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THE PRACTICALITY OF USING ROCK MASS CLASSIFICATION IN A NARROW TABULAR OREBODY AT DEPTH

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DECLARATION

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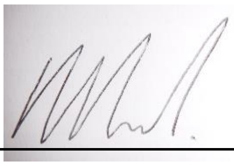

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ABSTRACT

Rock mass classification is in use throughout most of the mining industry, but not in common practice with narrow tabular gold-bearing orebodies at great depth. Many classification systems exist, but only one classification system was designed for mining applications, which is the MRMR system designed by Laubscher (1990). MRMR was mainly developed for caving applications, but through a thorough literature review and underground investigations, it was found to be the best suited for this particular application. The system has been modified to suit the narrow tabular reef mining environment at great depth, to assess the hanging wall objectively and to incorporate relevant mining adjustments, and rock mass rating properties. Therefore, although rock mass classification is not widely practised in the narrow tabular gold-bearing orebodies, this modified system is well suited for the environment. The system has been designed to be simple and easy to use.

DEDICATION

This work is dedicated to my wife, Candice, for her unwavering support through the process of completing this research report, and her ability to keep me going when I faltered.

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I want to express my gratitude to the following people and organisation for their support during the compiling of this research report:

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MPa	Megapascals
N	Newtons
kg	Kilograms
m	Metres
MJ/m^2	Energy Release Rate in Megajoules per metre squared.

1 INTRODUCTION

1.1 Problem Statement

The study will aim to understand whether a rock mass classification system can be practically implemented in a narrow tabular reef mining environment at great depth, and to assist in the prevention of falls of ground in the face area of a stope, by predicting ground conditions ahead of the advancing face.

1.2 Justification for Research

A rock mass can be considered a naturally occurring complex geological material and the behaviour of the ground is dependent on the conditions and the naturally occurring features (Rehman, et al., 2018). Rock mass classification systems have been in use in the mining industry as well as the civil engineering industry for many years. These systems provide a useful tool for engineers as they can provide a starting point for the design of a tunnel and force the user to examine the rock mass in a systematic way (Hoek, 2006). Rock mass classification in the South African context has been used for varying applications from massive to open-pit mining and used at the Lesotho Highlands Water Project to empirically classify the water tunnels (De Graaf & Bell, 1997). Laubscher developed the first real mining rock mass classification system in 1975 for cave mining operations, modified by Laubscher and Taylor (1976) and termed the mining rock mass rating in 1990 (Laubscher, 1990), (Dyke, 2006).

Although these systems have been in use for several years they are based on data gathered from civil engineering projects and massive mining methods such as sub-level cave mining and not the specific gold mining environment that the research described in this report will be based upon.

1.3 Previous Work

Previous work includes that of Hanekom (2003), Watson (2004) and Watson and Gerber (2018), but nothing specific to the deep gold mining industry. Watson developed the modified stability number for the Bushveld Complex but not for great depth. Gumede (2006) attempted to define common joint characteristics in South African gold mines, which was further extended by Stacey and Gumede (2007) to evaluate the risk of rock fall accidents based on measured joint data, and although these two projects mapped joint data, they were not included in a type of rock mass classification system.

1.4 Research Methods

Research design: The research will be quantitative. Data will be collected from underground working areas and will be inserted into a self-designed program that will calculate the various parameters for the rock mass classification. Data collection will be from primary and secondary sources such as the in situ stress database for Southern Africa (Wesseloo & Stacey, 2006).

Methods and sources: Data collected for the various parameters will be fracture frequency per metre (ff/m), rock material strength (UCS) to obtain the intact rock strength (IRS), joint spacing for a varying number of joint sets, and joint condition, which includes; large scale joint expression, small scale joint expression, joint wall alteration zone and joint infilling. The parameters for joint condition can relate to wet or dry circumstances. Adjustments will then be made for joint orientation, stress regime, weathering and blasting, and possibly other parameters if appropriate.

The discrete fracture network (DFN) method developed by Haines (1983) and Grady (1983), which has been developed over the last 30 years, will be assessed as a potential approach to further assist in the assessment of the joint and fracture network in the stope, in conjunction with ff/m or on its own. Input for the parameters include joint dip direction, dip angle, dip and strike lengths, and joint spacing, and ranges of each (Stacey, et al., 2015). Gumede (2006) mapped conventional stopes in two gold mines and used the data to create a DFN, and subsequently evaluated the potential for rock falls (Stacey, et al., 2015).

1.5 Sources of Data

The data will be collected from 23 working places, which will consist of ledges and stoping panels. This will contribute towards adding the transferability of findings across real-world mining conditions. Furthermore, additional information will be collected from laboratory uniaxial strength tests as well as data from existing sources, such as the in situ stress database for Southern Africa (Wesseloo & Stacey, 2006). The data will be analysed through a self-designed rock mass rating program as indicated in section 1.4, and comparisons will be done for the different areas that are assessed.

1.6 Structure of the Research Report

An introduction to the research is included in Chapter 1. To provide background to the research, a review of relevant literature is conducted in Chapter 2. Chapter 3 describes the data collection process and outlines the methodology. A justification for the use of a modified version of the Mining Rock Mass Rating (MRMR) system is described in Chapter 4. Chapter 5 analyses the data collected and the findings thereof. Chapter 6 describes the research conclusions, and Chapter 7 is the recommendation for future work.

2 LITERATURE REVIEW

2.1 Definition of a Rock Mass

The material in which mining takes place can be described as a non-homogeneous construction material built up of fragments and blocks of varying size. There is great diversity both in the composition of the intact rock and in the nature and extent of its discontinuities, and rock masses exhibit a wider range in structure, composition and mechanical properties compared to most other construction materials (Palmstrom, 1996). The rock mass can be further defined as a discontinuous medium comprising of partitioned solid bodies or aggregates of blocks, more or less separated by planes of weakness, which commonly fit together tightly, with water and soft and/or hard infilling materials present in the spaces between the blocks (Dyke, 2008).

Rock mass classification methods have been developing for more than 100 years since Ritter (1879) attempted to formalise an empirical approach to tunnel design, in particular for support requirements (Hoek, 2006). Rock mass classification systems can be very useful practical engineering tools, not only because they provide a starting point for the design of tunnel support but also because they force users to examine the properties of the rock mass in a very systematic manner (Hoek, 2006). Although there are challenges and difficulties associated with classifying a rock mass quantitatively, Laubscher (1990) states that *"a classification system must be straightforward and have a strong practical bias so that it can form part of* *the normal geological and rock engineering investigations to be used for mine design and communication"*.

2.2 Rock Mass Classification Systems

Through the introduction, a succinct discussion has been put forward on rock mass classification, a short history on it, its importance during design, mining and execution, and how it can be of value for this project. Through a literature review, it is expected that the mining rock mass rating system developed by Laubscher (1990) will be the most appropriate system to use as a basis for the design of a classification system for this research project. This literature review is presented in the following sub-sections.

2.2.1 Terzaghi's rock mass classification system (1946)

Rock mass classification can trace its history further back than the 1940s but the first real classification system was developed in 1946 by Terzaghi (1946) and was used for the design of steel supports in tunnels (Stacey, 2019) This was the first classification system to recognise the importance of the geological structure. Terzaghi's (1946) paper contains descriptive engineering geology information for:

- intact rock,
- stratified rock,
- moderately jointed rock,
- blocky and seamy rock,
- crushed but chemically intact rock,

- squeezing rock
- swelling rock (Hoek, 2006).

2.2.2 Rock quality designation (RQD)

Rock quality designation is defined as *"a modified core recovery percentage in which all pieces of 'sound' core over 4 inches long (100 mm) are summed and divided by the length of the core run"* (Pells, et al., 2017). RQD has been historically widely used as a rating system on its own and as a parameter for other rating systems such as Barton, et al., (1974) Q-rating and Bieniawski (1989) geomechanical classification system. Since its inception, it has been used as a fundamental tool in rock mass characterisation. It was originally devised whilst working in granite at the Nevada test site for nuclear bombs (Deere & Deere, 1989) as an index for classifying the quality of rock core obtained from small diameter (about 50mm) core drilling.

Table 2-1: RQD and The Relationship with Rock Quality (Deere & Deere, 1989)

RQD (Rock Quality Designation)	**Description of Rock Quality**
0 – 25%	Very Poor
25 – 50 %	Poor
50 – 75%	Fair
75 – 90%	Good
90 – 100%	Excellent

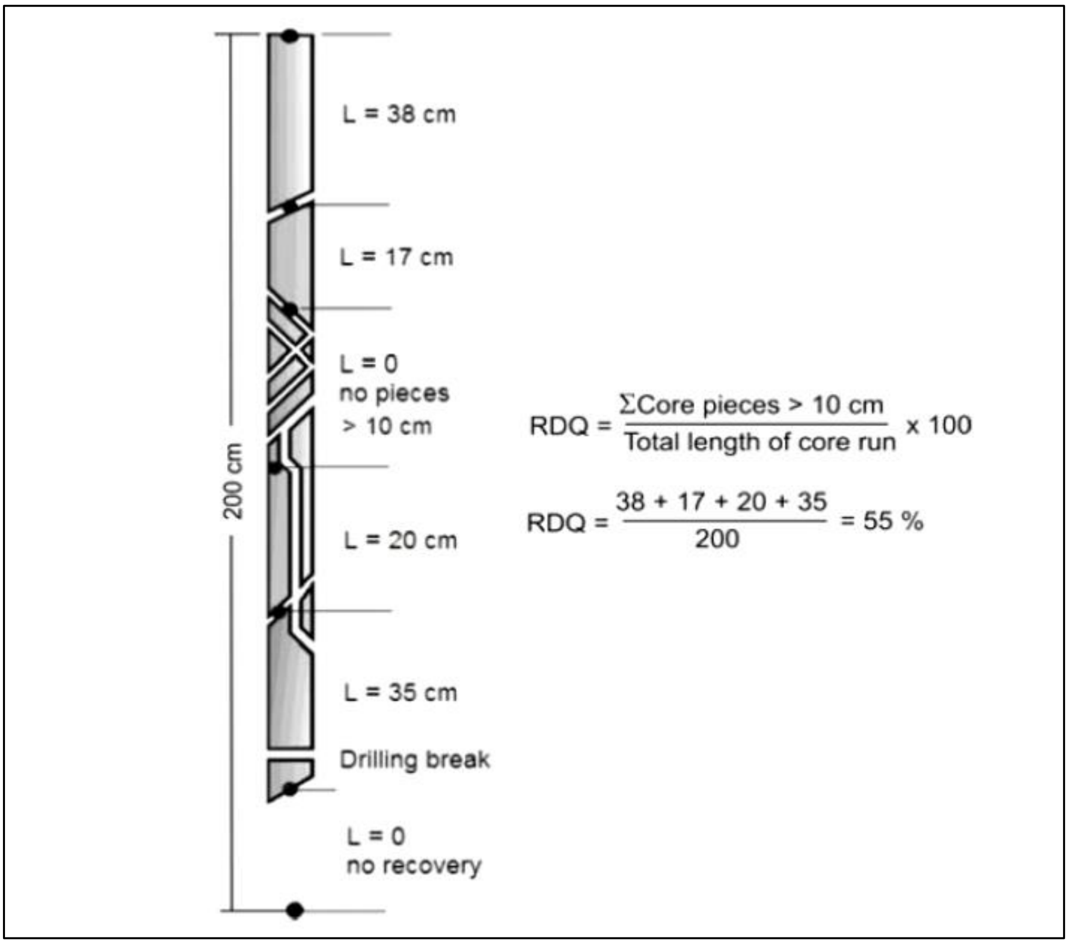


Figure 2-1: Procedure for measurement and calculation of RQD after Deere 1989 (Hoek, 2006).

The direction in which the core is drilled can significantly affect the RQD as it is direction-dependent (Hoek, 2006) and further, an underestimation of the rock quality could be obtained by ignoring pieces of core less than 100mm in the length. Earth-like core or fresh rock pieces are discarded if their length is too short (Palmstrom, 2005). Some limitations of the method were also highlighted by Bieniawski (1973) who stated that RQD disregards the influence of joint orientation, continuity and gouge material. Bieniawski (1973) highlighted the simplicity of RQD, relative inexpensiveness to use and ease of reproduction. An article published by (Pells, et al., 2017) titled: *Rock Quality Designation (RQD): time to rest in peace* argues that RQD has several inherent limitations. The work conducted on South African unlined spillways was part of a major study funded by various Australian authorities responsible for dam maintenance and construction (Pells, et al., 2017). The project involved the mapping and rock mass classification of unlined spillways of various South African dams after which the results of the investigation by Pells and Pells (2014) was compared to that of van Schalkwyk, et al., (1994) (Pells, et al., 2017).

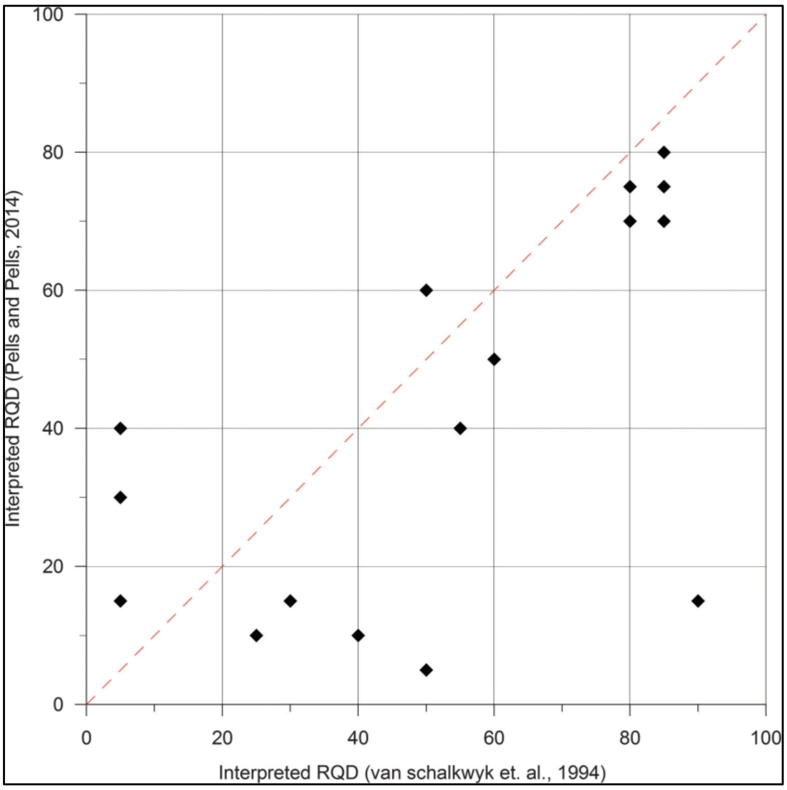


Figure 2-2: Comparison between interpreted RQD values and various unlined spillway sites (Pells, et al., 2017).

A summation of the article's conclusion states the following, *"different parts of the world use RQD in different ways making it inconsistent with the original methodology of Deere. In the dominant classification systems, RQD is mostly obtained from surface exposures which can be fraught with subjectivity by the user."* The rock mass rating system and mining rock mass rating system has recommended that RQD be replaced by fracture frequency (Pells, et al., 2017) and therefore rock quality designation will not be used and instead fracture frequency per metre will be implemented in the present research. The advantage of the fracture frequency per metre (FF/m) technique is that it is more sensitive for a wide range of joint spacing compared to that of RQD, as it takes into account joint spacing of less than 100mm (Laubscher, 1990).

2.2.3 Q-Rating

When the NGI Tunneling Index or Q-rating was developed by (Barton, et al., 1974) it covered some 200 hundred case studies on tunnels and revealed an interesting, but satisfactory correlation between permanent support and the rock mass quality *Q.* The numerical value of Q ranges from 0.001 for poor quality squeezing ground to 1000 for exceptionally good practically unjointed rock (Barton, et al., 1974). The Q-rating is a function of six parameters, each of which has a rating, which can be estimated from surface mapping and subsequently during excavation (Barton, et al., 1974). The six parameters are:

- the rock quality designation (RQD),

- the joint set number (J_n),
- the roughness of the weakest joints (J_r),
- the degree of alteration or filling on the weakest joints (J_a),
- the stress reduction factor (SRF), and
- joint water inflow (J_w).

Table 2-2: Q equation and the three main factors which describe the stability of underground openings (Norwegian Geotechnical Institute, 2015).

$Q = \frac{RQD}{Jn} x \frac{Jr}{Ja} x \frac{Jw}{SRF}$	**Equation 2-1**
RQD ÷ J_n	This component represents the overall structure of the rock mass and happens to be a simple measure of the block size.
J_r ÷ J_a	The quotient represents the roughness and degree of alteration of the joint walls or infilling material. Using $\tan^{-1}(J_r/J_a)$ a fair approximation of shear strength can be found. Inter-block shear strength.
J_w ÷ SRF	The quotient consists of two stress parameters taking into account water pressure which has a negative effect on joint stability and SRF which takes into account loosening load in clay or sheared rock, rock stress in competent rock and squeezing and swelling. Active stress.

The rating values for each parameter are given in the tables below:

Table 2-3: Rock Quality Designation (RQD) (Barton, 2002).

Rock Quality Designation		**RQD (%)**
A	Very Poor	0-25
B	Poor	25-50
C	Fair	50-75
D	Good	75-90
E	Excellent	100

Table 2-4: Joint set number (Jn) (Barton, 2002).

Joint Set Number		**J_n**
A	Massive, no or few joints	0.5-1
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, earthlike	20

Table 2-5: Joint Roughness Number (Jr) (Barton, 2002).

Joint Roughness Number		**J_r**
a) Rock-wall contact, and b) rock-wall contact before 10cm shear		
A	Discontinuous joints	4
B	Rough or irregular, undulating	3
C	Smooth, undulating	2
D	Slickensided, undulating	1.5
E	Rough or irregular, planar	1.5
F	Smooth, planar	1.0
G	Slickensided, planar	0.5
No rock-wall contact when sheared		
H	A zone containing clay mineral thick enough to prevent rock-wall contact	1.0
J	Sandy, gravely or crushed zone thick enough to prevent rock-wall contact	1.0

Table 2-6: Joint Alteration Number (Ja) (Barton, 2002).

Joint alteration number		**Φr approx. (deg)**	**J_a**
a) Rock-wall contact (no mineral fillings, only coatings)			
A	Tightly healed, hard, non-softening, impermeable filling	-	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	8-16	4
b) Rock-wall contact before 10cm shear (thin mineral fillings)			
F	Sandy particles, clay-free disintegrated rock.	25-30	4
G	Strongly over-consolidated non-softening clay mineral fillings (continuous but <5mm thickness)	16-24	6
H	Medium or low-consolidated, softening, clay mineral fillings (continuous but <5mm thickness)	12-16	8
J	Swelling-clay fillings (continuos but <5mm thickness) J_a depends on per cent of swelling clay-size particles and access to water.	6-12	8-12
c) No rock-wall contact even when sheared (thick mineral fillings)			
KLM	Zones or bands of disintegrated or crushed rock and clay (See G, H, J for a description of clay)	6-24	6, 8, or 8-12
N	Zones or bands of silty or sandy clay, small clay fraction (non-softening)	-	5
OPR	Thick, continuos zones or bands of clay (see G, H, J for a description of clay condition)	6-24	10, 13, or 13-20

Table 2-7: Joint Water Reduction Factor (Jw) (Barton, 2002).

Joint water reduction factor		**Approx. water pres. (kg/cm^2)**	**J_w**
A	Dry excavations or minor inflow, i.e, <51/min locally	<1	1
B	Medium inflow or pressure, occasional outwash of joint fillings	1-2.5	0.66
C	Large inflow or high pressure in competent rock with unfilled joints	2.5-10	0.5
D	Large inflow or high pressure, considerable outwash of joints fillings	2.5-10	0.33
E	Exceptionally high inflow or water pressure at blasting, decaying with time	>10	0.2-0.1
F	Exceptionally high inflow or water pressure continuing without noticeable decay	>10	0.1-0.05

Table 2-8: Stress Reduction Factor (SRF) (Barton, 2002)

Stress reduction factor				**SRF**
	a) Weakness zones intersecting excavation, which may cause loosening of rock mass when the tunnel is excavated			
A	Multiple occurrences of weakness zones containing clay or chemically disintegrated rock, very loose surrounding rock (any depth)			10
B	Single weakness zones containing clay or chemically disintegrated rock (depth of excavation ≤50m)			5
C	Single weakness zones containing clay or chemically disintegrated rock (depth of excavation >50m)			2.5
D	Multiple shear zones in competent rock (clay-free), loose surrounding rock (any depth)			7.5
E	Single shear zone in competent rock (clay-free), (depth of excavation ≤50m)			5
F	Single shear zone in compotent rock (clay-free), (depth of excavation >50m)			2.5
G	Loose, open joints, heavily jointed or 'sugar cube' (any depth)			5
	b) Competent rock, rock stress problems			
		σ_c/σ_1	**σ_Θ/σ_c**	**SRF**
H	Low stress, near surface, open joints	>200	<0.01	2.5
J	Medium stress favourbale stress conditions	200-10	0.01-0.3	1
K	High stress, very tight structure. Usually favourable to stability, may be unfavourable to wall stability	10-5	0.3-0.4	0.5-2
L	Moderate slabbing after >1hr in massive rock	5-3	0.5-0.65	5-50
M	Slabbing and rockburst after a few minutes in massive rock.	3-2	0.65-1	50-200
N	Heavy rock burst (strain-burst) and immediate dynamic deformation in massive rock	<2	>1	200-400
			σ_Θ/σ_c	**SRF**
	c) Squeezing rock: the plastic flow of incompetent rock under the influence of high rock pressure			
			σ_Θ/σ_c	**SRF**
O	Mild squeezing rock pressure		1-5	5-10
P	Heavy squeezing rock pressure		>5	10-20
				SRF
	d) Swelling rock: chemical swelling activity depending on the presence of water			
R	Mild swelling rock pressure			5-10
S	Eavy swelling rock pressure			10-15

Analysis of the rock mass quality led to suitable permanent support estimations for the varying rock mass qualities which are based on support pressure, rock mass quality Q and span of the excavation (Barton, et al., 1974). The Q-system, initially based on 212 case studies, was derived from an era when plain shotcrete (S), or steel-mesh reinforced shotcrete (Smr), or cast concrete arches and varying rock bolts were used for tunnel and cavern support. Subsequently, in 1993 the support recommendations based on an additional 1050 case studies was updated and included the update of steel-fibre reinforced sprayed concrete (Sfr) in place of (Smr). Despite the significant increase in case studies very little was changed in the rock mass quality index *Q* and just three of the strength/stress parameters were added to include the support of massive intact blocks under high stress (Barton, 2002).

The Q-rating system has been updated through the years and has been further developed for tunnelling with a tunnel boring machine (Barton, 2000) as well as the assessment of the stability of raise bored shaft (McCracken & Stacey, 1989).

2.2.4 Geomechanics classification or rock mass rating (RMR)

The Geomechanics classification system was originally developed by Bieniawski (1973). Rock mass rating, RMR_{73} contained 5 main parameters for the classification of a jointed rock mass, in which each parameter has a rating. The parameters are the strength of the intact rock material, RQD, joint spacing, joint condition and groundwater.

Each rating carried a different weighting and when totalled, ultimately gave a rock mass rating out of 100. The system was further refined to improve it in 1989 and 2014 with further adjustments being done in between. These variations are due to parameters that were added or removed such as modifications for groundwater, joint orientation, spacing and condition, excavation method and stress-strain behaviour (Rehman, et al., 2018). The most noticeable difference between the Q tunnelling index and Bieniawski's RMR system is that the latter lacked a stress parameter (Hoek, 2006).

The parameters carry different values and weightings according to their individual importance. To use the system, the rock mass is divided into structural regions so that features are more or less uniform with each region. Even though the rock mass is discontinuous some uniformity exists in different regions, for example, the joint spacing and direction, or similar types of rock. Commonly the areas are divided by geological features such as fault or dyke (Bieniawski, 1979). The five individual parameters are assessed and added together to obtain a rock mass rating. Thereafter further adjustments can be called for, for different applications such as mining where further adjustments are made for stress and blasting (Bieniawski, 1979).

Table 2-9: The 1989 RMR classification system (Bieniawski, 1989)

Parameter			**Ranges of Values**						
1	Strength of intact rock material	Point load index	>10MPa	4-10 Mpa	2-4 Mpa	1-2 Mpa	For this low range – a uniaxial compressive test is preferred		
		Uniaxial compressive strength	>250MPa	100-250Mpa	50-100 Mpa	25-50 Mpa	5-25 Mpa	1-5 Mpa	<1 Mpa
	Rating		**15**	**12**	**7**	**4**	**2**	**1**	**0**
2	Drill core quality RQD		90%-100%	75%-90%	50%-75%	25%-50%	<25%		
	Rating		**20**	**17**	**13**	**8**	**3**		
3	Spacing of joints		>2mm	0.6-2m	200-600mm	60-200mm	<60mm		
	Rating		**20**	**15**	**10**	**8**	**5**		
4	Condition of Joints		Very rough surface Not continuous No separation Weathered wall rock	Slightly rough surfaces Separation <1mm Slightly weathered walls	Slightly rough surfaces Separation <1mm Highly weather walls	Slicken-sided surfaces, or Gouge <5mm thick, or Separation 1-5mm continuous	Soft gouge >5mm thick, or Separation >5mm continuous		
	Rating		**30**	**25**	**20**	**10**	**0**		
5	Ground-water	Inflow per 10m tunnel length (l/min)	None	10	10-25	25-125	>125		
		Joint water pressure/major principal stress	0	0.0-0.1	0.1-0.2	0.2-0.5	>0.5		
		General conditions	Complete-ly dry	Damp	Wet	Dripping	Flowing		
	Rating		**15**	**10**	**7**	**4**	**0**		
6	Strike and dip orientations of joints*		Very favourable	Favourable	Fair	Unfavourable	Very unfavour-able		
	Rating		**0**	**-2**	**-5**	**-10**	**-12**		

* The effect of joint strike and dip orientation

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

Traditionally the geomechanics classification system was only used for tunnelling, but it has been extended for other applications such as the design of rock slopes, dam foundations and mining (Bieniawski, 1979). The geomechanics classification system is a useful tool for assessing rock mass conditions in a range of different engineering applications (Bieniawski, 1979).

2.2.5 Discrete Fracture Network

Stacey et. al. (2015) describes a discrete fracture network (DFN) developed by Haines (1983) and Grady (1983) which was used to quantify potential rockfalls and thereby develop and determine appropriate rock bolt lengths and spacings for a contractual dispute. In later years a computer analysis programme was developed called JPLOT, which uses actual gathered data from field mappings, taking into account statistical variances in these data, to create a graphical representation of joint traces from two-dimensional sections (Stacey, et al., 2015).

The method described by (Stacey, et al., 2015) and developed by Haines (1984) will be further elaborated on later in this section. The input parameters for the system include joint set dip direction, dip angle, dip and strike length, joint spacing and ranges of each (Stacey, et al., 2015). Data collected from field mapping can be used or, if not available, 'standard' distributions of these parameters, based on published worldwide field mappings can be assumed (Stacey, et al., 2015). Substantial advances have been made since the early development of DFN by Haines (1984), and sophisticated three-dimensional applications have been developed to assess wedge and key-block failure (Stacey, et al., 2015). The method described in the journal article by (Stacey, et al., 2015) has proved to be very successful in many different applications and it is stated that it complies with the simplicity principle of the rock mechanics design principles of Bieniawski (1992).

The application of a DFN method requires that joint traces are plotted to scale on appropriate two-dimensional sections through the excavation (Stacey, et al., 2015), in a mining stope this would be limited to the hanging wall and face but in some instances could include a sidewall if present. Haines (1984) states that various discontinuity patterns should be examined on the sidewalls, faces, roofs or benches of the proposed excavation. Furthermore, the generating method for the patterns has been developed for computer use and five parameters should be captured for the generation of discontinuity patterns. The five parameters are listed below:

- Dip Direction (degrees 0° - 360°),
- Dip Angle (degrees 0° - 90°),
- Strike Length (metres),
- Dip Length (metres), and
- Spacing (metres).

The dimensions of the required values are given in parenthesis. The Monte Carlo Sampling Technique is adopted through the generation process where random sampling from the known field derived distributions of the five structural parameters is carried out (Haines, 1984). The geometry of the excavation is then plotted onto the traces using the same scale, and potentially unstable blocks and wedges are identified (Stacey, et al., 2015). This process is repeated numerous times to produce distributions of outputs such as unstable block size, surface area, volume, depth etc. It is a manual process to identify the unstable block and requires engineering judgement (Stacey, et al., 2015).

Several case studies are presented by (Stacey, et al., 2015) in the paper;

- Evaluation of potentially unstable blocks in a tunnel;
- Prediction of potential breakout in a ventilation shaft;
- Predication of cavability and fragmentation in a block cave mining project;
- Prediction of the stability of ore passes under high-stress conditions;
- Prediction of back-break in rock slopes

The discussion on the prediction of rock falls in deep tabular gold mine stopes is of particular interest as it is applicable to rock mass classification in a narrow tabular orebody (Gumede, 2006). The application of a DFN to predict rock falls in tabular deep level stopes is of particular interest in the context of this research report as it could potentially predict the size and probability of a rock fall occurring. It is known that rock falls in these types of excavations usually result from the interaction between joints as well as joints and stress-induced fractures (Stacey, et al., 2015). Prior to the work done by Gumede (2006) and Stacey and Gumede (2007), no published data on joint parameters existed for the gold mines mining within the Witwatersrand Basin (Stacey, et al., 2015). Following work done by Gumede (2006), which included mapping of conventional mining stopes to generate the jointing statistical data, the information was used to create DFN's and evaluated the potential for rock falls. The information was also used to predict the fall out thickness. Stacey & Gumede (2007) took the analysis further and used a key-block based programme to assess the probability of gravity-driven rock falls and evaluated support effectiveness (Stacey, et al., 2015).

This particular case study deals with an open-pit mining scenario with attention given to the extent and volume of material involved in three-dimensional jointed slopes. Failure of rock slopes are normally a result of geological planes of weakness, and in-situ stresses are normally quite low and not normally considered to be of major importance. Data on the back-break of model slopes were available from small-scale centrifugal loading and the DFN method was then applied to the models using two sets of joint data (Stacey, et al., 2015). Following the analysis, it was found that there was close agreement between the physical models and DFN predictions using actual joint geometries. The method was implemented in a real-life vertical open pit to predict the extent of the failure of a back-break.

2.2.6 Geological strength index (GSI)

Hoek and Brown recognized that a rock mass failure criterion would have no practical value unless it could be related to geological observations that could be made quickly by an engineer or engineering geologist (Marinos, et al., 2005). The GSI system was introduced by Hoek (1994) as an alternative means to the RMR system, to determine rock mass strengths but not as a replacement. The system is based on the first four parameters of the RMR system and was introduced to overcome the limitations of RMR in poor quality rock masses, and to avoid duplicating the influence of groundwater and joint orientations in RMR or the stress reduction factor (SRF) and joint water factor (J_w) in the Q system (Bertuzzi, et al., 2016). The original GSI was a number based on four parameters; intact rock strength, RQD, spacing of discontinuities and $JCond_{89}$ (Bertuzzi, et al., 2016).

The GSI is a form of rock mass characterisation that was developed for engineering rock mechanics to satisfy the need for reliable input data, in particular those related to rock mass properties required as inputs into numerical analysis or closed-form solutions for the design of tunnels, slopes and foundations in rock (Marinos, et al., 2005). The visual assessment of the rock mass and the geological characteristics are used as a direct input to the applicable parameters relevant for the prediction of rock mass strength and deformability (Marinos, et al., 2005). The approach allows for the rock mass to be considered as a mechanical continuum medium including the influence the geology has on its mechanical properties. It provides the engineering geologist or engineer with a method to visually determine difficult to describe rock masses (Marinos, et al., 2007).

The GSI system places greater emphasis on geological observations of the rock mass, the material and its structure and does not include RQD due to its ineffectiveness in weak rock masses (Marinos, et al., 2005). GSI is a function of the rock mass properties and entails an assessment of the lithology, structure and condition of discontinuity surfaces in the rock mass and it is estimated from visual examination of an exposed rock surface and rock core samples. The GSI combines two important parameters of the geological process to determine a number for the given rock mass, and these are the blockiness of the mass and the condition of the discontinuities, and includes the main geological constraints that govern a formation (Marinos, et al., 2005). Using a visual indicator could assist the geologist or rock engineer when conducting an assessment in a narrow stope.

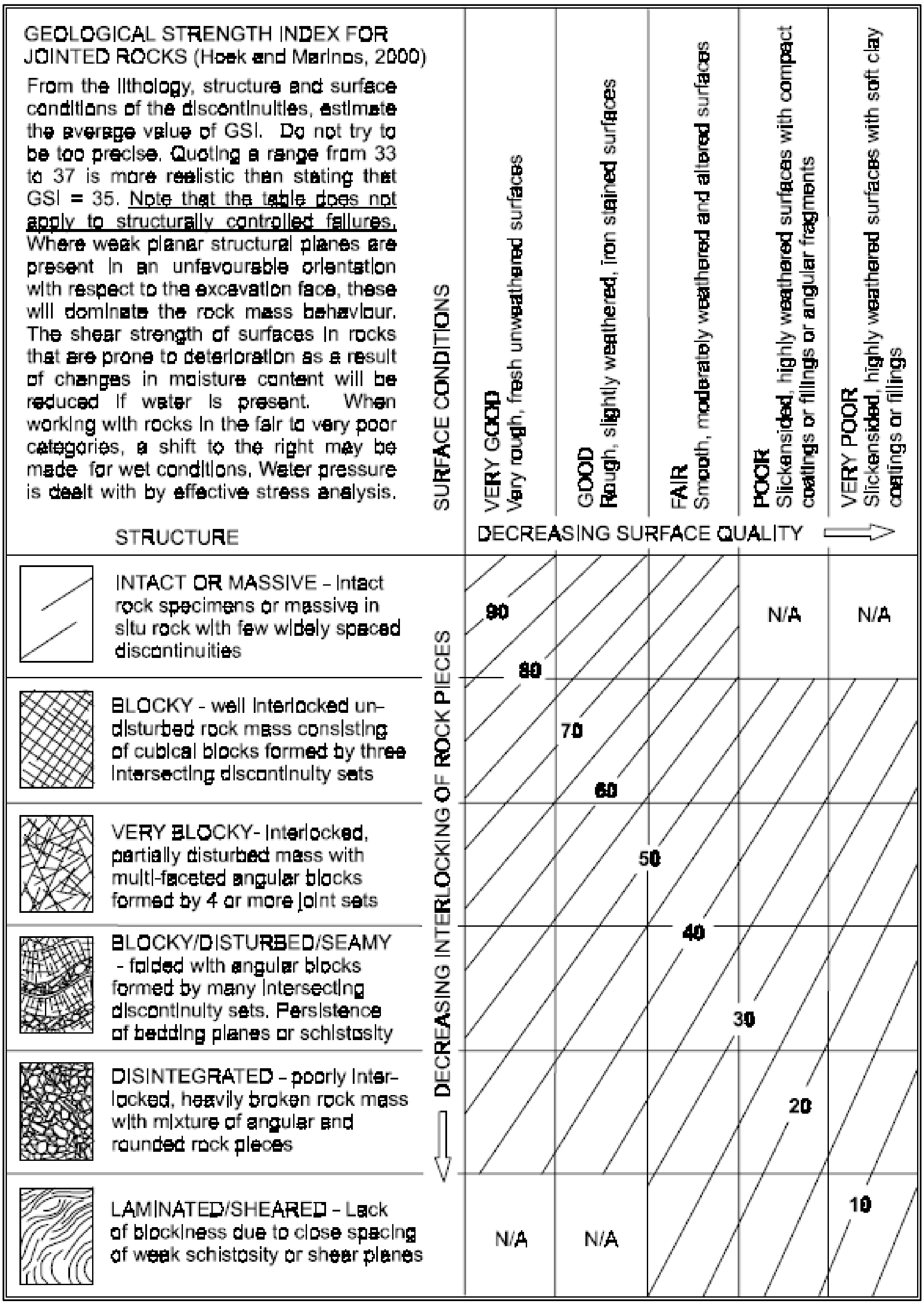


Figure 2-3: Geological Strength Index (GSI) for a Jointed Rock Mass (Marinos & Hoek, 2000)

2.2.7 Modified rock mass rating for mining (MRMR)

Laubscher (1977) (1984), Laubscher and Taylor (1976) have described a Modified Rock Mass Rating (MRMR) system for mining (Laubscher, 1990).

The MRMR classification system was introduced in 1974 as a modification to the CSIR geomechanics classification system (Bieniawski, 1973). The system was initially based on the RMR system as defined by Bieniawski (1973) but was modified as Laubscher (1976) found that the RMR system did not appropriately suit the design and concept behind his rating system, and was too inflexible for mining applications (Dyke, 2006). The system was developed to form the Mining Rock Mass Rating, which takes into account the intact rock strength, spacing of fractures and joints, joint condition and water. These ratings give a value out of 100, and they are then further adjusted for induced mining stresses, stress changes, joint orientation, blasting and weathering (Laubscher, 1990).

A set of support recommendations is associated with the resulting MRMR value. In using Laubscher's MRMR system it should be remembered that many of the case histories upon which it is based are derived from caving operations. Originally, block caving in asbestos mines in Africa formed the groundwork for the modifications but, subsequently, other case histories from around the world have been added to the database (Hoek, 2006).

Laubscher (1990) noted that a classification system must be straightforward, and have a strong practical bias, so that it can form part of the normal geological rock-mechanics investigations to be used for mine design and communication. The approach he adopted involved the assignment to the rock mass of an *in situ* rating based on measurable geological parameters (Laubscher, 1990). The geological parameters that must be assessed include the intact rock strength (IRS), joint/fracture spacing and joint condition/water (Laubscher, 1990).

- Intact Rock Strength (IRS)

  IRS is the uniaxial compressive strength of the rock between fractures and joints. Taking into account the variability between a weaker and stronger rock, an average value is allocated based on the weaker rock, as it is assumed that the weaker rock would have a stronger influence on the average value when compared to the stronger rock (Laubscher, 1990), see Figure 2-4.

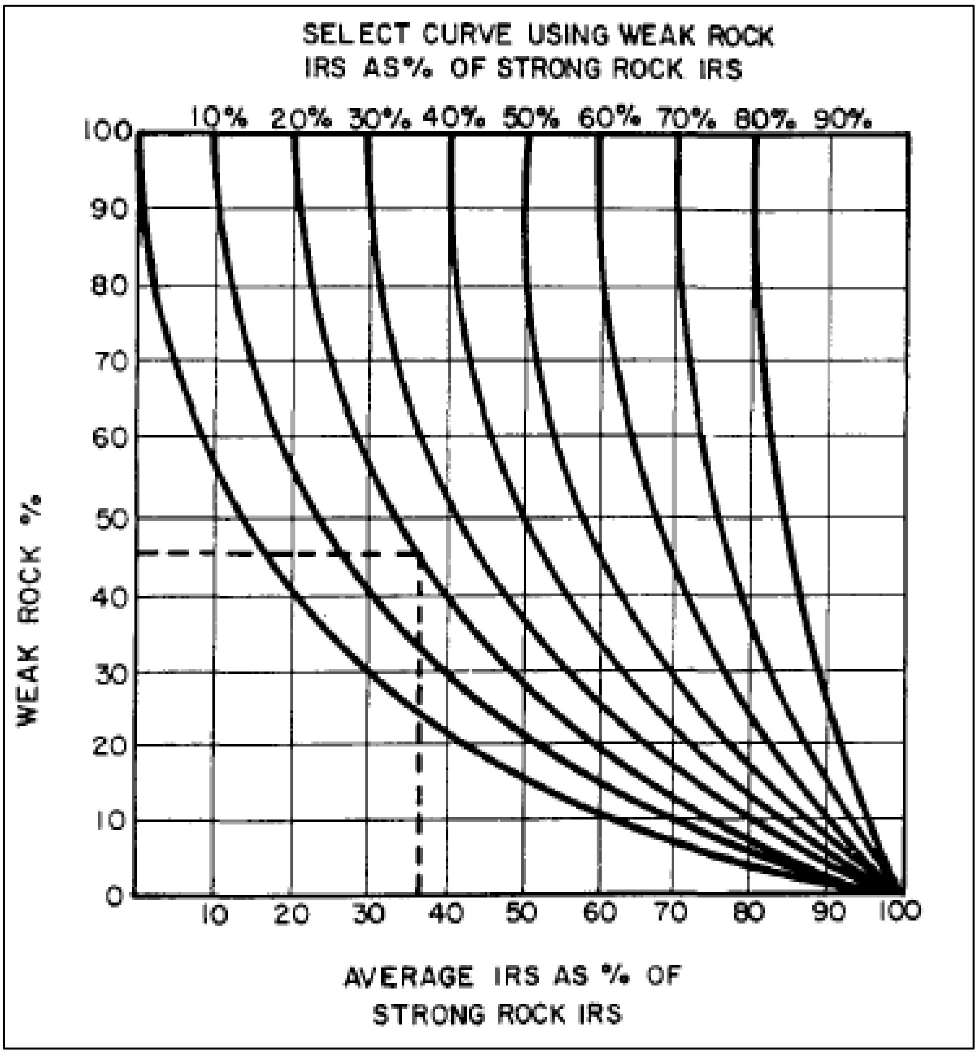


Figure 2-4: Determination of average IRS where the rock mass contains weak and strong zones (Laubscher, 1990).

- Spacing of Fractures and Joints (RQD + JS or FF)

  The measurement of all the discontinuities and partings is defined as spacing, which does not include cemented joints (Laubscher, 1990), although cemented joints were taken into account in 2000 in IRMR (Jakubec & Laubscher, 2000). Furthermore, cemented features can affect the IRS and must therefore be taken into account when determining the value.

Laubscher (1990) states that a joint is an obvious break in the rock which is continuous if its length is greater than the width of the excavation. Fractures and partings do not necessarily have continuity. A maximum value of three joint sets is used on the basis that they will define the rock block appropriately. Any other joints will merely define the shape (Laubscher, 1990). Two techniques were developed to assess this parameter:

- use the rock quality designation (RQD) and the joint spacing separately, the maximum ratings being 25 and 15 respectively.
- measure all the discontinuities and record these as fracture frequency per metre (FF/m) with a maximum rating of 40, *see Table 2-10*.

- Joint Condition and Water

  The joint condition parameter is an assessment of the frictional properties of the joints and is based on expression, surface properties, alteration zones, filling and water (Laubscher, 1990).

Table 2-10: MRMR Parameters and Adjustments (Laubscher, 1990). Note that Fracture Frequency per Metre (FF/m) can be substituted for RQD and Joint Spacing.

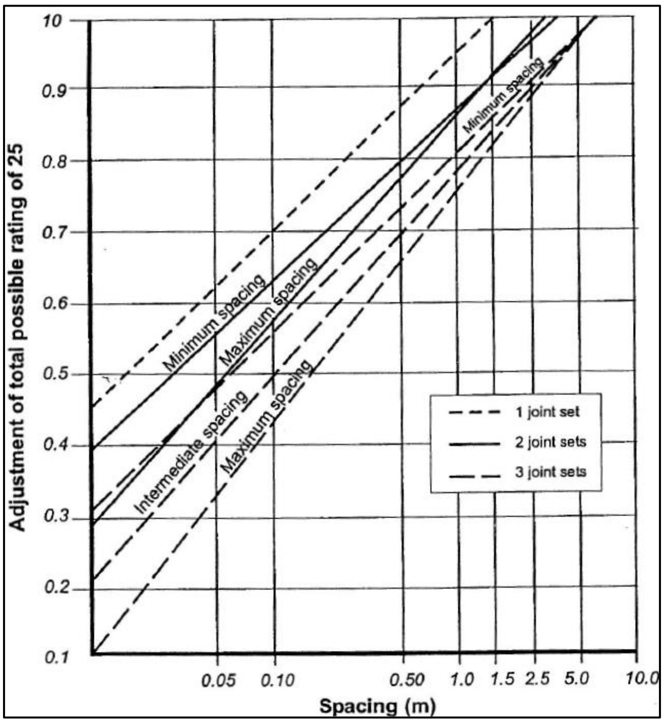
<table>
<tr><td colspan="2">Parameter</td><td colspan="11">Range of Values</td></tr>
<tr><td>1</td><td>RQD</td><td colspan="2">100-97</td><td>96-84</td><td>83-71</td><td>70-56</td><td>55-44</td><td>43-31</td><td>30-17</td><td>16-4</td><td colspan="2">3-0</td></tr>
<tr><td></td><td>Rating (=RQD x 15/100)</td><td colspan="2">15</td><td>14</td><td>12</td><td>10</td><td>8</td><td>6</td><td>4</td><td>2</td><td colspan="2">0</td></tr>
<tr><td>2</td><td>UCS (Mpa)</td><td>185</td><td>184-165</td><td>164-145</td><td>144-125</td><td>124-105</td><td>104-85</td><td>84-65</td><td>64-45</td><td>44-25</td><td>24-5</td><td>4-0</td></tr>
<tr><td></td><td>Rating</td><td>20</td><td>18</td><td>16</td><td>14</td><td>12</td><td>10</td><td>8</td><td>6</td><td>4</td><td>2</td><td>0</td></tr>
<tr><td>3</td><td>Joint Spacing</td><td colspan="11">MRMR joint spacing ratings chart. (Laubscher, 1990)

</td></tr>
<tr><td></td><td>Rating</td><td colspan="11">25 x adjustment from the chart.</td></tr>
</table>

Table 2-10(continued): MRMR Parameters and Adjustments (Laubscher, 1990). Note that Fracture Frequency per Metre (FF/m) can be substituted for RQD and Joint Spacing.

4	**Parameter**	**Description**		**Dry Condition**	**Wet Conditions**		
					Moist	**Moderate Pressure 25-125 l/min**	**Severe Pressure >125 l/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)	Coarse Sheared	95	90	70	50
			Medium Sheared	90	85	65	45
			Fine Sheared	85	80	60	40
		Soft sheared material (eg.tacl)	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	
	Rating	**40 x A x B x C x D**					

The RMR is adjusted/modified to take into account the effects of mining:

- for weathering, as certain types of rocks weather and this should be taken into account when deciding on an opening and support design. Weathering is time-dependent and influences the timing of support installation (Laubscher, 1990).
- It is further adjusted for joint orientation as the orientation of joints can have a negative effect on excavation stability. The size, shape and orientation of an excavation can affect the behaviour of the rock mass depending on how it is orientated to that of the jointing (Laubscher, 1990).
- A third adjustment, for mining-induced stresses is made, as these also affect excavation stability. Good confinement enhances stability and the maximum positive adjustment is 120%. Poor confinement, associated with numerous, closely spaced joint sets, does not promote stability, and the maximum negative adjustment is 60% (Laubscher, 1990).
- The last adjustment is for excavation technique or blasting effects, blasting can create new fractures and loosen the rock mass, causing movement on joints (Laubscher, 1990).

To be noted is that further revision of the MRMR system was done. The revised system introduced new factors to the system prior to 1999. The changes that were amended/added to the MRMR system are the introduction of rock block strength; introduction of 'cemented' joint adjustments; changes in the joint condition rating and expressing the water impact as an MRMR adjustment (Jakubec & Laubscher, 2000). MRMR has also been used for the design of slopes in open-pit mines by (Haines, et al., 1991).

2.3 Key Debates and Controversies

Certain points were raised by Jakubec & Laubscher (2000) that are important to note. These observations are not only applicable to the MRMR system, and it can be argued that they are valid for other rating systems as well. Common mistakes in classifying rock masses highlighted by Jakubec and Laubscher (2000) are of the following:

- Using ratings as an average across certain geological domains.
- Confusing mining-induced and naturally occurring defects eg. Joints and fractures.
- Ignoring variability of values of individual parameters.
- Averaging joint conditions for individual discontinuity sets.
- Wrongly adjusting for alteration and weathering.

Important points to note are: correctness of collected data; strength anisotropy should not be ignored as it can lead to an under or overestimation of the rock mass competency (Jakubec & Laubscher, 2000).

2.4 Summary and Conclusion

In summary rock mass classification is a useful tool in defining the rock mass. For this report, a rock mass classification system suitable for a narrow reef tabular environment had to be chosen as a basis to build a rating system.

The system considered to be best suited is the modified/mining rock mass rating system by Laubscher (1990), as it was designed for mining scenarios from the start. Although it was designed on data from cave mining it can be modified to suit narrow tabular reef mining at great depth.

3 DATA COLLECTION

3.1 Kusasalethu Geotechnical Setting

Kusasalethu Gold Mine is situated approximately 90km to the west of Johannesburg, near the border of Gauteng and the North-West, in the West Witwatersrand Basin. It extracts the Ventersdorp Contact Reef (VCR) (Harmony Gold, 2021) which is overlain by the Alberton Porphyry Formation (Roberts & Schweitzer, 1999). The current mining extracts the orebody at depths of 2700m – 3388m below the collar elevation and employs the sequential grid mining method (Harmony Gold, 2021). The sequential grid method employs a series of raise lines spaced 200m apart, separated by 30m wide dip stabilizing pillars. The access/ventilation tunnels are placed ‘deep’ in the quartzite footwall, >80m below the reef plane and are excavated ahead of the mining operations. Long cross-cuts every 200m link the main haulages to the reef plane where raises are developed (Jager & Ryder, 1999), see *Figure 3-1, 3-2*. The mining sequence followed helps to manage the stress levels and limit the incidence of seismicity.

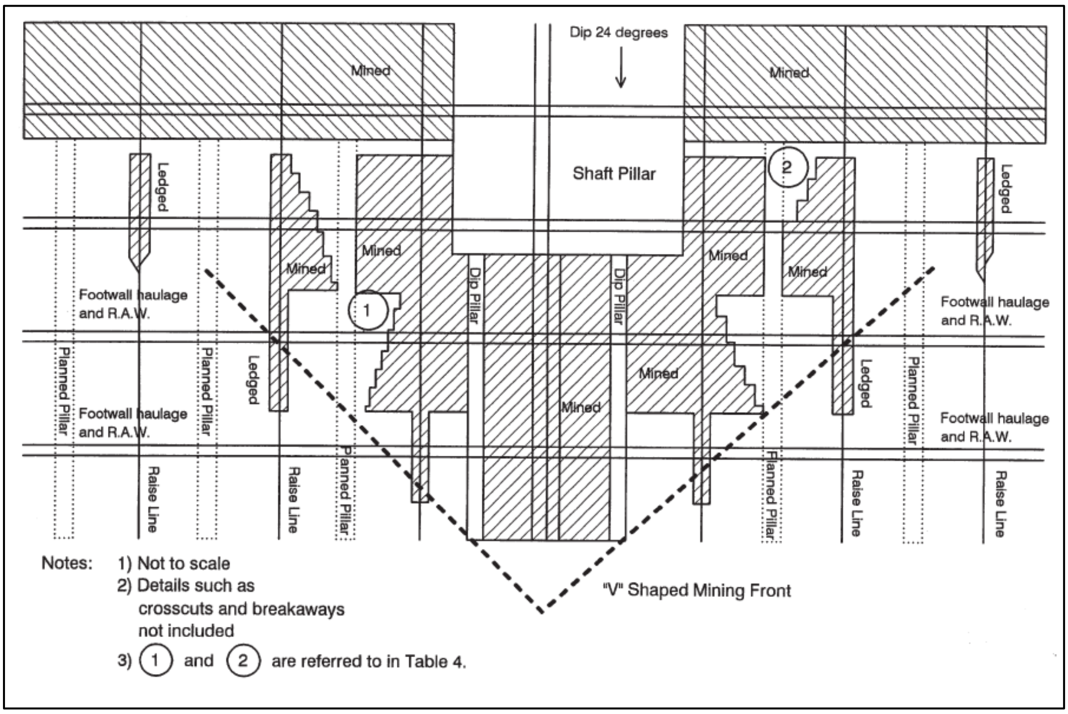


Figure 3-1: Overall strategy for sequential grid mining (Handley, et al., 2000).

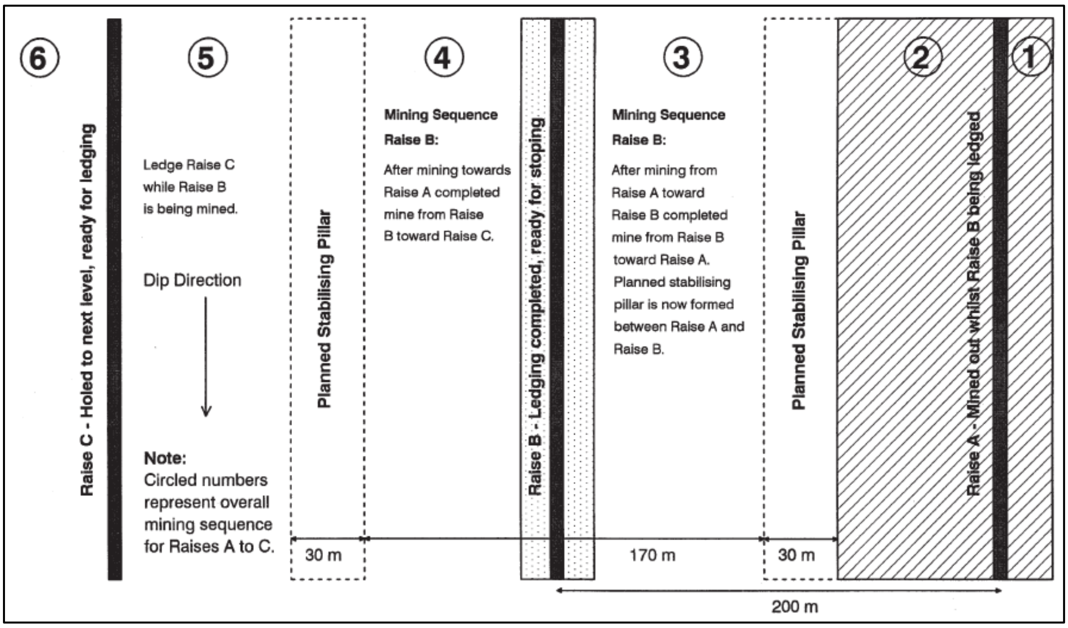


Figure 3-2: Stoping sequence for sequential grid mining (not to scale) (Handley, et al., 2000).

Kusasalethu Mine is situated on the far southern section of the West Rand goldfields. It forms part of the central portion of the greater Witwatersrand basin and mining is focused on the Ventersdorp Contact Reef (VCR), *see Figures 3-3, 3-4*. This is a conglomerate reef band with a strike of north 65^0 east and a dip of 23° to the south. The reef consists of various terraces separated by slopes, all of which may be structurally deformed by duplicated reef zones. The grade is highly variable with un-pay zones typically occupying sand-filled channels. The reef is characterised by a relatively large amount of faulting with throws of less than 10 metres (Harmony Gold Mining Company Limited, 2019).

The hanging wall is Ventersdorp Lava, which is strong with a UCS of 300MPa. Conditions vary considerably across the above-mentioned reef type. Varying conditions are caused by pilloids; inter-pilloid breccias and joints associated with slopes and duplicated reef zones. Other contributing factors are a relatively large amount of flat faulting, which extends into the hanging wall due to the brittle nature of the hangingwall lava. The footwall is competent quartzite, (UCS 180 – 250MPa) which extends to a depth of approximately 430m below reef on the eastern boundary and about 550m below reef on the western boundary, enabling haulages and most other primary related development to be sited deep in the footwall in strong competent host rock (Harmony Gold Mining Company Limited, 2019). When a major geological feature is present, the pillars may be shifted to include the feature which would then act as part of the regional stability design and will form part of the pillar system. Low-grade areas can also be left unmined, which further improves the stabilizing pillar system. Backfilling is practised to improve the overall stability, reduce closure rates and improve regional as well as local support when incorporated with additional support units such as elongates or Rapid Yielding Hydraulic Props (RYHP) (Jager & Ryder, 1999), (Handley, et al., 2000).

Figure 3-3: Generalised stratigraphic column for the Carletonville Goldfields (Handley, et al., 2000).

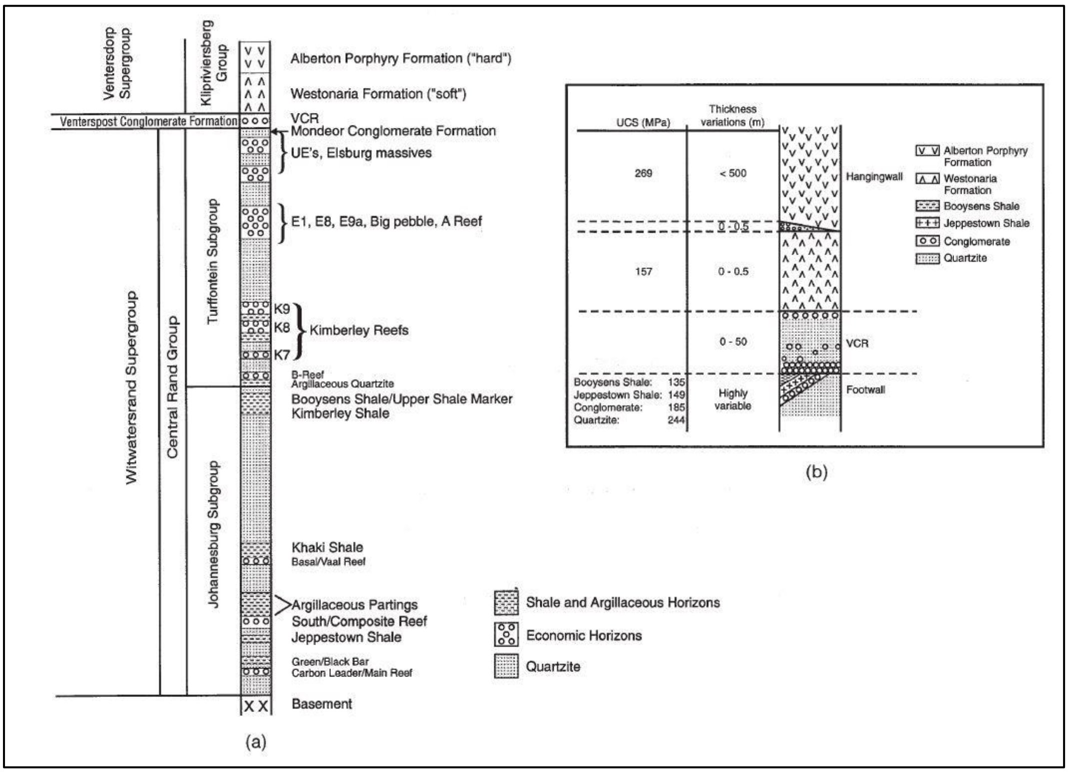


Figure 3-4: Stratigraphic position of the Ventersdorp Contact Reef (a). An idealised stratigraphic section through the Ventersdorp Contact Reef and under- and overlying rocks, together with the thickness and uniaxial compressive strength (UCS) variations are provided in (b) (Roberts & Schweitzer, 1999).

3.2 Data Collection

To understand and develop a system that is simple and user friendly, a series of data needed to be collected. Data has been collected using two methods including underground mapping/observations, using a simplified version of scan line mapping, and analysis of the fall of ground database from 2020 – 2021. Furthermore, research done by Gumede (2006) was considered as it mapped prominent joint sets in mines neighbouring Kusasalethu.

Underground mapping was based on the scan line system, and breaks the mining panel into three distinct areas: the top, middle and bottom 5.0m of the face, with observations being commonly made within 2.0m from the mining face. This method initially tended to have a bias towards at least two fracture sets, however, due to the proximity to the topmost or bottommost portion of the panel, and could skew the result. As a result, the portion of the hanging wall that was sampled was changed to every 5.0m starting at least 3.0m – 5.0m from the top of the panel. Furthermore, mapping was deemed difficult due to various constraints such as low stoping widths, permanent in-stope netting obstructing the hanging wall, low levels of illumination, and long travel time to working areas. Figure 3-5 indicates a standard stoping panel in an underhand mining configuration and Figure 3-6 indicates the planned mapping positions.

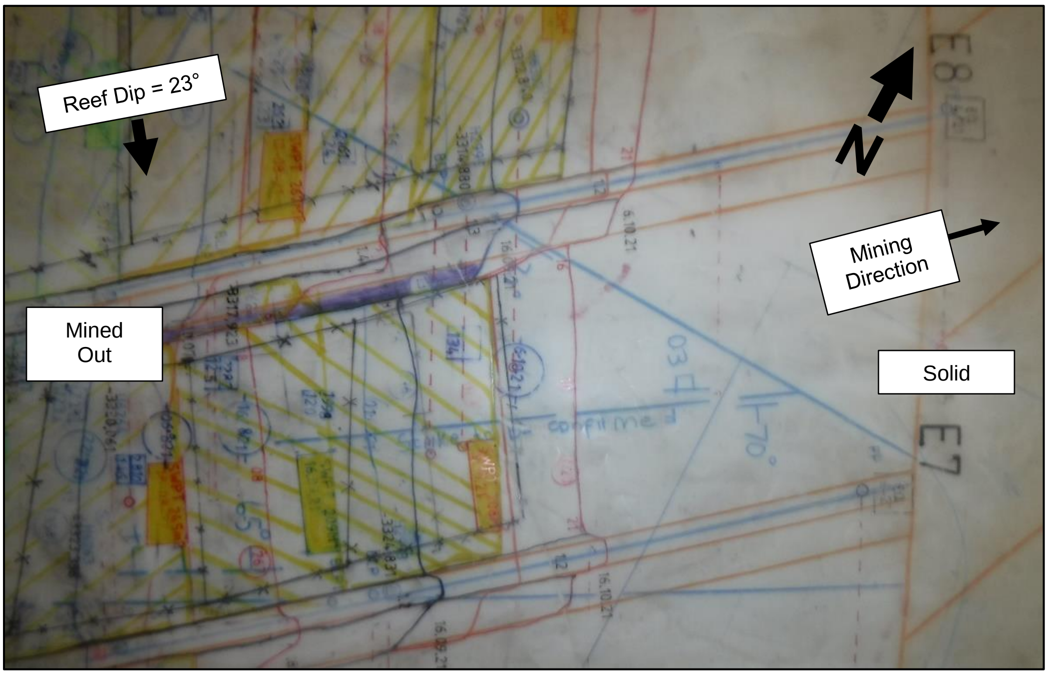


Figure 3-5: Standard stoping panel in a top leading (underhand) mining sequence, utilising an advanced strike gully and a siding.

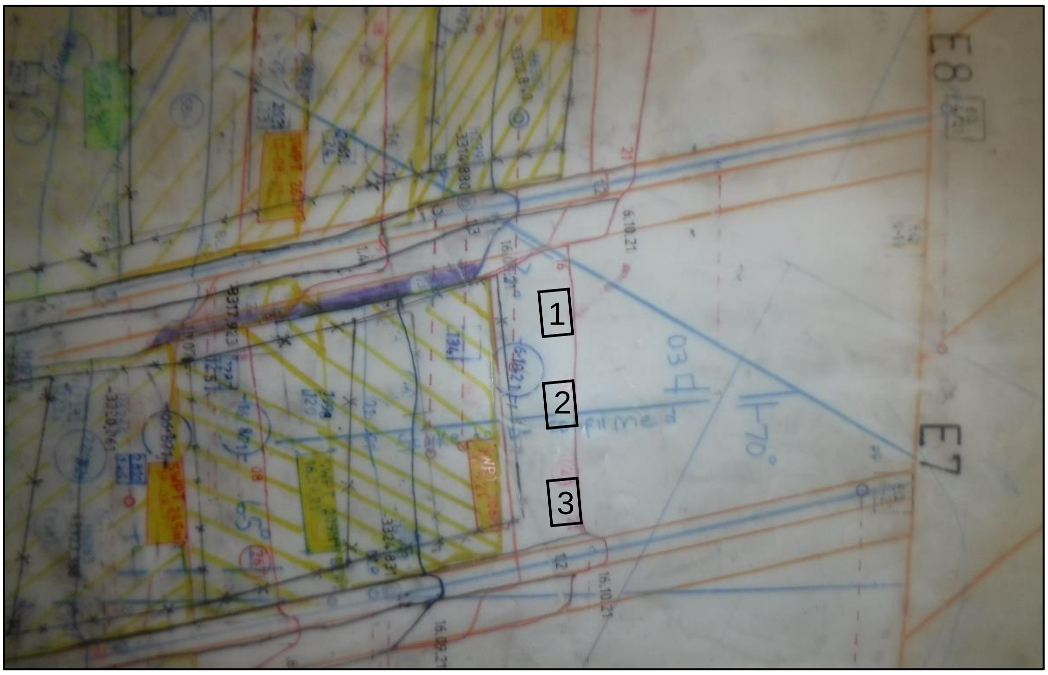


Figure 3-6: Required mapping positions within the panel.

From the data collection, the following observations were made and recorded with regard to prominent joint set orientation and dip, infiling, roughness and spacing. The joint length was most difficult to determine due to the limited space in a panel, but on average a joint trace length of approximately 2.0m – 2.5m was observed - this could be longer but was not possible to determine.

The strike of an observed joint was measured in relation to the orientation of the panel face and captured upon returning to surface, see Table 3-1 for the orientations of prominent joint sets identified at the mine. The last surveyed and measured face position was determined using the survey sheet (scale = 1:200), with north = 0°, and the strike of the joint determined with an allowance of ±5° and then captured in the database. The method described gives a good indication of the joint's strike direction. The influence of jointing on falls of ground cannot be ignored, hence the purpose of capturing this data. The common practice at the mine with regard to face orientation is to lay out the panel's strike gullies 5° above the reef strike direction to accommodate the egress of water from the panels and to assist in cleaning operations. The general face orientation for panels mining in the easterly direction is 150° from north, or 5° south of south-east, and for the western panels is 335° from north, or 25° west of north. The deviation from the aforementioned orientation is normally 5° and could vary by as much as 7°.

The database consists of at least 23 data collection points in various areas of the mine ranging in depth from 2800m – 3300m below surface. The data has been collected from different sources such as fall of ground investigations, ledging and regular breast mining operations.

Table 3-1 Two prominent joint sets were identified and their mean values with regards to spacing, orientation and dip.

	Prominent Joint Set 1	**Prominent Joint Set 2**
Joint Surface	Rough Undulating/Planar	Rough Planar
Joint infilling	Tight No Infill 1mm calcite	No infill/Calcite infill 0mm – 2mm
Joint Dip and Direction	80° W or E	75° E
Joint Orientation (0° = North)	317°	302°
Joint Spacing	0.2m – 1.0m	0.3m – 1.5m

Fracture data was collected due to its major influence on the rock mass surrounding the stopes at depth, and the inherent influence of a fracture as a boundary of a fall of ground. A similar approach was used in capturing the fracture data like that used for the joint data. Fracture orientation, fracture dip and direction, and fracture spacing were recorded. It must be reiterated that fracture orientation generally conforms to the shape of the excavation, especially in terms of extension-type fractures, which can become the dominant set of discontinuities in a stope (Jager & Ryder, 1999). Extension fractures, which are the most common type of fractures experienced in deep-level mining, form in induced tension, but in a wholly compressive stress field. They develop on a plane normal to the minor principal stress and are sensitive to changes in stress orientation. These types of fractures are planar and clean, and often start or stop against bedding planes and joints (Jager & Ryder, 1999). Due to the ubiquitous nature of the fractures around the stope, measurements were taken in three areas of the panel. The prominence of the fractures was captured, the general spacing between the fractures of each set, and the dip of the fractures. The fracture orientation as stated before conforms with the outline of the excavation and therefore does not have a uniform overall strike direction. The common fracture sets observed are highlighted in *Table 3-2 below*.

Table 3-2: Common Fracture Sets Observed Underground.

	Fracture Set 1	**Fracture Set 2**	**Fracture Set 3**
Prominence	Very (always observed)	Very (always observed)	Average (mostly observed along static abutments)
Dip and dip direction	60° - 85° (Mean = 68°) commonly dips away from the direction of mining)	55° - 85° (Mean = 80°) Dip is dependent on the direction of mining.	Tend to curve with a flat dip of 20°-40°.
Fracture Orientation.	Face parallel, greatly influenced by the overall panel face shape.	Normally perpendicular to fracture set 1	Curved as it is found at the intersection of the abutment and advancing face.
Fracture Spacing	0.05m – 0.4m	0.02m – 0.3m	0.02m – 0.4m

The main types of fractures focused on are extension fractures. Although shear fractures are present, they are not as common as extension fracturing and have not been recorded as part of this study. Furthermore, due to the compressive conditions at depth, tensile fracturing is also not as common compared with a low-stress mining environment (Jager & Ryder, 1999).

Observations from underground visits have indicated that stress fracturing and jointing have a major influence on the stability of the rock mass as can be seen in *Figure 3-7 below*:

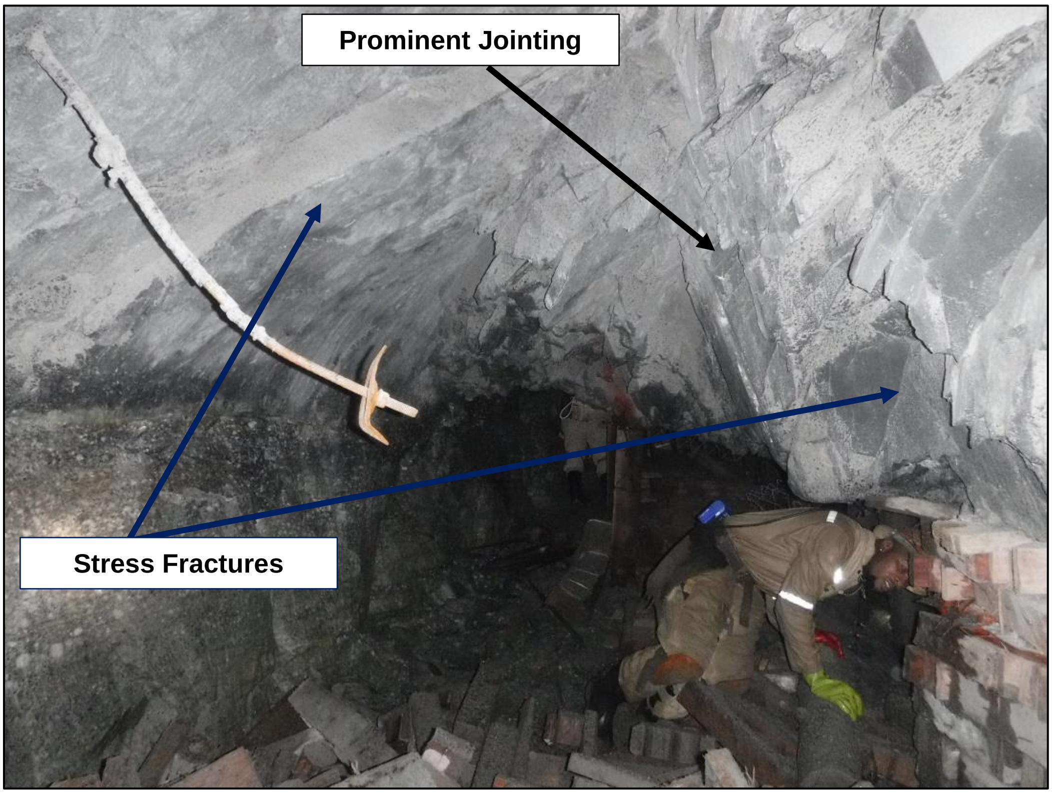


Figure 3-7: Influence of stress fractures and prominent jointing on a fall of ground in one of the stoping panels at the mine.

The average fall-out thickness measured at Kusasalethu Mine is 1.2m and this has been recorded in the mine's fall of ground database upon which the support designs were based (Harmony Gold Mining Company Limited, 2019).

As can be seen in *Figure 3-7,* the breaks in the rock mass affect the overall stability and are taken into account and captured using the fracture frequency per metre system. This simplified system requires the measurement of all the discontinuities that are intersected along the scan line (Laubscher, 1990), which is done at each point in the panel where the data is collected. The user should know whether it is a one, two or three-joint system being sampled. The sampling for fracture frequency per metre (FF/m) is done along the hanging wall of the panel at the specified sampling points (Laubscher, 1990). Measurements of fracture frequency are made along the hanging wall of the panel and, if necessary, the north or south side wall of the panel, depending on the orientation of the features (Laubscher, 1990).

4 JUSTIFICATION OF THE MODIFICATION TO THE MRMR PARAMETERS AND THE ASSOCIATED ADJUSTMENTS

The mining method which is used at Kusasalethu Mine required adjustments to the parameters set out by Laubcsher (1990) as this is the rock mass classification method adopted for use. The influence of various parameters had to be included, and those which are not relevant to be excluded, to create the most effective method for rock mass classification for the mine and mining method. Some of the parameter ratings needed to be adjusted to accommodate the specific influence of the parameter on the quality of the rock mass.

4.1 Using The Modified Rock Mass Rating by Laubscher (1990)

The modified rock mass rating is described by Laubscher (1977), (1984) and Laubscher and Taylor (1976). As explained earlier in the report the MRMR system takes the basic RMR parameters, as defined by Bieniawski, and adjusts them for stresses, the effects of blasting and weathering (Hoek, 2006). The MRMR system uses the following parameters:

- Intact Rock Strength,
- Spacing of Fractures and Joints (FF/m used in this case),
- Joint Condition and Water.

The above-mentioned parameters give the rock mass rating for the system. Thereafter, adjustments for the following are applied to the RMR to take into account mining:

- Weathering,
- Joint Orientation,
- Mining Induced Stresses,
- The Effects of Blasting.

The justification for using this method of rock mass classification is its simplicity, and that it considers the rock mass rating and those factors influencing the rock mass by mining within it. The MRMR system as published is not entirely compatible with the mining method used, type of orebody and mining depth, and modifications had to be made to the system, keeping it simple and easy to use. After analysing empirically what affects the rock mass conditions the most at Kusasalethu Mine the following parameters were added and removed:

- Intact rock strength remained in place even though the general UCS of the rock in which the Ventersdorp Contact Reef is situated is strong. If low-strength rock is intersected, how it reacts to the high-stress levels may be somewhat different to that of a high-strength rock in terms of fracturing and competence.

- The joint condition is captured in the database, but is not included in the RMR, as the observed joint and fracture conditions are similar throughout the mine, (*see chapter 3.2 and Appendix D).* Water was not included as it is very seldom encountered during mining. If water is encountered in the future, its effect will be assessed individually and recommendations given for the specific area. There are currently no such water conditions in the mine.
- Both joint and fracture data had to be collected, as the combination of these two parameters, or if assessed individually, have a large impact on the rock mass conditions. Stress fracturing is extremely prevalent at the depths at which mining is taking place. The number of joint and fracture sets found in a panel directly influences the rock mass conditions, i.e. the more weakness planes, the greater likelihood of failure occurring in the hanging wall.
- Referring to the point above, fracture frequency per metre is used in place of RQD and joint spacing, as it tends to be more sensitive (Laubscher, 1990), whereas RQD is not, as it discounts weakness plane spacing of less than 100mm (Palmstrom, 2005), (Hoek, 2006), the explanation of which is well covered in the article by Pells, et al. (2017).
- Prominent jointing has remained as part of the RMR adopted, because it is commonly found in all the panels and is accounted for in the main rock mass classification systems in use today, as indicated in literature by Rehman, et al., (2018) and Hoek (2006). As

stress fracturing is very prominent at great mining depths due to the high levels of stress encountered, and directly affects the stability of the hanging wall, and could also become the dominant set of discontinuities (Jager & Ryder, 1999), a parameter for stress-induced fracturing was added. Jointing and stress fracturing are the main contributors to instability at Kusasalethu Mine.

In terms of the mining adjustments, the following were added or removed:

- Mining induced stress was assessed slightly differently from the published method and simplified. What has been included is the average face stress in place of mining-induced stresses. Laubscher (1990) indicated that the adjustment for mining-induced stresses can range from 60% to 120% but due to the high levels of stress at great depths, the consequence is that a highly fractured hanging wall is substantially more stable compared to shallower depths as the horizontal stresses tend to clamp the discontinuities (Jager & Ryder, 1999) resulting in a more stable hanging wall. Therefore, due to the high-stress regime at great depth, the common adjustment would be 120%, indicating good confinement as described by Laubscher (1990). By using the adjustments described by Laubscher (1990) the rating would almost always increase positively, and taking cognisance of the aforementioned, it was decided best to use the average face stress, as it influences the occurrence of stress fracturing in the hanging wall of the panel, which is easily measurable from numerical modelling results and underground observations.

- The process of breaking the rock using conventional blasting methods also influences the rock mass conditions in a stoping panel, and therefore this parameter could not be ignored and has been retained as an adjustment. Pre-conditioning blasting can also affect the condition of the rock mass either negatively or positively, and therefore had to be included, as the mine practises face perpendicular pre-conditioning. When done correctly, pre-conditioning can improve the overall hanging wall conditions and lead to improved face advances (Toper, et al., 2003). When done incorrectly, the pre-conditioning blast has a negative effect on the rock mass conditions, and can result in poor panel face shapes and destabilisation of the hanging wall.
- Discontinuity orientation was retained as an adjustment, as adversely orientated weakness planes affect the stability of the rock mass. For example, if a discontinuity has a dip of 30° then the very same stress that enhances stability with a more steeply dipping feature, can result in instability, and cognisance must be taken of this factor.

4.2 Parameters and Adjustment Values

The UCS parameter is self-explanatory and the rating values were adjusted from those given by Laubscher (1990) to account for the higher rock strengths experienced at the mine. The overall contribution to the rock mass rating out of 100 was reduced from 20 to 10 as the Ventersorp Contact Reef at Kusasalethu Mine is overlain by the Alberton Porphyry Formation and has a quartzite conglomerate footwall below the reef, which remains relatively unaltered (Roberts & Schweitzer, 1999). The lava extrusion in this area ranges from 160m – 400m, and is incorporated into a geotechnical area described by Roberts & Schweitzer (1999) and remains relatively unaltered unless it is close to a large fault or igneous intrusion.

As previously stated, prominent jointing and fracturing were accounted for equally under the rock mass rating. Laubscher (1990) only accounted for the effect of fracturing under the FF/m portion of the MRMR system and emphasised the effect of jointing and joint condition. The parameters that have been included in the current system deal with the strike orientation of the features as well as the dip. These two factors, along with the fracture spacing, which is added later, encompass the joint and fracture parameters of the rating system. The jointing and fracture properties in terms of joint surface condition and infilling are captured separately, but not included in the rating as it was found that the prominent joint sets have similar properties throughout the mine. Where anomalous properties are identified they are addressed with the relevant remedial action in terms of support and blasting techniques. Two prominent joint sets have been identified on the mine and when found in a stoping panel, influence falls of ground and ground conditions.

5 ANALYSIS OF THE DATA

Analysis of the data was conducted using the rock mass rating system developed for the VCR stope at Kusasalethu Mine, which is a modified version of the MRMR system developed by Laubscher (1990), subsequently updated by Jakubec & Laubscher (2000). The system is simple and easy to use for rock engineering and geology personnel.

5.1 Mining Rock Mass Rating for the Ventersdorp Contact Reef

The $MRMR_{vcr}$ that was designed for use in stoping environments at Kusasalethu Mine is based on the MRMR system as well as the RMR system as it looks at the main parameters influencing the ground conditions at Kusasalethu Mine. Keeping in line with the principles stated by Laubscher (1990) the classification system was kept as straightforward and simple as possible. Firstly, the 'unconfined' uniaxial compressive strength of the rock was taken into consideration and is the strength of the rock between joints and fractures (Laubscher, 1990). The values of the uniaxial compressive strength were modified to cater for the higher host rock strengths found in the VCR stratigraphy. As Kusasalethu Mine falls within a geotechnical area described by Roberts and Schweitzer (1999), *see Figure 5-1,* the values for the UCS are ascertained from there as well as a previous study conducted by the Council for Scientific and Industrial Research (CSIR) (Güler, 1996).

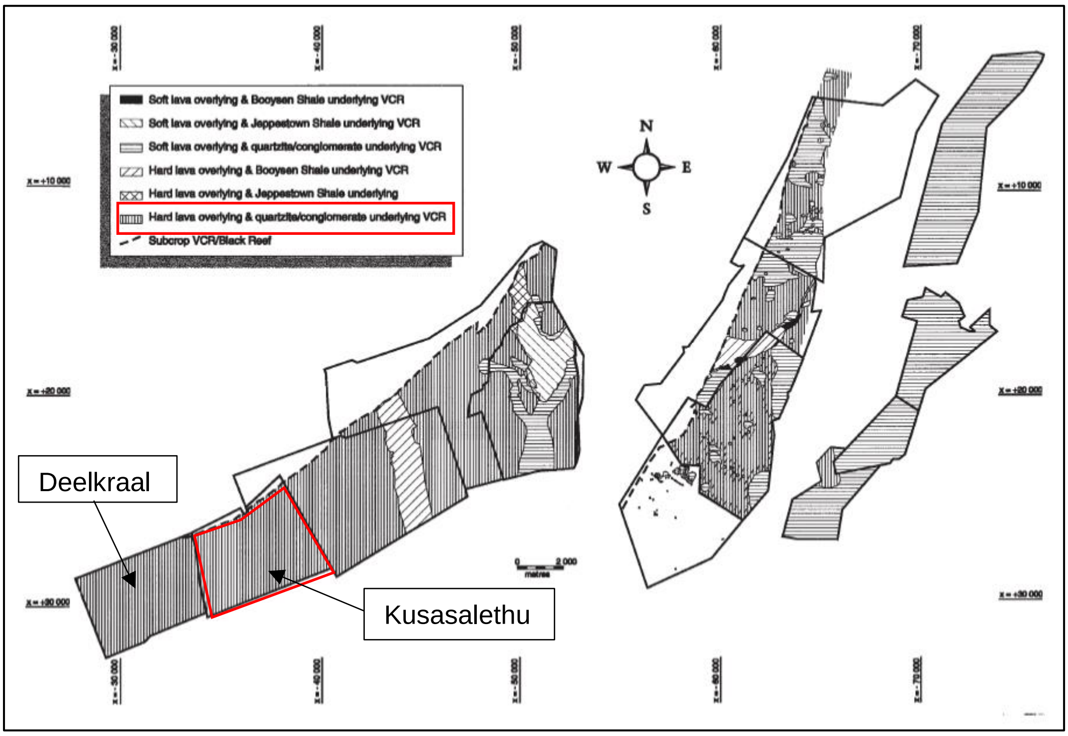


Figure 5-1 Ventersdorp Contact Reef geotechnical area map (Roberts & Schweitzer, 1999).

5.2 Mining Rock Mass Rating for VCR process explanation

Rock strength values reported by Güler (1996) for the ‘hard-lava’ are around 250MPa, and 210 - 260 MPa is indicated by Ryder and Jager (2002). The underlying quartzite-conglomerate can have UCS values of around 250MPa as well (Güler, 1996), although the main focus for the project is the classification of the hanging wall. The values and ratings for the UCS portion of the system are indicated in Table 5-1.

Table 5-1: UCS Values and Associated Ratings.

Value (MPa)	<300	299-270	269-240	239-200	199-160	159-110	109-80	>80
Rating out of 10	10	8.75	7.5	6	4.75	3	1.5	0.5

Once the user has completed the rating for the UCS, *see Table 5-1*, the influence of prominent jointing is considered for the next step. From underground observations and experience, two influential joint sets were identified at the shaft, with random joints being present as well. From knowledge of what the two main contributors to hanging wall instability are at the mine, the rating for prominent jointing was defined, which followed a similar process to that of Laubscher (1990), where a cumulative percentage adjustment of a possible total value of 25 is given, *see Table 5-2*. The data input into this portion of the system comes from underground observations that have been captured at various locations at the mine. The value adjustments were rated on the influence of the prominent joint set on the ground conditions.

Table 5-2: Prominent Jointing and Accumulative Percentage Adjustments.

Accumulative Percentage adjustment of a possible rating of 25				
Jointing Present	**Y**	**N**		
Rating	**25** x (joint dip 1,2,3) x (joint strike 1,2,3) x dip direction	No adjustement to the total rating		
Joint Set 1 Dip (JS1D)	**0°-30°**	**30°-60°**	**60°-90°**	**N/A**
Percentage Rating Downgrade	87% (0.87)	95% (0.95)	100% (1)	100% (1.0)
Joint Set 2 Dip (JS2D)	**0°-30°**	**30°-60°**	**60°-90°**	**N/A**
Percentage Rating Downgrade	0.8% (0.8)	90% (0.9)	98% (0.98)	100% (1.0)
Joint Set 3 Dip (JS3D)	**0°-30°**	**30°-60°**	**60°-90°**	**N/A**
Percentage Rating Downgrade	70% (0.7)	87% (0.87)	95% (0.95)	100% (1.0)
Joint Set 1 Strike (JS1S)	**0°-30°**	**30°-60°**	**60°-90°**	**N/A**
Percentage Rating Downgrade	99% (0.99)	95% (0.95)	98% (0.98)	100% (1.0)
Joint Set 2 Strike (JS2S)	**0°-30°**	**30°-60°**	**60°-90°**	**N/A**
Percentage Rating Downgrade	98% (0.98)	90% (0.9)	97% (0.97)	100% (1.0)
Joint Set 3 Strike (JS3S)	**0°-30°**	**30°-60°**	**60°-90°**	**N/A**
Percentage Rating Downgrade	95% (0.95)	87% (0.87)	93% (0.93)	100% (1.0)
Dip Direction	**Favourable**	**Unfavourable**		
Percentage Rating Downgrade	100% (1.0)	90% (0.9)		

Once the joint rating is complete for the specific panel, a fracture rating is allocated. In deep mines, stress fracturing is a regular occurrence around stope faces and along abutments, and is due to the stress exceeding the strength of the rock mass. Stress fracturing is such that the fractures can become the dominant set of discontinuities in the rock mass (Jager & Ryder, 1999). The most common fractures observed were extension fractures with generally close spacing of ~15cm. To allocate a fracture rating, a total value of 25 is allocated with cumulative adjustments to obtain a final rating, similar to the process for the prominent jointing, *see Table 5-3*. The down-rating values become more significant when more fracture sets are present in the rock mass, as each breaks the rock mass further, into smaller blocks that can dislodge.

Table 5-3: Stress Fracturing and Cumulative Percentage Adjustments.

Accumulative Percentage adjustment of a possible rating of 25				
Fracturing Present	**Y**	**N**		
Rating	25 x (Fracture Dip 1,2,3) x Fracture Strike (1,2,3) x dip direction.	No adjustment to the rating.		
Fracture Set 1 Dip (FS1D)	**0°-30°**	**30°-60°**	**60°-90°**	**N/A**
Percentage Rating Downgrade	87% (0.87)	95% (0.95)	100% (1.0)	100% (1.0)
Fracture Set 2 Dip (FS2D)	**0°-30°**	**30°-60°**	**60°-90°**	**N/A**
Percentage Rating Downgrade	80% (0.8)	90% (0.9)	98% (0.98)	100% (1.0)
Fracture Set 3 Dip (FS3D)	**0°-30°**	**30°-60°**	**60°-90°**	**N/A**
Percentage Rating Downgrade	70% (0.7)	87% (0.87)	95% (0.95)	100% (1.0)
Fracture Set 1 Strike (FS1S)	**0°-30°**	**30°-60°**	**60°-90°**	**N/A**
Percentage Rating Downgrade	99% (0.99)	95% (0.95)	98% (0.98)	100% (1.0)
Fracture Set 2 Strike (FS2S)	**0°-30°**	**30°-60°**	**60°-90°**	**N/A**
Percentage Rating Downgrade	98% (0.98)	90% (0.9)	97% (0.97)	100% (1.0)
Fracture Set 3 Strike (FS3S)	**0°-30°**	**30°-60°**	**60°-90°**	**N/A**
Percentage Rating Downgrade	95% (0.95)	87% (0.87)	93% (0.93)	100% (1.0)
Dip Direction	**Favourable**	**Unfavourable**		
Percentage Rating Downgrade	100% (1.0)	80% (0.8)		

To incorporate the number of breaks in the rock mass fracture frequency per metre is used. This technique requires the measurement of all breaks in the hanging wall, fractures and jointing, along a determined scan line. The ratings originally set out by Laubscher (1990) have remained unchanged, *refer to Table 5-4*.

Table 5-4: Fracture Frequency per Metre Ratings after Laubscher (1990).

FRACTURE FREQUENCY per METRE FF/m (40 points allocated)			
	Rating		
Average Per Metre	**1 Set**	**2 Set**	**3 Set**
0.1	40	40	40
0.15	40	40	40
0.2	40	40	38
0.25	40	38	36
0.3	38	36	34
0.5	36	34	31
0.8	34	31	28
1	31	28	26
1.5	29	26	24
2	26	24	21
3	24	21	18
5	21	18	15
7	18	15	12
10	15	12	10
15	12	10	7
20	10	7	5
30	7	5	2
40	5	2	0

Once the RMR value of the classification system is obtained, the adjustments are applied for mining to obtain the final ‘mining’ rock mass rating, which is similar to the approach in the MRMR by Laubscher (1990). The adjustment values were based on parameters that affected the overall quality of the rock mass significantly and were kept to a total of 4. Average Face Stress (AFS) is the first ‘mining’ adjustment that is taken into account, *see Figure 5-3 and Figure 5-4*. The rationale behind this adjustment is the effect the face stress has on stress fracturing. As the levels of stress increase so does the intensity of stress fracturing and the occurrence thereof which is easily observed underground, *see Figure 5-2.*

Figure 5-2: Closely spaced stress fracturing (5cm - 15cm) observed in a panel with face stress values exceeding 350MPa.

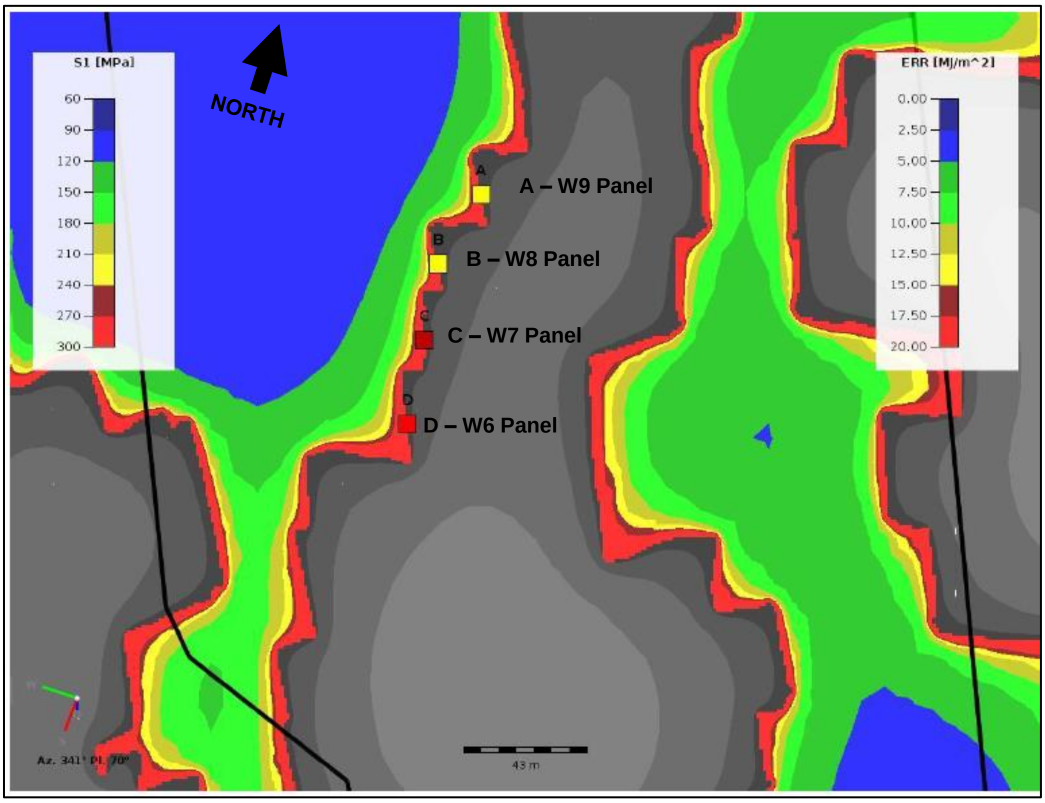


Figure 5-3: **Numerical modelling results showing the high-stress levels at 113 level 36 raise panel W6, the same panel in which *figure 5-2* was taken, *image courtesy of the Institute for Mine Seismology. (grey colouring indicates the mined-out areas)***

The fracturing observed correlates with the high incidence of stress found at this specific raise line and in turn manifests as more intense stress fracturing as well as an increased occurrence of seismicity. Due to the above-mentioned factors, it was deemed necessary to add this adjustment, *see Table 5-5 for the adjustment values*.

Sigma1 [MPa]**

	Name	Jan	Feb	Mar	Apr*	May*
A	.._W_9_VCR	272	303	356	336	324
B	.._W_8_VCR	367	343	308	310	320
C	.._W_7_VCR	280	304	347	352	333
D	.._W_6_VCR	386	391	400	411	420

* planned mining

** mean value of cells within 10 m

ERR [MJ/m^2]**

	Name	Jan	Feb	Mar	Apr*	May*
A	.._W_9_VCR	11	11	14	16	15
B	.._W_8_VCR	14	16	13	14	14
C	.._W_7_VCR	11	14	17	19	18
D	.._W_6_VCR	20	20	22	23	24

* planned mining

** mean value of cells within 10 m

Figure 5-4: Modelled stress and energy release rate (ERR) values.

Table 5-5: Average Face Stress Adjustment

Average Face Stress (values obtained from numerical modelling) Design = 250MPa (Harmony Gold Mining Company Limited, 2019)	
<Design	100% (1.0)
=Design	98% (0.98)
>Design	90% (0.9)

A slight downgrade has been added for values equal to the design as the stress fracturing is still influential at these stress levels as the magnitude of the principal stresses will determine whether stress fracturing will occur in the rock (Jager & Ryder, 1999).

A direct contributor to poor and unstable ground conditions is the quality of the production blast and the pre-conditioning blast, both of which are included as separate adjustments. Due to blasting creating new fractures as well as loosening the rock mass, possibly causing movement along joint and fracture planes, the production blast adjustments have remained in line with what is used in Laubscher's MRMR rating system (Laubscher, 1990). Bored tunnels were excluded as the method is not used at Kusasalethu Mine. It has been found that at Kusasalethu Mine when the hanging wall beam is damaged by the blast, due to poor drilling and/or charging discipline, very poor and difficult to control ground conditions are created and more often than not a fall of ground results, *see Table 5-6 for the blasting adjustments and Table 5-7 for the pre-conditioning adjustments*.

Table 5-6: Production Blasting Adjustment

Production Blasting	
Good	97% (0.97)
Poor	90% (0.9)

Table 5-7: Pre-conditioning Blasting Adjustment

Pre-conditioning Blasting	
Good	100% (1.0)
Poor	90% (0.9)

To be noted is that the mine only uses conventional blasting methods, which involve the drilling of holes using handheld drilling machines and charging of the holes with an emulsion-type explosive. The pre-conditioning standard for the mine requires the holes to be drilled in the middle of the mining face, 2.4m apart along the length of the face and drilled to a depth of 2.4m. Two-thirds of the hole is charged up with explosives and the remainder with stemming material. In the case of pre-conditioning blast holes, the timing of the pre-conditioning holes relative to the production holes is very important. Poor timing may lead to ineffective pre-conditioning and/or misfires, which in turn may result in blast damage to the rock mass and the loss of any advantages that can normally be gained from pre-conditioning. The preconditioning hole should be timed such that detonation of the hole will take place ahead of the production holes (Middindi Consulting (PTY) LTD (In cooperation with SIM Mining Consultants, D Arnold, N Ndeweni), 2015).

The last adjustment that has been used for the rock mass rating system is that for joint orientation. As the size, shape and orientation affect the behaviour of the rock mass the orientations of the joint and fractures planes need to be taken into consideration, as well as the interaction (Laubscher, 1990), which was covered earlier in the report. The attitude of the joints, and in this case stress fractures as well, and whether the base of the block is exposed, have a significant influence on the stability of the excavation, and the rating should be adjusted accordingly (Laubscher, 1990). The adjustment depends on the attitude of these breaks in the rock mass with respect to the vertical axis of the block. Laubscher (1990), considers gravity as the most influential force, and that the instability of the block depends on the number of the joints that dip away from the vertical axis. It is pertinent to note that hanging wall strata are clamped together by high horizontal stresses at great depth and can be self-supporting under static conditions. The presence of low-angle jointing and fracturing can nevertheless give rise to ground control problems (Jager & Ryder, 1999). In stating the above, gravity-related falls of ground contribute to more than 60% of the total falls of ground recorded annually at Kusasalethu Mine. The values for the joint/fracture plane adjustments have been kept the same as those of Laubscher (1990), *see Table 5-8*.

Table 5-8: Percentage Adjustments for Joint Orientation (Laubscher, 1990)

No. of joints defining the block	**No. of faces inclined away from the vertical**				
	Rating Adjustment				
	70%	**75%**	**80%**	**85%**	**90%**
3	3	-	2	-	-
4	4	3	-	2	-
5	5	4	3	2	1
6	6	5	4	3	2,1

The categories or classes under which the ratings fall are indicated in *Table 5-9*.

Table 5-9: RMR_{vcr} and $MRMR_{vcr}$ class ratings remain the same as the original (Laubscher, 1990).

Class Rating	**5**	**4**	**3**	**2**	**1**
RMR_{vcr} & $MRMR_{vcr}$ value	0-20	21-40	41-60	61-80	81-100
Basic Decription	Poor	Poor to Average	Average	Average to Good	Good

5.3 Baseline Panel for Comparison

To obtain meaningful values from the classification system a theoretical, but practical baseline value, had to be put in place. The baseline value represents a panel with fair to good conditions which will not require any changes with regard to support and/or mining techniques. The baseline panel consists of:

- The UCS value remains fairly constant throughout the mine and ranges from around 250MPa – 300MPa for the Ventersdorp Lava hanging wall (Roberts & Schweitzer, 1999).
- 1 Prominent joint set striking near-parallel to the face, indicating a strike of 300° from north (north = 0°). This is the most common joint set found at the mine.

- 2 fracture sets, the first of which is common extension fracturing, striking parallel to the face and with a dip of 60° away from the direction of mining. The extension fractures have a spacing of 10cm – 40cm. The 2nd fracture set is normally identified at the top or bottom of the panel, has a dip of around 70° over the excavation/panel, has a strike that conforms to the direction of mining, and is near perpendicular to the strike of the first set.
- Lastly, the number of fractures per metre is included as 5 – 7 and 3 fracture sets.

The final $MRMR_{vcr}$ value is calculated as 65 – 70, which indicates an average – good rock mass before the mining adjustments are taken into account (see *Appendix B* for the full worksheet).

The mining adjustments that are indicated for the baseline panel are highlighted below:

- The average face stress is equal to the design, therefore an adjustment of 98% is allocated.
- The production and pre-conditioning blasting is indicated as good and a 97% adjustment is applied for the production blast and a 100% adjustment for the pre-conditioning blast.
- Lastly, an adjustment is applied for the number of joint/fracture planes that are orientated away from the vertical. Taking into

consideration that at least 4 of the planes defining the block will have an attitude orientated away from the vertical, with the top and bottom contacts being horizontal, the values given are 4 joints/fractures defining the plane of which at least 4 are orientated away from the vertical. This gives an adjustment value of 70%. Note that one of the prominent joint sets defined in Chapter 3 has a dip ranging from 80° - 90°, and this may change the value of the number of planes orientated from the vertical to 2 or 3.

The final adjusted rating, taking into account mining, will be in the range of 45 – 55 resulting in an average rock mass. This provides a valuable indication of the effects of mining on the rock mass as well as the orientations of the weakness planes.

5.4 MRMR$_{vcr}$ Jointing Adjustment Change

The initial joint rating values were harsh, and skewed the RMR$_{vcr}$ value before the mining adjustments. Therefore, the jointing adjustments were equated to those of the fractures. The rationale behind the change is that, individually, the jointing and fracturing affect the rock mass stability as they both create breaks within the rock, 'weakness points' along which failure can take place. The interaction between these two sets of features further creates weakness within the rock mass and therefore their impact is seen as equal even though the incidence of fracturing is more common than that of jointing. The change in the RMR_{vcr} value and the $MRMR_{vcr}$ value is indicated in *Figures 5-5 and 5-6*.

5.5 $MRMR_{vcr}$ Correlation with Poor Ground Conditions and Falls of Ground

The data collection was done during underground visits to 23 separate working panels, including one raise inspection prior to the commencement of ledging. The types of visits that were conducted are normal panel audits, poor ground condition investigations, fall of ground investigations, pre-ledge inspections and fracture mapping. The different visits, RMR values, MRMR values are indicated in *Table 5-10*:

Table 5-10: Summary of the underground investigations conducted.

Workplace Name	BASELINE	113-36 E2	105-18 Raise	102-43 W13	113-35 E1	113-36 W5	102-16 E8	102-18 E2	102-18 E1	105-24 E4	109-34 E1a	109-32 E1	105-18 E9
Workplace Type	Breast Panel	Breast Panel	Raise Tunnel	Breast Panel	Breast Panel	Breast Panel	Ledging Panel	Breast Panel	Breast Panel	Breast Panel	Breast Panel	Breast Panel	Breast Panel
Investigation Type	Baseline Panel	F.O.G	Pre-ledge Inspec-tion	F.O.G	Pre-work Assess-ment	F.O.G	Routine Panel Audit	Routine Panel Audit	Poor Ground Condition Investiga-tion	F.O.G	F.O.G	Poor Ground Condition Investiga-tion	Fracture Mapping
Joint Sets	1	1	1	1	1	1	1	1	1	1	1	2	0
Fracture Sets	2	1	1	1	1	1	1	1	1	2	1	3	3
RMRvcr	68	69	69	74	64	74	73	77	71	69	57	54	67
MRMRvcr	46	38	57	46	52	41	49	48	44	42	46	32	41

Workplace Name	109-24 W2	109-39 W7	113-38 E11	113-36 W7	105-39D W13	113-30N E6	105-18 E9	109-24 W2	109-24 W4	105-42 E6	105-39D	113-31N E8
Workplace Type	Breast Panel	Breast Panel	Breast Panel	Breast Panel	Breast Panel	Breast Panel	Breast Panel	Breast Panel	Breast Panel	Breast Panel	Breast Panel	Breast Panel
Investigation Type	F.O.G	Poor Ground Condition Investiga-tion	F.O.G Follow-up	Poor Ground Condition Investiga-tion	Poor Ground Condition Investiga-tion	F.O.G	Fracture Mapping	F.O.G	F.O.G	Pre-work	Pre-work	DMR Visit
Joint Sets	1	2	2	1	1	1	0	1	1	2	1	0
Fracture Sets	2	2	2	1	1	3	3	2	2	1	2	1
RMRvcr	67	74	67	76	76	65	67	67	61	63	62	80
MRMRvcr	45	49	37	43	44	40	41	45	35	32	39	62

From *Table 5-10*, it can be seen that the RMR_{vcr} value for most panels where falls of ground occurred is around 68, and 42 after adjustments. This is an indication that even though the RMR_{vcr} value might fall under the average – good category, but with the combination of weakness plane interaction, poor mining practices, and high-stress levels, a rating downgrade of up to 26 can result. This places these panels into the average and average–poor categories, *see Figure 5-7*. Even though the ground conditions may appear to be 'good', with incorrect mining practices, which are not limited to blasting alone, these values are detrimentally affected and in practice result in poorer ground conditions, which become more difficult to control. Falls of ground occurred in working places that fall under the 'good' category and therefore it must not be assumed that falls of ground occur only in those areas with poorer conditions, albeit more likely, complacency in terms of standard mining practices must be avoided at all costs.

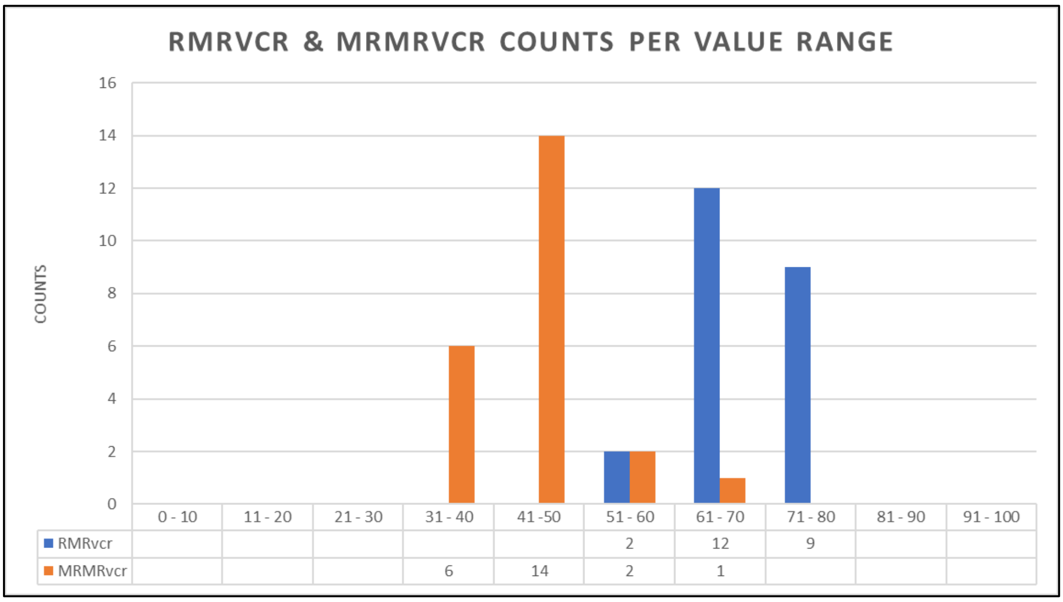


Figure 5-5: Counts per value range for RMR_{vcr} and $MRMR_{vcr}$

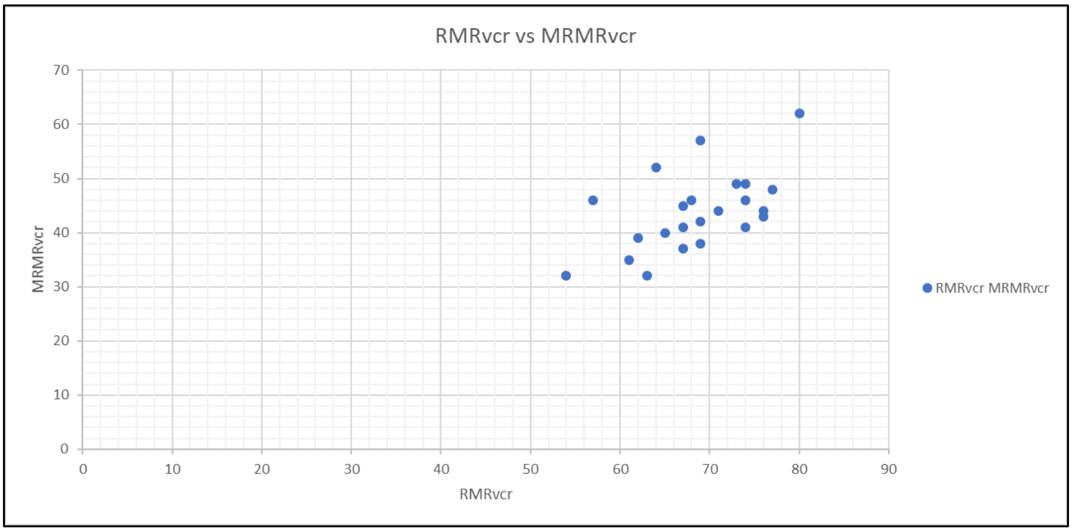


Figure 5-6: RMR_{vcr} vs. $MRMR_{vcr}$

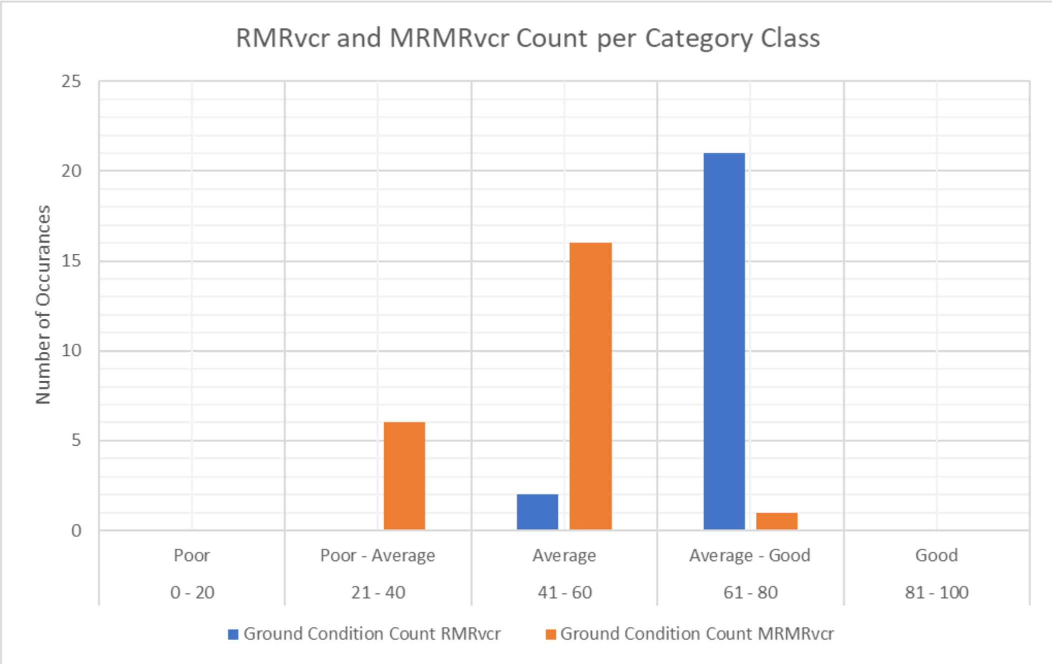


Figure 5-7: Count per category class.

6 CONCLUSION

6.1 Practicality and Applicability of the $MRMR_{vcr}$ System

The $MRMR_{vcr}$ rock mass classification system is effective in determining the basic rock mass conditions underground.

- The system remains simple, which Laubscher (1990) referred to. It is easy to use, with only a few but important factors to consider, which significantly influence the condition of the rock mass.
- How the system has been set up provides easy observational points in a mining panel with reference to the panel face, from which basic joint and fracture parameters can be determined.
- This system can easily be used by rock mechanics and geology departments, and helps with the understanding of the rock mass and its reaction to mining.

It is envisaged that this system will be in continuous use over the coming months and years at Kusasalethu Mine, with additional data being added to the system, refining it further and cementing its applicability in the narrow tabular orebody mining environment. The system is robust and adequately defines the rock mass conditions within which mining takes place, giving the user and production personnel a better understanding of the rock mass quality, the effects of interacting weakness planes and their frequency of occurrence, and the effects of mining.

6.2 Quantitative Analysis

6.2.1 Standard rock mass quality and conditions

The baseline panel gives a good indication of the common conditions encountered at Kusasalethu Mine and has a RMR_{vcr} value of 68 and a $MRMR_{vcr}$ value of 46 which compare well with the RMR_{vcr} average value of 69 and $MRMR_{vcr}$ average value of 44. The RMR values indicate that the majority of panels fall into the average–good category and after adjustments are downgraded to the lower part of the average category which is a good indication of the physical conditions underground. The average downgrade on the RMR value is 25 points after the adjustments, and this shows that the effects of mining, stress and weakness plane orientation could have a negative effect on the stability of the hanging wall in a panel.

The values do not necessarily indicate that a fall of ground will occur when these conditions are encountered, but rather that a combination of adversely orientated weakness planes can easily result in a fall occurring, if not treated correctly with controllable mining practices such as blasting and, by association, support installation.

6.2.2 Average RMR_{vcr} and $MRMR_{vcr}$ ranges

The results of the research show that the RMR_{vcr} range from the data collected at Kusasalethu Mine is 54 – 76 and that the $MRMR_{vcr}$ range from the data collected is 32 – 57. This confirms that conditions throughout the mine vary, which is expected due to the nature of deposition of the reef and extrusion of the lava hanging wall. Furthermore, it is clear that the occurrence of factors, natural and man-made, affect these values immensely.

7 RECOMMENDATIONS

The author recommends that the system remains in use and is continually updated by the personnel within the rock engineering department as well as the geology department with more observational data to increase the size of the database, to refine and improve the rating system, and to gain a better understanding of the various geotechnical areas on the mine.

Furthermore, consideration should be given to the $MRMR_{vcr}$ system in proactively reacting to varying conditions in a working place to reduce the number of falls of ground that occur at the mine. Further research into the system can be done so that appropriate support regimes can be implemented for the different conditions found, before they become worse or result in a fall of ground.

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A. Appendix - Excerpt From the Fall of Ground Database

Report	Date	Time	Shift Time	Reef Type	Stope / Dev	Excavation	Dist to face [m]	Location	Depth	Source	Type [Injury]	Activity	Seismicity	Length	Width	Thickness	Mass [kg]	Size	Support to Std.	Sup. Design=ok	correct Cut	FOG Boundaries	Temp. Sup. Dist.	Perm. Sup. Dist.	Pack. Sup. Dist.	Backfill Sup. Dist.	Tendon Sup. Dist.	Jointing	Flow planes present	Fault / shear zone/dyke	HW layer	Mynolite	Quartze veins	Brow	Stress	Barring to std.	Entry Exam.	Previous FOG	Face Shape	Layout	Blasting Quality	Isb helped
KUS929-21	2021/10/05	pm	W	V	S	P	0.0	F	3037	H		A Gravity FOG occurred with the blast. Wedges bound by miltiple intersecting joints dislodged from the hanging wall. 1.6m thick brow was observed 4.1m behind the panel face at the mddle of the panel. Excessive lava was exposed above the reef top contact. A brow that is prependicular to the panel face was observed at the middle of the panel. Panel face shape is not straight as the bottom of the panel is lagging the middle of the panel. Overhanging face and brow dislodged from the hanging wall with the blast.	N	7.00	2.80		0.00	U	N	Y	U	Upper and Lower Boundaries - Blast fractures. Eastern Boundary - Blast and Stress fractures. Western Boundary - Stress Fractures. Southern Boundary - E-W striking joint plane dipping 90 degree spaced 2.0m apart on dip. Northern Boundary - Blast Fractures.	1.4-2.0	1.7	2.6	4.1	2.1	1	N	Y	Y	N	N	Y	Y	N	N	N	P	Y	P	
KUS937-21	2021/10/10	19:14	W	V	S	P	1.0	B	3196	H		A seismic event of 1.5Mag event occurred during night shift with the blast.	Y	16.00	3.50	0.50	75600.00	L	N	Y	U	Upper Boundary - Blast and stress fractures. Lower Boundary - VCR Reef. Easterna dn Western Boundaries - N-S Striking joint, dipping at 90 degrees and spaced 0.8 - 1.5m apart on strike and stress fractures. Southern and Northern Boundaries - E-W striking joint plane dipping 90 degrees spaced 1.8 - 3.0m apart on dip and stress fractures.	U	1.5	3	3.2	1.4	2	N	Y	Y	N	Y	N	Y	N	N	N	P	Y		
KUS938-21	2021/10/11	pm	W	V	S	P	0.0	F	2895	H		Gravity FOG during the blast	N	3.00	1.50	0.20	2430.00	S	N	Y	U	Upper Boundary - Blast and stress fractures and layered lavas. Lower Boundary - VCR Reef. Eastern and Western Boundaries - N-S striking joint with a spacing of 0.7 - 1.2m and dip of 90 degrees were observed in the face. Southern Boundary - A Fault with a NW-SE strike dipping 90 degrees with an unknown throw was observed in the face along which the FOG occurred. Northern Boundary - Blasting and stress fractures.	U	1.7	U	3.2	1.5	1	U	Y	Y	N	Y	N	Y	N	N	Y	P	Y	P	
KUS941-21	2021/10/10	pm	B	V	S	P	0.0-2.4	F	2874	H		Gravity FOG assumed to occurred after the blast THAT WAS NOT REPORTED	N	3.00	2.40	0.50	9720.00	S	N	Y	U	Upper Boundary - Extension Fractures. Lower Boundary - Top Reef Contact. Eastern, Western, Southern and Northern Boundaries - Extension Fractures.	NA	2.5	3	3.5	1.6	0	U	Y	Y	N	N	N	Y	Y	N	N	G	Y	U	
KUS951-21	2021/10/13	10:40	I	V	D		2.0	F	3011	H	R	Gravity FOG injured 2 employees.	N	3.80	3.70	1.10	41758.20	L	N	Y	U	Upper Boundary - Weak parting plane(lava flow plane). Lower Boundary - Top reef contact. Eastern and Western Boundaries - Extension Fractures. Southern Boundary - Prominent joint dip 65 degree N, strike 84 degree W of N. Northern Boundary - Extension Fracture dip 85 degree, Strike 80248 W of N	NA	U	5.8	NI	U	1	Y	Y	Y	N	N	N	Y	N	N	N	U	Y	P	
KUS940-21	2021/09/22	08:30	I	V	S	R	0.0 - 20.0	B	3237	H		Gravity FOG In the over-stoped area leading to Rasie 2 and 1 had completely collapsed.	N	20.00	12.00	2.00	1296000.00	L	N	Y	U	Upper Boundary - Weak parting plane(lava flow plane). Lower Boundary - Blasted stope hanging wall. Eastern and Western Boundaries - Unknown. Southern Boundary - Prominent quartz vein intrusions. Northern Boundary - Unknown.	U	U	U	U	U	1	Y	Y	Y	N	Y	N	Y	U	Y	N	U	Y	U	

B. Appendix - Mining Rock Mass Rating for the Ventersdorp Contact Reef

Rock Mass Rating Ventersdorp Contact Reef (Adjusted Values)

			UCS		Jointing																				
Date	Working Place	Excavation Type	Value	Rating	Present	No. of Sets	Dip 1	% Rating	Favourable or unfavourable	% Rating	Strike 1	% Rating	Dip 2	% Rating	Favourable or unfavourable	% Rating	Strike 2	% Rating	Dip 3	% Rating	Favourable or unfavourable	% Rating	Strike 3	% Rating	Total Joint Rating
BASELINE	BASELINE	Breast Panel	240-269	7.5	Y	1	60°-90°	100%	Favourable	100%	0°-30°	99%	N/A	100%	Favourable	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	24.75
07/01/2020	113-36 E2	Breast Panel	270-299	8.75	Y	1	60°-90°	100%	Unfavourable	90%	30°-60°	95%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	21.38
21/01/2020	105-18 Raise	Raise	240-269	7.5	Y	1	60°-90°	100%	Unfavourable	90%	30°-60°	95%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	21.38
05/02/2020	102-43 W13	Breast Panel	240-269	7.5	Y	1	60°-90°	100%	Unfavourable	90%	0°-30°	99%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	22.28
18/03/2020	113-35 E1	Breast Panel	270-299	8.75	Y	1	60°-90°	100%	Unfavourable	90%	0°-30°	99%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	22.28
19/03/2020	113-36 W5	Breast Panel	270-299	8.75	Y	1	60°-90°	100%	Unfavourable	90%	0°-30°	99%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	22.28
29/04/2020	102-16 E8	Ledging Panel	270-299	8.75	Y	1	60°-90°	100%	Favourable	100%	0°-30°	99%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	24.75
07/06/2020	102-18 E2	Breast Panel	270-299	8.75	Y	1	60°-90°	100%	Unfavourable	90%	0°-30°	99%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	22.28
22/06/2020	102-18 E1	Breast Panel	270-299	8.75	Y	1	60°-90°	100%	Unfavourable	90%	0°-30°	99%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	22.28
29/06/2020	105-20 E4	Breast Panel	240-269	7.5	N	1	N/A	100%	Favourable	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	25.00
01/07/2020	109-34 E1a	Breast Panel	270-299	8.75	Y	1	60°-90°	100%	Unfavourable	90%	0°-30°	99%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	22.28
22/07/2020	109-32 E1	Breast Panel	270-299	8.75	Y	2	60°-90°	100%	Unfavourable	90%	0°-30°	99%	60°-90°	98%	Unfavourable	90%	60°-90°	97%	N/A	100%	N/A	100%	N/A	100%	19.06
22/10/2020	109-39 W7	Breast Panel	240-269	7.5	Y	2	60°-90°	100%	Favourable	100%	0°-30°	99%	60°-90°	98%	Unfavourable	90%	60°-90°	97%	N/A	100%	N/A	100%	N/A	100%	21.17
18/11/2020	113-38 E11	Breast Panel	240-269	7.5	Y	2	60°-90°	100%	Favourable	100%	30°-60°	95%	60°-90°	98%	Unfavourable	90%	60°-90°	97%	N/A	100%	N/A	100%	N/A	100%	20.32
15/03/2021	113-36 W7	Breast Panel	270-299	8.75	Y	1	60°-90°	100%	Favourable	100%	30°-60°	95%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	23.75
14/04/2021	105-39D W13	Breast Panel	270-299	8.75	Y	1	60°-90°	100%	Unfavourable	90%	30°-60°	95%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	21.38
13/05/2021	113-30N	Breast Panel	270-299	8.75	N	1	N/A	100%	Favourable	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	25.00
04/06/2021	105-18 E9	Breast Panel	<300	10	N	1	N/A	100%	Favourable	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	25.00
27/08/2021	109-24 W2	Breast Panel	270-299	8.75	Y	1	60°-90°	100%	Favourable	100%	0°-30°	99%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	24.75
27/08/2021	109-24 W2	Breast Panel	240-269	7.5	Y	1	60°-90°	100%	Favourable	100%	0°-30°	99%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	24.75
04/09/2021	105-42 E6	Breast Panel	240-269	7.5	Y	2	60°-90°	100%	Favourable	100%	0°-30°	99%	60°-90°	98%	Favourable	100%	60°-90°	97%	N/A	100%	N/A	100%	N/A	100%	23.53
15/09/2021	105-39D W11	Breast Panel	270-299	8.75	Y	1	60°-90°	100%	Favourable	100%	30°-60°	95%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	23.75
06/10/2021	113-31N E8	Breast Panel	270-299	8.75	N	1	N/A	100%	Favourable	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	25.00

		Fracturing																					Fracture Frequency per Metre			
Date	Working Place	Present	No. of Sets	Dip 1	% Rating	Favourable or unfavourable	% Rating	Strike 1	% Rating	Dip 2	% Rating	Favourable or unfavourable	% Rating	Strike 2	% Rating	Dip 3	% Rating	Favourable or unfavourable	% Rating	Strike 3	% Rating	Total Fracture Rating 1 Set	Average per Metre	Number of Sets	Rating	RMR vcr
BASELINE	BASELINE	Y	2	60°-90°	100%	Favourable	100%	0°-30°	99%	60°-90°	98%	Unfavourable	90%	60°-90°	97%	N/A	100%	N/A	100%	N/A	100%	21.17	5	3 Set	15	68
07/01/2020	113-36 E2	Y	1	30°-60°	95%	Favourable	100%	0°-30°	99%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	23.51	7	2 Set	15	69
21/01/2020	105-18 Raise	Y	1	60°-90°	100%	Favourable	100%	0°-30°	99%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	24.75	7	2 Set	15	69
05/02/2020	102-43 W13	Y	1	30°-60°	95%	Favourable	100%	0°-30°	99%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	23.51	3	2 Set	21	74
18/03/2020	113-35 E1	Y	1	60°-90°	100%	Favourable	100%	0°-30°	99%	60°-90°	98%	Unfavourable	90%	60°-90°	97%	N/A	100%	N/A	100%	N/A	100%	21.17	7	3 Set	12	64
19/03/2020	113-36 W5	Y	1	60°-90°	100%	Favourable	100%	0°-30°	99%	N/A	100%	Favourable	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	24.75	5	2 Set	18	74
29/04/2020	102-16 E8	Y	1	60°-90°	100%	Favourable	100%	0°-30°	99%	N/A	100%	Favourable	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	24.75	7	2 Set	15	73
07/06/2020	102-18 E2	Y	1	60°-90°	100%	Favourable	100%	0°-30°	99%	N/A	100%	Favourable	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	24.75	3	2 Set	21	77
22/06/2020	102-18 E1	Y	1	60°-90°	100%	Favourable	100%	0°-30°	99%	N/A	100%	Favourable	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	24.75	7	2 Set	15	71
29/06/2020	105-20 E4	Y	2	60°-90°	100%	Favourable	100%	0°-30°	99%	60°-90°	98%	Unfavourable	90%	60°-90°	97%	N/A	100%	N/A	100%	N/A	100%	21.17	7	2 Set	15	69
01/07/2020	109-34 E1a	Y	1	60°-90°	100%	Favourable	100%	0°-30°	99%	60°-90°	98%	Favourable	100%	60°-90°	97%	30°-60°	87%	Unfavourable	90%	30°-60°	87%	16.03	10	3 Set	10	57
22/07/2020	109-32 E1	Y	3	60°-90°	100%	Favourable	100%	0°-30°	99%	60°-90°	98%	Favourable	100%	60°-90°	97%	30°-60°	87%	Unfavourable	90%	30°-60°	87%	16.03	10	3 Set	10	54
22/10/2020	109-39 W7	Y	2	60°-90°	100%	Favourable	100%	0°-30°	99%	60°-90°	98%	Unfavourable	90%	30°-60°	90%	N/A	100%	N/A	100%	N/A	100%	19.65	1	3 Set	26	74
18/11/2020	113-38 E11	Y	2	60°-90°	100%	Favourable	100%	0°-30°	99%	60°-90°	98%	Unfavourable	90%	60°-90°	97%	N/A	100%	N/A	100%	N/A	100%	21.17	3	3 Set	18	67
15/03/2021	113-36 W7	Y	1	60°-90°	100%	Unfavourable	90%	0°-30°	99%	N/A	100%	Favourable	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	22.28	3	2 Set	21	76
14/04/2021	105-39D W13	Y	1	60°-90°	100%	Favourable	100%	0°-30°	99%	N/A	100%	Favourable	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	24.75	3	2 Set	21	76
13/05/2021	113-30N	Y	3	60°-90°	100%	Favourable	100%	0°-30°	99%	30°-60°	90%	Unfavourable	90%	30°-60°	90%	60°-90°	95%	Favourable	100%	60°-90°	93%	15.94	5	3 Set	15	65
04/06/2021	105-18 E9	Y	3	60°-90°	100%	Favourable	100%	0°-30°	99%	30°-60°	90%	Unfavourable	90%	60°-90°	97%	60°-90°	95%	Favourable	100%	60°-90°	93%	17.18	5	3 Set	15	67
27/08/2021	109-24 W2	Y	2	60°-90°	100%	Favourable	100%	0°-30°	99%	30°-60°	90%	Unfavourable	90%	30°-60°	90%	60°-90°	95%	Favourable	100%	60°-90°	93%	15.94	3	3 Set	18	67
27/08/2021	109-24 W2	Y	2	30°-60°	95%	Favourable	100%	0°-30°	99%	30°-60°	90%	Unfavourable	90%	30°-60°	90%	N/A	100%	N/A	100%	N/A	100%	17.14	7	3 Set	12	61
04/09/2021	105-42 E6	Y	2	30°-60°	95%	Favourable	100%	0°-30°	99%	60°-90°	98%	Unfavourable	90%	60°-90°	97%	N/A	100%	N/A	100%	N/A	100%	20.12	7	3 Set	12	63
15/09/2021	105-39D W11	Y	2	30°-60°	95%	Favourable	100%	0°-30°	99%	30°-60°	90%	Favourable	100%	30°-60°	90%	N/A	100%	N/A	100%	N/A	100%	19.05	10	3 Set	10	62
06/10/2021	113-31N E8	Y	1	60°-90°	100%	Favourable	100%	0°-30°	99%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	N/A	100%	24.75	5	1 Set	21	80

RMR VCR Adjustments (Adjusted Values)

		AFS		Blast Prod		Pre-con Blast		Weakness Plane Orientation				
Date	Working Place	Design	% Rating Adj	Good/Poor	% Rating Adj	Good/Poor	% Rating Adj	No. of Joints Defining the block	No. of Faces Inclined Away from The Vertical	% Rating Adj	MRMR vcr (adjusted Values)	Comments/Additional Observations/Concerns
BASELINE	BASELINE	=Design	98%	Good	97%	Good	100%	4	4	70%	46	
07/01/2020	113-36 E2	<Design	100%	Good	97%	Poor	90%	4	4	70%	42	
21/01/2020	105-18 Raise	>Design	90%	Good	97%	Good	100%	4	2	85%	51	
05/02/2020	102-43 W13	<Design	100%	Poor	90%	Poor	90%	4	2	85%	51	
18/03/2020	113-35 E1	=Design	98%	Good	97%	Good	100%	4	2	85%	52	
19/03/2020	113-36 W5	=Design	98%	Poor	90%	Poor	90%	3	3	70%	41	
29/04/2020	102-16 E8	=Design	98%	Good	97%	Good	100%	4	4	70%	49	
07/06/2020	102-18 E2	<Design	100%	Poor	90%	Poor	90%	4	2	85%	53	
22/06/2020	102-18 E1	<Design	100%	Poor	90%	Poor	90%	4	2	85%	49	
29/06/2020	105-20 E4	<Design	100%	Good	97%	Good	100%	4	4	70%	47	
01/07/2020	109-34 E1a	=Design	98%	Good	97%	Good	100%	4	2	85%	46	
22/07/2020	109-32 E1	<Design	100%	Good	97%	Poor	90%	6	5	75%	35	
22/10/2020	109-39 W7	<Design	100%	Good	97%	Good	100%	5	4	75%	54	
18/11/2020	113-38 E11	=Design	98%	Poor	90%	Poor	90%	4	4	70%	37	
15/03/2021	113-36 W7	>Design	90%	Poor	90%	Poor	90%	4	4	70%	39	
14/04/2021	105-39D W13	<Design	100%	Poor	90%	Poor	90%	3	2	80%	49	
13/05/2021	113-30N	<Design	100%	Good	97%	Good	100%	3	3	70%	44	
04/06/2021	105-18 E9	=Design	98%	Poor	90%	Good	100%	4	4	70%	41	
27/08/2021	109-24 W2	=Design	98%	Good	97%	Good	100%	3	3	70%	45	Obervations were only made in the siding, ASG and toe of the panel. Blasting using 1.5m long drill steel.
27/08/2021	109-24 W2	>Design	90%	Poor	90%	Poor	90%	4	4	70%	31	
04/09/2021	105-42 E6	<Design	100%	Poor	90%	Poor	90%	3	3	70%	36	
15/09/2021	105-39D W11	>Design	90%	Poor	90%	Good	100%	3	3	70%	35	Panel wil be mining from the down-dip to the breast direction
06/10/2021	113-31N E8	>Design	90%	Good	97%	Good	100%	3	2	80%	56	

C. Appendix - Captured Field Data

				Joint Data																		
Date	Workplace	Investigation Type	Face Orientation in relation to North (0°)	JS 1 Dip	JS 1 Dip Direction	JS 1 Strike (0° = NORTH)	Joint Infill (type) + Thickness	Joint Surface	JS 1 Spacing	JS 2 Dip	JS 2 Dip Direction	JS 2 Strike	Joint Infill (type) + Thickness	Joint Surface	JS 2 Spacing	JS 3 Dip°	JS 3 Dip Direction	JS 3 Strike°	Joint Infill (type) + Thickness	Joint Surface	JS 3 Spacing	Random Y/N If Y give details
07/01/2020	113-36 E2	FOG	336	80	SW	300	Calcite (1mm)	Rough Undulating	1.4m													N
21/01/2020	105-18	Pre-Ledge	335	85	W	335	None	Rough Undulating	0.15m - 0.3m													N
05/02/2020	102-43 W13	FOG	339	85	E	330	Tight No Infill	Rough Planar	1.0m													N
18/03/2020	113-35 E1	SRA	332	85	E	14	Tight No Infill	Rough Planar	1.0m													N
19/03/2020	113-36 W5	FOG	341	85	E	340	Calcite (1mm)	Rough Planar	0.2m-0.5m													N
29/04/2020	102-16 E8	Normal Audit	336	80	W	350	Tight No Infill	Rough Planar	0.1m - 0.2m													N
07/06/2020	102-18 E2	Audit	335	85	E	8	Tight No Infill	Undulating	0.3m - 0.6m													N
22/06/2020	102-18 E1	Poor Ground Investiagation	336	85	E	6	Tight No Infill	Undulating	0.3m - 0.6m													Y - Details not captured
29/06/2020	105-20 E4	FOG	332	No Joint Data Captured																		
01/07/2020	109-34 E1a	FOG	333	70	E	340	Tight No Infill	Smooth	0.5m													N
22/07/2020	109-32 E1	Poor Ground Investiagation	330	75	E	329	No Infill	Undulating	0.4m	75	E	40	2mm Calcite Infill	Smooth	1.5m							N
22/10/2020	109-39 W7	Poor Ground Investiagation	28	65	SE	2	No Infill	Rough	0.1m - 0.3m	60	NE	302	No Infill	Rough	1.0m							N
18/11/2020	113-38 E11	FOG follow-up	341	70	W	43	No Infill	Rough Undulating	0.57m	80	E	358	No Infill	Rough	0.3m	23	N		5cm + quartz	Smooth Undulating	?	Y - JS 3
15/03/2021	113-36 W7	Poor Ground Investiagation	30	80	NE		No Infill	Undulating	3m													N
14/04/2021	105-39D W13	Poor Ground Investiagation	10	80	NW	45	No Infill	Smooth Undulating	Fault parallel (0.5m)													N
13/05/2021	113-30N E6	FOG	337																			
04/06/2021	105-18 E9	Fracture Mapping	326																			
27/08/2021	109-24 W2	FOG	21	85	W	330	No Infill	Rough Undulating	0.5m													
04/09/2021	105-42 E6	Pre-work	335°	80	E	305	No Infill	Smooth Undulating	0.3	90	E	245°	No infill	Smooth undulating	0.4 - 0.7							
15/09/2021	105-39D W11	Pre-work	12°	80	S	330	Quartz	Smooth Undulating	1.0 - 1.5													
06/10/2021	113-31N E8	DMR Visit	330																			

		Fracture Data															
Date	Workplace	FS 1 Dip	FS 1 Dip Direction	FS 1 Strike	Type	FS 1 Spacing	FS 2 Dip	FS 2 Dip Direction	FS 2 Strike	Type	FS 2 Spacing	FS 3 Dip	FS 3 Dip Direction	FS 3 Strike	Type	FS 3 Spacing	Random Y/N If Y give details
07/01/2020	113-36 E2	80°	W	330° (face parallel)	Extension	0.2m											N
21/01/2020	105-18	None Observed	None Observed														N
05/02/2020	102-43 W13	50°	E	330° (face parallel)	Extension	0.3m											N
18/03/2020	113-35 E1	70°	W	334°(face parallel)	Extension	0.3m	80	NW	64°	Extension	0.15m						
19/03/2020	113-36 W5	65°	E	335° (face parallel)	Extension	0.3m											N
29/04/2020	102-16 E8	80°	W	330° (face parallel)	Extension	0.1m-0.25m											N
07/06/2020	102-18 E2	70°	W	338° (face parallel)	Extension	0.3m											N
22/06/2020	102-18 E1	65° - 70°	W	338° (face parallel)	Extension	0.1m - 0.25m											N
29/06/2020	105-20 E4	80°-90°	W	339° (face parallel)	Extension	25cm	80°	NW	69°	Extension	0.02m - 0.05m						N
01/07/2020	109-34 E1a	70	W	340° (face parallel)	Extension	0.1m - 0.2m	80	NW	60°	Extension	0.1m	60	E	Curved	Extension	0.1m	N
22/07/2020	109-32 E1	65	W	330° (face Parallel)	Extension	0.3m	80	NNW	60°	Extension	0.3m	55	E	Curved	Extension	0.2m	N
22/10/2020	109-39 W7	65	E	332° (face parallel)	Extension	0.05m - 0.1m	60	NE	Curved	Extension	0.05m - 0.15m						N
18/11/2020	113-38 E11	65	W		Extension	0.3m	85	N	242°	Extension	0.15m						N
15/03/2021	113-36 W7	80	E	Face parallel	Extension	0.05m - 0.15m	80	W (reverse)	Face Parallel	Extension (secondary)	0.1m						Y - Spacing 2cm, dip 85°, paralle to face.
14/04/2021	105-39D W13	70	E	Face parallel	Extension	0.05m - 0.15m (top) & 0.2m - 0.4m (middle + Bot)											N
13/05/2021	113-30N E6	65	W	Face parallel	Extension	0.3m	55	W (reverse)	Curved over sidign and gully	Extension	0.1m	85°	N	(90° to FS 1)	Extension	0.1m - 0.2m	N
04/06/2021	105-18 E9																
27/08/2021	109-24 W2	60	E	Face Parallel	Extension	0.3m	45	E	Over Sding + Gully	Extension	0.4m						
04/09/2021	105-42 E6	45	W	Face Parallel	Extension	0.15											
15/09/2021	105-39D W11	55	NE	Curved	Extension	0.1	60	E	Face Parallel	Ext	0.075						
06/10/2021	113-31N E8	70	W	Face Parallel	Extension	0.2											

D. Appendix - Numerical Modelling Input Parameters.

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Input Stress State

20 Jun 2021 - 20 Sep 2021

General		Stress Tensor	
Coordinate System:	SWD	Sxx [MPa/m]:	0.013750
Status:	SOLVED	Syy [MPa/m]:	0.013750
Rockmass		Szz [MPa/m]:	0.027500
Datum [m]:	186.00	Sxy [MPa/m]:	0.000000
Young's Modulus [GPa]:	70.00	Sxz [MPa/m]:	0.000000
Poisson's Ratio:	0.20	Syz [MPa/m]:	0.000000