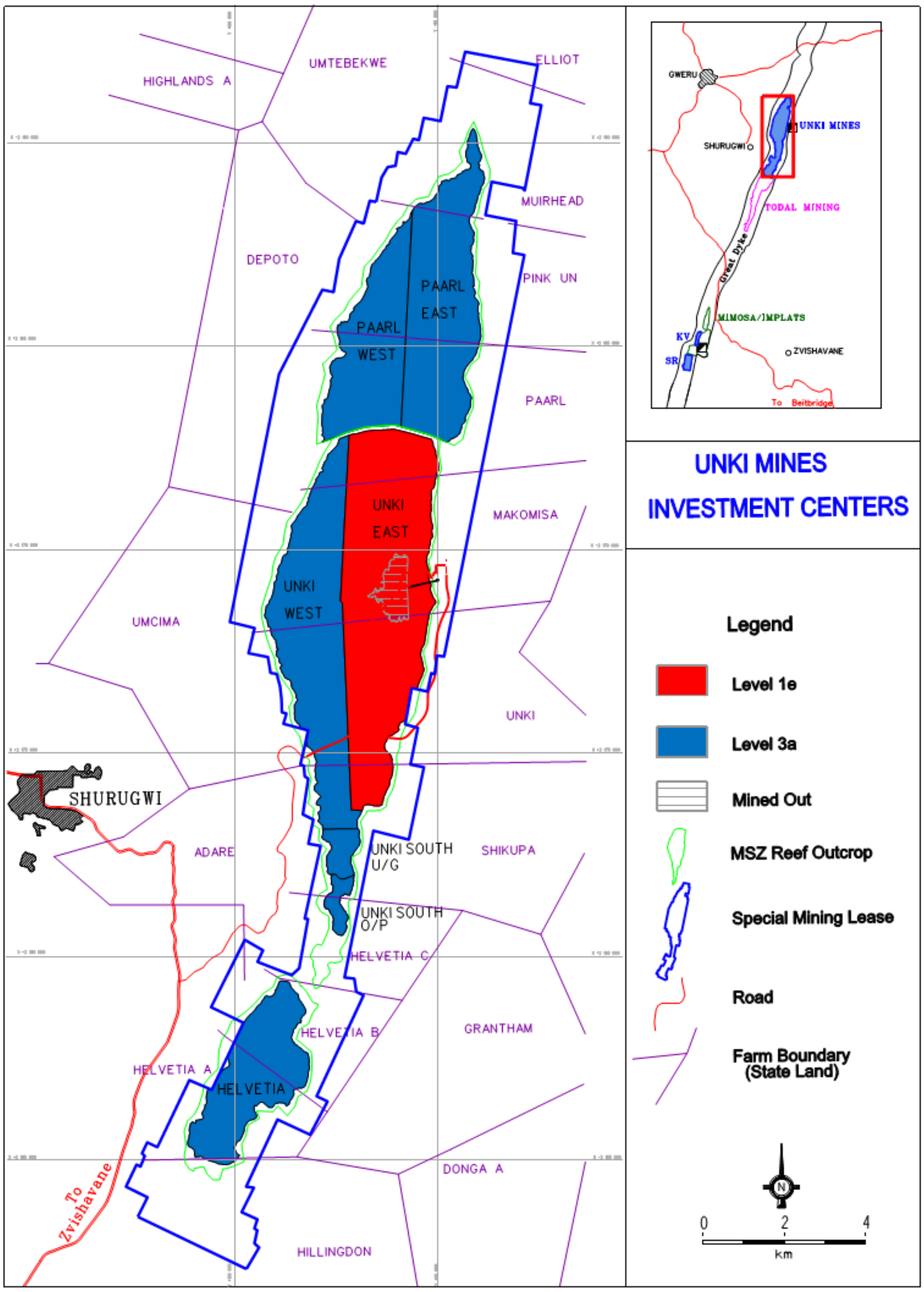

Figure 3: Unki Mining Lease showing the different investment centres (Adopted from Unki Geology Technical Support Document to Business Plan 2017)

2.2 Stratigraphy (Adopted from Unki Geology CBE, 2005)

The target ore body, the main sulphide zone, occurs within the topmost pyroxenite layer usually referred to as Pyroxenite Layer 1 (P1) of the Great Dyke. The main sulphide zone reef is hosted within plagioclase pyroxenite. It typically occurs as a 2-4 metre layer in which the visibility of the sulphides varies greatly. The reef is a sub-layer of the plagioclase pyroxenite, and occurs at 2-3 metres beneath a hanging wall websterite/plagioclase pyroxenite contact.

This contact is gradational and has little or no parting potential (Unki Geology CBE, 2005). A marker horizon corresponding to the drop-in copper and nickel sulphides and coinciding with peak platinum values is identified and used as the reef reference and known locally as the Base of Base Metal Subzone. A persistent coarse grained (pegmatoidal) plagioclase pyroxenite layer occurs about 0.45 m to 2 m above the base of Base Metal Subzone (Figure 4) and shows some parting potential. Its characteristic large grain size allows splitting along crystal cleavage planes (Mandingaisa and Musa 2017). Of greater significance is an intermittent chromitite stringer which marks the top of the topmost pyroxenite layer (Figure 4). This contact with the base of the overlying gabbronorite is usually developed between 7 m to 9 m above the BMSZ (Musa et al, 2015). As recorded in the Unki Geotech CBE (2005), *"the chromitite stringer presents a geotechnical constraint in stoping span design. The current design allows for a maximum 12 m panel span as a wider span would require external support in the form of stiff, strong packs or dense stick support to prevent the collapse of the hanging wall beam. "*

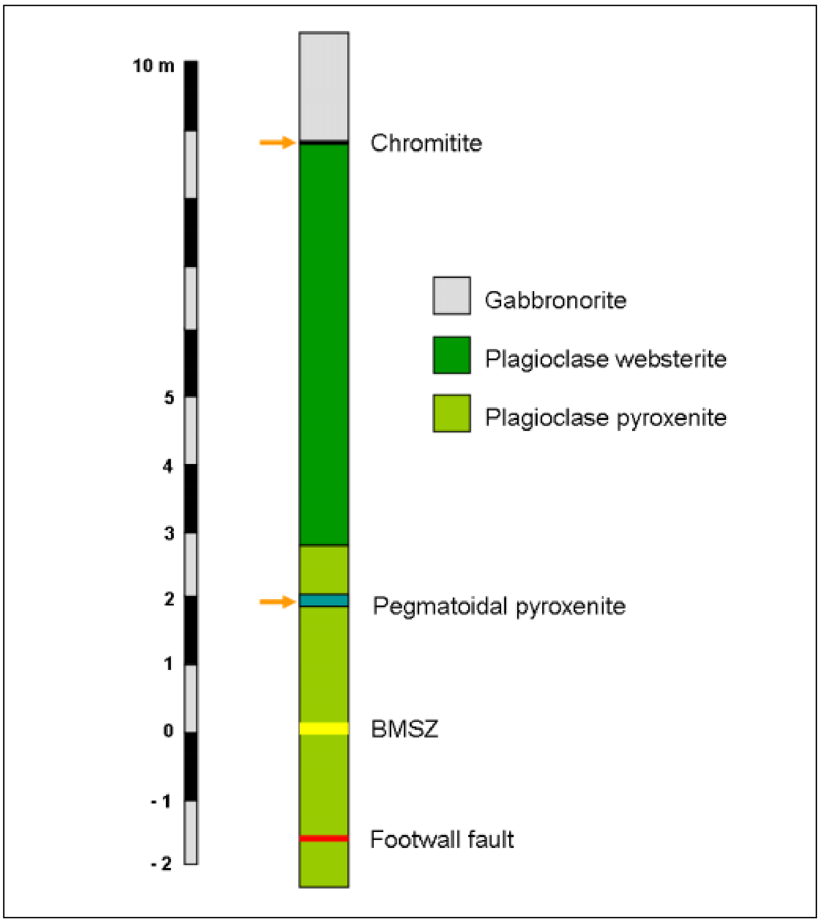

Figure 4: Stratigraphic column of the Unki area. Potential zones of stratigraphic weakness are indicated by the orange arrows (Unki Geotech CBE, 2005)

2.3 Major Geological Structures

2.3.1 Major Faults

According to Hlasi and Mwatahwa (2017), the Unki East and West investment centres are bounded to the north by a major fault termed the Paarl fault. It has a displacement of approximately 100 m. Figure 5 illustrates the structural complexity and discontinuities along the Unki Special Mining Lease (SML). The lease is categorised into high, medium and low complexity zones shown as red, yellow and green in Figure 5.

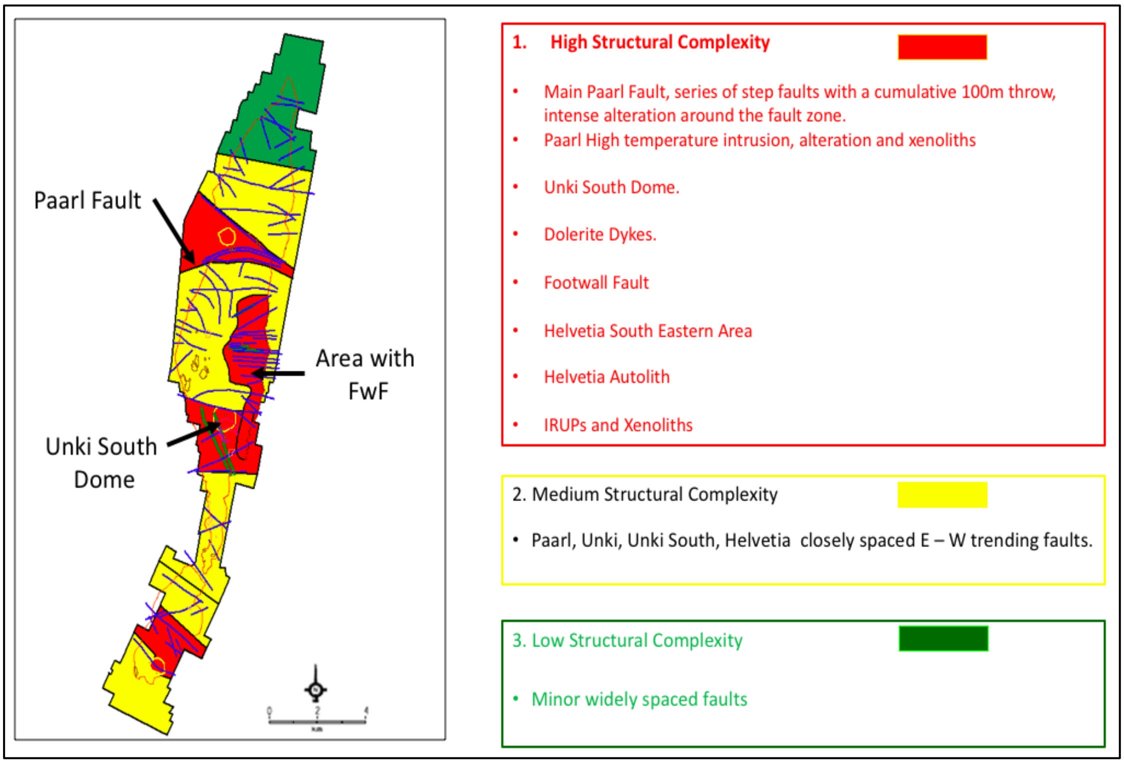

Figure 5: Illustration of the structural complexity and discontinuities along the Unki Special Mining Lease (adopted from Hlasi and Mwatahwa, 2017)

2.3.2 Vertical faults, including micro-faulting

Faulting is common throughout the uppermost pyroxenite layer on all sub-chambers of the Great Dyke. These faults are not always identified through exploration drilling due to their steep dips and relatively small throw (in the order of <0.5 to 1.5 metres). It can be expected that such previously unknown faults will be intersected from time to time during mining. Micro-faulting of the Main Sulphide Zone reef is relatively common, i.e. the magnitude of displacement is so small that the reef layer or any part of it is not displaced outside the panel face as discussed in Unki Mine Design Criteria Geotechnical - Rock Engineering Report (2007).

Mandingaisa (2018), states that where smaller faults occur (less than 0.2 m) the bords should be able to negotiate through these faults with relative ease. Depending on the position of the fault, an additional pillar may be required plus an extra row of bolts on either side of the feature. For the larger throw faults, where ramping up and cutting through the hanging wall are required, long anchors are installed to support the brows and adequate sized pillars may need to be cut on either side of the fault for greater stability. These pillars should be at least 5 m wide. The long anchors are time consuming to install and a change in the product was introduced during the research work by the author resulting in value addition documented in latter chapters of this report.

2.3.3 Hanging wall fault (Ramp and Flat Structures)

The ramp and flat structures and hanging wall fault are layered structures affecting the hanging wall 4 m to 8 m above the Base of the Base Metal Subzone (BMSZ) horizon. It is critical that these zones be considered when panel spans and supports are designed. The major areas in Unki East with ramp and flat structures, whose effects are similar to that of the hanging wall fault are also delineated.

2.3.4 Footwall fault (FwF)

Brown and Mwatahwa (2005), state that *"the footwall fault is a reef parallel fault that occurs within the footwall of the main sulphide zone reef. It occurs between 0.6 m and 2.9 m below the BMSZ, with an average depth of 1.6 m. The fault undulates, pinches and swells, and does not cross the BMSZ but is associated with sympathetic jointing and faulting that pose ground challenges where they splay into the hanging wall. Current mining has shown that splay faults from the FwF cross the BMSZ and form wedges and 'domes' in the hanging wall. The FwF consists of highly altered, mylonitised and brecciated plagioclase pyroxenite."* **Error! Reference source not found.** Figure 6 is an image showing the position of the footwall fault (FwF) relative to the Base of the Base Metal Subzone (BMSZ, yellow reference line). Diamond drilling has shown that the footwall fault is a localized feature that only affects the eastern portion of the immediate Unki mining area (Musa et al 2015). Figure 7 and Figure 8 show the distribution of the footwall fault relative to mining cut and the Unki East mining area.

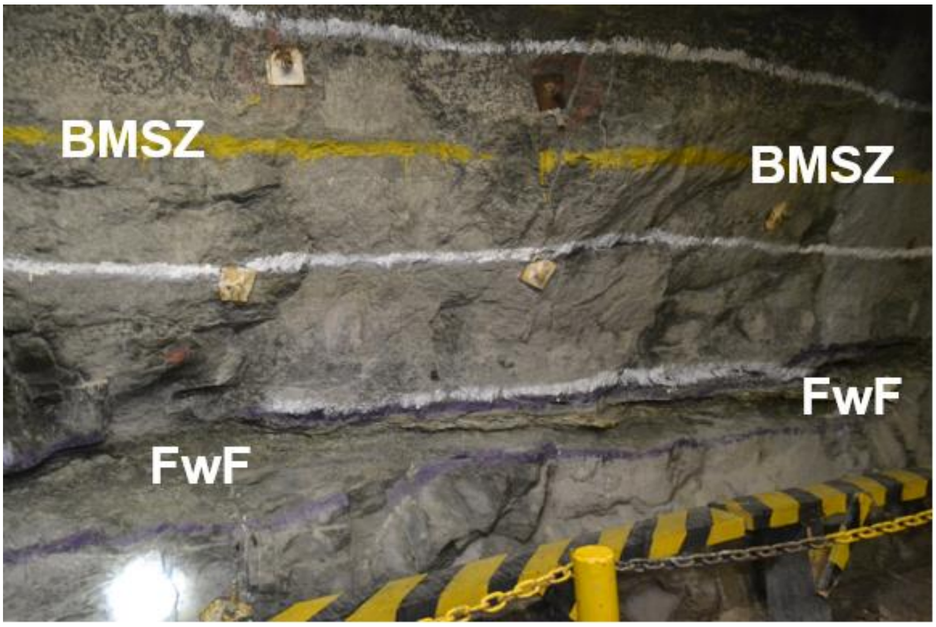

Figure 6: Image showing the position of the FwF relative to the BMSZ (yellow line), (photo from Mandingaisa and Musa, 2017)

Figure 7: Distribution of the FwF showing its localised nature in the Unki mining lease (after Musa et al, 2015)

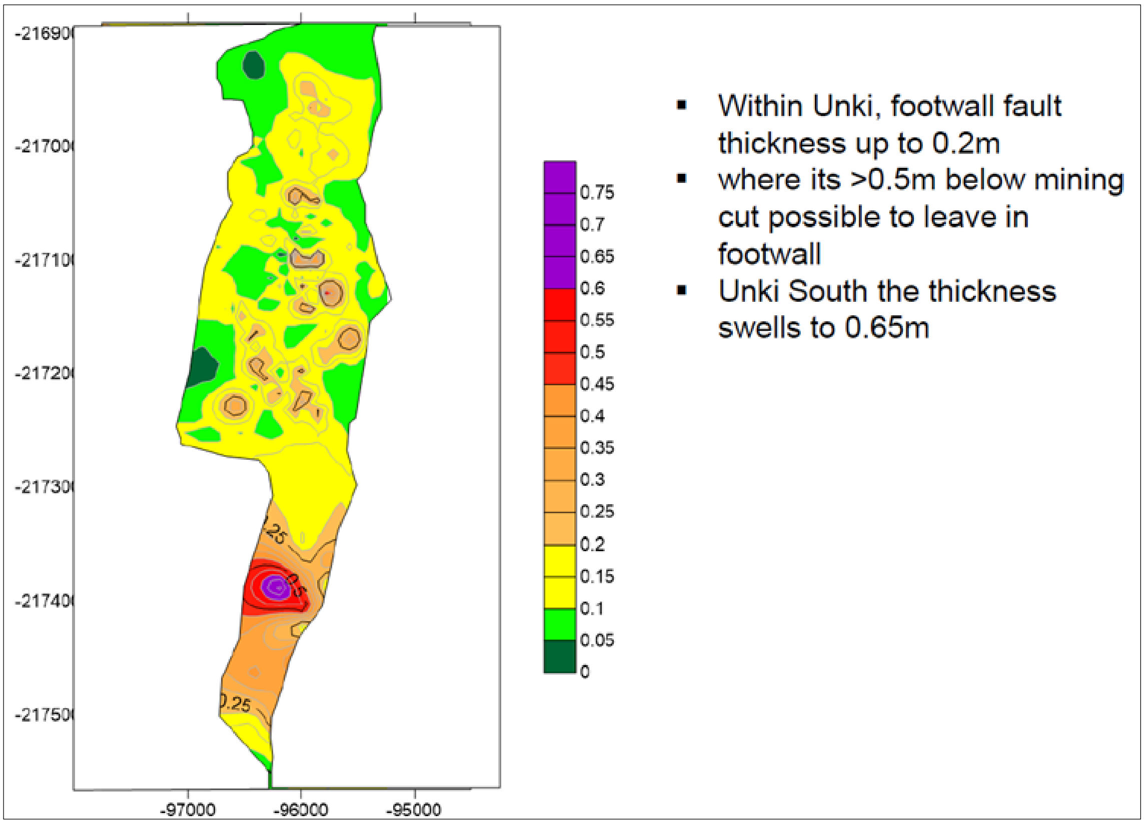

Figure 8: Footwall thickness map illustrating distribution (after Musa et al, 2015)

2.3.5 Impacts of the footwall fault

Van Aswegen (2011), stated that it was *"shown in case studies that reef parallel structures, exposed in pillars, will have a net negative effect of reducing the ultimate strength of a pillar. The magnitude of the strength reduction varies depending on the properties of the geological structure. This reduction can vary from approximately 40% for a structure with a low cohesion and low friction angle to very little for a structure with a high friction angle and high cohesion."* For Unki Mine, FLAC3D inelastic numerical modelling was conducted by Leach (2011) which indicated a reduction in pillar strength by 31% for a friction angle of 15 degrees and cohesion of 1.5 MPa. Later laboratory tests done at RockLab indicated that the friction angle on the footwall fault averages 21 degrees. According to Mandingaisa and Musa (2017), in pillar design at Unki, a 35% reduction in pillar strength due to the footwall fault should be applied and considered a conservative design approach.

2.3.6 Joint Information

It is expected that the joint orientations and properties for Unki West and the Paarl area will be very similar to the joints that have been intersected in Unki East mining area. This deduction is based on similar depths at which the areas will be mined and their proximity to each other. However, those in the Unki South and Helvetia area could exhibit differences in the infill properties due to shallow depths. This information must however be updated as soon as the areas under consideration have been exposed by mining or further drilling.

According to Mandingaisa and Musa (2017), the general occurrence and properties of the joints that were intersected in current mined out areas and their interaction with other geological structures is as follows:

a. Two joint sets trend north–south with one steeply dipping and the other shallowly dipping and two minor east-west trending steeply-dipping joint sets.
b. The overall joint frequency is lower on the southern part of the Mine than in the northern part of the Mine
c. Narrow, potentially unstable wedges may be formed in the stope hanging wall, normal to the stope faces. The same structures may also form narrow, potentially unstable, triangular-shaped wedges, oriented sub-parallel to the faces.
d. "Dome" structures intersecting the four above-mentioned joint sets may form blocky ground and induce unstable hanging wall conditions leading to potential Falls of Ground (FOG). Updated mapping suggests that these "domes" may in fact be low angle reef parallel discontinuities.
e. Random joints in the hanging wall are generally shallow-dipping, (<45°) and, although the joint frequency is low, these pose major rock engineering ground control challenges. The joints are random and are not expected to influence the mining cut considerably.

Generally, the hanging wall rock immediately above the BMSZ is competent and mostly un-weathered. A summary of the general joint orientations are given in Table 1.

Table 1: Summary of Major Joint set orientation collected to date

Joint Number	**Dip/Dip Direction**	**Strike direction**	**Comments**
J1	60/101	N-S	Low angle joints
J2	79/360	E-W	Variation of the same set
J3	78/179	E-W	
J4	75/284	N-S	Not very prominent if a large area is considered. Mainly in South Section

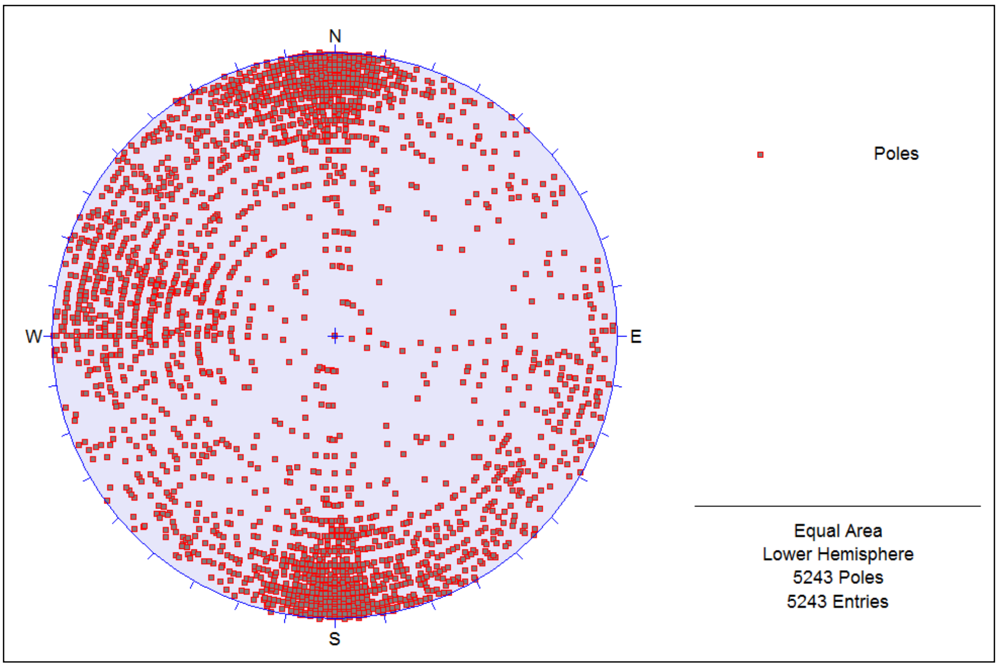

Figure 9: Lower hemisphere equal area stereographical projection of jointing pole concentrations (Mandingaisa, 2018)

Figure 9 shows the poles of all the joints that were used for the analysis of joint orientations while *Figure 10* shows the contouring of the joint orientation poles density and the general joint orientations. *Figure 11* is a rose plot of the joint orientation showing the predominance of the E-W trending joints and the N-S trending joints.

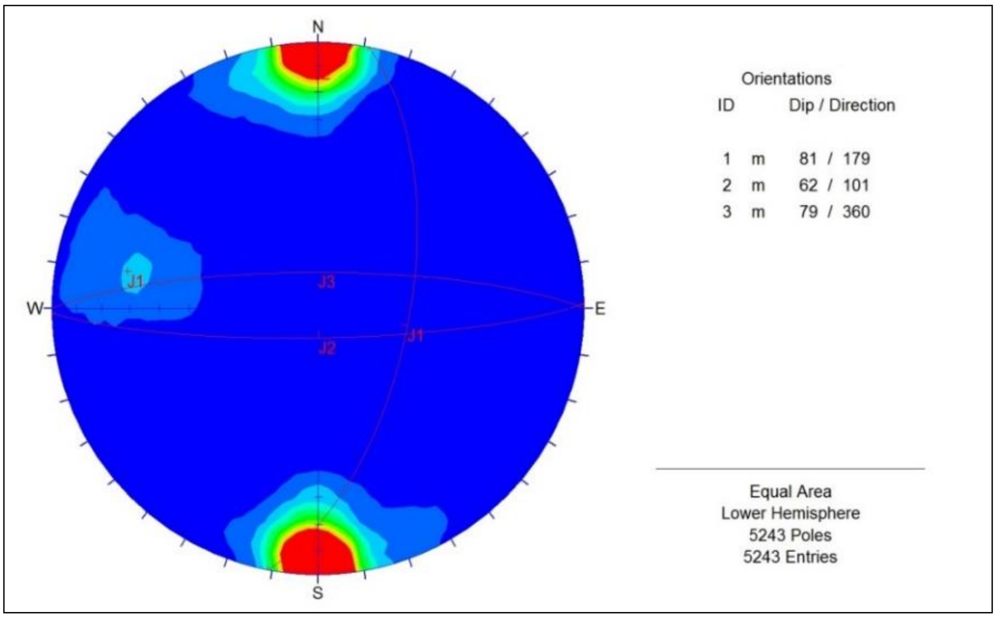

Figure 10: Lower hemisphere equal area stereographical contouring of joint pole orientations (Mandingaisa, 2018)

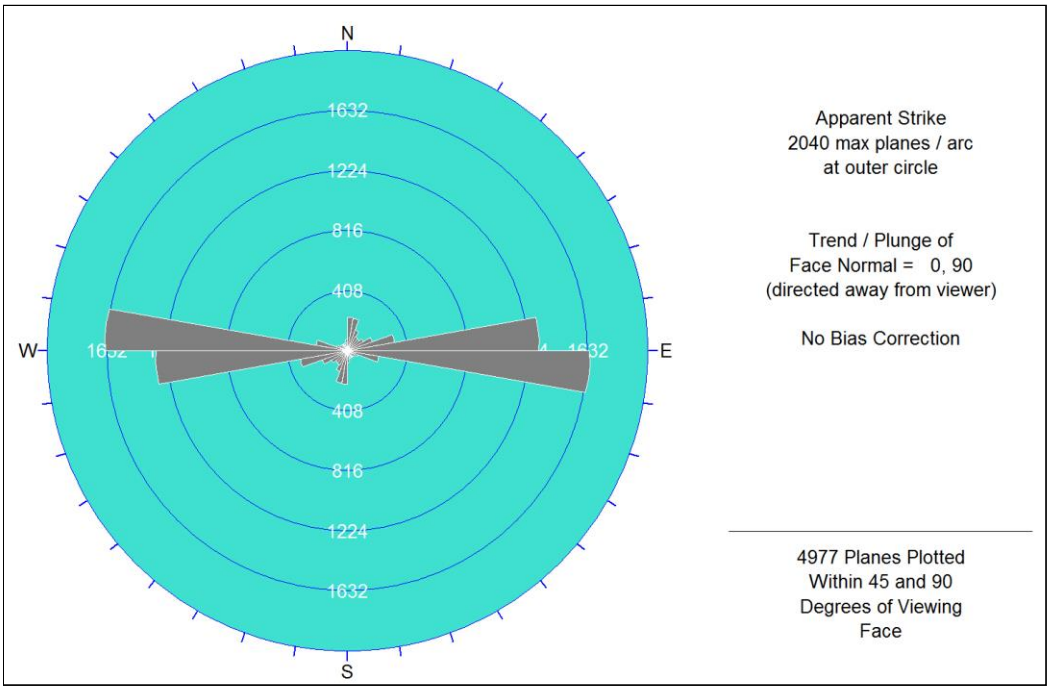

Figure 11: Rose diagram of the major joint sets as mapped in the current production area and the declines (Mandingaisa, 2018)

The following plots in *Figure 12***Error! Reference source not found.** and *Figure 13****Error! Reference source not found.*** show the lower hemisphere projection of the orientations of joints in the north and south sections of the mine respectively.

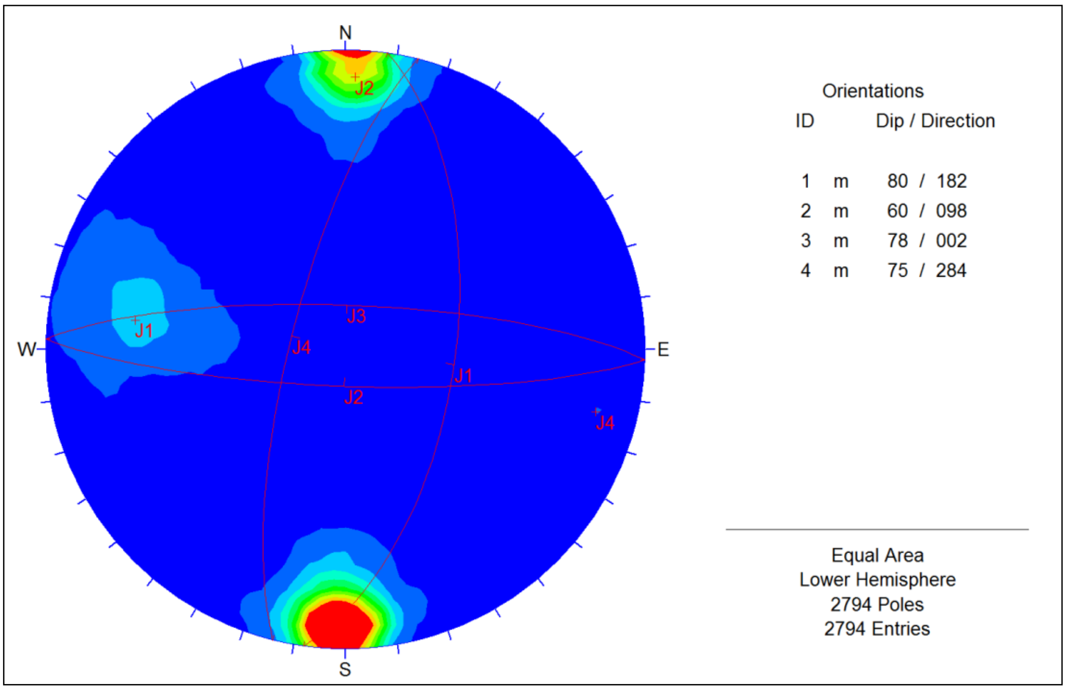

Figure 12: Plot of poles and planes for the major joint sets in the South Section (Mandingaisa, 2018)

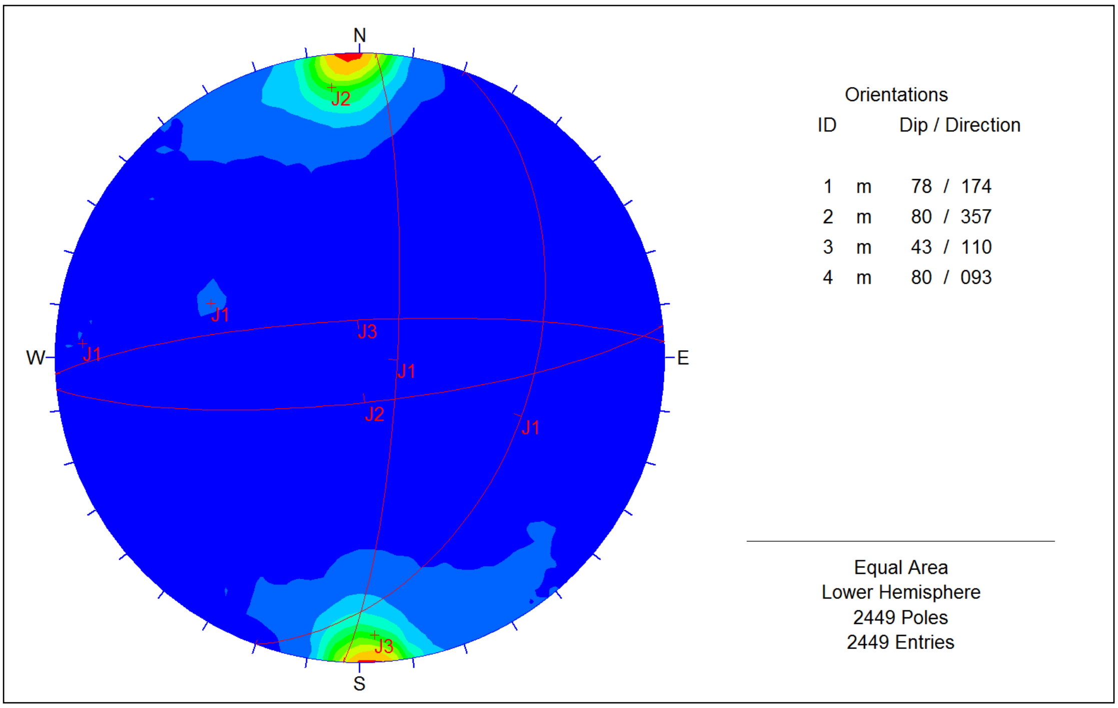

Figure 13: Plot of poles and planes for the major joint sets in the North section (Mandingaisa, 2018)

It can be noted from the two plots that:

a. In the southern section of the Mine, J4 appears but is not too prominent and J1 is steeply dipping

b. In the northern section J4 is not a prominent set but there are two variations of J1 i.e. low and high dip angle N-S striking joint sets

2.4 Stress regime

Most of the mining (roughly estimated at more than 70%) will be done at depths of between 140 m and 240 m. There is a very small area that will be situated at depths of between 240 and 350 m. Unki is a shallow mine (<300 m) and all mining will be in a shallow, low stress zone. **Error! Reference source not found.** Figure 14 shows the direction of the principal horizontal stress as determined from borehole geophysics data by Trofimczyk (2008).

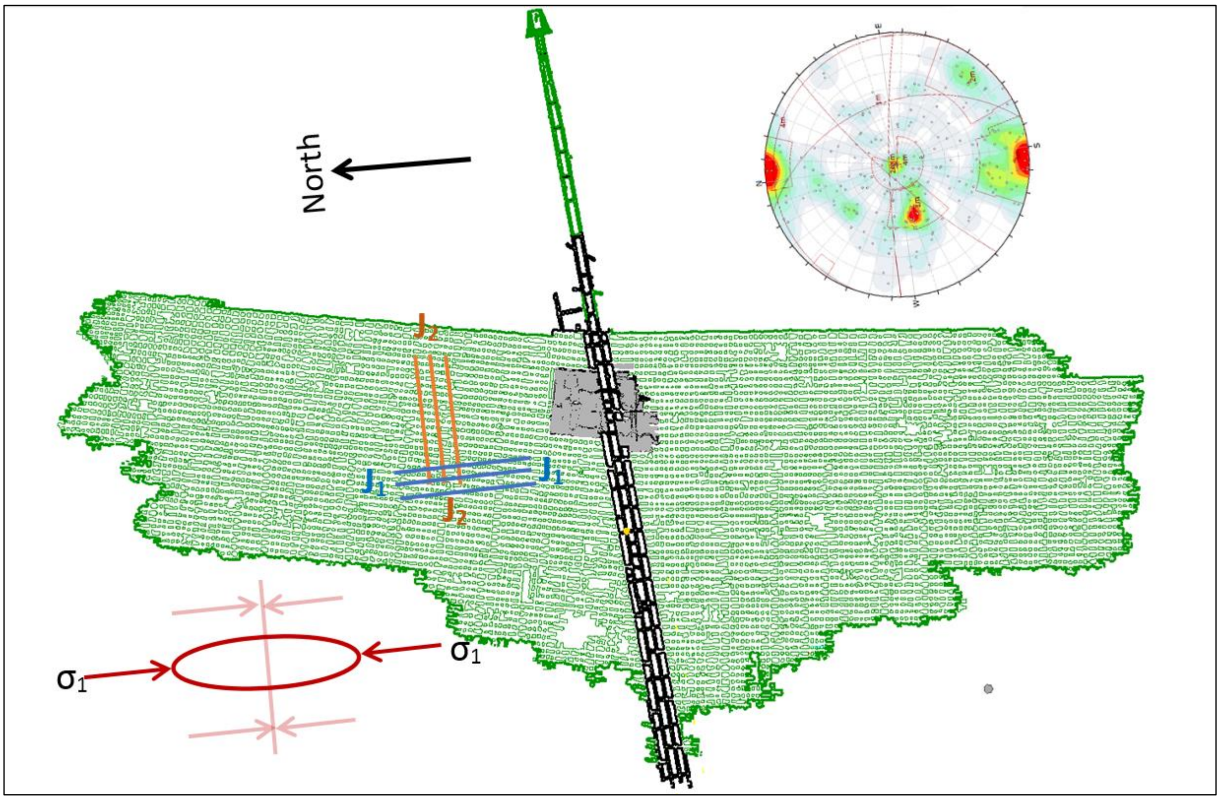

Figure 14: Schematic showing the direction of the horizontal principal stresses (Mandingaisa, 2018) (NB. The stereoplot is rotated such that the North label on the plot corresponds to North on the schematic)

The k-ratio, from underground observations, is approximately 1 with no significant variations. The unit weight of the overburden is approximately 29.4 kN/m^3.

2.4.1 Regional Support

Following the investigations and FLAC3D modelling by Leach (2011) on the effect of the footwall fault and tailings storage facility loading, barrier pillars were introduced from 175 metres below surface (mbs). These were introduced in the current mining areas to improve stability and maintain the factor of safety above 1.6. This created a challenge in the calculation of factor of safety for the smaller pillars as Tributary Area Theory (TAT) becomes inapplicable in the calculation of the average pillar stress, due to the bridging/abutment effect of the larger pillars. Institute of Mine Seismology (IMS) is sub-contracted every year to model the APS of the pillars.

2.4.2 Unki Hazard Identification and Treatment System (HITS) ABS-P system

The Hazard Identification and Treatment System (HITS) forms part of a daily geotechnical risk assessment carried out in all production bords before a bord that has been blasted is cleaned. HITS is a task action response plan employed as part of its risk assessment process. This involves classification of ground based on the presence of different triggers (listed in Table 2) into class A, B and S. The system is referred to as the (HITS) ABS-P, where A, B, and S are risk-based ground classes. 'A' is the low risk, 'B' is medium risk and 'S' is high risk. 'P' stands for a precautionary stage where support instructions have been issued for a 'B' or an 'S' type trigger. The Hazard Identification and Treatment System is prescriptive on the team and level of personnel to respond to each of these risks. A different blast pattern was developed to cater for the high-risk poor rock mass quality 'S' type conditions. The following pie chart in Figure 15***Error! Reference source not found.*** shows the risk-based ground distribution.

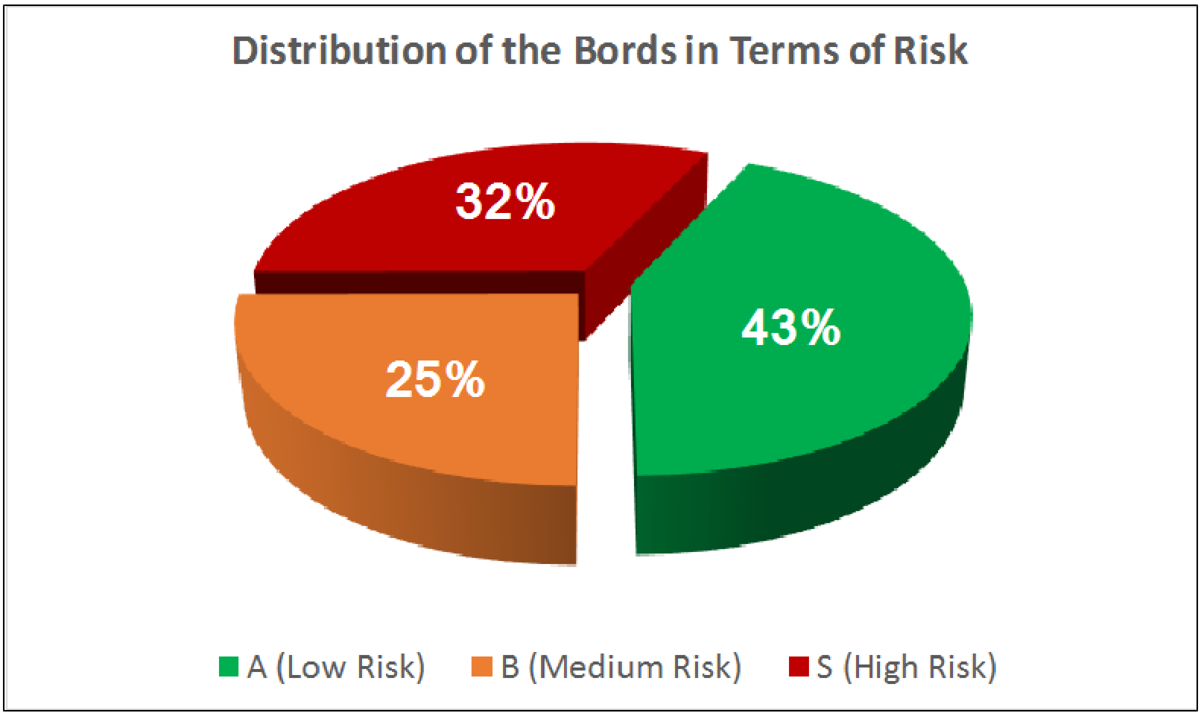

Figure 15: Risk distribution in the active bords (Mandingaisa, 2018)

The 'S' type ground conditions are associated with ground challenges that normally require the installation of long tendons in the form of pre-tensioned full column grouted cable anchors. These are often associated with delays in production cycles, as the installation process requires that the bolter operator drills holes, and a separate team comes and installs the cable anchors using a suite of tensioning cropping and grouting.

Table 2: Hazard Identification and Treatment System ABS-P Triggers involved in the assessment

	A	B	S
1	Support missing	Whole line of support missing	
2	Brow smaller than 150mm	Brow larger than 150mm but smaller than 500mm	Brow larger than 500mm
3	Normal ground	Dyke	Flat dipping feature less than 60º to horizontal
4		Fault	Dome
5			Water dripping from rock wall
6		Fall of ground less than support spacing	Fall of ground greater than support spacing
7		Material change in face composition for example pothole	
8		Abnormal blocky ground	Pillar scaling

3 LITERATURE REVIEW

3.1 Rock Mass Quality

Stacey (2001) defines rock mass quality or rock mass classification as a means of quantifying the quality of the rock mass by ascribing a numerical value to the quality, and further states that rock mass classification has significantly functioned as an essential pillar of empirical designs, leading to frequent use and attention by rock engineers. Several systems were developed by different engineers from different fields of expertise ranging from civil engineering, tunnelling to mining. The main uses of rock mass classification have been to assess rock mass quality, and to estimate rock support requirements, excavation stability, raise bore stability, tunnel boring rates etc. Rahmannejad and Mohammadi (2007) state that the maximum benefit of rock mass classification systems is derived during preliminary and feasibility stages of a project during which very little information in terms of rock mass properties and quality is known. The systems provide a way to estimate the rock mass strength and deformation properties, and rock mass characteristics, as discussed by Stacey (2016). It should be noted that a clear understanding of the different fields of application for these systems is essential for successful and correct application.

This research will apply the geomechanics classification system (Rock Mass Rating, RMR) as developed and updated by Bieniawski (1973) and modified by Bieniawski in 1989, as well as the NGI Q system as developed and updated by Barton et al (1974), Barton (2002) and NGI (2015) to quantify rock mass quality and its impact on stope width, and ultimately to evaluate the effects of rock mass quality on profitability of mining and the business plan. The limitation of this review of rock mass quality characterization is that only the classification systems intended for this research will be discussed in detail. However, a general consideration will be made for other systems of rock mass classification.

Stacey (2009) reports that rock mass classification ascribes a value to the quality of the rock mass, thereby allowing the provision of estimates of rock stability, support requirements, strength and deformation properties. According to Stille and Palmstrom (2003), these rock mass classification systems '*are an effective means of obtaining an enhanced understanding or more comprehensive overview of a phenomenon or set of data in order to make effective decisions regarding them*'.

Some of the main classification and characterization systems, their types and main applications are summarized in Table 3 below as published by Palmstrom (2000).

Table 3: Some of the main classification and characterization systems (Palmstrom, 2000)

Name of classification	Form and Type*)	Main applications	Reference
The Terzaghi rock load classification system	Descriptive and behaviouristic form Functional type	For design of steel support in tunnels	Terzaghi, 1946
Lauffer's stand-up time classification	Descriptive form General type	For input in tunnelling design	Lauffer, 1958
The new Austrian tunnelling method (NATM)	Descriptive and behaviouristic form Tunnelling concept	For excavation and design in incompetent (overstressed) ground	Rabcewicz, Müller and Pacher, 1958 - 64
Rock classification for rock mechanical purposes	Descriptive form General type	For input in rock mechanics	Patching and Coates, 1968
The unified classification of soils and rocks	Descriptive form General type	Based on particles and blocks for communication	Deere et al., 1969
The rock quality designation (RQD)	Numerical form General type	Based on core logging; used in other classification systems	Deere et al., 1967
The size-strength classification	Numerical form Functional type	Based on rock strength and block diameter; used mainly in mining	Franklin, 1975
The rock structure rating (RSR) classification	Numerical form Functional type	For design of (steel) support in tunnels	Wickham et al., 1972
The rock mass rating (RMR) classification	Numerical form Functional type	For use in tunnel, mine and foundation design	Bieniawski, 1973
The Q classification system	Numerical form Functional type	For design of support in underground excavations	Barton et al., 1974
The typological classification	Descriptive form General type	For use in communication	Matula and Holzer, 1978
The unified rock classification system	Descriptive form General type	For use in communication	Williamson, 1980
Basic geotechnical classification (BGD)	Descriptive form General type	For general use	ISRM, 1981
The Geological Strength Index (GSI)	Numerical form Functional type	For design of support in underground excavations	Hoek, 1994
The Rock Mass index (RMi) system	Numerical form Functional type	For general characterisation, design of support, TBM progress	Palmström, 1995

*) Definition of the following expressions:

Descriptive form: the input to the system is mainly based on descriptions
Numerical form: the input parameters are given numerical ratings according to their character
Behaviouristic form: the input is based on the behaviour of the rock mass in a tunnel
General type: the system is worked out to serve as a general characterisation
Functional type: the system is structured for a special application (for example for rock support)

3.1.1 Classification and characterisation

Palmstrom (1995) differentiated between rock mass characterization and classification. The differentiation between rock mass classification and characterisation is believed by most researchers in this field to be important and of significant advantage if done correctly. As defined by Palmstrom 1995, '*Rock Mass characterization*' refers to the process of assigning a number to the rock mass quality or a description. Particular features, qualities, trends and traits are described or represented by a number as characterisation. *'Rock Mass classification'*: As the name entails this involves grouping rock mass features into classes.

In strict terms *'characterization'* would have more descriptive terms than *'classification'*. However, in practice the two terms are used loosely and interchangeably. Figure 16 shows the relationship and process flow from field observations or measurements to characterisation and the ultimate application.

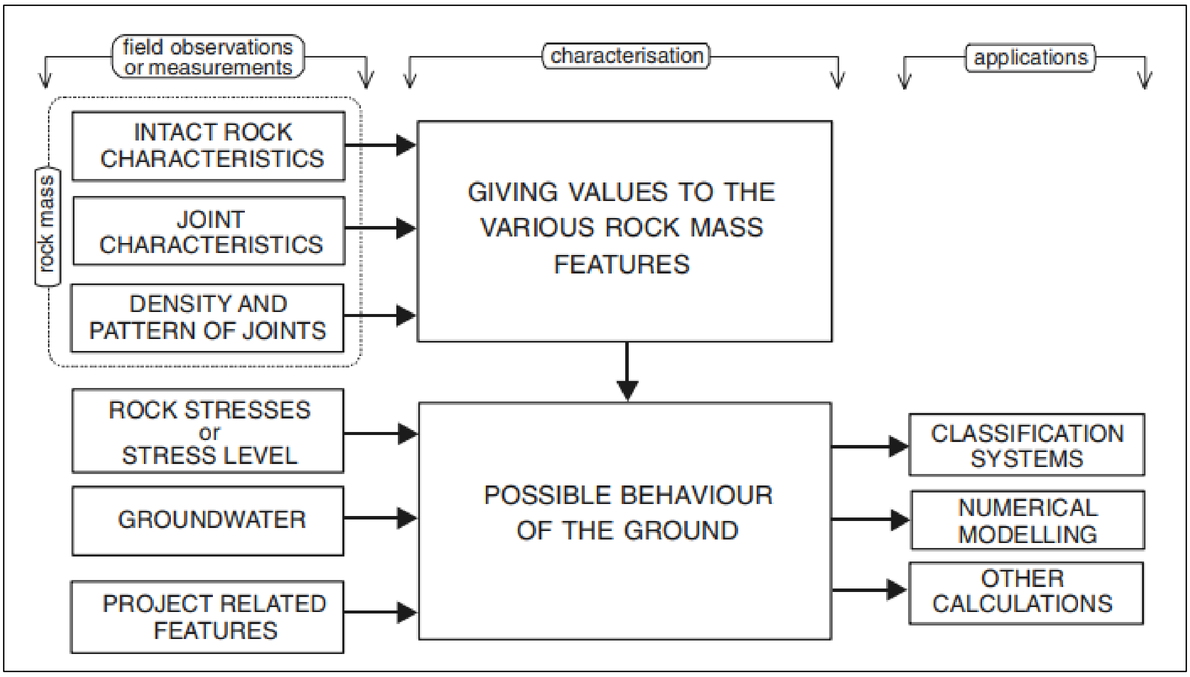

Figure 16: Illustration of characterization of joints as the core of rock engineering (Palmstrom et al, 2001)

3.1.2 Rock Quality Designation (RQD)

As early as 1967, Deere et al (1967) developed the Rock Quality Designation (RQD) Index. This index is defined by the lengths of intact core pieces greater/longer than 10 cm expressed as a percentage of the total length of core in the core run.

This is given by the following formula:

$$RQD = \frac{\sum Length\ (L)\ of\ core\ pieces > 10\ cm\ in\ length}{Total\ length\ of\ core\ run} * 100\% \qquad \text{Eqn. (1)}$$

Usually NX core size is used in the determination of RQD but this can be done on other core sizes. The definition of the index clearly indicates that measurement of RQD is unidirectional. In the determination of this index, only joints are considered and all core mechanical breaks due to drilling or handling practice are ignored. *Figure 17* illustrates the calculation of RQD, after Deere (1989).

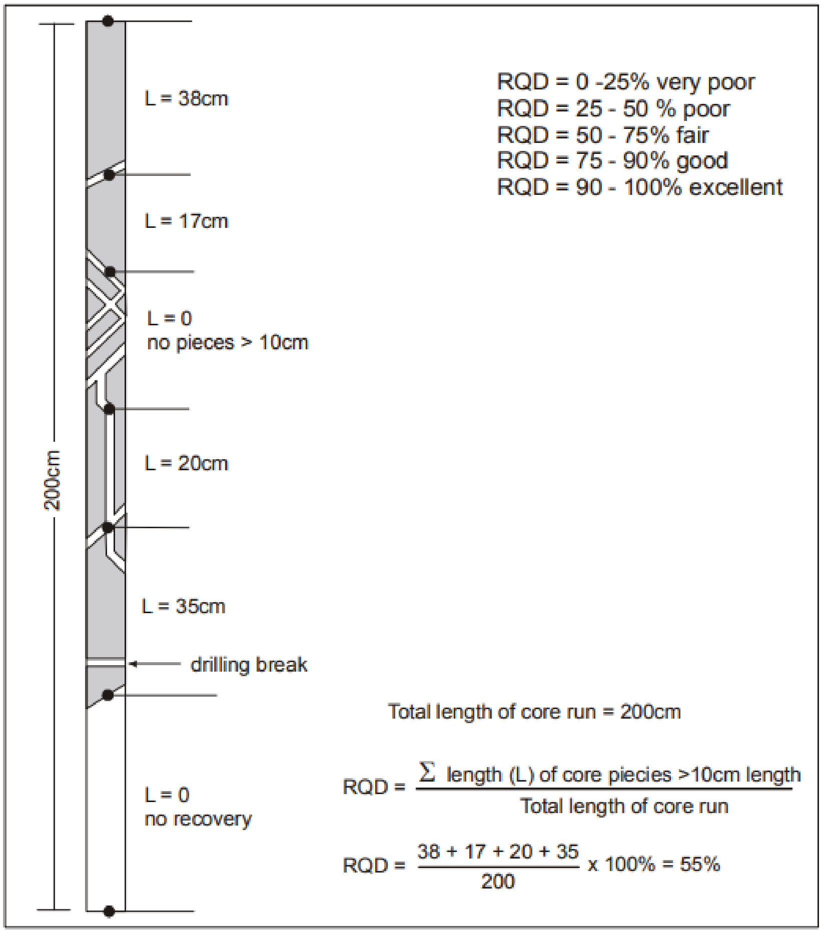

Figure 17: The procedure of determination of RQD using diamond drill core (after Deere, 1989)

Deere (1989) established correlation between RQD and rock mass quality as given in *Table 4.*

Table 4: Correlation of RQD and rock mass quality (After Deere, 1989)

RQD (%)	**Rock Quality**
< 25	Very Poor
25- 50	Poor
50- 75	Fair
75- 90	Good
90- 100	Excellent

3.1.3 Volumetric Joint count (Jv)

Similarly, Palmstrom (1982) came up with the concept of volumetric joint count (J_v). Palmstrom et al (2001) define J_v as a measure of the number of joints intersecting a volume of rock mass or simply joints per cubic metre where the joints exist as joint sets. However, it cannot be taken for granted that all joints will exist in sets as random joints are also common. Grenon and Hadjigeorgiou (2003) believed that ignoring random joints would result in erroneous quantification of the existence of joints in the rock mass. Palmstrom et al (2001) presented a correction to that effect that included random joints. This correction will not be applied for purposes of this research. The volumetric joint count will be calculated from the following formula published by Palmstrom (1982):

$$\mathrm{Jv} = \frac{1}{\mathrm{J1}} + \frac{1}{\mathrm{J2}} + \frac{1}{\mathrm{J3}} + \cdots \qquad \text{Eqn. (2)}$$

Where J1, J2, J3…. are average joint spacings for joint set 1, joint set 2, joint 3 etc.

As alluded to earlier, RQD is unidirectional, and it is essential to try and represent a three-dimensional volume of a rock mass for it to be closer to reality. This can be done by using holes that penetrate a rock mass in different directions. The first attempts were carried out by Palmstrom (1974) leading to the introduction of the Jv concept, giving rise to equation 3 below which relates RQD to the volumetric joint count:

$$\mathrm{RQD} = 115 - 3.3\mathrm{Jv} \qquad \text{Eqn. (3)}$$

With RQD=0 for Jv > 35, and RQD=100 for Jv < 4.5

However, it should be noted that the correlation between Jv and RQD has a significant variation though it was converted to a linear relationship by Palmstrom (1974).

Further work by Palmstrom (2005) introduced a new relationship between RQD and Jv as RQD = 110 – 2.5J_v (for J_v between 4 and 44), which he believed is an improvement from RQD = 115 – 3.3J_v. Palmstrom (2005) still believes that RQD should be used with great care as it still has challenges in block size characterisation. For this reason, Palmstrom (2005), Bieniawski, (1984) and Milne et al. (1998) all agree that "*RQD is a practical parameter for core logging, it is not sufficient on its own to provide an adequate description of a rock mass*".

Palmstrom's 2005 formula (Eqn. 4) is applied in the correlation of RQD and Jv for purposes of this research as the author agrees with the arguments and the improved correlation laid out by Palmstrom et al (2001).

$$RQD = 110 - 2.5Jv \qquad \text{Eqn. (4)}$$

(for Jv between 4 and 44), and RQD=100 for Jv<4

3.1.4 NGI – Q system

Barton et al (1974) developed the Norwegian Geotechnical Institute (NGI) Q system of rock mass classification. The system provides a means of rock mass character determination and estimation of support requirements. The Q-values are determined from the following formula:

$$Q = \left(\frac{RQD}{Jn}\right) * \left(\frac{Jr}{Ja}\right) * \left(\frac{Jw}{SRF}\right) \qquad \text{Eqn. (5)}$$

Where: RQD = Quality Designation
J_n = joint set number
J_r = joint roughness
J_a = joint alteration
J_w = joint water reduction factor
SRF = stress reduction factor

The components of equation can be subdivided as the following:

a. Rock block size (RQD/J_n),
b. Inter-block/joint shear strength (J_r/J_a)
c. Confining/active stress (J_w/SRF)

For purposes of this research, equation 4 is applied in the calculation of RQD. The rest of the parameters are obtained by allocating numerical values (ratings) corresponding with the number of the observed properties present in the rock mass. Tables for use in this research, are as outlined in NGI (2015) and are provided in *Table 5* to *Table 10* and *Figure 18*.

The value of the Rock Quality Index Q, is then obtained by substituting all the selected values for the six parameters into equation 5. Q is measured on a scale of 1-1000. This large variability has seen criticism from other scholars. The criticism however, does not override the benefits derived from correct applications of this system in the field of mining and tunnelling. The system is also used for support recommendations.

Table 5: Rock Quality Designation (RQD) and volumetric joint count (Jv)

1	RQD (Rock Quality Designation)		RQD
A	Very poor	(> 27 joints per m^3)	0-25
B	Poor	(20-27 joints per m^3)	25-50
C	Fair	(13-19 joints per m^3)	50-75
D	Good	(8-12 joints per m^3)	75-90
E	Excellent	(0-7 joints per m^3)	90-100
Note: i) Where RQD is reported or measured as ≤ 10 (including 0) the value 10 is used to evaluate the Q-value ii) RQD-intervals of 5, i.e. 100, 95, 90, etc., are sufficiently accurate			

Table 6: Joint Set Number (Jn) – values

2	Joint set number	J_n
A	Massive, no or few joints	0.5-1.0
B	One joint set	2
C	One joint set plus random joints	3
D	Two joint sets	4
E	Two joint sets plus random joints	6
F	Three joint sets	9
G	Three joint sets plus random joints	12
H	Four or more joint sets, random heavily jointed "sugar cube", etc	15
J	Crushed rock, earth like	20
Note: i) For tunnel intersections, use 3 x J_n ii) For portals, use 2 x J_n		

Table 7: Joint Roughness Number (Jr) Values

3	Joint Roughness Number	J_r
a) *Rock-wall contact, and* ***b)*** *Rock-wall contact before 10 cm of shear movement*		
A	Discontinuous joints	4
B	Rough or irregular, undulating	3
C	Smooth, undulating	2
D	Slickensided, undulating	1.5
E	Rough, irregular, planar	1.5
F	Smooth, planar	1
G	Slickensided, planar	0.5
Note: i) Description refers to small scale features and intermediate scale features, in that order		
c) *No rock-wall contact when sheared*		
H	Zone containing clay minerals thick enough to prevent rock-wall contact when sheared	1
Note: ii) Add 1 if the mean spacing of the relevant joint set is greater than 3 m (dependent on the size of the underground opening) iii) J_r = 0.5 can be used for planar slickensided joints having lineations, provided the lineations are oriented in the estimated sliding direction		

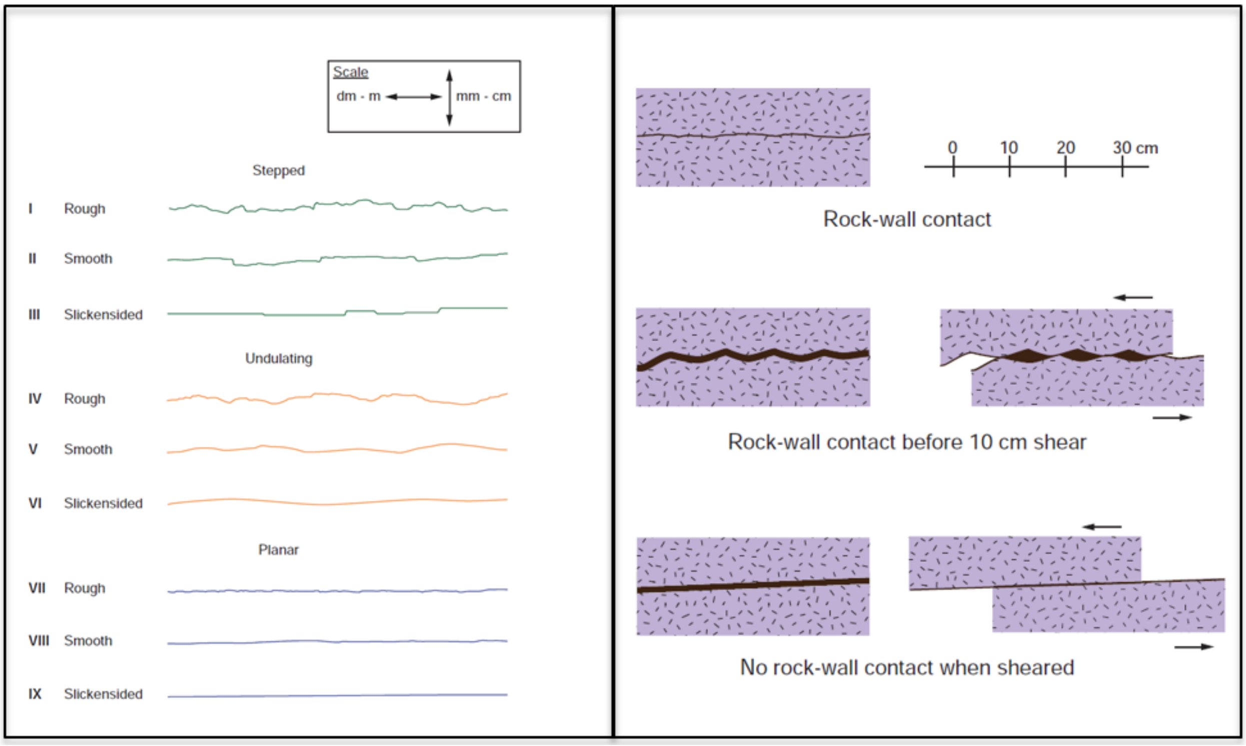

Figure 18: Joint Roughness and Rock-wall contact

Table 8: Joint alteration number (Ja) - values

4	Joint Alteration Number	Φ_r approx.	J_a
a) Rock-wall contact (no mineral fillings, only coatings)			
A	Tightly healed, hard, non-softening, impermeable filling, i.e., quartz or epidote.		0.75
B	Unaltered joint walls, surface staining only.	25-35°	1
C	Slightly altered joint walls. Non-softening mineral coatings; sandy particles, clay-free disintegrated rock, etc.	25-30°	2
D	Silty or sandy clay coatings, small clay fraction (non-softening).	20-25°	3
E	Softening or low friction clay mineral coatings, i.e., kaolinite or mica. Also chlorite, talc gypsum, graphite, etc., and small quantities of swelling clays.	8-16°	4
b) Rock-wall contact before 10 cm shear (thin mineral fillings)			
F	Sandy particles, clay-free disintegrated rock, etc.	25-30°	4
G	Strongly over-consolidated, non-softening, clay mineral fillings (continuous, but <5mm thickness).	16-24°	6
H	Medium or low over-consolidation, softening, clay mineral fillings (continuous, but <5mm thickness).	12-16°	8
J	Swelling-clay fillings, i.e., montmorillonite (continuous, but <5mm thickness). Value of J_a depends on percent of swelling clay-size particles.	6-12°	8-12
c) No rock-wall contact when sheared (thick mineral fillings)			
K	Zones or bands of disintegrated or crushed rock. Strongly over-consolidated.	16-24°	6
L	Zones or bands of clay, disintegrated or crushed rock. Medium or low over-consolidation or softening fillings.	12-16°	8
M	Zones or bands of clay, disintegrated or crushed rock. Swelling clay. J_a depends on percent of swelling clay-size particles.	6-12°	8-12
N	Thick continuous zones or bands of clay. Strongly over-consolidated.	12-16°	10
O	Thick, continuous zones or bands of clay. Medium to low over-consolidation.	12-16°	13
P	Thick, continuous zones or bands with clay. Swelling clay. J_a depends on percent of swelling clay-size particles.	6-12°	13-20

Table 9: Joint water Reduction factor (Jw) - values

5	Joint Water Reduction Factor	J_w
A	Dry excavations or minor inflow (humid or a few drips)	1.0
B	Medium inflow, occasional outwash of joint fillings (many drips/"rain")	0.66
C	Jet inflow or high pressure in competent rock with unfilled joints	0.5
D	Large inflow or high pressure, considerable outwash of joint fillings	0.33
E	Exceptionally high inflow or water pressure decaying with time. Causes outwash of material and perhaps cave in	0.2-0.1
F	Exceptionally high inflow or water pressure continuing without noticeable decay. Causes outwash of material and perhaps cave in	0.1-0.05
Note: i) Factors C to F are crude estimates. Increase J_w if the rock is drained or grouting is carried out ii) Special problems caused by ice formation are not considered		

Table 10: Stress Reduction Factor (SRF) - values

6	Stress Reduction Factor			SRF
a)	*Weak zones intersecting the underground opening, which may cause loosening of rock mass*			
A	Multiple occurrences of weak zones within a short section containing clay or chemically disintegrated, very loose surrounding rock (any depth), or long sections with incompetent (weak) rock (any depth). For squeezing, see 6L and 6M			10
B	Multiple shear zones within a short section in competent clay-free rock with loose surrounding rock (any depth)			7.5
C	Single weak zones with or without clay or chemical disintegrated rock (depth ≤ 50m)			5
D	Loose, open joints, heavily jointed or "sugar cube", etc. (any depth)			5
E	Single weak zones with or without clay or chemical disintegrated rock (depth > 50m)			2.5
Note:	i) Reduce these values of SRF by 25-50% if the weak zones only influence but do not intersect the underground opening			
b)	*Competent, mainly massive rock, stress problems*	σ_c/σ_1	σ_θ/σ_c	**SRF**
F	Low stress, near surface, open joints	>200	<0.01	2.5
G	Medium stress, favourable stress condition	200-10	0.01-0.3	1
H	High stress, very tight structure. Usually favourable to stability. May also be unfavourable to stability dependent on the orientation of stresses compared to jointing/weakness planes*	10-5	0.3-0.4	0.5-2 2-5*
J	Moderate spalling and/or slabbing after > 1 hour in massive rock	5-3	0.5-0.65	5-50
K	Spalling or rock burst after a few minutes in massive rock	3-2	0.65-1	50-200
L	Heavy rock burst and immediate dynamic deformation in massive rock	<2	>1	200-400
Note:	ii) For strongly anisotropic virgin stress field (if measured): when $5 \le \sigma_1/\sigma_3 \le 10$, reduce σ_c to 0.75 σ_c. When $\sigma_1/\sigma_3 > 10$, reduce σ_c to 0.5 σ_c, where σc = unconfined compression strength, σ_1 and σ_3 are the major and minor principal stresses, and σθ = maximum tangential stress (estimated from elastic theory) iii) When the depth of the crown below the surface is less than the span; suggest SRF increase from 2.5 to 5 for such cases (see F)			
c)	*Squeezing rock: plastic deformation in incompetent rock under the influence of high pressure*		σ_θ/σ_c	**SRF**
M	Mild squeezing rock pressure		1-5	5-10
N	Heavy squeezing rock pressure		>5	10-20
Note:	iv) Determination of squeezing rock conditions must be made according to relevant literature (i.e. Singh et al., 1992 and Bhasin and Grimstad, 1996)			
d)	*Swelling rock: chemical swelling activity depending on the presence of water*			**SRF**
O	Mild swelling rock pressure			5-10
P	Heavy swelling rock pressure			10-15

3.1.4.1 *Limitations and requirements for use of the Q system*

The Q system is no stranger to criticism as its application has been challenged by several scholars and practitioners. Palmstrom and Broch (2006), advocated that the Q-system doesn't work best in extremely poor and good rock and refined the usage of Q in support recommendations to the region shown in *Figure 19*.

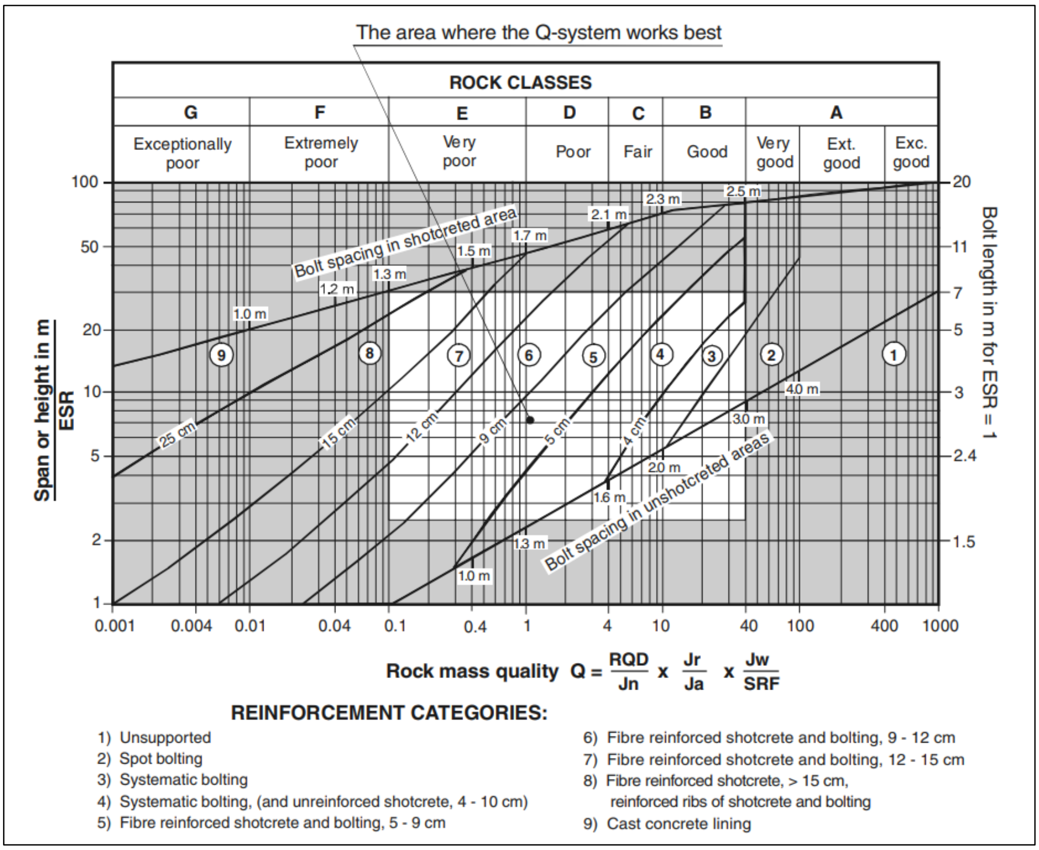

Figure 19: Zone in which Q works well in support estimation (unshaded), outside the unshaded zone other methods and rationale should be applied (after Palmstrom et al., 2002)

Stacey (2001) highlighted that the Q system does not take into account the orientation of joints but takes the movement potential of blocks as more important, rendering the system inapplicable and not relevant for situations where joint orientation plays an important role in block stability. Like other rock mass rating systems, the Q system is dependent on the experience of the user. It is important to note that the method is suitable for assessing rock mass conditions, stability of openings and the selection of ground support requirements provided the appropriate values are used for the stress and geotechnical parameters. This method is good for feasibility studies in which there is very little to limited information available. This review is critical as it provided the author with a guiding framework in the use of this system in this research.

Several modifications for different applications have been made to the RMC systems leading to a wide spectrum of applications and several successful applications of RMC in excavation and support design. However, a school of thought by Pells and Bertuzzi (2008) says "... tunnel design should be done by methods of applied mechanics, like any other structural design. Classification systems are good for communication and in many cases good for producing correlations in particular geological environments. However, on the basis of experience set out in this paper they should NOT be used as the primary tool for the design of primary support."

The Q system has been no exception to this type of criticism. Peck and Lee (2007), after studying support in 15 Australian mines, came to the conclusion that *'there is no simple relationship between Q and the installed support capacity'.* Thus, they disputed the capability of the empirical charts from the Q charts in recommending support for mining purposes. Palmstrom and Broch (2006) stated that, though several design charts based on RMC exist, these should never be used to replace proper design procedures.

3.1.5 Rock Mass Rating (RMR) system (Geomechanics classification)

Bieniawski (1973) decided on 5 parameters that influenced rock mass strength and estimated their importance which he referred to as ratings. The system was later modified from case histories. As the number of case histories increased, the system evolved with time. The classification in this system is obtained by determining ratings for each parameter and summing up the ratings of the 5 parameters and adjusting the result for joint orientation. However, some scholars prefer referring to the summing up of 6 parameters. This is immaterial but an issue of preference. The parameters are namely:

a. Rock mass strength (UCS)
b. Rock Quality Designation
c. Joint spacing
d. Joint roughness and separation
e. Ground water
f. Orientation adjustment

The values of RMR range from 0 to 100. The method of classification involves the subdivision of the area to be rated into zones of similar structural features or based on changes in rock type. According to Hoek (2006), the limits/boundaries of these zones are usually marked by a structure such as a fault or dyke, or by a change in lithology. In some instances, a change in fracture frequency or rock mass characteristics within the same rock unit, may necessitate splitting the rock mass into smaller zones. This classification system does not consider the stresses present in the rock mass. Table *11* gives the parameters and their ratings.

Table 11: Rock Mass Rating (RMR) system after Bieniawski, 1989)

A. CLASSIFICATION PARAMETERS AND THEIR RATINGS

	PARAMETER		Range of values // ratings						
1	Strength of intact rock material	Point-load strength index	> 10 MPa	4 - 10 MPa	2 - 4 MPa	1 - 2 MPa	For this low range uniaxial compr. strength is preferred		
		Uniaxial com-pressive strength	> 250 MPa	100 - 250 MPa	50 - 100 MPa	25 - 50 MPa	5 - 25 MPa	1 - 5 MPa	< 1 MPa
		RATING	**15**	**12**	**7**	**4**	**2**	**1**	**0**
2	Drill core quality RQD		90 - 100%	75 - 90%	50 - 75%	25 - 50%	< 25%		
		RATING	**20**	**17**	**13**	**8**	**5**		
3	Spacing of discontinuities		> 2 m	0.6 - 2 m	200 - 600 mm	60 - 200 mm	< 60 mm		
		RATING	**20**	**15**	**10**	**8**	**5**		
4	Condition of discon-tinuities	Length, persistence	< 1 m	1 - 3 m	3 - 10 m	10 - 20 m	> 20 m		
		Rating	**6**	**4**	**2**	**1**	**0**		
		Separation	none	< 0.1 mm	0.1 - 1 mm	1 - 5 mm	> 5 mm		
		Rating	**6**	**5**	**4**	**1**	**0**		
		Roughness	very rough	rough	slightly rough	smooth	slickensided		
		Rating	**6**	**5**	**3**	**1**	**0**		
		Infilling (gouge)	none	Hard filling		Soft filling			
			-	< 5 mm	> 5 mm	< 5 mm	> 5 mm		
		Rating	**6**	**4**	**2**	**2**	**0**		
		Weathering	unweathered	slightly w.	moderately w.	highly w.	decomposed		
		Rating	**6**	**5**	**3**	**1**	**0**		
5	Ground water	Inflow per 10 m tunnel length	none	< 10 litres/min	10 - 25 litres/min	25 - 125 litres/min	> 125 litres /min		
		p_w / σ1	0	0 - 0.1	0.1 - 0.2	0.2 - 0.5	> 0.5		
		General conditions	completely dry	damp	wet	dripping	flowing		
		RATING	**15**	**10**	**7**	**4**	**0**		

p_w = joint water pressure; σ1 = major principal stress

B. RATING ADJUSTMENT FOR DISCONTINUITY ORIENTATIONS

		Very favourable	Favourable	Fair	Unfavourable	Very unfavourable
RATINGS	Tunnels	**0**	**-2**	**-5**	**-10**	**-12**
	Foundations	**0**	**-2**	**-7**	**-15**	**-25**
	Slopes	**0**	**-5**	**-25**	**-50**	**-60**

C. ROCK MASS CLASSES DETERMINED FROM TOTAL RATINGS

Rating	100 - 81	80 - 61	60 - 41	40 - 21	< 20
Class No.	I	II	III	IV	V
Description	VERY GOOD	GOOD	FAIR	POOR	VERY POOR

D. MEANING OF ROCK MASS CLASSES

Class No.	I	II	III	IV	V
Average stand-up time	10 years for 15 m span	6 months for 8 m span	1 week for 5 m span	10 hours for 2.5 m span	30 minutes for 1 m span
Cohesion of the rock mass	> 400 kPa	300 - 400 kPa	200 - 300 kPa	100 - 200 kPa	< 100 kPa
Friction angle of the rock mass	< 45°	35 - 45°	25 - 35°	15 - 25°	< 15°

3.1.6 Correlation between RMR and NGI Q system

Palmstrom and Broch (2006) conclude that "*For the correlation between the RMR and Q classification systems, often used when experience in one system is applied in the other, the conditions are shown in Figure 20. Because of the rather poor correlation between the two systems the equation RMR = 9 ln Q + 44 or similar should be applied with great care to avoid errors*". The following relationships between Q and RMR have been proposed from a number of case studied by Bieniawski (1976) and Barton (2002)

$RMR = 9 \ln Q + 44$ (Bieniawski, 1976)

$RMR = 10 \log_{10} Q + 50$ (Barton, 1974)

The Q system was originally developed from civil engineering case histories but has found a lot of applications in mining. Figure 20 illustrates the relationship between RMR and Q and the variability in this relationship.

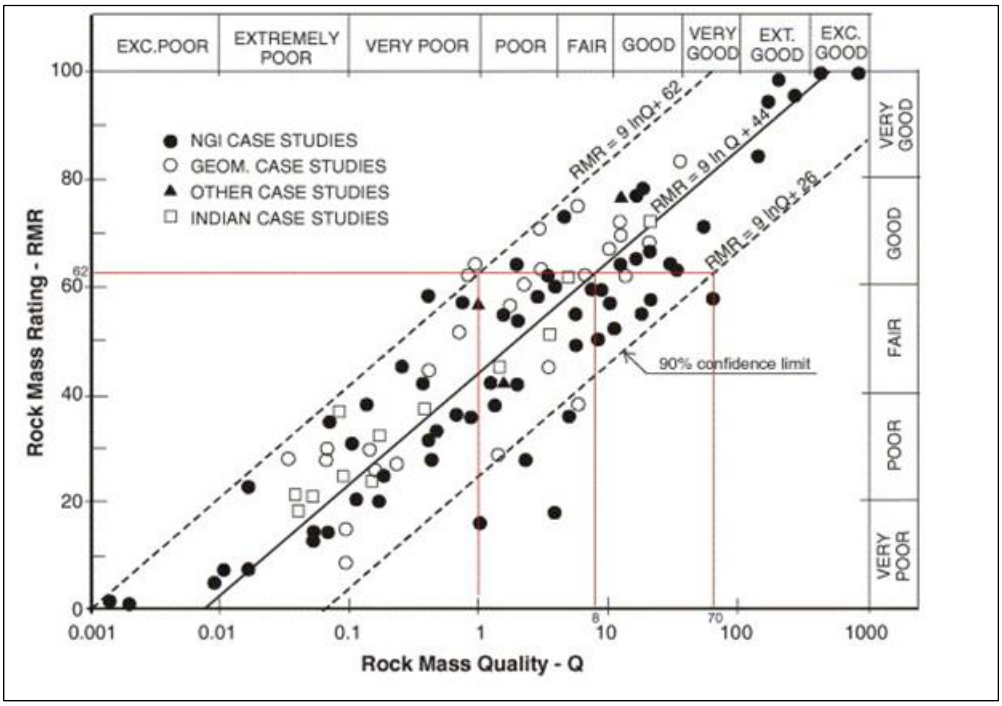

Figure 20: RMR vs Q correlation showing large variability (from Bieniawski, 1984)

3.1.7 Laubscher's Mining Rock Mass Rating (MRMR) System

Laubscher (1977) developed the MRMR system by modifying Bieniawski's (1974) RMR system. MRMR takes into account the RMR system parameters, however it combines groundwater and joint condition parameters.

Stacey (2016) documents that the parameters for MRMR are:

- Rock material strength (UCS)
- RQD
- Joint spacing
- Joint condition and groundwater

Tables are used to determine the ratings of these parameters as shown in the table below extracted from Stacey (2016). Table 12 gives the ratings for the parameters used.

Table 12: Mining Rock Mass Classification (extracted from Stacey, 2016)

Parameter		**Range of values**										
1	RQD	100-97		96-84	83-71	70-56	55-44	43-31	30-17	16-4	3-0	
	Rating = (RQD × 15/100)	15		14	12	10	8	6	4	2	0	
2	UCS (MPa)	185	184-165	164-145	144-125	124-105	104-85	84-65	64-45	44-25	24-5	4-0
	Rating	20	18	16	14	12	10	8	6	4	2	0
3	Joint spacing	Refer Figure 7										
	Rating	25										
4	Joint condition including groundwater	Refer Table in Appendix 3										
	Rating	40										

Joint spacing ratings are obtained from Figure 21, and Appendix 2 is used to obtain a rating for joint condition including groundwater, as a proportion of maximum ratings 25 and 40 respectively.

The adjustments for joint conditions and ground water parameter are cumulative. An MRMR rating is obtained by summing the ratings. The following adjustments are then applied to the MRMR value to determine an adjusted or modified MRMR.

a) Weathering
b) Joint orientation
c) Blasting
d) Mining induced stress

Tables applied for the adjustments are given in Appendix 2. This results in an adjusted MRMR.

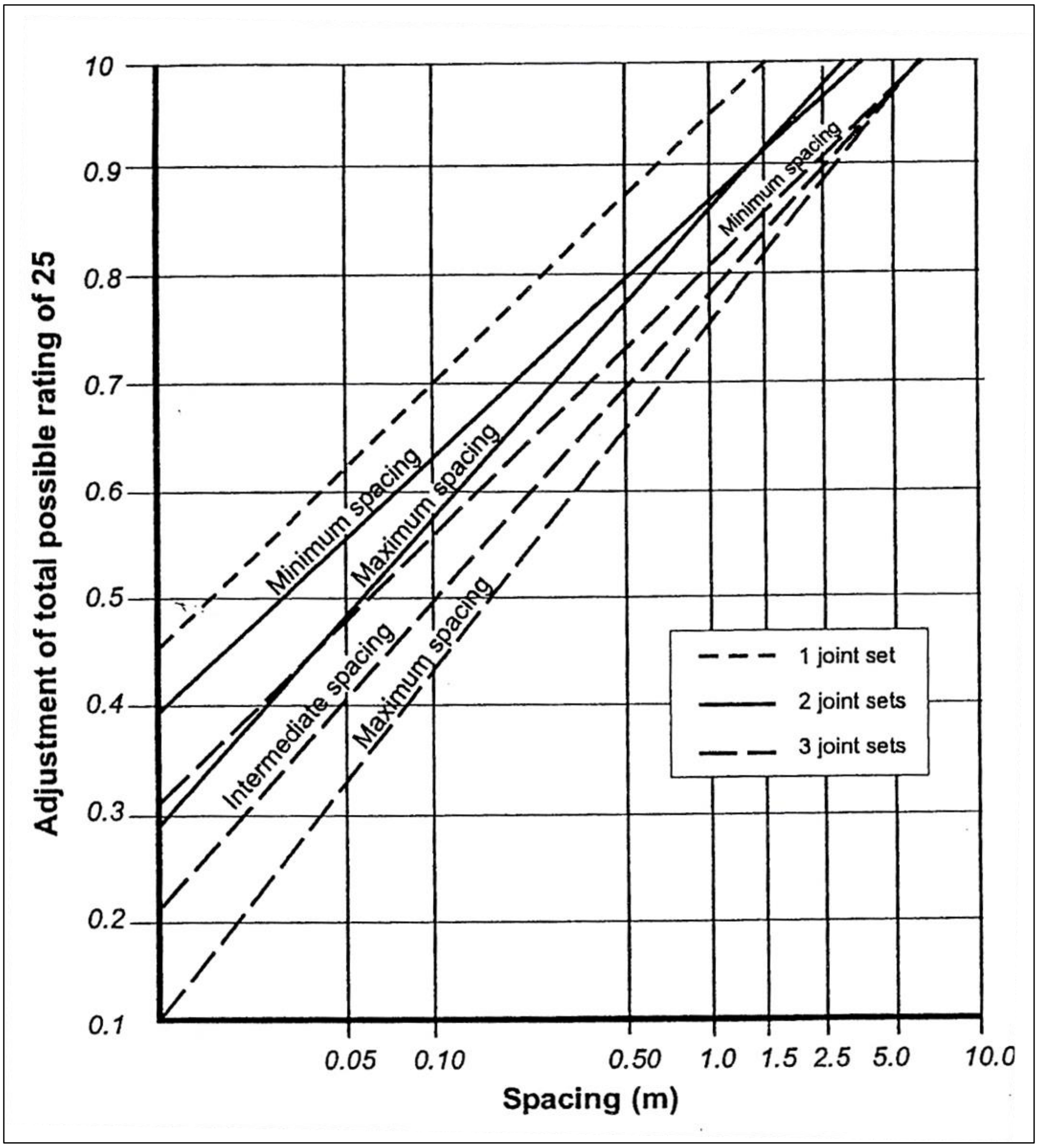

Figure 21: Determination of joint spacing rating (from Stacey, originally Laubscher, 1990)

3.1.8 Rock Mass Strength (RMS)

Laubscher (1990) defined a Rock Mass Strength given by the following formula

$$RMS = UCS * (MRMR - RUCS)/100 \qquad \text{Eqn. 6}$$

Where RUCS is the rating value in Table 12 corresponding with the rock UCS. Laubscher (1990) also defined a Design Rock Mass Strength (DRMS) which is the rock mass strength in a specific mining environment. It is obtained by applying adjustments to the RMS to consider the effects of weathering, joint orientation and method of excavation.

3.2 Drilling and Blasting (Stope width control)

Swart et al (2005) state that poor ground conditions are often aggravated by bad drilling and blasting practices. This leads to excessive stope widths which are detrimental to the ore grade produced from the stopes. Chikande and Zvarivadza (2016) indicated, in their research on the review of support systems used in poor ground conditions in platinum room and pillar mining, that stoping over-break is influenced mainly by poor ground conditions and the explosives in use. Swart et al (2005) also noted that good drilling and blasting techniques could help in stope width control narrow reefs, thereby reducing ore dilution and the associated hoisting and processing costs. However, in a production setup these facts are normally down-played, and poor ground is used to explain any dilution and loss of production. Hence the need to have a way to quantify the impact poor ground causes as well as to validate the explanations for failing to achieve the desired stope width. This research seeks to fill in this gap by providing the platinum mines on the Great dyke with a basis to challenge current underperformance explanations and drive productivity.

3.3 Ore dilution

Scoble and Moss (1994) state that the characteristics of the principal forms of dilution, both planned and unplanned, are controlled by quality factors relating to exploration, mine design and stoping practice. Jordan (2003) defines ore dilution as a complicated issue and a function of the orebody's dimensions and machine operational dimensions, dip and stoping width. Dilution in Jordan (2003) is presented as a major factor that links directly to the profitability and viability of any mining operation, i.e. the greater the dilution, the lower the mineral content per mined tonnage output. This results in less revenue per mined tonnage output, hence the impact on mine viability. By alluding to the fact that failure to mine to the correct stope width results in dilution, one accepts that there is need to understand the causes of dilution and to be able to quantify their impacts.

According to the Anglo Platinum Unki Mine Design Criteria (2005), unplanned dilution is overbreak in excess of the planned overbreak acceptable to achieve the 'best cut'.

"*Various factors influence the magnitude of the unplanned dilution, such as:*

- *Rolling reef – rolling reefs with a high amplitude and short-wave length will tend to have higher unplanned over break. Production sections were planned at a bearing 5 degrees above strike to minimize possible down dip development when rolling reef is encountered.*
- *Geotechnical conditions – poor rock mass quality conditions generally result in higher levels of hanging wall or footwall sloughing (increased dilution)*
- *Stoping width – in conjunction with the drilling method the stope width can influence the amount of dilution*
- *Drill steel length – generally increased drill steel length results in increased dilution*"

No unplanned dilution was allowed for in addition to the inefficiencies in each production section ramp up over 2 years. (Unki Mine Design Criteria, 2005)

It is in line with the spirit of the last statement above that this research is being conducted as no unplanned dilution was provided for in developing the mine's business case. The mine's business case was built with a broken ore grade of 3.82 grams per tonne yet the mine is operating nowhere near this value (at 3.46 grams per tonne). Figure 22 summarises ore dilution classification as detailed by Ercikdi et al (2003) in which dilution is either internal or external, with the internal dilution being planned or geological and external dilution is unplanned. The current research focuses on external dilution emanating from failures to comply with the designed mining cut.

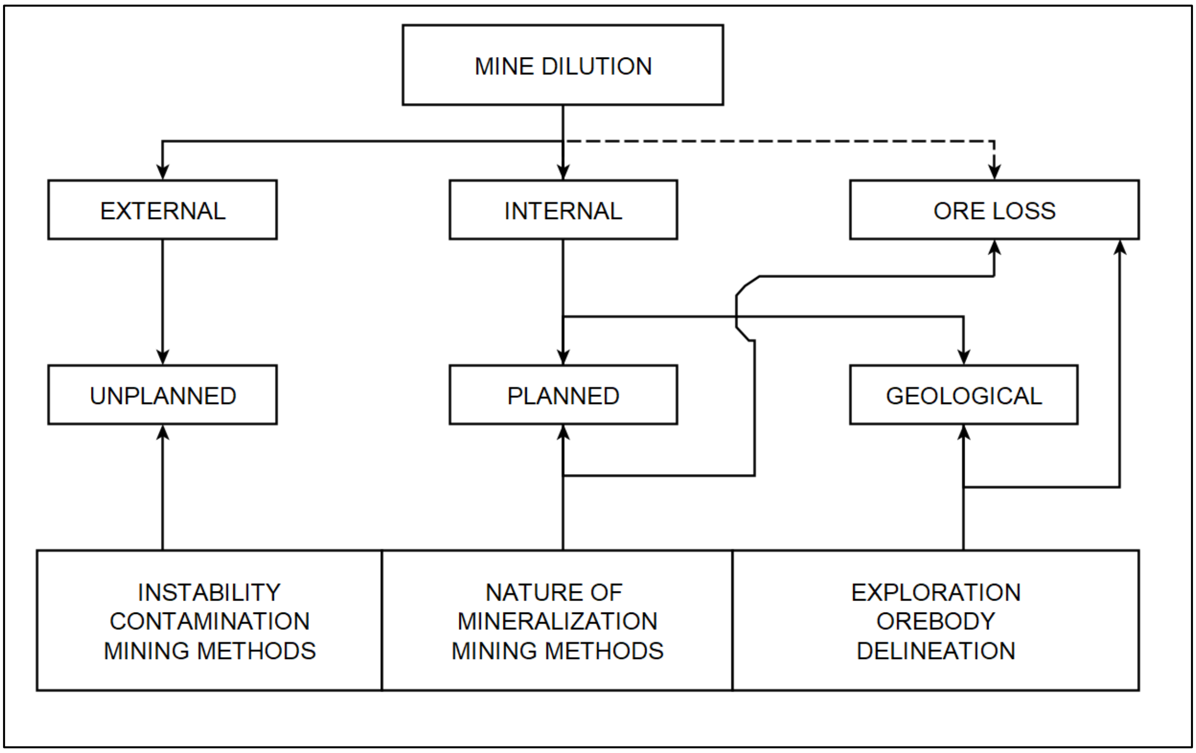

Figure 22: Classification of dilution (Ercikdi et al, 2003)

Ercikdi et al (2003) state that *"Geotechnical measurements are required to assess the response of the rock mass to the excavation process and are a key component of the mine design optimization process required to achieve safe and most economical extractions."*

3.4 Pillar design stability/Integrity

Salamon and Munro (1967), from thorough back analysis of in situ data, came up with a design formula for coal pillars in South Africa. However, very little back analysis has been done for hard rock pillars. By analysing competent rib pillars in uranium mines, Hedley and Grant (1972) developed the power formula constants (Pillar strength = $Kw_e^{0.5}/h^{0.75}$ where K=0.33 x UCS, w_e=4*Area/Perimeter (Wagner, 1974), h=height of pillar) used in hard rock pillar designs, based on the original work by Salamon and Munro (1967). Joughin and Swart (2000) and Wesseloo and Swart (2000) agree in their conclusions that, due to lack of anything better to apply in the designing of stable pillars, rock engineers apply the constants derived by Hedley and Grant (1972) in geotechnical environments different from the one in which they were derived. However, Stacey and Page (1986) suggested that the K value in this formula should be the Design Rock Mass Strength (DRMS) derived from Laubscher's (1990) Mining

Rock Mass Rating (MRMR) system. All redesigns at Unki mine utilises this approach in order to incorporate rock mass quality in the design process.

Pillar strength and integrity are usually measured in the form of safety factor. Sofianos et al (2013) defined safety factor as the structural capacity of a system beyond the expected actual loads i.e. how much stronger a system needs, to be for the expected load. It is measured as the ratio of capacity of the system versus the demand. The design of stable pillar systems in shallow narrow reef hard rock mines is dictated by the existence of large tensile stresses in the hanging wall and geological weaknesses in the hanging wall rock mass (Ozbay et al, 1995). The safety factor for pillars is given by the ratio of the pillar strength (described earlier in this section) to the average pillar stress (**ρgh/[1-e]** where **ρ**=rock density, **g**=acceleration due to gravity, **h**=depth below surface and **e** =extraction) as defined in Swart et al (2005). The pillars in shallow, narrow reef hard-rock tabular mines are designed not to yield or fail, with a safety factor of not less than 1.6 a figure adopted from coal rock engineering according to Ryder and Ozbay (1990).

According to Mortazavi et al (2009), a pillar fails in a process that has been demonstrated by many researchers. The failure follows phases given below.

"a) Pillar is in virgin conditions.

b) Stresses rise at pillar sidewalls with stress at the core relatively unchanged.

c) Pillar sidewalls fracture and fail and stresses in pillar core start increasing.

d) Sidewall fracture zones grow larger and pillar core approaches yield conditions

e) The fracture zones convergence

f) Ground around failed pillar destresses"

(Extracted from Mortazavi et al, 2009)

Ozbay, Salamon and Lee (2001) summarise the failure of a pillar as mainly governed by confinement of pillar core, the strength of pillar rock mass, the mechanical behaviour of the pillar-host rock and the pillar end conditions. Mortazavi et al (2009), after simulating different w/h ratios of pillars in the lab and conducting analysis of failures of pillars through numerical modelling, came to the conclusion that pillar w/h ratio is an important factor influencing pillar behaviour since pillar shape governs pillar stiffness and strength properties. In the light of this work it is important that the w/h ratio of pillars showing signs of instability (scaling or stress fracturing) be examined, as well as the associated rock mass quality they were developed in. This will enable the impact of rock mass quality, to be estimated from the rehabilitation costs, if the pillars were developed in a poor rock mass. If the pillars were developed in good rock mass quality or better, the factors affecting the strength of the pillars will need to be investigated and the rehabilitation costs be allocated to those factors.

3.5 Geotechnical and Business Risk

In order to appreciate risk, it is important to define basic terms used in risk management. Brown (2012) quotes the definitions from Standards Australia (2009). All the definitions of risk quoted in this research report are as quoted by Brown (2012). Standards Australia (2009) defines:

Risk as "*the effect of uncertainty on objectives* "

Risk source as an *"element which alone or in combination has the potential to give rise to a risk"*.

Hazard as *"a source of potential harm"*

Level of risk as *"magnitude of a risk or combination of risks, expressed in terms of the combination of consequences and their likelihood"*

Event as an *"occurrence or change of a particular set of circumstances"*

Consequence of an event as the *"outcome of an event affecting objectives"*

Likelihood is the "*chance that something will happen*"

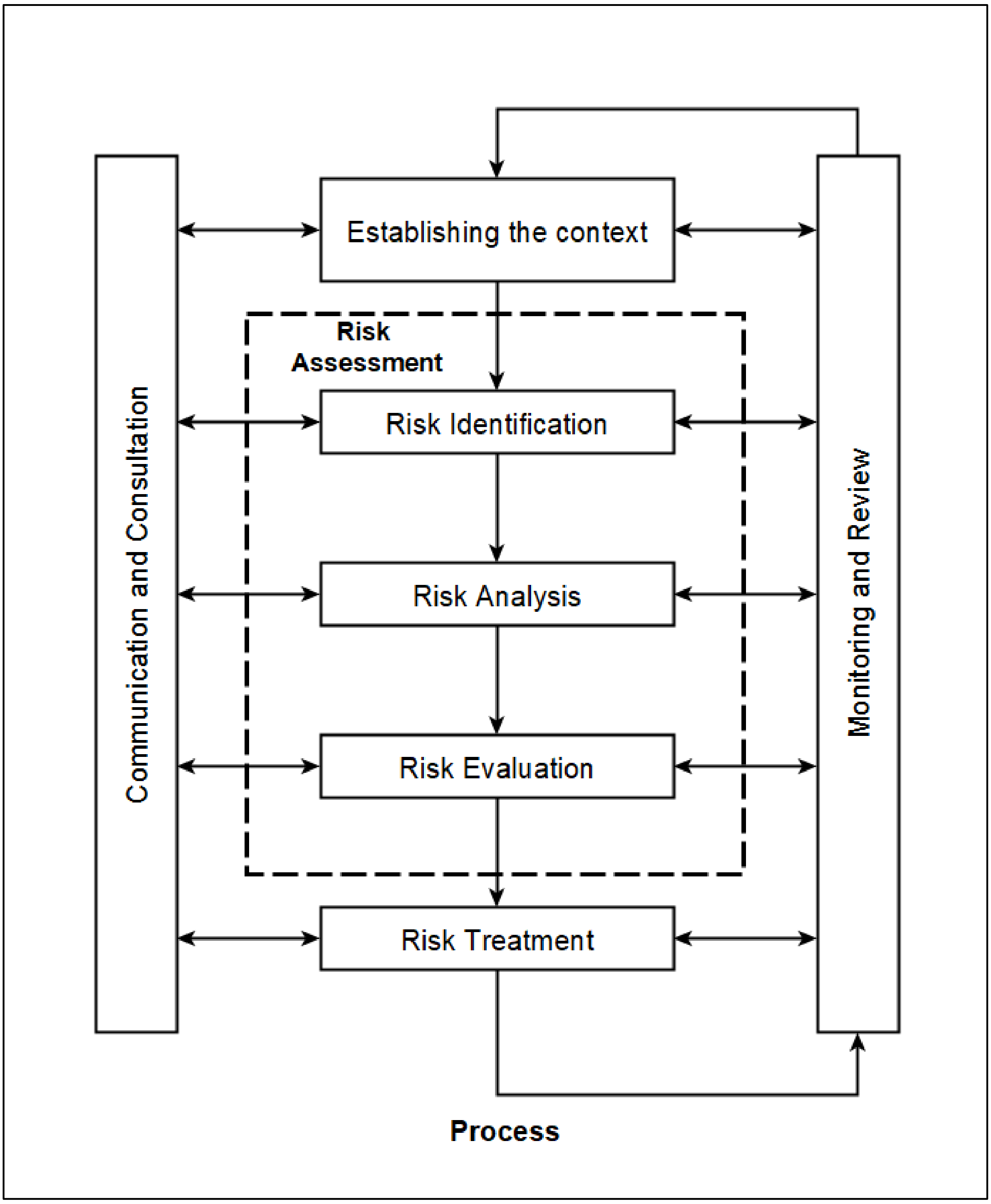

Figure 23: Risk Management Process (Standards Australia, 2009)

According to Brown (2012), the risk identification, risk analysis and risk evaluation steps are together known as risk assessment as illustrated in Figure 23. In the process of risk management in a mining research, Steffen (2002) is of the opinion that the major risks are related to understanding the geology in respect of mineralogical, structural, rock mass quality and rock mass performance. According to Steffen (2002) "….. *the thesis within the financial community is that greater rewards are possible when a higher risk is tolerated, which therefore establishes the risk/reward relationship. A formal definition of risk followed is the following: Risk = {P* *(event) x (Consequences of the event)}. In the case of mining enterprises, the consequences could be safety of personnel, financial or company image. This definition forms the cornerstone on which mining related risks are evaluated quantitatively*." Fall of ground is a safety-associated geotechnical risk attributable to the existence of poor rock mass quality, hence a contributing factor to business risk. Based on Steffen's (2002) definition, its contribution therefore can be evaluated quantitatively. Risk can also be subjective or objective with the objective, risk being definitive and the subjective risk being open ended (Palisade, 2016)

Collan (2011) concludes that key uncertainties of mining researches in the literature are identified as price and orebody uncertainty, both of which should be included in a proper investment analysis. Orebody uncertainty includes geotechnical risk, which incorporates poor rock mass conditions, giving rise to the need for a process of assessing, quantifying and ascribing relevant financial values to its impacts. Steffen (2002) concludes that future price remains a major uncertainty and a risk factor, as there is very little that can be done to control it. The author of this document believes that the long-term impacts of operational encountered geotechnical risk is often down-played as management usually does not have financials to visualise the impact of this risk to the business.

During the mining process, in short term planning it is important to ensure that the plan is underpinned by a concise risk assessment for it to be a robust plan. Failure to adhere to such a process leads to surprises that often culminate in non-achievement of the plan.

4 METHODOLOGY

This research involves the investigation and quantification of the effects of rock mass quality on stope width control and pillar stability. The process adopted involves, a review of available and applicable literature as discussed in the previous chapter, the collection of stope width and rock mass quality data to establish the distribution rock mass quality, validation of the data, manipulation and processing of the data to provide information to aid decision making.

4.1 Selection of study zone

The selection of the teams and mining areas included in this research was designed to allow comparison of efficiencies and value loss/gain whilst operating in poor rock mass and good quality rock mass. Stope width control was used as the value control lever since it is directly related to the extracted ore grade (value) as discussed is chapter 1. The focus in this choice was to establish the impact of rock mass quality, quantify the related value at risk, mitigate the risk and create cost savings in the process.

4 teams were selected and included in this study. The periods chosen for the analysis were Q1-Q2 of 2016 in which Team 1 and Team 2 were achieving similar stoping heights, and Team 3 and Team 4 were also achieving similar stope widths but better than Team 1 and Team 2. However, Team 1 and Team 3 in Q1 were working in what is regarded as fair ground according to Bieniawski (1989), but considered relatively poor rock mass at Unki. This area is characterized by low angle shears and low angle N-S trending joints as J_1. The areas are located in the upper sections of the mine. Team 1 was mining in the southern section of the mine while Team 3 was mining in the North Section. In the second quarter of the year, Team 3 was moved to the lower part of the mine (good quality rock mass) to allow for redevelopment of the upper north section (in poor rock mass) after intersecting bifurcating low angle shears. See Figure 24 and Figure 25 for these locations.

Teams 2 and 4 were working in the lower sections where the ground is considered to be in the good to very good category as per the correlations of rock mass quality and RMR laid out by Bieniawski (1989). However, Team 1 and Team 2 working in different rock mass quality failed to meet the requisite stope width. The challenges were ascribed to poor rock mass quality prompting for this investigation, to quantify the actual value loss related to rock mass quality and mitigate against the loss of value i.e. generate savings for the company.

This led to a point where dimensional blasting was made top priority and answers to the following questions were being sought:

a. Why was the performance of Team 1 and Team 2 with respect to stope width similar while operating in different rock mass quality areas? What was the contribution of rock mass quality to the poor achievements by these two tems?
b. What value was the company losing due to the geotechnical risk of rock mass quality that could be mitigated, if any?
c. What measures could be put in place to correct the situation and realise value?

A 100-day programme on dimensional blasting improvement which formed part of this research was launched and spearheaded by the author of this research. Figure 24**Error! Reference source not found.** shows the rock mass quality plot for the south section and the team deployment for Team 1, Team 3 (Q2-Q4 only) and Team 4. Figure 25 shows the rock mass quality contours for the North section and the areas where team 2 and 3 (Q1 only) were working during the period of this research.

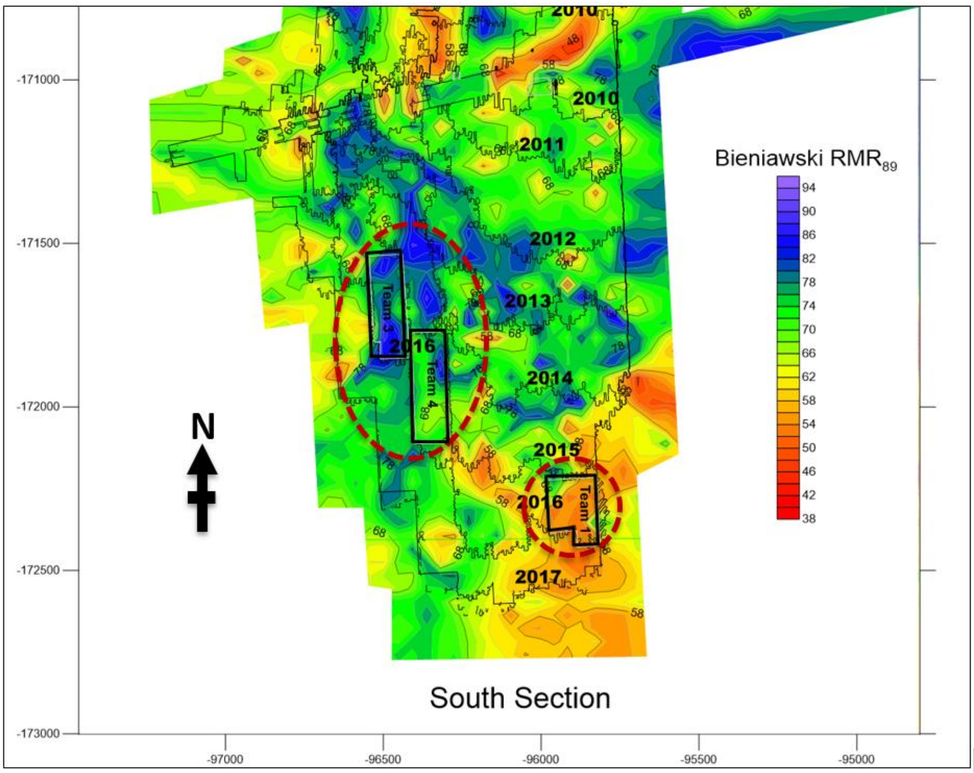

Figure 24: Rock mass quality (RMR contour) plot showing Q1-Q4 deployment or Team 1, Team 3 (Q2-Q4 only) and Team 4

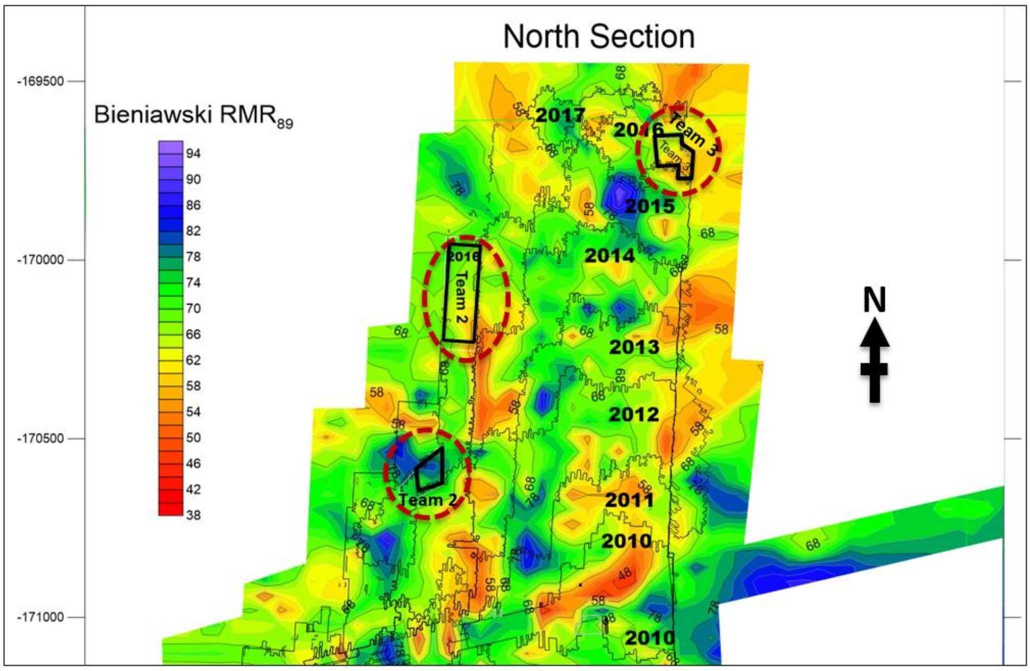

Figure 25: Rock mass quality (RMR89 contour) plot showing deployment or Team 2 and Team 3 (Q1 only)

4.2 General mine layout

As discussed in chapter 2 the standard mine layout comprises of pillars measuring 5 m on dip and 6 m on strike. The bords are designed at 12m span and a budget stope width of 204 cm. Strict adherence of mining to correct dimensions for both pillars and stope width are key to the achievement of the value related team efficiencies. As mining progresses with depth larger pillars are introduced for regional stability spaced at 96 m apart. The achievement of the design dimensions can be hampered by the existence of poor rock mass quality. This impact needs to be understood and quantified in monetary terms for sound decision making by management in the business planning process. The data collected for stope width and rock mass quality was aimed at understanding the value that can be created by putting measures to mitigate against this risk.

4.3 Data Collection and manipulation

The collected data, in the form of rock mass rating, stope widths per blast, support quantities for pillar rehabilitation, costs of support materials and production data, was collated into

Excel spreadsheets. For each set of data on a monthly basis an average and a standard deviation were computed. New data and the data in the existing mine databases were considered to come up with the areas and zones in which this research was carried out. The following databases were used:

a. Bieniawski RMR_{89} data
b. Stope width data
c. Team production figures

4.3.1 Rock mass quality data

The modal rock mass quality parameters were determined per each bord worked based on the collected data for the period of the research project. Table 13 details the rock mass quality in terms of RMR_{89} and Q rating based on the modal values of the data collected. More data is available in Appendix 3.

Table 13: Rock mass quality of the bords mined based modal ratings

Section	Bord	RQD rating	Rock strength	Spacing rating	Joint condition	Groundwater	Initial RMR	Correction factor (orientation)	Final RMR	Q
1N	1NB2	17	12	15	25	15	84	-10	**74**	**28,0**
	1NB3	13	12	15	10	15	65	-12	**53**	**2,7**
	1NB3A	17	12	10	20	15	74	-10	**64**	**9,2**
	1NB4	13	12	15	10	15	65	-2	**63**	**8,3**
	1NB5	17	12	15	20	15	79	-10	**69**	**16,1**
	1NB6	17	12	15	20	15	79	-10	**69**	**16,1**
	1NB6A	17	12	8	20	15	72	-12	**60**	**5,9**
	1NS	13	12	10	10	15	60	-5	**55**	**3,4**
2S	2SB1	20	12	10	10	15	67	-12	**55**	**3,4**
	2SB5	20	12	10	20	15	77	-12	**65**	**10,3**
3S	3SB1	20	12	15	10	15	72	-5	**67**	**12,9**
	3SB3	20	12	15	10	15	72	-12	**60**	**5,9**
	3SB4	20	12	15	10	15	72	-5	**67**	**12,9**
	3SB5	20	12	15	20	15	82	-12	**70**	**18,0**
	3SS	20	12	15	20	15	82	-12	**70**	**18,0**
6N	6NB2	17	12	10	20	15	74	-10	**64**	**9,2**
	6NB3	17	12	15	25	15	84	-10	**74**	**28,0**
	6NB4	17	12	15	20	15	79	-10	**69**	**16,1**
	6NB6	17	12	15	20	15	79	-10	**69**	**16,1**
	6NS	17	12	15	20	15	79	-5	**74**	**28,0**
6S	6SB1	20	12	15	20	15	82	-5	**77**	**39,1**
	6SB3	20	12	15	20	15	82	-5	**77**	**39,1**
	6SB6	20	12	15	20	15	82	-5	**77**	**39,1**
7N	7NB2	17	12	15	20	15	79	-2	**77**	**39,1**
	7NB3	17	12	15	20	15	79	-5	**74**	**28,0**
	7NB4	17	12	15	20	15	79	-10	**69**	**16,1**
	7NB5	20	12	10	15	15	72	-5	**67**	**12,9**
	7NS	20	12	15	25	15	87	-5	**82**	**68,2**
7S	7SB1	20	12	10	20	15	77	-5	**72**	**22,4**
	7SB2	20	12	15	20	15	82	-10	**72**	**22,4**
	7SB3	20	12	15	20	15	82	0	**82**	**68,2**
	7SB4	20	12	15	20	15	82	-5	**77**	**39,1**
	7SB4	17	12	15	20	15	79	0	**79**	**48,9**
	7SB5	20	12	20	20	15	87	0	**87**	**118,8**
	7SB6	13	12	10	10	15	60	-10	**50**	**1,9**

4.3.2 Stope width data

The stope widths data was collected in the areas mined by the selected teams and the results were later used to calculate pillar width to height (w/h) ratio and the grades of ore extracted. The stope width data was collected on daily blasts and the average monthly stope width data is reported in Table 14

Table 14: Average monthly actual stoping width

Average Monthly Stoping widths (cm)				
	Team 1	**Team 2**	**Team 3**	**Team 4**
Jan	208	209	213	203
Feb	213	207	216	203
Mar	213	213	212	203
Apr	212	210	204	206
May	208	211	203	206
Jun	210	202	200	201
Jul	209	201	201	201
Aug	203	209	204	204
Sep	205	209	201	203
Oct	209	208	205	202
Nov	203	207	206	205
Dec	204	203	206	204

The daily stope width measurements used for statistical analysis have not been included in this report as they are voluminous. Only the resultant analysis graphs are included in section 4.3.3 and chapter 5. Stope width data was also used to check whether the failing pillars were mined according to design or not by calculating w/h ratio and safety factor for the pillars. Almost all the scaling pillars had safety factor above 1.6 which is the minimum defined in the design criteria for the mine. Table14, Table 15, Table 16 and Table 17 in section 4.2.3 tabulate the scaling pillars per each team and the resultant safety factor and w/h ratio. The same information was also collected for the stable pillars.

4.3.3 Impact of rock mass quality on pillars

In order to study the impact of the poor rock mass quality on pillars, the pillars left in situ in the same period were assessed over a period of 1 year. The following data was collected for the pillars mined in that period.

a. Number of pillars scaling, rehabilitated or pillars showing stress fracturing
b. The reasons for the pillar damaged or scaling were recorded.
c. Pillar safety factor and w/h ratio was calculated for all these pillars.

Table 15, Table 16, Table 17 and Table 18 contain the data collected on the scaling pillars per each team and the calculated safety factor and width to height ratios. More data is presented in Appendix 4 including the calculations of the cost implications.

Table 15: Safety factor and w/h ratio for scaling pillars in area worked by Team 1

	Actual Pillar (m^2)	Safety Factor	W:H ratio	Perimeter
Team 1	41	2,0	3,1	23,9
	33	1,9	2,4	22,7
	31	1,9	3,2	22,5
	76	2,2	2,6	36,8
	53	2,1	3,6	28,1
	89	2,2	2,6	40,2
	63	2,0	2,3	34,0
	57	2,0	2,5	31,4
	125	2,5	3,5	55,2
	31	1,8	2,2	21,9
	37	1,9	2,3	23,4
	33	1,8	2,3	23,5
	27	1,7	2,1	20,3
	48	2,0	3,8	26,8
	40	1,9	2,1	25,6

Table 16: Safety factor and w/h ratio for scaling pillars in area worked by Team 2

	Actual Pillar (m^2)	Safety Factor	W:H ratio	Perimeter
Team 2	30	1,9	2,1	20,9
	27	1,8	2,0	19,4
	31	1,9	2,2	22,4
	38	2,0	3,5	24,2
	38	2,0	2,5	23,3
	32	2,0	2,5	21,4
	33	2,0	2,5	22,7
	28	1,9	2,2	20,3
	32	2,0	2,7	22,1
	28	1,9	2,1	20,6
	23	1,8	2,2	18,0
	28	1,9	2,1	20,6
	32	1,9	2,9	22,0
	49	2,1	2,5	26,8
	33	1,9	2,1	24,0

Table 17: Safety factor and w/h ratio for scaling pillars in area worked by Team 3

	Actual Pillar (m²)	Safety Factor	W:H ratio	Perimeter
Team 3	66	3,5	2,2	35,1
	38	3,3	2,6	23,1
	68	3,3	1,7	46,7
	27	3,0	2,1	19,7
	52	3,6	3,1	27,5
	37	3,2	2,2	23,0
	24	2,8	1,7	18,7
	39	3,2	2,5	24,1
	23	2,8	1,7	18,2
	18	2,7	1,6	16,9
	47	2,9	1,3	20,5
	36	3,2	2,3	23,0
	42	2,8	1,2	20,4

Table 18: Safety factor and w/h ratio for scaling pillars in area worked by Team 4

	Actual Pillar (m²)	Safety Factor	W:H ratio	Perimeter
Team 4	49	1,7	2,8	26,0
	34	1,6	2,3	23,4
	132	2,1	3,6	49,0
	31	1,6	2,4	23,0
	67	1,9	3,4	33,0
	93	2,0	3,5	37,8
	41	1,6	2,3	27,0
	28	1,5	2,5	19,9
	109	2,0	3,2	47,3
	118	2,0	3,0	53,0
	75	1,7	2,1	42,3
	98	1,9	2,8	49,0
	30	1,5	2,3	21,1

4.4 Ground Support

An evaluation was conducted on the support applied to the areas associated with poor ground conditions, in a bid to optimize the support installation process to improve efficiencies. This evaluation also tested if automation of the installation of support would improve the viability of mining the areas that are associated with known poor rock mass quality. Often this requires the installation of longer tendons (cable anchors) using the conventional/manual methods, automation of which could prove efficient but normally comes with increased cost per unit.

In this research, the support installation optimisations replaced the AMS barrel cable anchors which require manual installation with Flexibolts that can be installed mechanically using a bolter. A financial evaluation was conducted (detailed in Section 5.6.4) to assess the benefits of the introduction of mechanically installed Flexibolts to the system and the impact on the mining cycle.

4.5 Impact on ore grade (value)

The impacts of rock mass quality and malpractices was on grade was calculated from the achieved ore grades against the budget ore grades. The variance in grade was converted to a loss in 4E (sum of platinum, palladium, rhodium and gold) ounces and the corresponding revenue loss. After the implementation of the mitigation measures the same calculations were carried out to look at the savings generated from the research initiative. Refer to Section 5.6.4 for the profitability assessment on the introduction of Flexibolts.

5 RESULTS AND ANALYSIS

The data collected was subjected to different analysis techniques and software. All contouring of rock mass rating data was carried out using Surfer 32. The rock mass rating and stope width control data was analysed using Palisade @Risk software by generating Excel models and applying Monte Carlo simulation to simulate the outcomes. The next section describes the process of data analysis and decision making applied using the @Risk software.

5.1 Palisade @Risk software

Palisade Corporation developed @Risk software which is an Excel based package that allows analysis of business and technical situations impacted by risk. The techniques of risk analysis have been recognized as powerful tools to help decision makers to successfully manage situations that are subject to uncertainty. The following steps were followed to analyse the risk using @Risk software.

a. Develop a model - the problem definition or situation definition was carried out by developing an Excel Model.
b. Identify inputs and outputs – inputs were classified into certain and uncertain inputs. Possible values of the variables were specified as well as their probability distribution functions. Important outputs to be analysed will also be identified.
c. Analysing the Model with Simulations – several scenarios were run, each with the uncertain values depending of the probability distribution functions of the uncertain input variables, to determine the probability distribution functions of the outputs.
d. Decision making – simulated results were analysed and engineering judgement applied to generate informed decisions.

Solutions were then sought to mitigate the effects of rock mass quality on both stope width and pillar stability and implemented to check effectiveness. Ground support optimisation for the poor rock mass quality areas was also conducted and as well as a cost benefit analysis.

5.2 Data review

5.2.1 Regression and Correlation analysis

Regression analysis was used to evaluate the outliers which were excluded from the collected data used in the analysis. A correlation analysis was also carried out, on the following.

a. Rock mass quality and the stope width.
b. Pillar strength and rock mass quality.
c. Rock mass quality and pillar factor of safety.

Very low correlations could be ascertained for the majority of the studies areas except in the area where Team 1 operated. This suggested that there were other factors that influenced the failures to meet the desired stope width. Investigations were carried out to establish these factors.

5.2.2 Monte Carlo simulation

Monte Carlo simulation was applied in Palisade @Risk software to generate probability distribution functions and simulate the outcomes. This is discussed in the next chapter.

5.3 Rock Mass Quality

5329 Bieniawski RMR_{89} data points were recorded for rock mass classification underground. Each point has a corresponding NGI Rock Quality Index Q value associated with it. The mine wide database was used to determine the mean RMR_{89} and Q as well as the associated standard deviation for the mine, giving a mean RMR_{89} of 69.5 and standard deviation of 10.7, while the mean Q was 17.5 with a standard deviation of 18.7. Q was not considered for contouring due to its logarithmic nature leading to wide variability.

Four teams were selected for this research out of the 8 teams that were in production. It is important to note and appreciate that the aim of this research was to quantify the effects of rock mass quality on stope width control and pillar stability by ascribing monetary value to the effects and creating a basis for management to make sound decisions. The selected teams comprised of 2 teams that were poorly performing in terms of stope width control and mining in 'poor' ground (i.e. Team 1 and Team 2) and another 2 teams (Team 3 and Team 4) that were considered to be mining in good ground and achieving the lowest stope width at the time of the research. The ground conditions they were working in and their impacts on pillar failures and stope width achievements were evaluated and tracked for purposes of this research. This quality was measured in terms of the RMR_{89}.

Team 2 and Team 4, according to the data reported in the previous chapter, working in ground with RMR_{89} in the upper quartile ranges of >70 RMR were considered as the base case. *Figure 26***Error! Reference source not found.** shows the frequency histogram and the normalized graph (light blue line) shows that the data collected. *Figure 27* shows the frequency histogram and the cumulative frequency polygon for the RMR values.

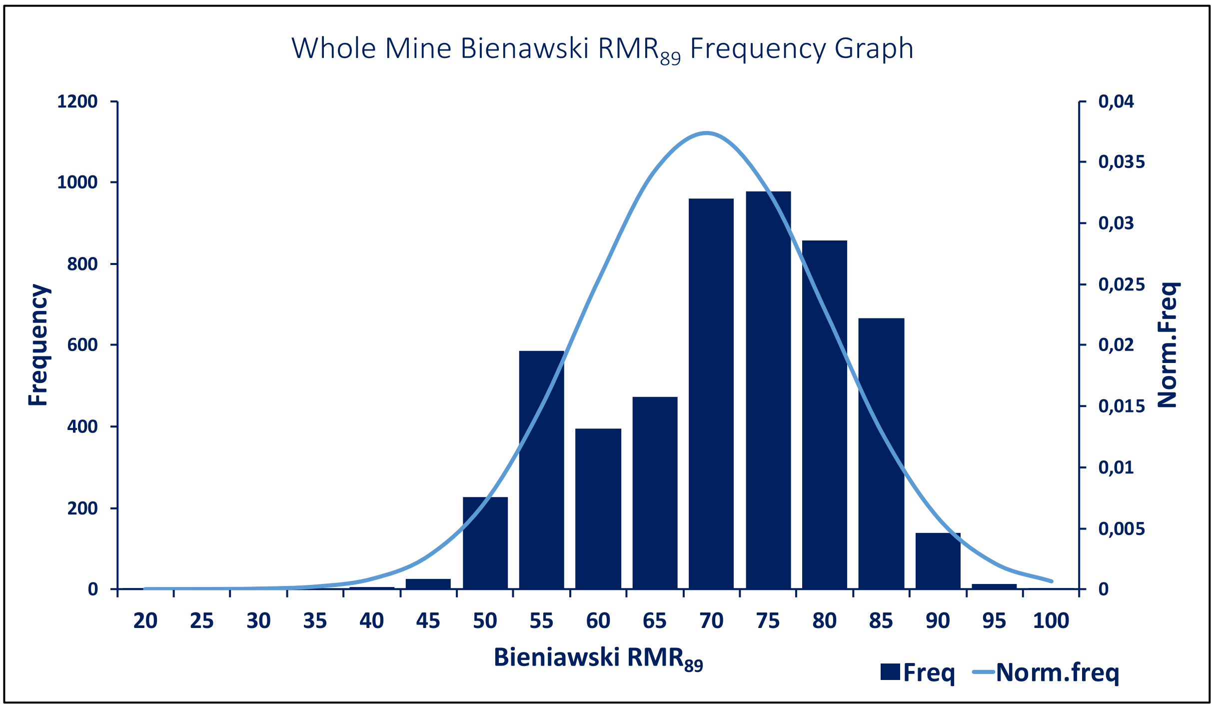

Figure 26: Whole mine RMR frequency distribution

The RMR values generally indicate that the mine is in fair to good ground conditions based on Bieniawski's 1989 correlation between RMR and rock mass quality. The RMR values were contoured to in Surfer 16 to indicate the variability of rock mass quality across the whole mined out area (Figure 28). This also allows for forward planning, as the values can be researched into the immediate areas adjacent to the mined out areas, leading to ease of predictions. A visual tool that can be applied for planning purposes (*Figure 28*).

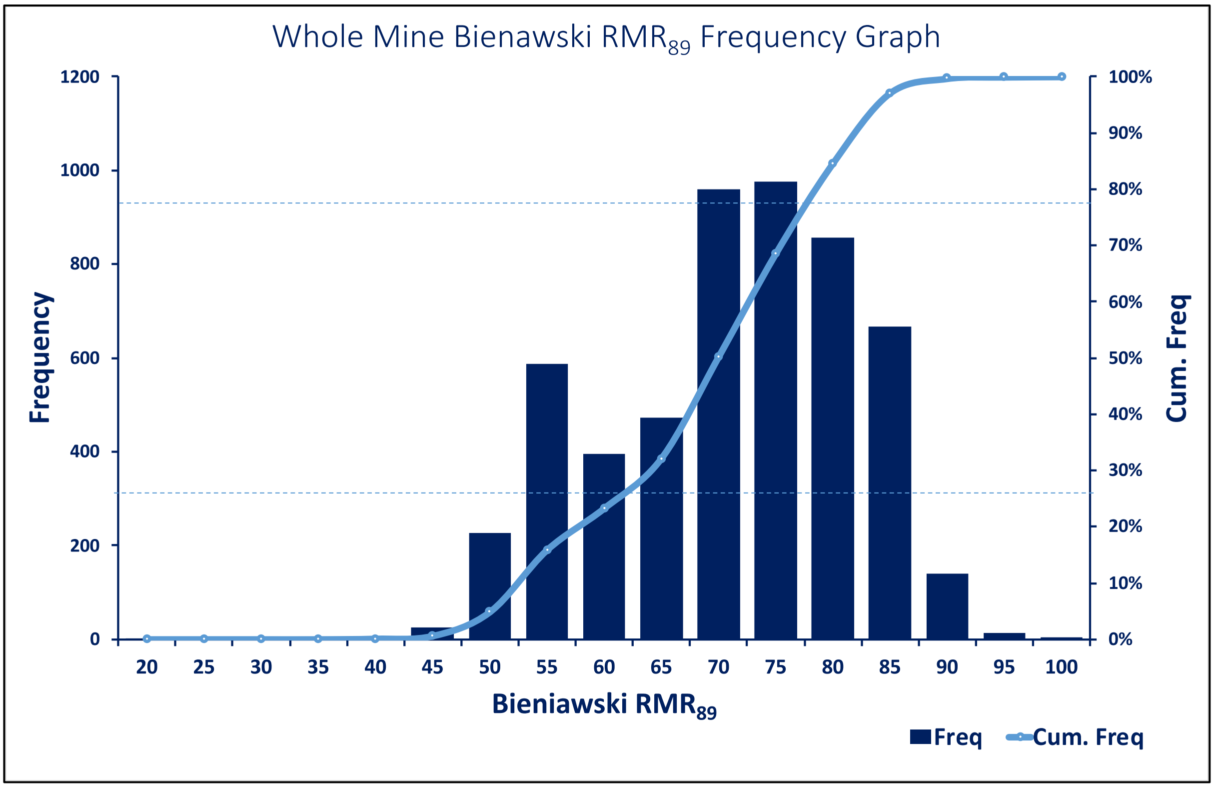

Figure 27: RMR histogram and cumulative distribution curve

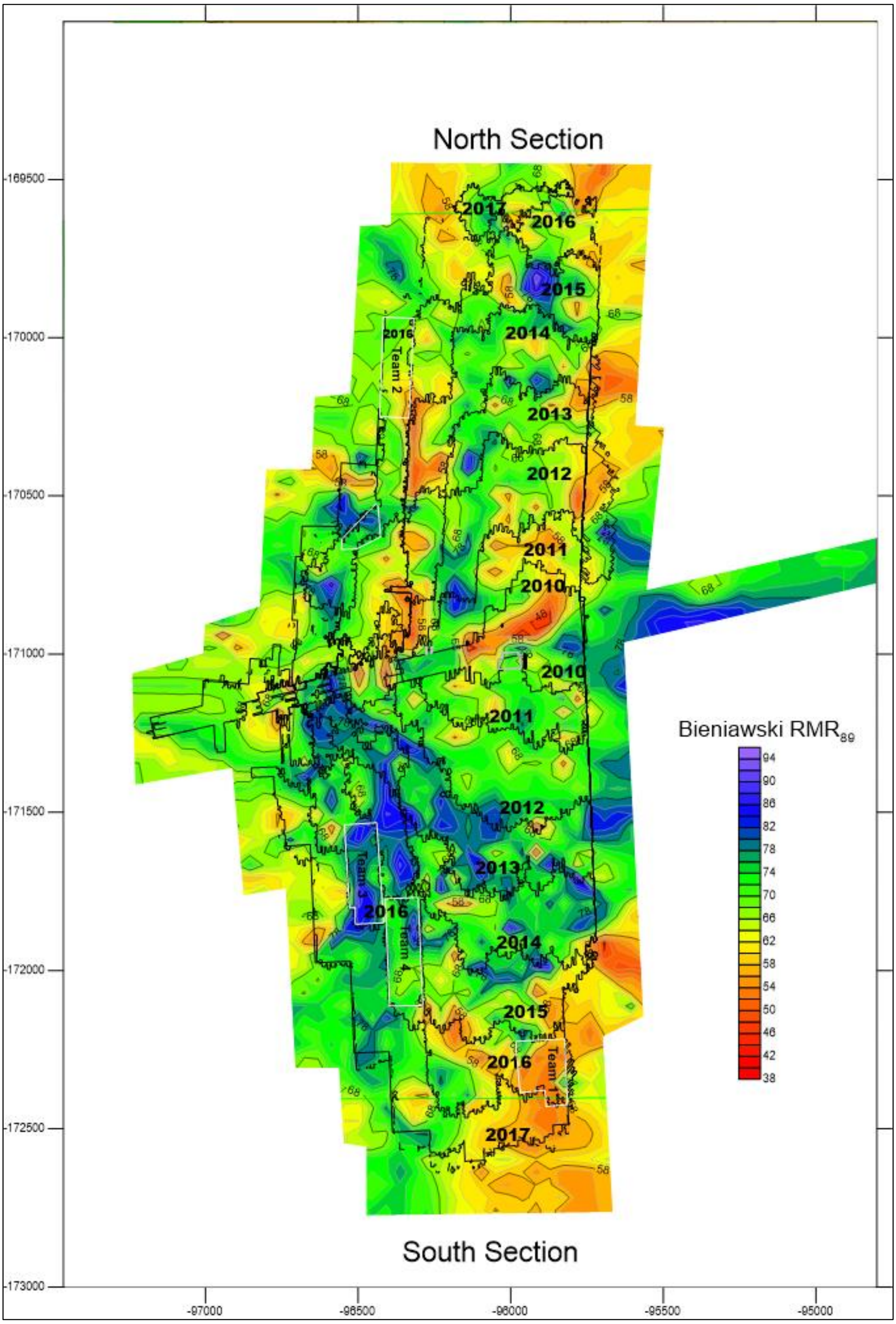

Figure 28: Unki East shaft rock mass quality map (applying Bieniawski, RMR, 1989)

5.4 RMR values analysis using Palisade @Risk software

The data collected was modelled in excel and @Risk software, an add-in to Microsoft Excel was used to analyse the risk using Monte Carlo simulation. The software showed possible outcomes for different scenarios and gives how likely these outcomes are to occur. This add-in finds the distribution that best fits the collected data and calculates the statistical likelihood of occurrence. Histograms were the chosen output from @Risk for this analysis. The main focus was placed on how the distribution of the data shows the performance of each team in the different rock mass conditions.

The analysis is within the 90% confidence interval. Reference will always be made to P75 and C80. As defined in the abbreviations, P75 denotes the higher quartile achievement i.e. the lower limit to the highest values achieved 25% of the time while C80 is the upper limit to the lowest values achieved 20% of the time. Given these values one can look at assess the achievement of the teams, consider their value addition and value loss. The preferred will be to achieve the budget stope width and grade hence it should be achieved most of the time. The RMR histograms indicate show how much time the team intersected different rock mass qualities, the SW and grade histograms provide corresponding results while working in that rock mass. The grade achieved and the tonnage mined in that period can be converted to give the number of ounces mined. The impact of the rock mass quality can thus be known in monetary terms once the loss or gain in the ore quality can be computed.

This means you can judge which risks to take on and which ones to avoid—critical insight in today's uncertain world. The histograms and distributions resulting from the processing of the data have the features shown in Figure 29**Error! Reference source not found.**.

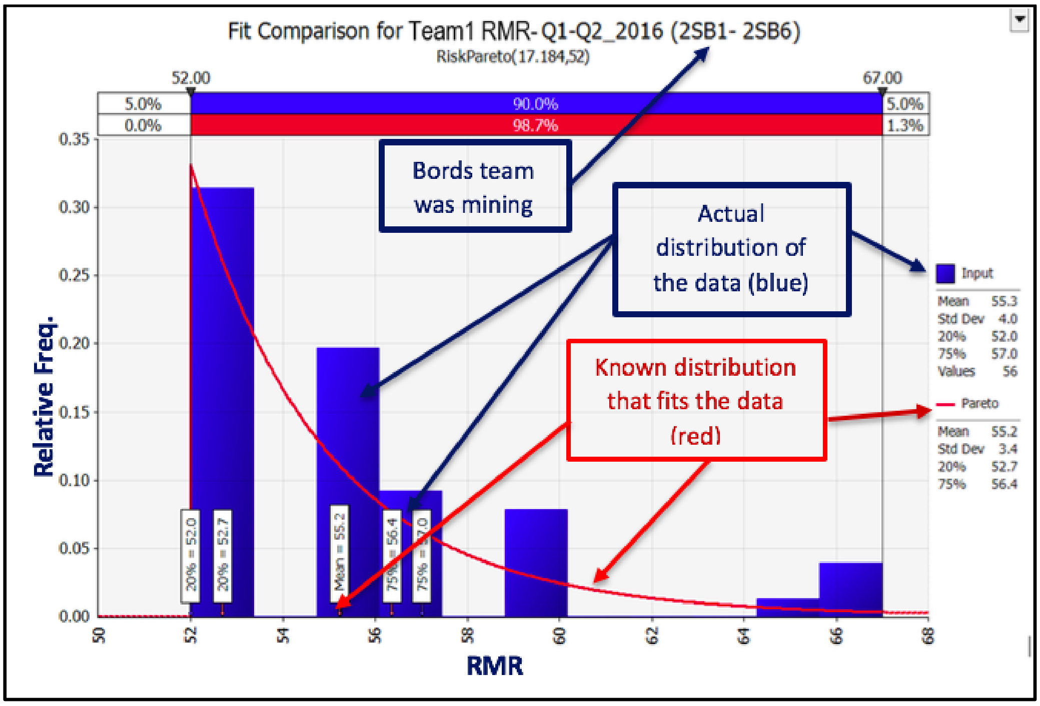

Figure 29: Explanation of the statistical distribution graphs

5.5 RMR values analysis using @Risk software

The RMR distribution histograms for Q1 to Q4 are illustrated in Figure *30* to SW and grade analysis using @Risk software
were the produced. The ground conditions in the sections were also modelled for Q1 and Q2 2016 for team 1, 2 and team 3. This indicated that team 1 operated in poor rock mass for the whole year, i.e. Q1-Q4 (Figure *30*). Team 1 was mining in 2 South Bord 1 to 2 South bord 6 (2SB1-2SB6). The distribution of the rock mass quality was 55.3 ± 4 for Q1-Q2 and 54.2 ± 2.9 for Q3-Q4.

Team 3 initially operated in poorer rock mass in Q1 seen in the distribution where for 20% of the time the team operated in RMR <56.1 reaching the lowest value at 47 (*Figure 32*). The team was then moved to a zone with good - very good quality ground conditions in Q2-Q4 as part of the initiatives of this research (team deployment initiative) giving the skew to the right in the distribution graph in *Figure 32*. The RMR approximated an extension value minimum distribution with mean 79.4, standard deviation 9.3 for Q1-Q2 and triangle distribution in Q3-Q4. The rock mass rating distribution had mean 77.2 and standard deviation of 5.3 (Q3-Q4). In Q3-Q4 the mean was 77.2 with standard deviation of 5.3. and approximated a triangular distribution with mean 77.7 and standard deviation of 5.2.

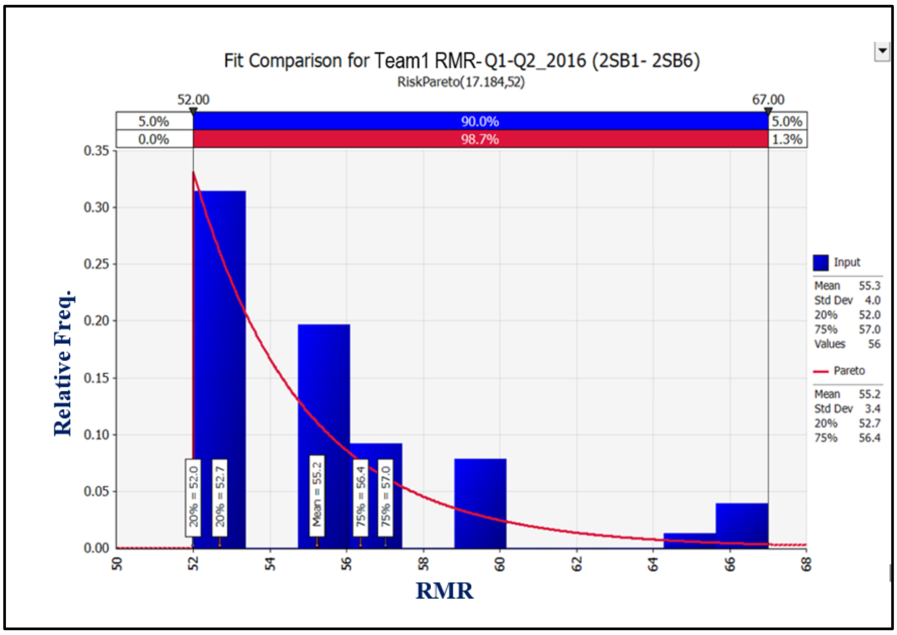

Figure 30: Team 1 Q1-Q2 RMR89 distribution

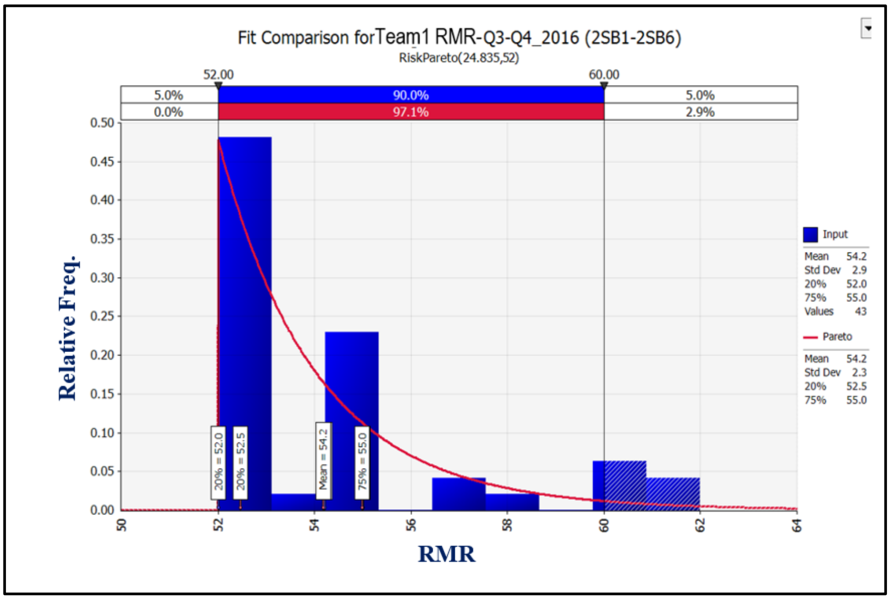

Figure 31: Team 1 Q3-Q4 RMR89 distribution

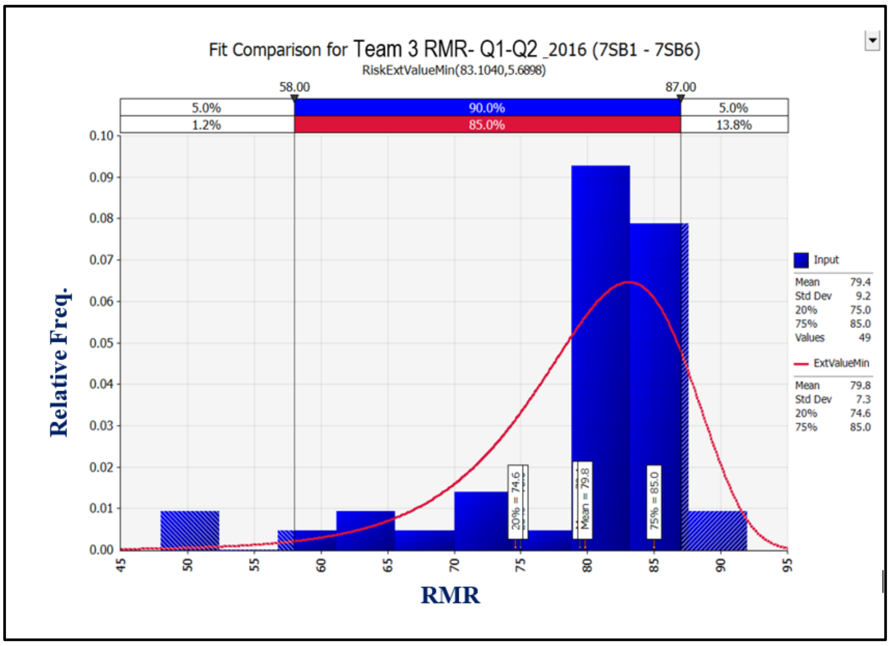

Figure 32: Team 3 Q1-Q2 RMR89 distribution

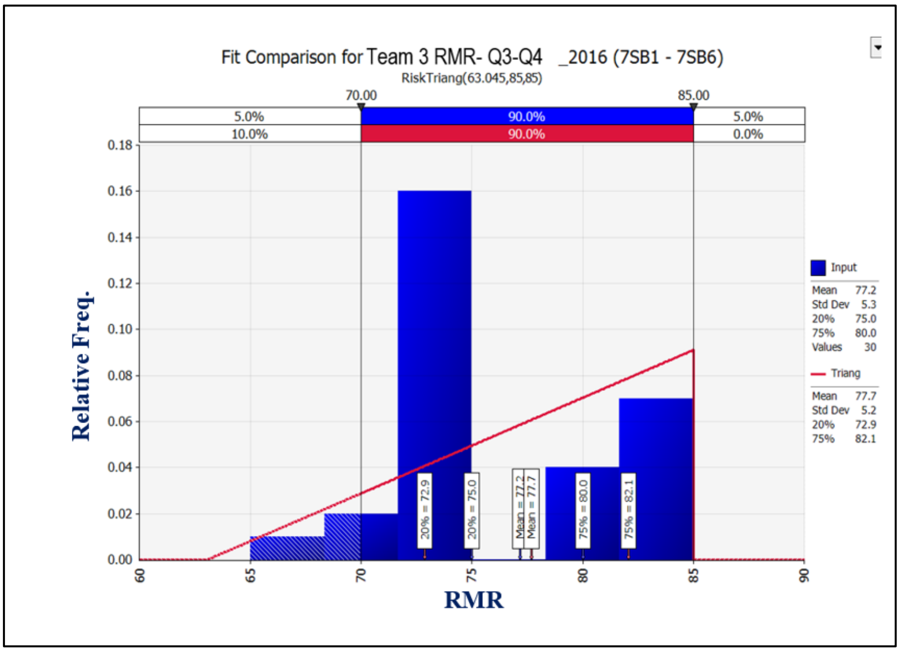

Figure 33: Team 3 Q3-Q4 RMR_{89} distribution

Team 2 failed to achieve the desired stope width and the assumption was that probably the team was mining in poor rock mass condition. The following histogram models show that there were other factors affecting the performance and hence the need to establish the challenges being faced by the team. The team operated in an area with mean RMR of 73.1 and a standard deviation of 8.3. The RMR in the area mined by team 2 in Q1-Q2 approximated a normal distribution with mean 73.1 and a standard deviation of 8.3

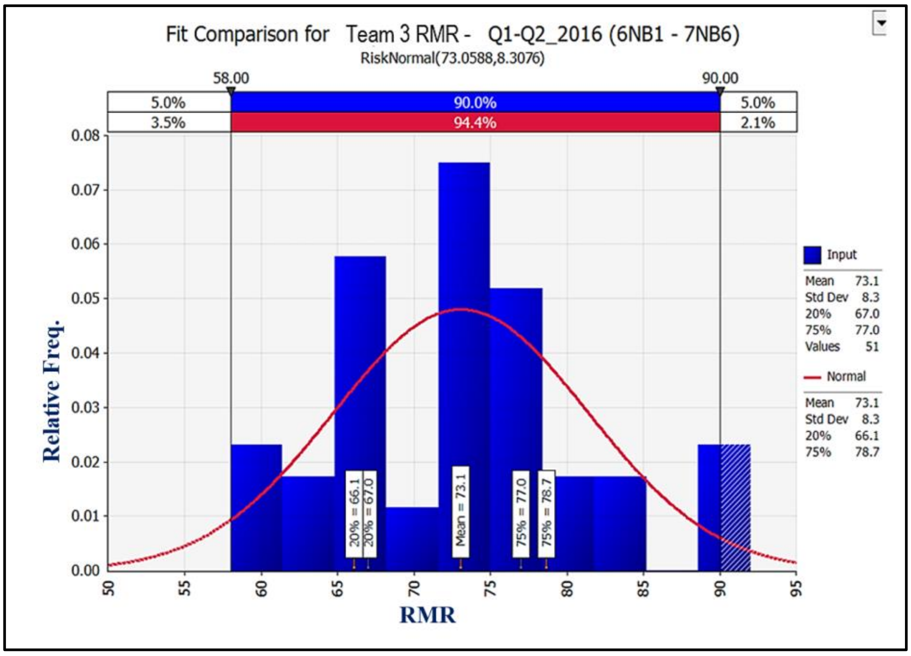

Figure 34: Team 2 Q1-Q2 RMR89 distribution+

5.6 SW and grade analysis using @Risk software

5.6.1 Team 1 Q1-Q2 Stope width

The team managed to achieve an average stope width of 210.4cm. Team 1 mined at stope widths above 2.15 cm for 75%of the time and 20% of Q1-Q2 the team mined at the desired stope width less than 202 cm. This is shown in Figure 35 by the values marked 75% and 20% respectively. Team 1's stope width achievement approximates a normal distribution with mean 210.6 cm and standard deviation of 9.6 cm (the red line). Figure 36 shows the corresponding grade for the stope widths mined within this Q1-Q2.

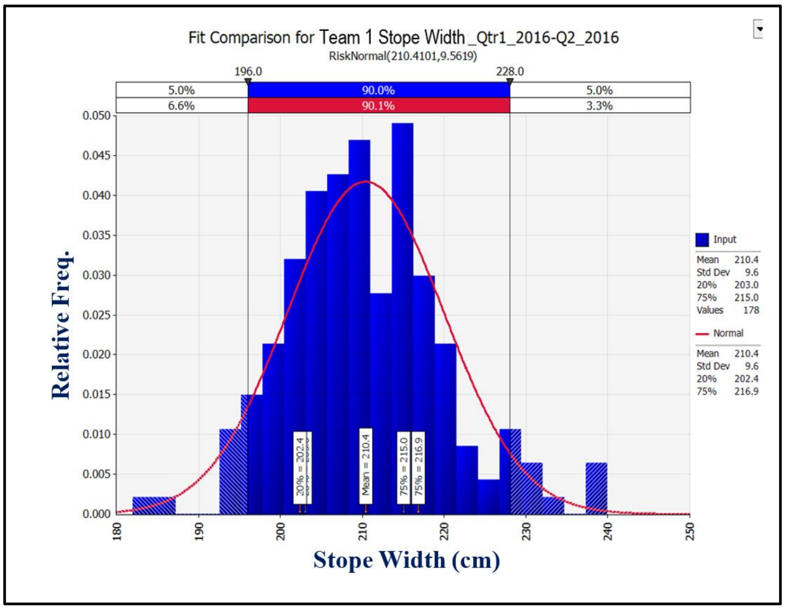

Figure 35: Team 1 actual stope width Q1-Q2 2016

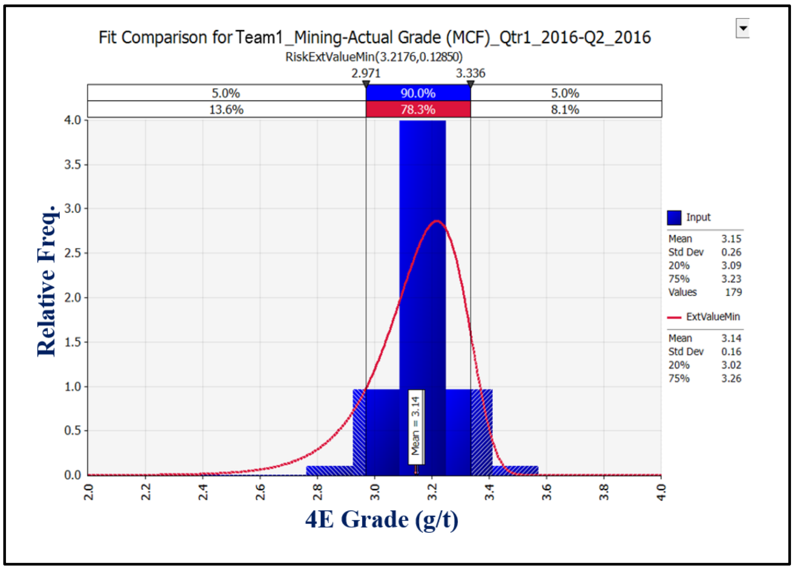

Figure 36: Team 1 mining actual grade for Q1-Q2 2016

5.6.2 Team 2 Q1-Q2 stope width and grade

Team 2 achieved mean stope width of 208 cm and standard deviation 14.3 cm. Its stope widths achievement approximated lognormal distribution of mean 208.7 cm and standard deviation of 14.2 cm. This team's achievement was almost similar to team 1 in terms of mean that is 208 cm vs 210 cm for team 1. However, team 2 was operating in good ground conditions. Figure 37 shows the histogram of the team's achievement.

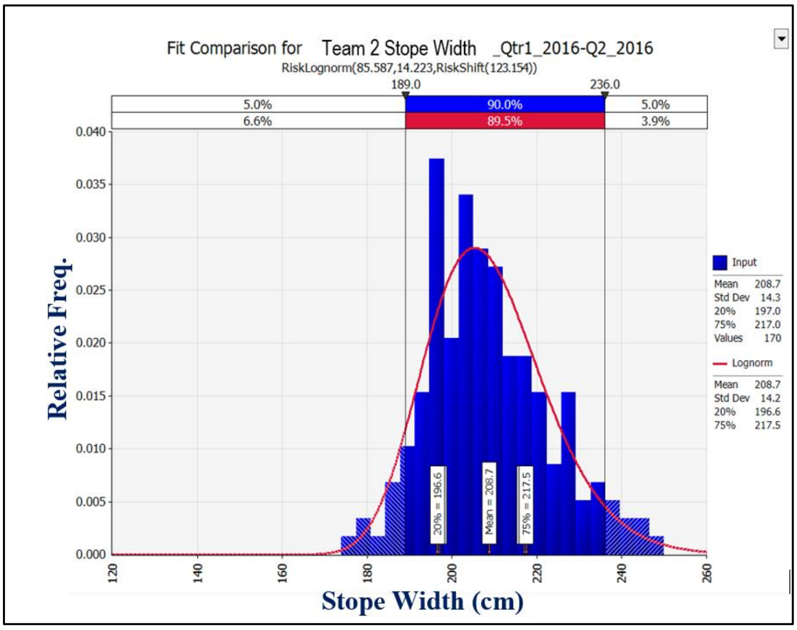

Figure 37: Team 2 actual SW Q1-Q2 2016

The corresponding grade distribution for the aforementioned stope width is shown in Figure 38. The model shows that the team averaged a grade of 3.43 g/t with a standard deviation of 0.16 g/t. The grades approximated a normal distribution. This achievement is against a planned grade of 3.60g/t i.e. a loss of 4.7 % in ore grade. This implies the team lost value by only operating above 3.54g/t for 25% of the time and below 3.30g/t for 20% of the time.

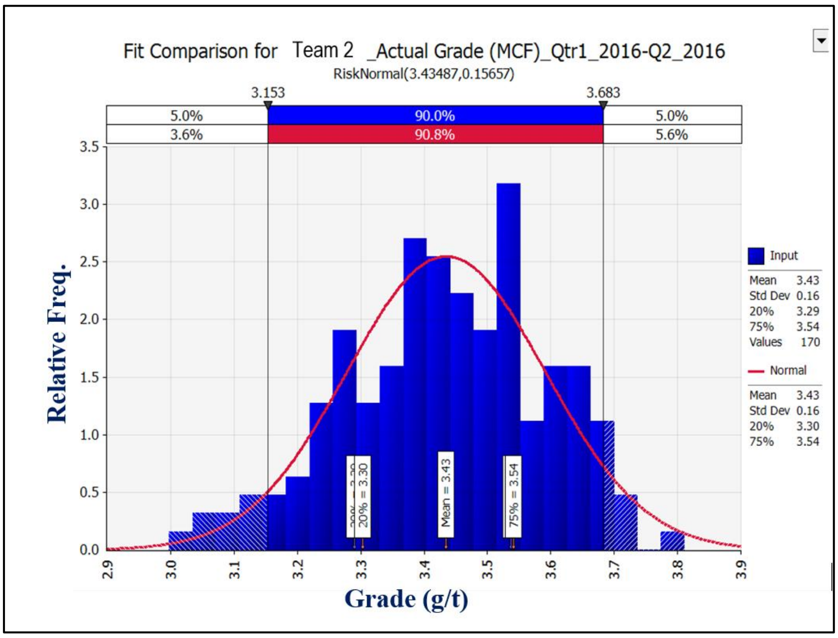

Figure 38: Team 2 actual grade for Q1-Q2 2016

5.6.3 Team 3 Q1-Q2 stope width and grade

Team 3 operated in an area characterized by dome structures, and bifurcating shears in places, but the block size is larger, giving rise to fair to good Q and RMR ratings. In quarter 2, the team operated in the lower sections as explained previously. The achieved mean stope width of 207.7cm with a standard deviation of 11.5 cm. The stope width achievement approximated a Pearson's distribution (*Figure 39*) was mean 207.7cm and 11.5 standard deviation. The P75 was 217.5 cm and C80 was 198.0 cm, implying that 25% of the time the team operated above 214.6 cm (undesired) and for 20% of the time the team operated below 196.6cm (desired).

The grade achievement approximated a normal distribution with mean 3.49g/t and 0.12g/t standard deviation against a budget grade of 3.65 g/t indicating a 4.4% loss in value. The P75 was 3.57 g/t and C80 was 3.49 g/t (*Figure 40*). The P75 is almost one standard deviation from the planned grade. The loss is attributable to Q1 when the team mined in the upper sections. During this period, the team achieved an average mining height of 213.5 cm and a grade of 3.48 g/t (average).

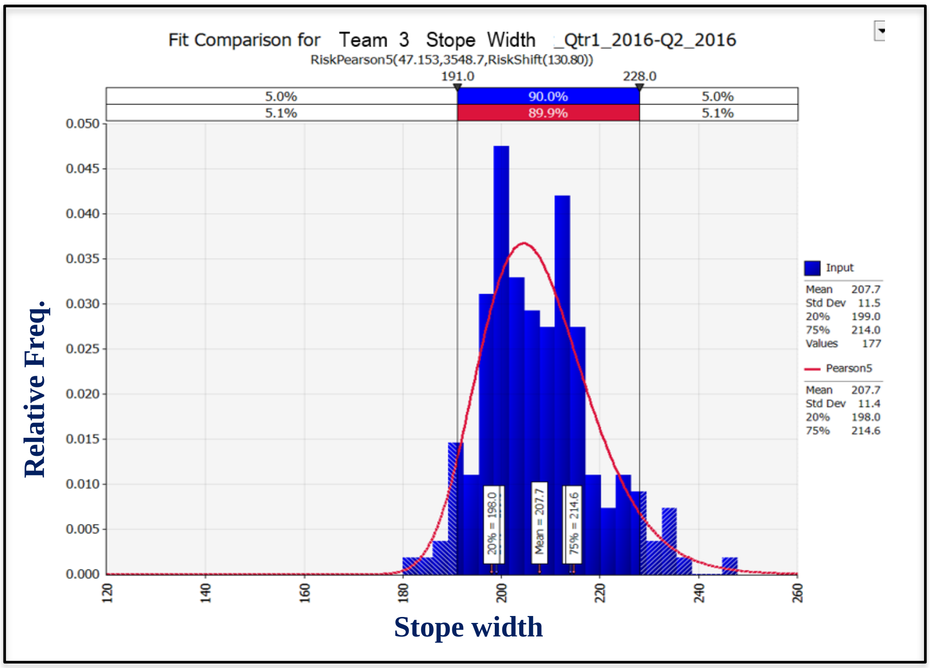

Figure 39: Team 3 stope width Q1-Q2 2016

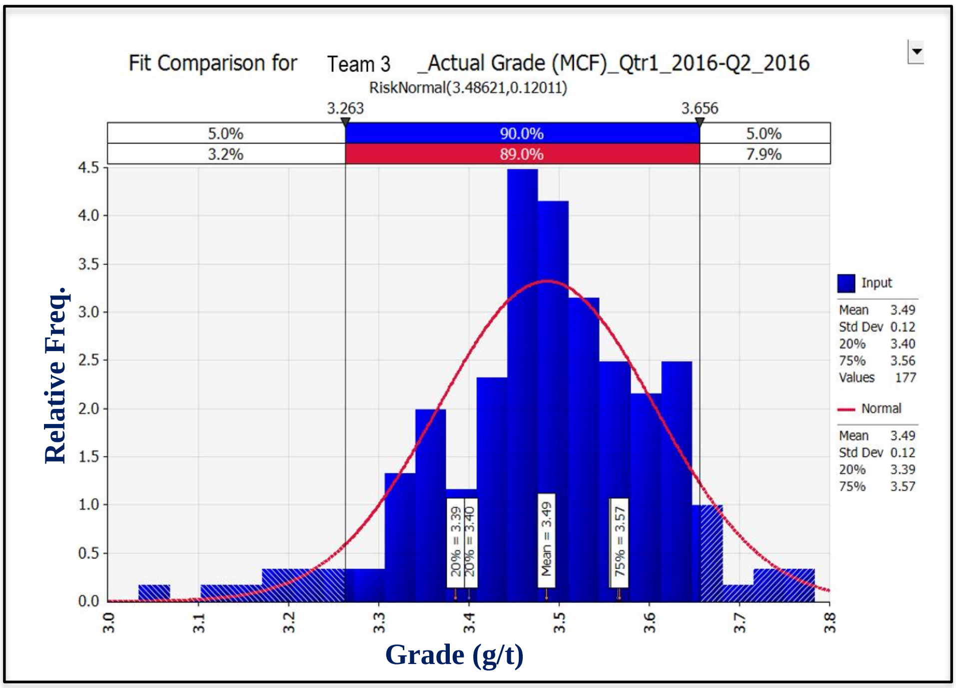

Figure 40: Team 3 mining actual grade Q1-Q2 2016

5.6.4 Team 4 Q1-Q2 stope width and grade

Team 4 was the best performing team at the mine in the period this research focused on. They achieved stope width of 203.4 cm and 8.5 cm standard deviation. The teams stope width approximated a logistic distribution with mean of 203.4 cm and standard deviation of 8.5 cm. The team's P75 was 208.6 cm and C80 of 196.9 cm. Team 4 only operated above the mean of the other 3 teams 25% of the time. *Figure 41* shows the distribution of the stope widths for team 4 in Q1-Q2.

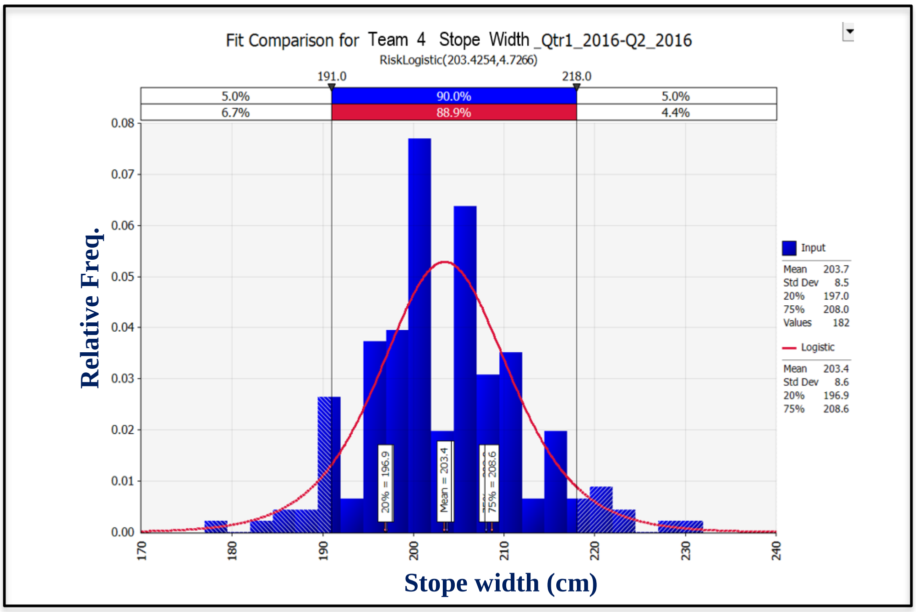

Figure 41 : Team 4 stope width Q1 - Q2 2016:

The team achieved mean grade of 3.50 g/t and a standard deviation of 0.10 g/t. The grades approximated a normal distribution with mean 3.50 g/t and a standard deviation of 0.10 g/t. The histogram in Figure 42 below shows the teams achievement.

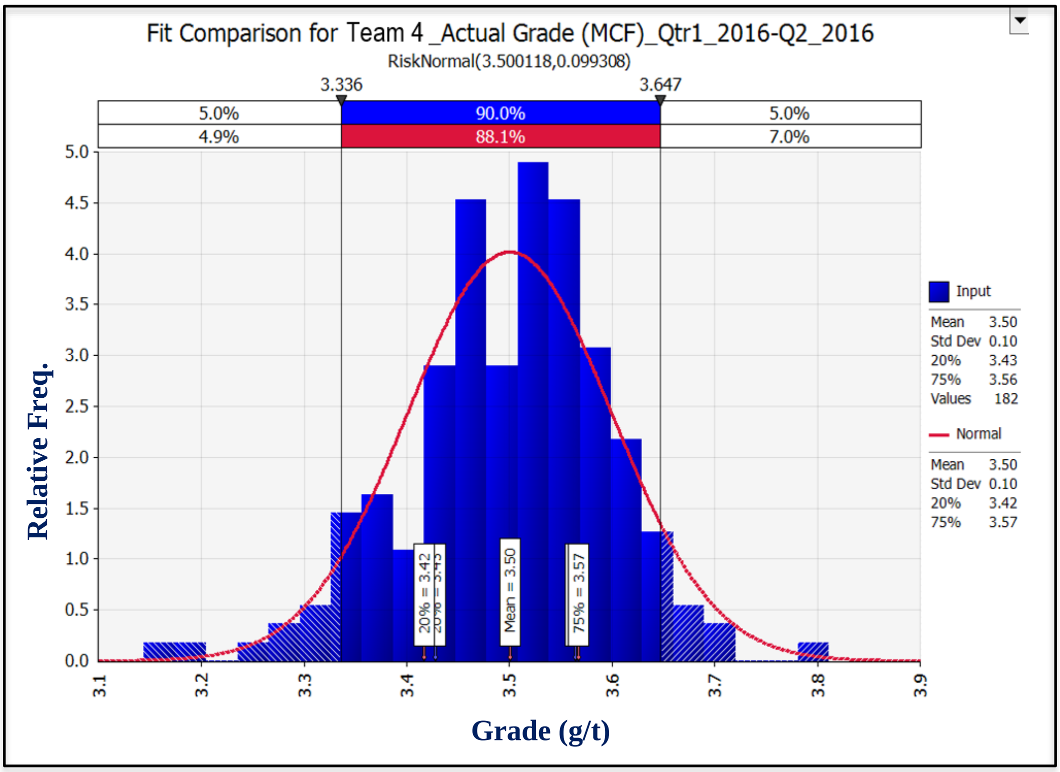

Figure 42: Team 4 actual grade Q1-Q2 2016

5.7 Interventions and their results

After observations that Team 1, Team 2 and Team 3 were failing to meet the desired mining height stope width, while mining through patches of poor rock mass(High Risk areas), a blast pattern was developed for use in poor ground zones the "High risk" type ground which incorporated air holes to try and post-split the perimeter of the charged faces. This saw a significant improvement in the stope width achievements recorded in Q3 and Q4. The stope width performance histograms in Figure 43, Figure 44 and Figure 45 show that Team 3, Team 1 and team 4 managed to achieve stope width within range of the budget thus delivering value after the interventions. This was significant improvement and higher value creation as compared to the first half of the year.

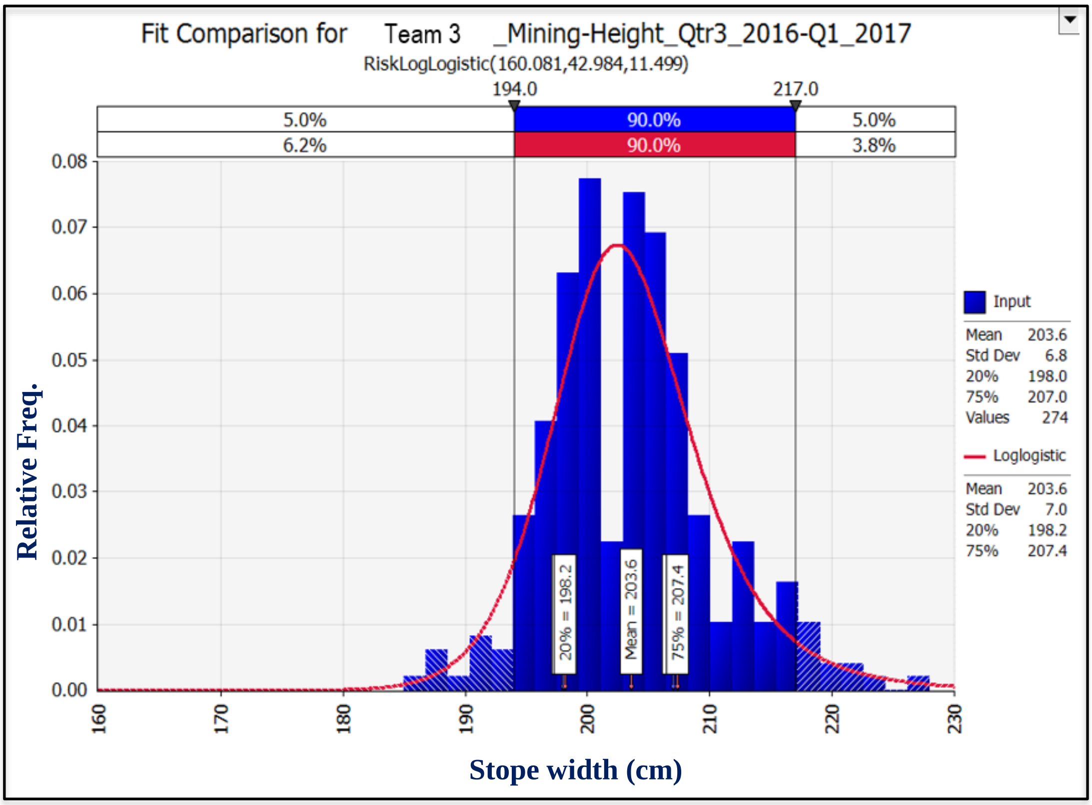

Figure 43: Team 3 Mining height Q3 2016 – Q1 2017

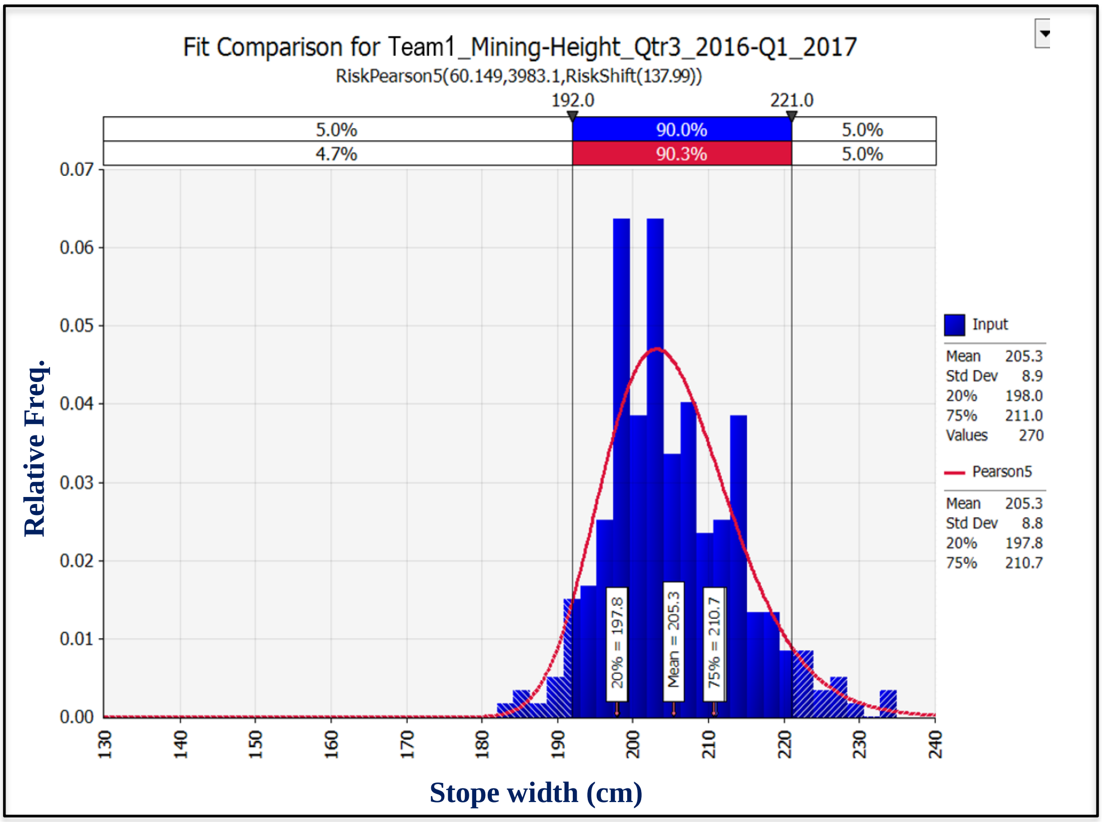

Figure 44: Team 1 mining height Q3 2016 - Q1 2017

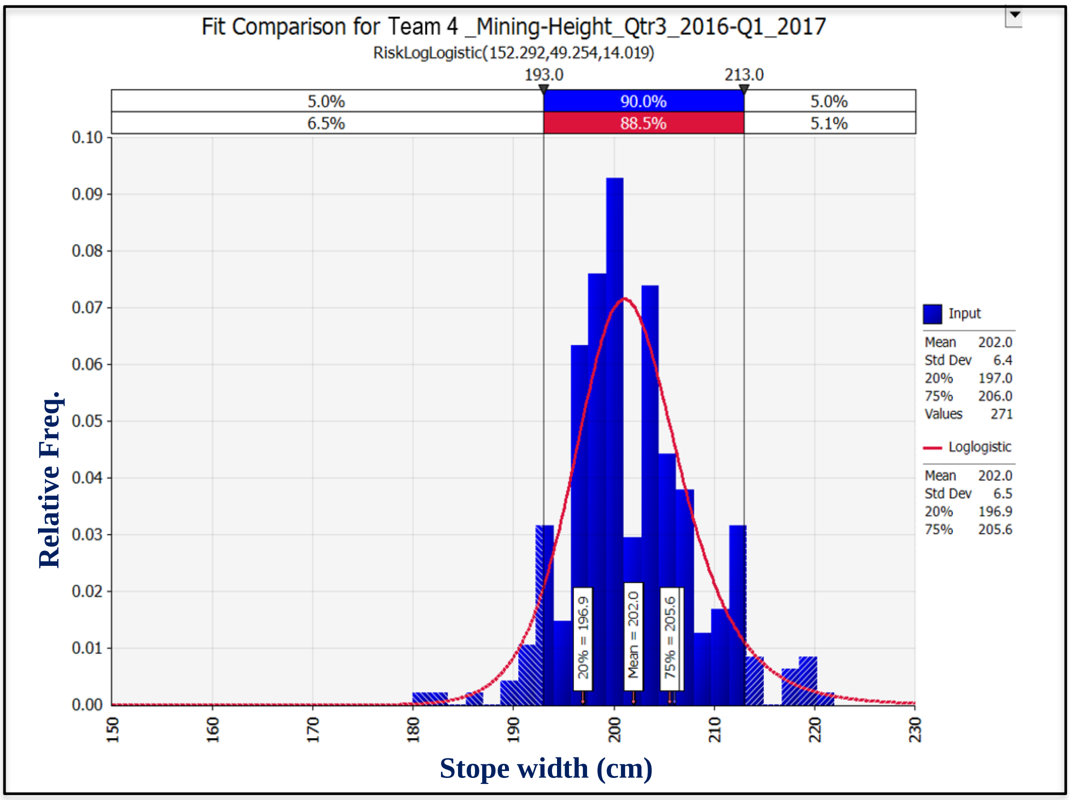

Figure 45: Team 4 mining height Q3 2016 - Q1 2017

5.7.1 Effects of practice and rock mass quality on stope width

Team 2 was operating in good ground conditions but failing to meet the desired stope widths. The following observations were the major challenges affecting stope width in the section.

a. Poor drilling of back holes (not parallel)
b. Overcharging of holes
c. Inconsistency in the use of poly pipes to decouple the charge in perimeter holes
d. Non-adherence to blasting pattern burden spacing
e. Batching unit on the charging unit was not working leading to overcharging of holes

These malpractices were being masked by the assumption that patches of poor rock masswere the major contributor. Upon correction of these malpractices the team started performing at the same level as Team 3 in terms of stope width achievement in Q3 and Q4 2016. This therefore shows that the mine incurred unwarranted losses due to incorrect reports.

After tracking charging, drilling and blasting practices i.e. the blasting pattern for 'High Risk' type ground was changed to include air holes in the perimeter and enforcing the use of poly pipes to decouple the charge. Team 1 was mining through poor ground conditions throughout 2016. The team implemented the new blasting pattern and was monitored for adherence to the support recommendations. In Q3 andQ4 the team's performance with regard to the stope width improved to the levels of the good performers (team 4) thereby adding value to the ore. Team 1 interventions were directed at proving that the effects of ground conditions could be controlled. This operation of the team was tracked by a data clerk to ensure all initiatives and standards were fully implemented as desired to remove the effects of malpractices. The following interventions were implemented:

a. Strict adherence in the use of poly pipes to decouple explosives in perimeter holes
b. Introduction of "High Risk" class blast design that saw the introduction of additional air holes in the perimeter of the design
c. Direct supervision by miner on the drilling of the holes and checks before end is blasted

For Teams 2, 3 and 4, the focus was on points (a) and (c), the implementation was carried out for 3 months with regular reviews. The following histograms show the marked changes in the adherence to stope width control. Of particular interest are the following observations:

a. Team 2 was working in RMR 66.1 in second half of the year compared to Q1-Q2 but achieved a mean stope width of 204 cm vs 209 cm
b. Team 1 was mining in similar ground conditions in Q3-Q4 and Q1-Q2 but achieved mean stope width of 205 cm vs 210 cm
c. Team 3 and Team 4 achieved mean stope width of 204 m and 202cm respectively
d. The teams were operating in the same ranges of stope width
e. The geotechnical attributable loss in ounces was thus for team 1 avoided by changes in blasting pattern, drilling monitoring and charging supervision

It can thus be proven that, despite the existence of geotechnical loss in ounces, which should be taken into consideration in strategic planning, solutions can be sought in basic mining principles that result in value add to the business.

5.7.2 Impact on grade and profitability

Team 1 which operated in poor ground conditions under direct supervision of the data capture clerks to remove the effect of malpractices showed that, the grade losses before interventions were at 5.7% (worst case). The team was delivering 3.04g/t instead of the planned 3.22g/t. This shows that given the average production expected per team at 17,200 tonnes and a basket price of R25,000 the team will generate R42.0m revenue monthly against a plan of R44,5m. That's a loss of R2.5m in revenue for one team. If calculated for 32% of the areas being mined as per the poor ground statistics captured from the TARP system, 3 production teams will be mining in poor rock mass at any given time. This translates to a risk or potential loss of R7.5m monthly (R90m annually), which, if managed well can be realised as revenue.

Team 1 proved that with management intervention the team can mine to a better stope width and reduce this loss in grade to zero. This indicates that if the geotechnical risk is managed well in short interval control, revenue can be realised leading to great value creation and viability for mining ventures.

The teams in good ground showed that malpractices can lead to loses of 7% in the worst-case scenario. Considering the same production per team this culminates to R3.07m loss. These malpractices were corrected and the teams performed and achieved the requisite mining height. This proved that the malpractices could easily be corrected by effective supervision and zero tolerance to substandard practices. Six teams operate in the good ground areas thus a total saving of R18.42m per month (R221.04m). Table 13 below showing the percentage value loss show the trends for each team.

Table 19: Percentage value loss per team

TEAM 1	JANUARAY	FEBRUARY	MARCH	APRIL	MAY	JUNE	JULY	AUGUST	SEPTEMBER	OCTOBER	NOVEMBER	DECEMBER
PLANNED GRADE	3,22	3,22	3,22	3,22	3,22	3,22	3,22	3,22	3,22	3,22	3,29	3,48
ACTUAL GRADE	3,04	3,22	3,1	3,04	3,17	3,18	3,21	3,3	3,29	3,37	3,39	3,33
VARIANCE	-0,18	0	-0,12	-0,18	-0,05	-0,04	-0,01	0,08	0,07	0,15	0,1	-0,15
% LOSS	-5,59	0,00	-3,73	-5,59	-1,55	-1,24	-0,31	2,48	2,17	4,66	3,04	-4,31
TEAM 2	JANUARAY	FEBRUARY	MARCH	APRIL	MAY	JUNE	JULY	AUGUST	SEPTEMBER	OCTOBER	NOVEMBER	DECEMBER
PLANNED GRADE	3,6	3,6	3,6	3,6	3,6	3,6	3,6	3,6	3,6	3,6	3,55	3,44
ACTUAL GRADE	3,44	3,4	3,49	3,39	3,32	3,42	3,41	3,51	3,45	3,62	3,61	3,56
VARIANCE	-0,16	-0,2	-0,11	-0,21	-0,28	-0,18	-0,19	-0,09	-0,15	0,02	0,06	0,12
% LOSS of Value	-4,44	-5,56	-3,06	-5,83	-7,78	-5,00	-5,28	-2,50	-4,17	0,56	1,69	3,49
TEAM 3	JANUARAY	FEBRUARY	MARCH	APRIL	MAY	JUNE	JULY	AUGUST	SEPTEMBER	OCTOBER	NOVEMBER	DECEMBER
PLANNED GRADE	3,6524	3,65	3,65	3,65	3,65	3,65	3,65	3,65	3,65	3,65	3,63	3,61
ACTUAL GRADE	3,56	3,6	3,56	3,38	3,35	3,46	3,51	3,49	3,49	3,67	3,62	3,51
VARIANCE	-0,0924	-0,05	-0,09	-0,27	-0,3	-0,19	-0,14	-0,16	-0,16	0,02	-0,01	-0,1
% LOSS	-2,53	-1,37	-2,47	-7,40	-8,22	-5,21	-3,84	-4,38	-4,38	0,55	-0,28	-2,77
TEAM 4	JANUARAY	FEBRUARY	MARCH	APRIL	MAY	JUNE	JULY	AUGUST	SEPTEMBER	OCTOBER	NOVEMBER	DECEMBER
PLANNED GRADE	3,58	3,58	3,58	3,58	3,58	3,58	3,58	3,58	3,58	3,58	3,59	3,61
ACTUAL GRADE	3,48	3,48	3,48	3,4	3,36	3,46	3,46	3,51	3,52	3,69	3,64	3,55
VARIANCE	-0,1	-0,1	-0,1	-0,18	-0,22	-0,12	-0,12	-0,07	-0,06	0,11	0,05	-0,06
% LOSS	-2,79	-2,79	-2,79	-5,03	-6,15	-3,35	-3,35	-1,96	-1,68	3,07	1,39	-1,66

5.7.3 Pillar strength impact

Unstable pillars identified for rehabilitation were recorded during the period of research. A total of 56 pillars from the 1008 pillars cut by the 4 teams in 2016 showed signs of deterioration and were rehabilitated with mesh and straps. This indicates a total of 5% of the pillars cut in the year for the 4 teams was affected by geological structures leading to rehabilitation requirement. Due to the inclusion of barrier pillars in the mine design, creating a situation where there is a mixture of large and small pillars in the layout. This makes Tributary Area Theory (TAT) inapplicable in accurate calculation of the Average Pillar Stress (APS). Institute of Mine Seismology (IMS) was then engaged to model APS for the pillars that were unstable for purposes of checking the factor of safety. The calculated w/h ratios and factor of safety for the failing pillars are tabulated in Appendix 4.

It can be noted from Appendix 4 that all the pillars that showed signs of instability, potential disintegration or stress fracturing have factor of safety of 1.6 or better. Given such FoS values according to the design criteria which requires that all pillars to have a FoS greater than 1.6, these pillars should not have failed. However, observations underground show that these pillars are affected by the low angle joints and the challenges increase when joints intersects with shears within a pillar. Smaller pillars with much lower FoS as low as 1.2, have not shown any signs of deterioration or stress fracturing indicating that the presence or absence of geological structures within the pillar play a major role in the failure mechanisms of these pillars. A rehabilitation research was initiated and the costs were collected for future forward planning.

The process of rehabilitating these pillars was costed and the results of the material costs excluding labour are presented in Appendix 6. The total rehabilitation cost for the pillars excluding labour was R602,556.66 (i.e. R0.6m) This is one of the costs of geotechnical risks that need to be factored into the planning process.

5.7.4 Cable anchor installation optimization

The poor ground ('High risk' type) ground which has been described and often prescribed to have cable anchors installed as part of the ground support system take long to come back into the production cycle due to the cable anchor installation process. The AMS cable anchor installation process involves the use of cable anchor tensioners, stressing jacks and croppers. These pieces of equipment are maintained by mine personnel. As mentioned in the literature review the process of these installation is time consuming and affects the mining cycle. An opportunity was identified during the research period to optimize this process by the introduction of a slightly more expensive product. This approach is often faced with resistance by management based on cost of product. Rwodzi (2011) in his research report concludes his illustration of value creation through safety with the following statement *"Some managers appear to believe that cost-cutting measures on a mine should include minimising as much as possible on any expenditure including ground support initiatives. Others are reluctant to opt for new strategies proposed as being more adequate by the Rock Engineer. However, the risk-cost approach described in this research report has clearly indicated the value created by investing in safety spending."*

The following table shows the expenditure on cable anchor installation equipment from Q1 2016 to Q1 2018.

Table 20: Cable anchor installation equipment costs

	Q1 -Q4 2016	**Q1 - Q4 2017**	**Q1 2018 (3 months)**	**Total**
ZAR	530 433.76	647 477.01	294 177.56	**R 1,472,088.33**
USD	37611.32	49250.49	24 678.80	**US$ 111,540.61**

NB. These costs are material only costs excluding labour and equipment transportation time during breakdowns

On an average R0.6m is spent annually on cable anchor installation equipment. Like any other pieces of equipment, the maintenance cost increases with age of the equipment. In the bid to improve productivity and ergonomics trials were carried out, to introduce Flexibolts as a replacement to the AMS barrel cable anchors. Table 21 gives the comparisons in installation time and cost between Flexibolts and AMS barrel cable anchors.

Table 21: Comparison of installation time and costs for Flexibolts and AMS barrel cable anchors

Time benefit

Product	**Installation time**	**Time benefit**
AMS Full barrel cable anchors	30 minutes	Bench-mark
3.0m DT Flexibolts	11 minutes	19 minutes

Cost comparison

Product	**Unit price (USD)**	**Cost benefit (%)**
AMS Full barrel cable anchors	R242.19	Bench-mark
3.0m DT Flexibolts	R394.91	-38.7%

Table 21 gives the Flexibolt specifications as defined for use at Unki mine.

Table 22: Flexibolt specifications

	3.0M DT FLEXIBOLT					Spec Sheet 2.08
Description:	The Flexibolt Cable Anchor is an 18 mm, 7 wire strand with an ultimate tensile strength of 38 tons with M20 thread which comes standard in left hand thread with an ultimate tensile strength 30 tons. Its locking device mechanism consists of a distribution plate (150 mm x 6 mm) with no dog ear and 32 mm cross flat nut that can be tensioned by impact wrench or by hand. A mechanical 3 part shell provides the fixed end support inside a standard 35 mm to 41 mm hole. These will be supplied in 3.0 m lengths.					
Quality control	Has a 4 mm yellow load indicator.					
Grouting Mechanism	It has a GV mechanism with a non-return valve. The breathing tube consist of a 3.0 m black HDPE 8 mm x 12 mm pipe.					
Dome plate	Has a 150 mm X 6 mm GV washer with no dog ear. In addition, the plate must not deform beyond its elastic limits at loads less than the ultimate load of the anchor.					
Hole diameter ranges	35 - 41 mm					
Lengths	The bolt is 3.0 m long and does not require cropping on installation.					
Shells	Has a rope type shell with a spring activation positioned to fit into a 38 mm hole designed to yield at 27 tons.					
Consistency	100 % of the units must meet the minimum specifications.					
Corrosion Protection	Preferably but not mandatory there some means of corrosion protection for storage.					
LENGTH [m]	SUPPLY LENGTH	PRE-LOAD	FAILURE LOAD	PLATE THICKNESS	PLATE DIAMETER	HOLE DIAMETER
3.0m	3.0 m	40-50 kN	270 kN	>= 6 mm	>= 150 mm	35-41 mm
FUNCTIONAL USE - LAYMANS' TERMS **Flexibolts** are flexible tendons of different length that are installed into drill holes and then grouted. They are be pre-stressed and are normally used in large excavations where support is needed far into the hangingwall.						

The following conclusions were made from the Flexibolts trials:

a. The Flexibolt cost unit is more expensive than the AMS barrel cable anchor but has a lot of benefits and value add that supersede the unit cost.

b. Flexibolts can be installed by one man using the fletcher bolter as opposed to the AMS that requires 3 people during installation.

c. There is a gain of 19 minutes in the installation processes per cable anchor. And average of 40-50 cable anchors are installed daily in the mine giving a total saved time of 38-man hours.

d. There is an opportunity to optimize labour given that the bolter operator will complete the installation

e. The ends in poor rock mass quality come back into cycle quicker with the introduction of Flexibolts.

f. Team 1 showed that productivity is increased by 25% by the introduction of Flexibolts. The team's production had dropped by 25%from 2700 m^2 to 2025 m^2 i.e. a drop of 142.1 4E ounces translating to R3.55m monthly for one team. The revenue savings for three teams in poor ground is R127.8m annually.

g. The Flexibolts introduction leads to the withdrawal of tensioning and cropping equipment 75% (R0.45m) of this will become an annual saving once the Flexibolts are fully deployed as most of this equipment will be decommissioned. Only the teams that install long hole tendons (cable anchors) on sidewalls will remain using this equipment.

6 CONCLUSIONS AND RECOMMENDATIONS

6.1 Conclusions

From this research, it can be concluded that

a. A total of R439.75m (nearly half a billion rand) was saved due to implementation of the strategies and interventions detailed in this research. Breakdown of savings was: stope width in poor ground R90m, Correction of malpractices R221.04m, Removal of cable anchor installation equipment R0.45m, improvement of efficiencies by introducing Flexibolts R127.8m and pillar rehabilitation -R0.6m.
b. Poor rock mass quality is accountable for the failure of 5% of pillars at the mine.
c. A substantial loss is made in the process of rehabilitating the pillars. A process normally not budgeted for. R1.4m loss in material costs in 2016 was incurred by the mine studied in this research.
d. Poor rock mass quality can be ascribed to a potential loss of about 5.7% in grade. However, Team 1 proved that this loss can be averted if appropriate interventions are put in place. If left unattended it can culminate into a loss of R7.5m per month (i.e. R90m annually)
e. Malpractices on the other hand can be more costly than geotechnical risk at an average of R3.07m. Any reasons for failure to achieve the mining cut should thus be interrogated and corrected if value is to be preserved.
f. The impact of rock mass quality on stope width can be managed
g. The rehabilitation of pillars affected by impact of rock mass quality will need to be incorporated into the financial budget and management should be made aware of this cost. This should cost must be considered in the decision to continue mining an area or leave it for redevelopment. Team deployment can be guided by this cost if there is good redundancy.
h. The higher cost of Flexibolts is countered by the 19 minutes gain in time per cable anchor, reduction in number of personnel involved in the cable anchor installation process, and the safety aspect that the cable anchor will be installed under the canopy of the bolter

6.2 Recommendations

6.2.1 Geotechnical risk assessment inclusion in business plan

The failure to achieve mining cut while mining through poor ground areas gives rise to the risk of loss in ounces due to dilution. The hoisting of low grade material is detrimental to the business and needs to be managed. The losses alluded to in the earlier sections have the potential of causing huge cumulative losses, culminating in retrenchments, or at the worst mine closures.

It is therefore proposed that this risk be managed in the business planning process and be inbuilt in both the short term and long-term plan. However, for a mining company to be able to build this operational geotechnical risk into its business planning process, and manage the associated malpractices and misuse of the phrase *"mining through poor ground"*, it is critical to carry out a proper investigation to understand the associated operational geotechnical risk, and what proportion of the losses can be ascribed to it. It is also important to note that the operation needs to have reached the production steady state to carry out such an assessment and to begin incorporating effectively the operational geotechnical risk. This enables the identification and separation of ground related challenges from production ramp up related challenges, and the isolation of excuses from genuine challenges.

6.2.1.1 Long term Planning:

The following process is hereby proposed

a. Identify the geotechnical risk related potential revenue loss sources
b. Set thresholds of losses depending on the magnitude
c. Understand what percentage geotechnical challenge has on revenue impact
d. Formulate these into the risk matrix specifying what the rating for likelihood and consequence are. That is, develop a matrix. Risk=Likelihood * consequence
e. List all possible geotechnical challenges and compute the risk ratings before implementing interventions
f. Compute the ratings with controls in place
g. Graphically represent the outcome on the risk matrix example in Appendix 6
h. Indicate to management what financial risk remains (residual risk). That is what is the number of ounces or amount of revenue that remains at risk
i. Does the risk remain too high? If so, look for better controls. If not, list mitigating factors and incorporate in business plan

6.2.1.2 Short term planning:

a. Create a rock mass rating database from both cover drilling holes and underground mapping.
b. Contour the rock mass rating.
c. Empirically relate the rock mass rating to the physical conditions underground so that one is aware what quality the numbers/contours refer to.
d. In order to create a predictive tool, several overlays can be done. E.g. Overlay the rock mass rating with geological structures, FOGs or known conditions, and identify all zones with the same possibilities. See appendix 6 for an example.
e. Limit predictions and extrapolations to reasonable distances (engineering judgement required) distances.
f. Communicate the risk areas as per the predictions and incorporate mitigation plans in short term plan.

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APPENDIX 1: Adjustments for joint condition and Groundwater

Parameter	Description		Dry Condition	Wet Conditions		
				Moist	Moderate pressure 25-125 1/mm	Severe Pressure >125 1/min
A Joint Expression (large scale irregularities)	Wavy	Multi-Directional	100	100	100	95
		Uni-Directional	95 90	95 90	90 85	80 75
	Curved		89 80	85 75	80 70	70 60
	Straight		79 70	74 65	60	40
B Joint Expression (small scale irregularities or roughness	Very rough		100	100	95	90
	Striated or rough		99 85	99 85	80	70
	Smooth		84 60	80 55	60	50
	Polished		59 50	50 40	30	20
C Joint Wall Alteration Zone	Stronger than wall rock		100	100	100	100
	No alteration		100	100	100	100
	Weaker than wall rock		75	70	65	60
D Joint Filling	No fill – surface staining only		100	100	100	100
	Non softening and sheared material (clay or talc free	Coarse Sheared	95	90	70	50
		Medium Sheared	90	85	65	45
		Fine Sheared	85	80	60	40
	Soft sheared material (eg talc)	Coarse Sheared	70	65	40	20
		Medium Sheared	65	60	35	15
		Fine Sheared	60	55	30	10
	Gouge thickness <amplitude of irregularity		40	30	10	
	Gouge thickness <amplitude of irregularity		20	10	Flowing material 5	

APPENDIX 2: MRMR Adjustments (after Laubscher, 1989)

Table 23: Weathering adjustment

Weathering adjustment (%)					
Description of weathering extent	**6 moths**	**1 year**	**2 years**	**3 years**	**4 + years**
Fresh	100	100	100	100	100
Slightly	88	90	92	94	96
Moderately	82	84	86	88	90
Highly	70	72	74	76	78
Completely	54	56	58	60	62
Residual soil	30	32	34	36	38

Table 24: Adjustments to MRMR due to join orientation

Number of Joints defining the block	Adjustment (%) Number of inclined away from the vertical				
	70	**75**	**80**	**85**	**90**
3	3	-	2	-	-
4	4	3	-	2	-
5	5	4	3	2	1
6	6	5	4	3	2 or 1

Table 25: Adjustments for blasting effects

EXCAVATION TECHNIQUE	Adjustments (%)
Boring	100
Smooth wall blasting	97
Good convectional blasting	94
Poor blasting	80

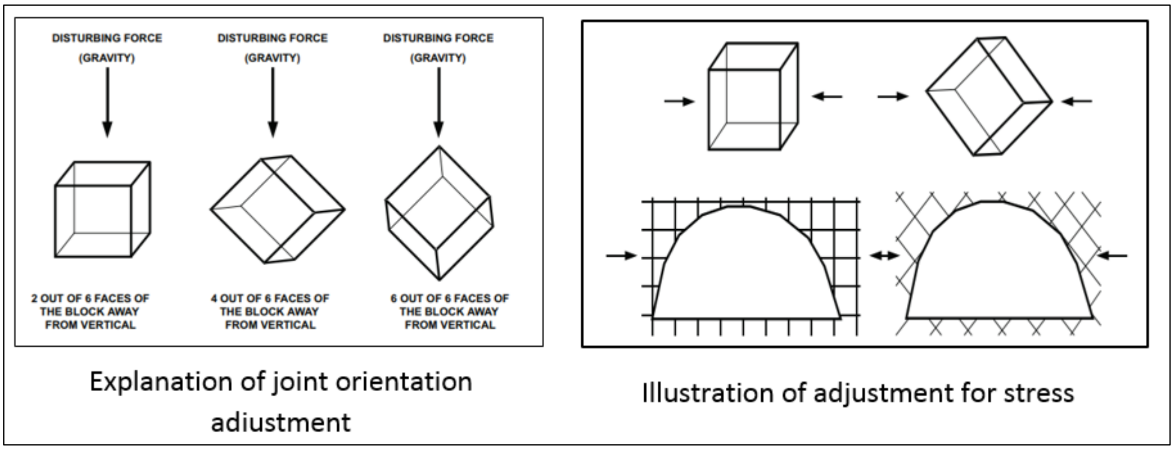


Figure 46: Joint orientation adjustment and adjustment for stress

APPENDIX 3: RMR in the areas being mined by team 1, 2, 3 & 4

Team 1	**2SB1- 2SB6**														**Avarage**	**STDEV**
Jan	55	55	55	52	60	57	59	59							**56.5**	**2.5**
feb	57	55	55	55	59										**56.2**	**1.6**
march	57	52	52	55	52	59									**54.5**	**2.8**
april	57	60	67	67	52	52	52	52	52	52	52				**55.9**	**5.8**
may	67	65	57	57	52	52	52	52	52	52	52	52	52		**54.9**	**5.1**
june	55	55	52	52	55	52	55	55	55	52	57	55	55		**54.2**	**1.6**
Jul	55	55	55	52	52	52	52	57	55	52	52	60			**54.1**	**2.5**
Aug	52	60	58	55	52	52	55	52	52	52	52				**53.8**	**2.7**
Sep	55	52	52	55	52	60									**54.3**	**2.9**
Oct	55	52	52	52	57	55									**53.8**	**2.0**
Nov	52	55	52	54	62										**55.0**	**3.7**
Dec	61	52	53												**55.3**	**4.0**
		Stdev	**3.6**												**55.4**	
Team 2	**6NB1 - 7NB6**															
Jan	90	77	80	64	77	69	77	77	72						**75.9**	**6.9**
feb	77	72	72												**73.7**	**2.4**
march	82	77	77	72	74	70									**75.3**	**3.9**
april	90	67	82	82	79	74	65	65	92	67	67	77	75	65	**74.8**	**9.0**
may	92	75	64	64	58	75	75	67							**71.3**	**9.8**
june	67	77	79	72	67	67	58	58	58	75	75				**68.5**	**7.5**
Jul	65	63	57	54	64	64	80								**63.9**	**7.6**
Aug	55	74	54	50	74	64	58	58	58	69	69	52	52	69	**61.1**	**8.2**
Sep	58	69	74	87	79	69	69	64	62	67	52	69	54	69	**67.3**	**8.9**
Oct	60	64	52	57	67	64	67	55	72	74	82	77	78	74	**67.4**	**8.9**
Nov	67	67	64	64	74	60	67	70	69	57	75				**66.7**	**5.2**
Dec	67	69	75	71	68	72	80	62							**70.5**	**5.1**
		Stdev	**9.0**												**66.1**	
Team 3	**7SB1 - 7SB6**															
Jan	75	75	70	75	70.2										**73.0**	**2.4**
feb	75	80	75	75	75										**76.0**	**2.0**
march	85	85	73	85	85										**82.6**	**4.8**
april	80	75	80	72	80										**77.4**	**3.3**
may	85	75	85	65	85										**79.0**	**8.0**
june	75	75	75	75	75										**75.0**	**0.0**
Jul	82	82	82	85	82	82	87	82							**83.0**	**1.8**
Aug	80	75	80	82	80	80	80	81							**79.8**	**1.9**
Sep	79	79	69	50	87	87	79								**75.7**	**11.9**
Oct	48	79	80	85	87	84	87	87	87	87	79	79	84		**81.0**	**10.1**
Nov	92	89	84	84	84	84	87								**86.3**	**3.0**
Dec	64	73	73	72	64	58									**67.3**	**5.7**
		Stdev	**7.9**												**77.2**	
Team 4	**6SB1- 6SB6**															
Jan	77	87	77	77	77	67									**77.0**	**5.8**
feb	77	72	77	67											**73.3**	**4.1**
march	72	72													**72.0**	**0.0**
april	67	67	77	72	77										**72.0**	**4.5**
may	67	77	77	67	77	77	77								**74.1**	**4.5**
june	67	72	67	67	67	77	77	77							**71.4**	**4.6**
Jul	67	57	67	77	67	67	77	67	67	72	77				**69.3**	**5.8**
Aug	57	77	77	77	52	67	52	72	77						**67.6**	**10.4**
Sep	77	77	77	67	55	52	77								**68.9**	**10.3**
Oct	77	77	55												**69.7**	**10.4**
Nov	76	76	76	69	76	76	76	76							**75.1**	**2.3**
Dec	75	72	72	59	59	70	70	74	73	69					**69.3**	**5.4**
		Stdev	**7.1**												**73.3**	

APPENDIX 4: Pillar w/h ratios and FoS

	Actual Pillar (m^2)	W:H ratio	Safety Factor
Team 1	41	3.1	2.0
	33	2.4	1.9
	31	3.2	1.9
	76	2.6	2.2
	53	3.6	2.1
	89	2.6	2.2
	63	2.3	2.0
	57	2.5	2.0
	125	3.5	2.5
	31	2.2	1.8
	37	2.3	1.9
	33	2.3	1.8
	27	2.1	1.7
	48	3.8	2.0
	40	2.1	1.9
Team 2	30	2.1	1.9
	27	2.0	1.8
	31	2.2	1.9
	38	3.5	2.0
	38	2.5	2.0
	32	2.5	2.0
	33	2.5	2.0
	28	2.2	1.9
	32	2.7	2.0
	28	2.1	1.9
	23	2.2	1.8
	28	2.1	1.9
	32	2.9	1.9
	49	2.5	2.1
	33	2.1	1.9
Team 3	66	2.2	3.5
	38	2.6	3.3
	68	1.7	3.3
	27	2.1	3.0
	52	3.1	3.6
	37	2.2	3.2
	24	1.7	2.8
	39	2.5	3.2
	23	1.7	2.8
	18	1.6	2.7
	90	2.6	3.8
	36	2.3	3.2
	90	2.6	3.8
Team 4	49	2.8	1.7
	34	2.3	1.6
	132	3.6	2.1
	31	2.4	1.6
	67	3.4	1.9
	93	3.5	2.0
	41	2.3	1.6
	28	2.5	1.5
	109	3.2	2.0
	118	3.0	2.0
	75	2.1	1.7
	98	2.8	1.9
	30	2.3	1.5

APPENDIX 5: Pillar rehabilitation costs

	Actual Pillar (m²)	Safety Factor	W:H ratio	Perimeter	bolt cost ZAR	Straps cost ZAR	Mesh Cost ZAR	Resin cost ZAR	Total cost ZAR	Total Cost US
Team 1	41	2.0	3.1	23.9	R 3,704.72	R 3,807.89	R 1,260.89	R 601.17	R 9,374.67	R 806.22
	33	1.9	2.4	22.7	R 3,525.20	R 3,623.37	R 1,199.79	R 572.04	R 8,920.40	R 767.15
	31	1.9	3.2	22.5	R 3,494.14	R 3,591.45	R 1,189.22	R 567.00	R 8,841.80	R 760.40
	76	2.2	2.6	36.8	R 5,714.86	R 5,874.02	R 1,945.03	R 927.36	R 14,461.26	R 1,243.67
	53	2.1	3.6	28.1	R 4,363.79	R 4,485.32	R 1,485.20	R 708.12	R 11,042.43	R 949.65
	89	2.2	2.6	40.2	R 6,242.86	R 6,416.72	R 2,124.73	R 1,013.04	R 15,797.35	R 1,358.57
	63	2.0	2.3	34.0	R 5,280.03	R 5,427.08	R 1,797.04	R 856.80	R 13,360.95	R 1,149.04
	57	2.0	2.5	31.4	R 4,876.26	R 5,012.07	R 1,659.62	R 791.28	R 12,339.23	R 1,061.17
	125	2.5	3.5	55.2	R 8,570.73	R 8,809.43	R 2,917.01	R 1,390.79	R 21,687.96	R 1,865.16
	31	1.8	2.2	21.9	R 3,400.96	R 3,495.68	R 1,157.50	R 551.88	R 8,606.02	R 740.12
	37	1.9	2.3	23.4	R 3,632.35	R 3,733.51	R 1,236.26	R 589.43	R 9,191.54	R 790.47
	33	1.8	2.3	23.5	R 3,655.64	R 3,757.45	R 1,244.18	R 593.21	R 9,250.49	R 795.54
	27	1.7	2.1	20.3	R 3,144.72	R 3,232.31	R 1,070.29	R 510.30	R 7,957.62	R 684.36
	48	2.0	3.8	26.8	R 4,154.14	R 4,269.84	R 1,413.84	R 674.10	R 10,511.92	R 904.03
	40	1.9	2.1	25.6	R 3,970.89	R 4,081.48	R 1,351.48	R 644.36	R 10,048.22	R 864.15
									R171,391.86	**$14,739.70**
Team 2	30	1.9	2.1	20.9	R 3,241.78	R 3,332.07	R 1,103.33	R 526.05	R 8,203.23	R 705.48
	27	1.8	2.0	19.4	R 3,016.92	R 3,100.94	R 1,026.79	R 489.56	R 7,634.21	R 656.54
	31	1.9	2.2	22.4	R 3,473.79	R 3,570.54	R 1,182.29	R 563.70	R 8,790.32	R 755.97
	38	2.0	3.5	24.2	R 3,764.35	R 3,869.19	R 1,281.18	R 610.85	R 9,525.57	R 819.20
	38	2.0	2.5	23.3	R 3,618.37	R 3,719.15	R 1,231.50	R 587.16	R 9,156.18	R 787.43
	32	2.0	2.5	21.4	R 3,321.76	R 3,414.27	R 1,130.55	R 539.03	R 8,405.61	R 722.88
	33	2.0	2.5	22.7	R 3,525.20	R 3,623.37	R 1,199.79	R 572.04	R 8,920.40	R 767.15
	28	1.9	2.2	20.3	R 3,152.49	R 3,240.29	R 1,072.94	R 511.56	R 7,977.27	R 686.05
	32	2.0	2.7	22.1	R 3,425.81	R 3,521.22	R 1,165.96	R 555.91	R 8,668.90	R 745.53
	28	1.9	2.1	20.6	R 3,199.08	R 3,288.17	R 1,088.79	R 519.12	R 8,095.16	R 696.18
	23	1.8	2.2	18.0	R 2,795.31	R 2,873.16	R 951.37	R 453.60	R 7,073.44	R 608.32
	28	1.9	2.1	20.6	R 3,199.08	R 3,288.17	R 1,088.79	R 519.12	R 8,095.16	R 696.18
	32	1.9	2.9	22.0	R 3,416.49	R 3,511.64	R 1,162.79	R 554.40	R 8,645.32	R 743.50
	49	2.1	2.5	26.8	R 4,160.35	R 4,276.22	R 1,415.96	R 675.11	R 10,527.64	R 905.38
	33	1.9	2.1	24.0	R 3,733.29	R 3,837.26	R 1,270.61	R 605.81	R 9,446.97	R 812.44
									R129,165.37	**$11,108.22**
Team 3	66	3.5	2.2	35.1	R 5,450.85	R 5,602.66	R 1,855.18	R 884.52	R 13,793.21	R 1,186.22
	38	3.3	2.6	23.1	R 3,587.31	R 3,687.22	R 1,220.93	R 582.12	R 9,077.58	R 780.67
	68	3.3	1.7	46.7	R 7,252.28	R 7,454.25	R 2,468.28	R 1,176.84	R 18,351.65	R 1,578.24
	27	3.0	2.1	19.7	R 3,061.18	R 3,146.43	R 1,041.86	R 496.74	R 7,746.20	R 666.17
	52	3.6	3.1	27.5	R 4,262.85	R 4,381.57	R 1,450.84	R 691.74	R 10,787.00	R 927.68
	37	3.2	2.2	23.0	R 3,574.89	R 3,674.45	R 1,216.70	R 580.10	R 9,046.15	R 777.97
	24	2.8	1.7	18.7	R 2,896.25	R 2,976.91	R 985.73	R 469.98	R 7,328.87	R 630.28
	39	3.2	2.5	24.1	R 3,741.06	R 3,845.25	R 1,273.25	R 607.07	R 9,466.62	R 814.13
	23	2.8	1.7	18.2	R 2,820.16	R 2,898.70	R 959.83	R 457.63	R 7,136.32	R 613.72
	18	2.7	1.6	16.9	R 2,616.72	R 2,689.60	R 890.59	R 424.62	R 6,621.53	R 569.45
	47	2.9	1.3	20.5	R 3,181.99	R 3,270.61	R 1,082.98	R 516.35	R 8,051.93	R 692.47
	36	3.2	2.3	23.0	R 3,571.79	R 3,671.26	R 1,215.64	R 579.60	R 9,038.29	R 777.29
	42	2.8	1.2	20.4	R 3,161.81	R 3,249.86	R 1,076.11	R 513.07	R 8,000.85	R 688.07
									R124,446.21	**$10,702.37**
Team 4	49	1.7	2.8	26.0	R 4,037.67	R 4,150.12	R 1,374.20	R 655.20	R 10,217.19	R 878.68
	34	1.6	2.3	23.4	R 3,633.90	R 3,735.11	R 1,236.78	R 589.68	R 9,195.47	R 790.81
	132	2.1	3.6	49.0	R 7,609.46	R 7,821.38	R 2,589.85	R 1,234.80	R 19,255.48	R 1,655.97
	31	1.6	2.4	23.0	R 3,571.79	R 3,671.26	R 1,215.64	R 579.60	R 9,038.29	R 777.29
	67	1.9	3.4	33.0	R 5,124.74	R 5,267.46	R 1,744.18	R 831.60	R 12,967.98	R 1,115.25
	93	2.0	3.5	37.8	R 5,870.15	R 6,033.64	R 1,997.88	R 952.56	R 14,854.23	R 1,277.46
	41	1.6	2.3	27.0	R 4,191.72	R 4,308.46	R 1,426.64	R 680.20	R 10,607.02	R 912.20
	28	1.5	2.5	19.9	R 3,096.12	R 3,182.34	R 1,053.75	R 502.41	R 7,834.62	R 673.78
	109	2.0	3.2	47.3	R 7,345.45	R 7,550.03	R 2,499.99	R 1,191.96	R 18,587.43	R 1,598.52
	118	2.0	3.0	53.0	R 8,228.46	R 8,457.63	R 2,800.52	R 1,335.25	R 20,821.86	R 1,790.68
	75	1.7	2.1	42.3	R 6,568.98	R 6,751.93	R 2,235.72	R 1,065.96	R 16,622.59	R 1,429.54
	98	1.9	2.8	49.0	R 7,609.46	R 7,821.38	R 2,589.85	R 1,234.80	R 19,255.48	R 1,655.97
	30	1.5	2.3	21.1	R 3,278.28	R 3,369.58	R 1,115.75	R 531.97	R 8,295.58	R 713.42
									R177,553.22	**$15,269.58**

APPENDIX 6: Risk Assessment example (Matrix and thresholds) (unpublished)

Likelihood					
1	1	3	6	10	15
2	2	5	9	14	19
3	4	8	13	18	22
4	7	12	17	21	24
5	11	16	20	23	25
Consequence	1	2	3	4	5

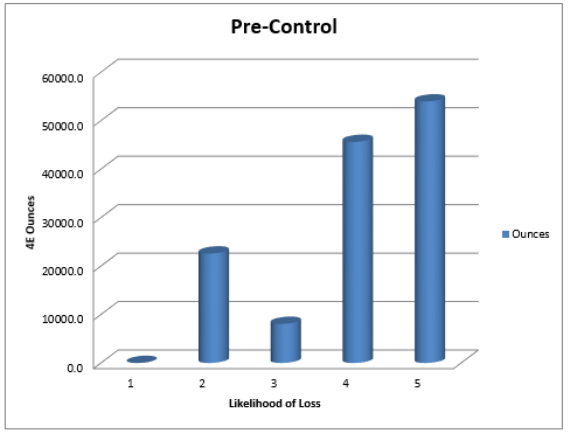
Loss (R Mill)	Consequence Rating
R 0	1
R 50	2
R 500	3
R 5,000	4
R 5,001	5

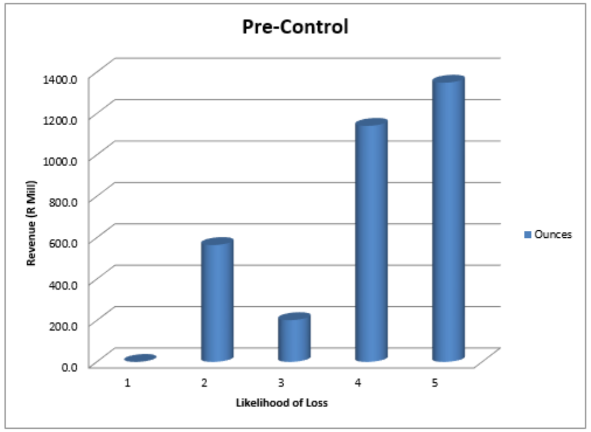
Parameters to consider
Poor ground conditions next to major structure
Poor ground conditions due to low RMR
Geologically very complex
High stresses
Stability problems in declines
History of losses in specific area
Infrastructure to clean
Seismicity
Ground water

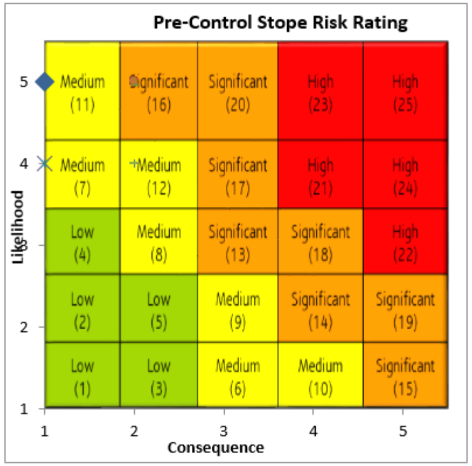
APPENDIX 6: Risk Assessment example (continued)

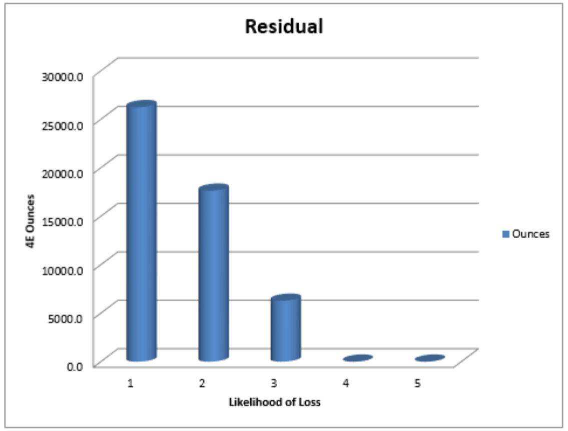
Pre-Control		
Likelihood	Ounces	Revenue
1	0.0	0.0
2	22596.0	564.9
3	8074.0	201.9
4	45628.5	1140.7
5	53973.6	1349.3

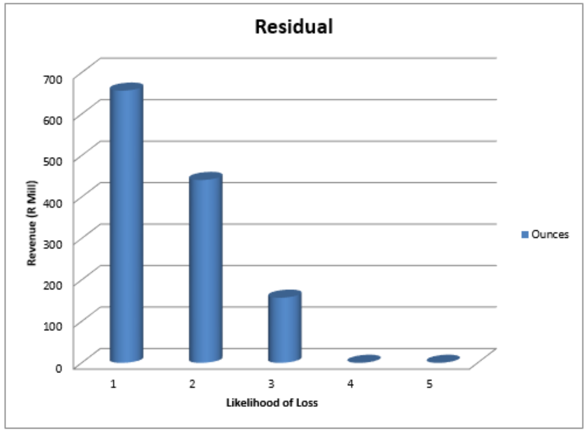
Residual		
Likelihood	Ounces	Revenue
1	26256.8	656
2	17638.7	441.0
3	6283.0	157.1
4	0.0	0.0
5	0	0

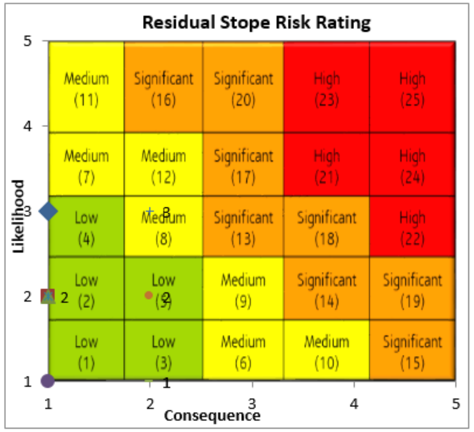












APPENDIX 6: Risk Assessment example (continued)

Platinum basket Price (R) R 25,000.00

UNKI MINE ROCK ENGINEERING
BP17 RISK ASSESSMENT

Working Place	Planned m²	Planned 4E Oz	Revenue (R Mill)	At risk Oz	At risk revenue (R Mill)	Pre-Control Risk: Likelihood of loss	Pre-Control Risk: Consequence of loss	Pre-Control Risk: Risk Rating	Hazards	Controls	At risk Oz	At risk revenue (R Mill)	Residual Risk: Likelihood of loss	Residual Risk: Consequence of loss	Residual Risk: Risk Rating
1N	18,917	13,068.32	R 326.71	13,068	R 326.71	5	2	16	*Mining challenge when negotiatong a series of step faults amounting to 2m throw*	*Redesigned the section. Reduced extraction to 75%*	1,307	R 32.67	3	1	4
2N	17,873	12,347.10	R 308.68	12,347	R 308.68	5	2	16	*Mining challenge when negotiatong a series of step faults amounting to 2m throw*	*Redesigned the section. Reduced extraction to 75%*	1,235	R 30.87	2	1	2
3N	17,339	11,978.20	R 299.45	11,978	R 299.45	5	2	16	*Mining challenge when negotiatong a series of step faults amounting to 2m throw*	*Redesigned the section. Reduced extraction to 75%*	1,198	R 29.95	2	1	2
4N	21,333	14,737.35	R 368.43	1,474	R 36.84	4	1	7	*Poor ground conditions next to major structures*	*Reduce bords to 6m and advances for 2 blast on either side of the major structures*	737	R 18.42	1	1	1
5N	28,175	19,463.96	R 486.60	1,946	R 48.66	4	1	7	*Poor ground conditions next to major structures*	*Reduce bords to 6m and advances for 2 blast on either side of the major structures*	973	R 24.33	2	1	2
6N	69,106	47,740.08	R 1,193.50	4,774	R 119.35	5	2	16	*Geologically very complex, History of losses in this area*	*All bords in "S" Classto be mined at 6m and all those in B at 8m*	2,387	R 59.68	2	2	5
7N	83,718	57,834.40	R 1,445.86	5,783	R 144.59	4	2	12	*Poor ground conditions next to major structures*	*Reduce bords to 6m and advances for 2 blast on either side of the major structures*	2,892	R 72.29	3	2	8
8N	102,484	70,798.40	R 1,769.96	7,080	R 177.00	4	2	12	*Poor ground conditions next to major structures*	*Reduce bords to 6m and advances for 2 blast on either side of the major structures*	3,540	R 88.50	1	2	3
9N	76,565	52,892.93	R 1,322.32	5,289	R 132.23	4	2	12	*Poor ground conditions next to major structures*	*Reduce bords to 6m and advances for 2 blast on either side of the major structures*	2,645	R 66.12	1	2	3
10N	76,977	53,177.55	R 1,329.44	5,318	R 132.94	5	2	16	*Poor ground conditions next to major structures*	*Reduce bords to 6m and advances for 2 blast on either side of the major structures*	2,659	R 66.47	2	2	5
11N	60,348	41,689.84	R 1,042.25	4,169	R 104.22	2	2	5	*Poor ground conditions next to major structures*	*Reduce bords to 6m and advances for 2 blast on either side of the major structures*	2,084	R 52.11	3	2	8
10S	73,651	50,879.87	R 1,272.00	5,088	R 127.20	2	2	5	*Poor ground conditions next to major structures*	*Reduce bords to 6m and advances for 2 blast on either side of the major structures*	2,544	R 63.60	1	2	3
11S	61,245	42,309.51	R 1,057.74	4,231	R 105.77	2	2	5	*Poor ground conditions next to major structures*	*Reduce bords to 6m and advances for 2 blast on either side of the major structures*	2,115	R 52.89	1	2	3
12S	57,488	39,714.09	R 992.85	3,971	R 99.29	2	2	5	*Poor ground conditions next to major structures*	*Reduce bords to 6m and advances for 2 blast on either side of the major structures*	1,986	R 49.64	1	1	1
13S	52,094	35,987.78	R 899.69	3,599	R 89.97	2	2	5	*Poor ground conditions next to major structures*	*Reduce bords to 6m and advances for 2 blast on either side of the major structures*	1,799	R 44.98	1	1	1
14S	46,288	31,976.86	R 799.42	3,198	R 79.94	3	2	8	*Poor ground conditions next to major structures*	*Reduce bords to 6m and advances for 2 blast on either side of the major structures*	1,599	R 39.97	2	1	2
Total	1,398,588	966,178	R 24,154	130,272	R 3,256.80					Oz at risk rating between >13	0.00	R -			
										Oz at risk rating between 13 and 21	0.00	R -			
										Oz at risk rating > 20	0.00	R -			