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Classification of Khoemacau (Zone 5) Carbon Rich Rockmass Using Q and Rmr Systems to Recommend Support for Ore Drives

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ABSTRACT
The Khoemacau project is situated on the Kalahari copper belt which stretches over 1000km from northeast Botswana into southwest Namibia and consists of deformed Meta volcanic and sedimentary rocks. High-grade copper and silver mineralization is dominantly related to shearing, folding and tensional failure along and close to the Ngwako Pan and D'Kar contact. Disseminated and hydrothermal vein-hosted sulfide mineralization.
Older CAR unit rocks seemed to deteriorate and the disking of the siltstones begins to break apart also showing signs of high shearing. This was a concern as it showed that this rock unit may pose problems during mining as it may lead to tunnel collapses and unstable conditions. So, in this research paper the quality of this rock mass will be assessed using the RMR and Q Classification method. The Carbon rich rock mass of Zone 5 was classified, and it is found to be inferior quality rock.
The recommended support for RMR classification suggested systematic bolts 4-5m long, spaced 1-1.5m in crown and walls with wire mesh combined with Shotcrete 100-150mm in crown and 100mm sides and steel sets light ribs spaced 1.5m where required while that of Q-system suggested systematic bolting, (and unreinforced shotcrete, 4-10cm).
The observations suggest that the use of two or more classification systems in design and rock engineering, will lead to better and more accurate results in terms of adequately classifying the rock mass and providing sufficient support. This is proved by the more intense support of (steel sets light ribs spaced 1.5m where required) that was suggested by using the RMR classification. In conclusion the empirical methods show that extraction will be feasible with the selected mining method (Sub-level open stoping).
Keywords: *Khoemacau Copper Belt, CAR, RMR, Q-System*

INTRODUCTION

The Khoemacau project is situated on the Kalahari copper belt which stretches over 1000km from northeast Botswana into southwest Namibia and consists of deformed Meta volcanic and sedimentary rocks. Zone 5 is located toward the north-east of the Khoemacau Project license area on the north-west side of a regional syncline. It has a deposit strike length of 4.2km with mineralization dipping at 56 to 61 degrees to the south-east with an average thickness of 20m. The location of the mine is shown in Fig.1

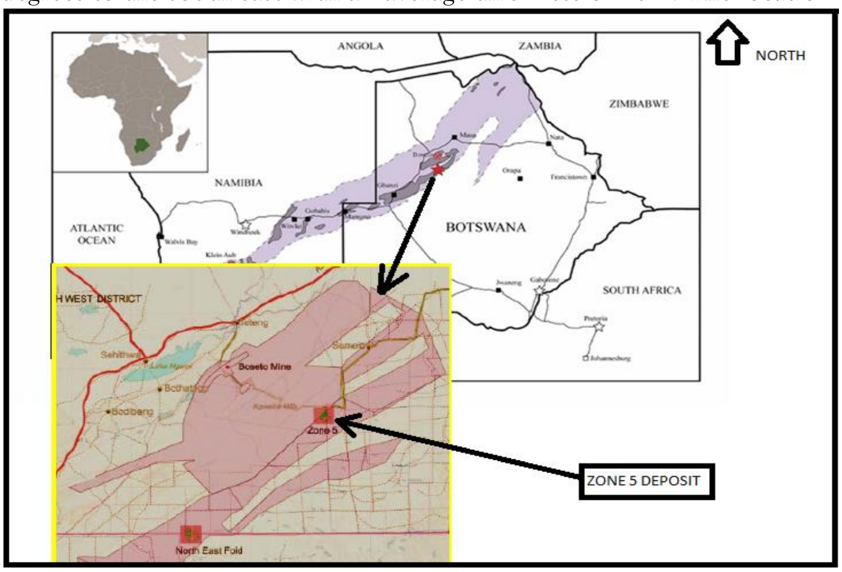

FIGURE 1: LOCATION OF STUDY AREA

GEOLOGY

The zone 5 deposit is comprised of different lithological units which include: Kalahari sands, Massive Sandstone (MSST), Alternating sequence marl/silt (ALT), Carbon Rich siltstones (CAR), Limestone (LST), Lower marl (LMRL) and Ngwako Pan Sandstone (NPF). Throughout the deposit, each one of the above lithologies exhibits some subtle petrographic variability in the form of either color, texture, grain size, deformation, thickness, alteration and mineralization. The Zone 5 deposit is divided into four zones. South, south central, north central, north (Fig.2).

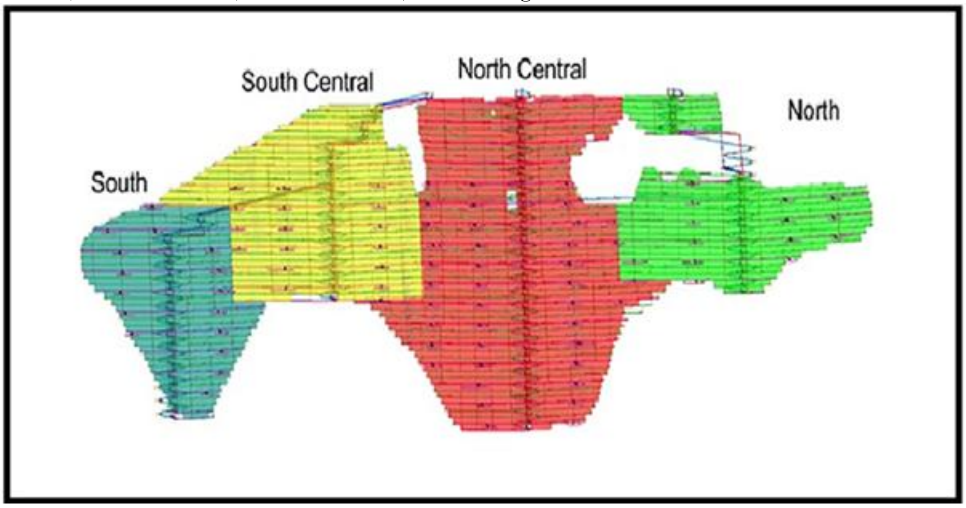

FIGURE 2 : ZONE 5 DEPOSIT DIVISIONS

STRATIGRAPHY

The basal volcanic sequence, Kgwebe Formation, is overlain by the Ghanzi Group metasediments. This Group, from oldest to youngest, forms part of the Neo-Proterozoic and consists of the Kuke, Ngwako Pan, D'Kar and Mamuno Formations. The lower ductile siltstones and carbonaceous units of the D'Kar Formation are the main host for copper and silver mineralization. The D'Kar sediments are composed of finely laminated and chemically reduced mudstones and siltstones intercalated with limestone and thin shale. Copper-silver mineralization at the Project occurs at the stratigraphic redox boundary between the contact of the Ngwako Pan and D'Kar Formations where a structurally controlled trap environment exists. Host rocks are unconformably overlain by unconsolidated Kalahari Sand and calcrete up to 60m thick. Folding and thrusting along north-east trends have structurally repeated stratigraphically controlled mineralization over hundreds of kilometers. Sulphide assemblages are commonly zoned. The sequence is developed vertically upward from the base of the D'Kar Formation. The typical zonation sequence consists of low Sulphur, low iron, copper sulphides (chalcocite and bornite) and passes upward with increasing iron content(chalcopyrite and pyrite). High-grade copper and silver mineralization is dominantly related to shearing, folding and tensional failure along and close to the Ngwako Pan and D'Kar contact. Disseminated and hydrothermal vein-hosted sulfide mineralization styles combine to produce continuity of high-grade copper and silver mineralization over tens of kilometers. These higher grade copper zones typically contain disseminated and massive chalcopyrite, bornite and chalcocite mineralization.

Zone 5 is located toward the north-east of the Project license area on the north-west side of a regional syncline. It has a deposit strike length of 4.2km with mineralization dipping at 56 to 61 degrees to the south-east with an average thickness of 20m. Mineralization is situated in the hanging wall sequence, extending up to 30 to 40m above the contact between the D'Kar Formation and Ngwako Pan Formation (NPF). Economic mineralization typically consists of massive bornite with accompanying chalcopyrite. These minerals are largely vein hosted and make up an extensive system of quartz and quartz carbonate vein shears and cleavages. A near surface layer of supergene oxidation has been identified at Zone 5. An undulating surface parallel to topography was identified and used to separate mixed oxide and sulfide mineralization from sulfide-only mineralization below the boundary. The mixed oxide/sulfide, sulfide only boundary lies approximately 70m below surface. Oxide mineralogy in this near surface zone includes malachite, chrysocolla and minor native copper. The generalized stratigraphy of the study area is shown in Figure.3

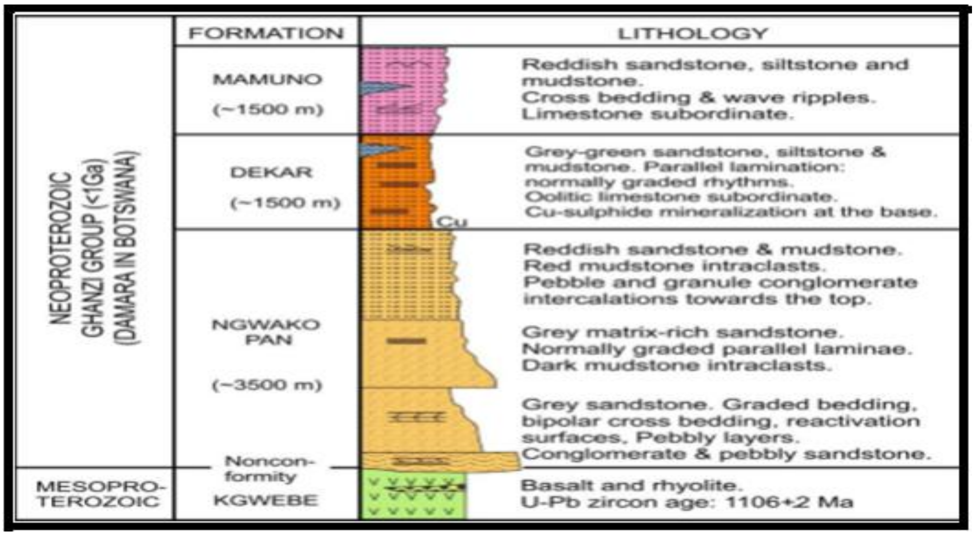

FIGURE 3: ZONE 5 STRATIGRAPHY

MINERALIZATION

Mineralization is situated in the hanging wall sequence extending up to 30-40 meters between D'kar and Ngwako and formation. The lower ductile siltstones and carbonaceous units of the D'Kar Formation are the main host for copper and silver mineralization. Copper-silver mineralization at the Project occurs at the stratigraphic redox boundary between the contact of the Ngwako Pan and D'Kar Formations where a structurally controlled trap environment exists. Host rocks are unconformably overlain by unconsolidated Kalahari Sand and calcrete up to 60m thick. Mineralization is characterized as a sediment hosted stratiform copper deposit. High-grade copper and silver mineralization is dominantly related to shearing, folding and tensional failure along and close to the Ngwako Pan and D'Kar contact. Disseminated and hydrothermal vein-hosted sulfide mineralization styles combine to produce continuity of high grade copper and silver mineralization over tens of kilometers. Higher grade copper zones typically contain disseminated and massive chalcopyrite, bornite and chalcocite mineralization. (Master, 2010). Mineralization is mostly in quartz, quartz carbonate vein shears and cleavages. The mineralization in the area is made up of sulphides which are chalcopyrite (30%) copper, bornite (60%) copper and chalcocite (80%) copper. The mineralization styles include disseminated, vein hosted and cleavage hosted mineralization. The copper originates from hydrothermal epimerization reactions in the redox boundary between the Ngwako and the D'kar formation.

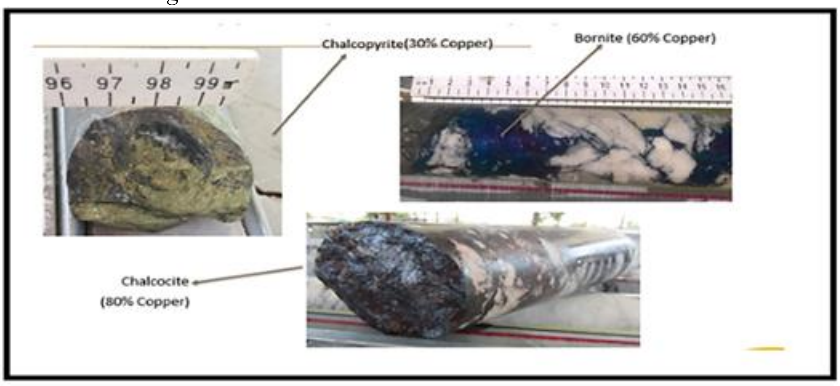

FIGURE 4: MINERALIZATION STYLES

STATEMENT OF THE PROBLEM

The siltstone in the mining area was found to contain high levels of carbon content which was mainly associated with the basin evolution; this is mainly because the zone five deposits are situated in an evolving basin. These siltstones were named carbon rich rock mass (CAR). The rock is formed and it remains strong, but it seems to continue to get weaker with its age. Older CAR unit rocks seemed to deteriorate and the disking of the siltstones begins to break apart also showing signs of high shearing. This was a concern as it showed that this rock unit may pose problems during mining as it may lead to tunnel collapses and unstable conditions. So, in this research paper the quality of this rock mass will be assessed using the RMR and Q Classification method by Barton and then determine the required support that will allow safe and economic extraction of the ore. With this focus the current work aims at the following:

- Classification of the Carbon rich rock mass, using Q and RMR systems.
- Assessment of the response of the CAR unit in ore drives.
- Recommendation of adequate support.

METHODOLOGY

In order to achieve the objectives of this project, the following methodology was carried out.

Field work comprising the following to obtain primary data.

1. Core logging in order to describe and characterize the joints and other discontinuities.
2. Physical identification of rock types during the core logging exercise.
3. Sampling of rock types for laboratory testing.
4. Acquisition of secondary data from the mine to supplement field data, this includes water reduction factor (Jw) and stress reduction factor (SRF).
5. Laboratory testing of the rock samples to determine their uniaxial compressive strengths.
6. The use of Dips7.0 to find the joint set number.
7. Analysis of data and classification of the rock mass using both the Q-system and RMR approach around circular openings.
8. The use of RS2 software to determine the stress distributions.

Uniaxial Compression Test

The uniaxial compression test was carried out in universal testing machine (UTM) and the following results were obtained:

Table 1: UCS VALUES

BOREHOLE ID	UCS (MPA)
HA-026-G	11
HA-014-G	9
HA-002-G	14
HA-009-G	10

Rock Mass Classification.

The rock mass classification schemes that are often used in rock engineering for assisting in the design and support of underground structures are RMR, Q and GSI systems. These systems form an essential part of foremost design approaches (the empirical and the numerical design methods) and are increasingly used in both design approaches as computing power improves. Empirical and numerical design approaches are considered very important in the viable and efficient design of support systems, stability analysis for tunnel, and underground excavations. They not only give information about the composition, strength, deformation properties and characteristics of a rock mass required for estimating the support requirements but also shows which information is relevant and required (Bieniawski, 1979). Based on rock mass classifications, strength and deformation parameters according to specific constitutive laws or the rock mass (e.g. Mohr-Coulomb or Hoek-Brown material models) can be deduced and applied in numerical simulations to consider stability, failure pattern and Factor-of-safety.

The Rock Quality (Q) System

Barton et al (1974) of the Norwegian Geotechnical Institute proposed a Tunneling Quality Index (Q) for the determination of rock mass characteristics and tunnel support requirements. The numerical value of the index Q varies on a logarithmic scale from 0.001 to a maximum of 1000 and is defined by six parameters:

$$Q = \left(\frac{RQD}{J_n}\right) * \left(\frac{J_r}{J_a}\right) * \left(\frac{J_w}{SRF}\right)$$

Where:

- RQD is the Rock Quality Designation
- J_n is the joint set number
- J_r is the joint roughness number
- J_a is the joint alteration number

- J_w is the joint water reduction factor

SRF is the stress reduction factor

SRF is a measure of loosening load in the case of an excavation through shear zones including clay bearing rock, rock stress in competent rock and squeezing loads in plastic incompetent rocks Singh and Goel (1999).

The first quotient (RQD/Jn) represents the overall structure of the rock mass and is a rough measure of the block size i.e. it represents the roughness and frictional characteristics of the joint walls or filling materials (Singh, 1999). The second quotient (Jr/Ja) is described as an indicator of inter-block shear strength. The third quotient (Jw/SRF) consists of active stress parameters.

B-Rock Mass Rating, RMR (Bieniawski, 1976)

The RMR classification system, Bieniawski (1976), was developed for the characterization of the rock mass as a design tool for tunneling. The ratings obtained can be adjusted to account for favorable/unfavorable orientation of discontinuities relative to excavation geometries and orientations. The following six parameters are used to classify a rock mass using the RMR system:

- Uniaxial compressive strength of rock material
- Rock quality designation (RQD)
- Spacing of discontinuities
- Condition of discontinuities
- Groundwater conditions
- Orientation of discontinuities

Each of the six parameters is assigned a rating value corresponding to the characteristics of the core logged during the field work. The sum of the six parameters (or five if unadjusted) gives the "RMR value", which lies between 0 and 100. The descriptions and corresponding ratings for these parameters are shown in the tables below. RMR rating (Bieniawski, 1989) classification is shown in Table.2 **below.**

Table 2: Rock Mass Rating, RMR (Bieniawski, 1989)

Parameter			Ranges of Values						
1	Strength of Intact rock material	Point load strength index	>10 MPa	4-10 MPa	2-4 MPa	1-2 MPa	For this low range – uniaxial compressive test is preferred		
		Uniaxial compressive strength	>250 MPa	100-250 MPa	50-100 MPa	25-50MPa	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		>2m	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 weathered Walls	Slickensided 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	Groundwater	Inflow per 10m Tunnel length (l/min)	None	10	10-25	25-125	>125		
		Joint water press-ure/major principal stress	0	0.0 – 0.1	0.1-0.2	0.2-0.5	>0.5		
		General Conditions	Completely dry	Damp	Wet	Dripping	Flowing		
	Rating		15	10	7	4	0		
6	Strike and dip orientations of joints*		Very favourable	Favourable	Fair	Unfavourable	Very unfavourable		
	Rating		0	-2	-5	-10	-12		

Rating adjustments for discontinuity orientation

Strike and dip directions		Very favourable	Favourable	Fair	Unfavourable	Very Unfavourable
Ratings	Tunnels and mines	0	-2	-5	-10	-12
	Foundations	0	-2	-7	-15	-25
	Slopes	0	05	-25	-50	

Support Recommendations Based on RMR Classification is shown in Table.3

Table 3: Support Recommendations Based on RMR Classification

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Rock mass class	Excavation	Support		
		Rock bolts (20 mm diam., fully bonded)	Shotcrete	Steel sets
1. Very good rock RMR: 81-100	Full face: 3 m advance	Generally no support required except for occasional spot bolting		
2. Good rock RMR: 61-80	Full face: 1.0-1.5 m advance; Complete support 20 m from face	Locally bolts in crown, 3 m long, spaced 2.5 m with occasional wire mesh	50 mm in crown where required	None
3. Fair rock RMR: 41-60	Top heading and bench: 1.5-3 m advance in top heading; Commence support after each blast; Commence support 10 m from face	Systematic bolts 4 m long, spaced 1.5-2 m in crown and walls with wire mesh in crown	50-100 mm in crown, and 30 mm in sides	None
4. Poor rock RMR: 21-40	Top heading and bench: 1.0-1.5 m advance in top heading; Install support concurrently with excavation - 10 m from face	Systematic bolts 4-5 m long, spaced 1-1.5 m in crown and walls with wire mesh	100-150 mm in crown and 100 mm in sides	Light ribs spaced 1.5 m where required
5. Very poor rock RMR < 21	Multiple drifts: 0.5-1.5 m advance in top heading; Install support concurrently with excavation; shotcrete as soon as possible after blasting	Systematic bolts 5-6 m long, spaced 1-1.5 m in crown and walls with wire mesh. Bolt invert	150-200 mm in crown, 150 mm in sides, and 50 mm on face	Medium to heavy ribs spaced 0.75 m with steel lagging and forepoling if required. Close invert

Table 4: Rock mass rating

RMR	ROCK QUALITY
0-20	Very poor
21-40	Poor
41-60	Fair
61-80	Good
81-100	Very good

RESULTS AND ANALYSIS

The 4 boreholes were classified according to both classification systems; the classes were used together with support charts to determine the recommended support for a borehole with the lowest ratings as this would account for the worst-case scenario when designing ore drives. The tables below show how the values of RMR and Q rating were obtained.

Software Dips 7.0 by RocScience

Software Dips 7.0 by RocScience was then used to determine the joint set number (Jn) as a parameter for Q-system and 3 mean joint sets were obtained for the 4 boreholes. The table below shows the number of joint sets and orientation obtained:

Table 5: Mean Set Orientation

SET ID	TREND	PLUNGE
1	180	31
2	210	32
3	294	68

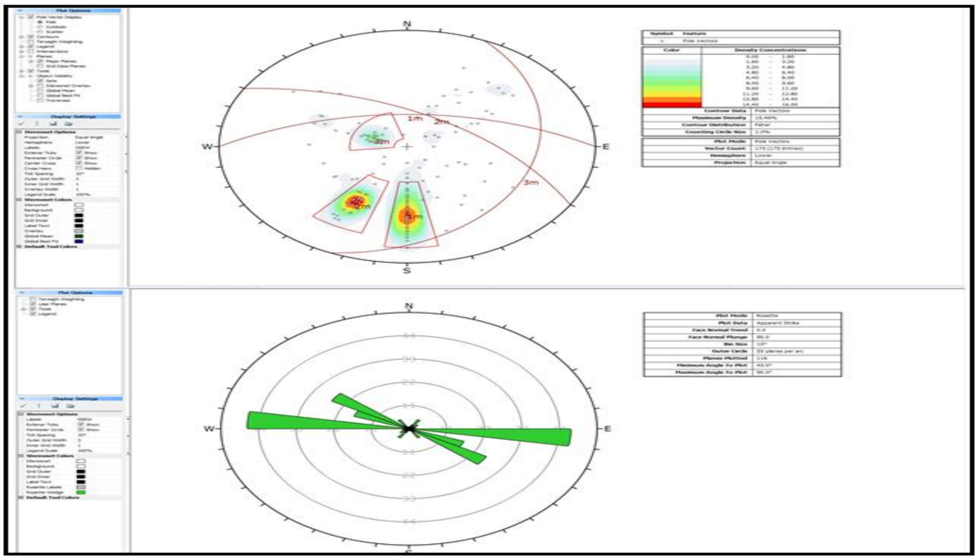

Fig.5: DIPS 7.0 Model Showing Joint Sets

Table 5: HA-026-G Identified joint sets

DISCONTINUITY	DIP	DIP DIRECTION
SET 1	57	031
SET 2	63	360

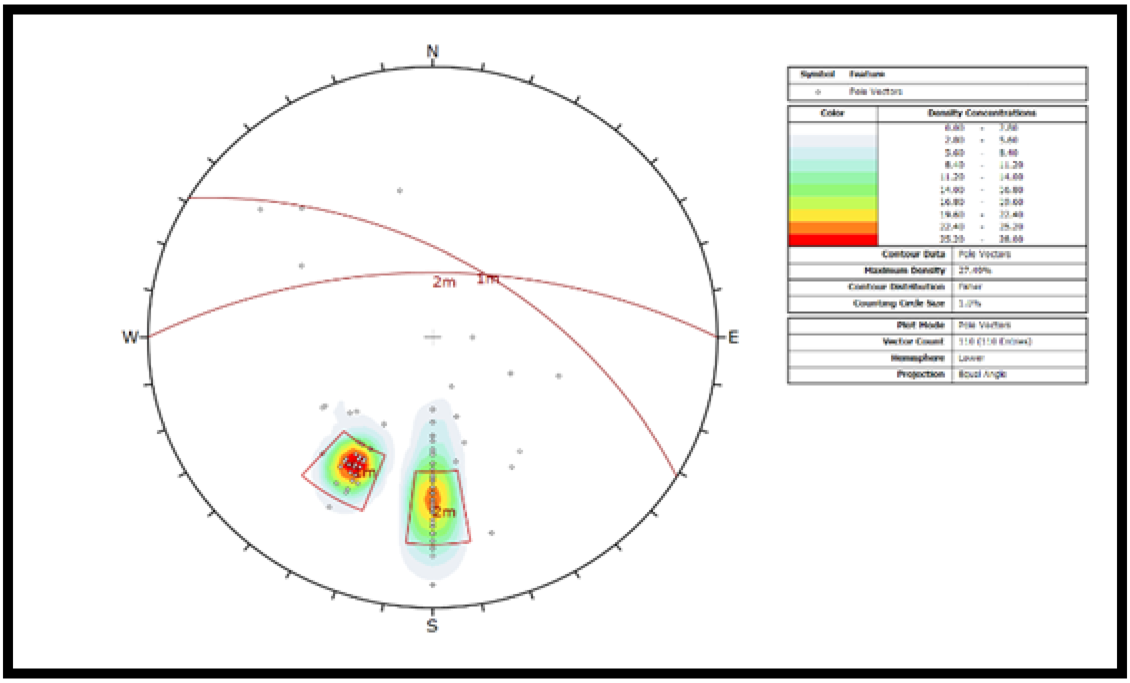

Figure 6: JOINT SET NUMBER

Table 6: HA-026-G Sample Calculations of RMR classification

PARAMETER	CARBON RICH ROCK	RATING
UCS	11MPA	15
RQD	71.21%	13
CONDITION OF DISCONTINUITIES		25
SPACING	410mm	10
GROUNDWATER	DRY	15
TOTAL RATING		78

The table below shows the Q value for borehole HA-026-G and how it was obtained after the calculation were completed.

Table 7: HA-026-G SAMPLE CALCULATIONS FOR Q-SYSTEM (BARTON)

PARAMETER	**DESCRIPTIONS**	**RATING**
RQD		71
Joint Number	2 sets	4
Joint Roughness	Rough or irregular, Planar	1.5
Joint Alteration	Slightly Altered	2
Joint Water	Dry	1
SRF		5
Q Value	{(RQD/Jn)x(Jr/Ja)x(Jw/SRF)	2.66
Excavation Category ESR	Permanent Tunnel (Ore drive)	1.6
Equivalent Dimension **(De)**	(Tunnel Height/ESR)	(5/1.6) = 3.12

$$Q = \left(\frac{RQD}{J_n}\right) * \left(\frac{J_r}{J_a}\right) * \left(\frac{J_w}{SRF}\right)$$

$$Q = \left(\frac{71}{4}\right) * \left(\frac{1.5}{2}\right) * \left(\frac{1}{5}\right)$$
$$\mathbf{Q} = 2.66$$

$$\mathbf{De} = \frac{\text{excavation span , diameter or height(m)}}{\text{excavation supprt ratio(ESR)}}$$

NB: Ore drive tunnels are 5m by 5m (lateral) therefore: $\mathbf{De = 3.12}$

The RMR and Q values were computed for each of the 4 boreholes. Borehole HA-026-G in Zone North region gave the highest rating and fall under good rock for RMR and Poor rock with Q system. However, the rest of the boreholes fall under fair rock for RMR rating and poor rock for Q system. This is an indication that the quality of the carbon rich rock mass at Zone 5 differs from one location to the next within the deposit.

Table 1: Results for RMR and Q rating for the 4 boreholes

Location (Zone 5)	Borehole ID	RMR	Q Value
North	HA-026-G	72	3.66
North central	HA-014-G	56	1.9
South	HA-002-G	51	1.7
South central	HA-009-G	49	1.2

The lowest rating borehole (HA-009-G) at Zone South central was chosen to represent the quality of the Carbon rich rock mass for the whole area. This was also account for the worst case scenario when designing a tunnel and the data is summarized in the tables below:

Table 2: summary of rock mass rating

Rock Type	Very good	Good rock	Fair rock	Poor rock	Very poor
Rating	100-81	80-61	60-41	40-21	<20
CAR			49		

Table 10: Rating

Rock Type	Extremely good	Very good	Good	Fair	Poor	Very poor	Extremely poor	Exceptionally poor
Rating	>100	40-100	10-40	4-10	1-4	0.1-1	0.01-0.1	0.001-0.01
CAR					1.2			

Table 3: Adjusted RMR value and Q value for ore drives

Lithological Domain	RMR	Adjusted RMR	Q value
CAR	49	37	1.2

Q SYSTEM CLASSIFICATION SUPPORT REQUIREMENTS

Plotting the Q value of 1.2 against the 3.12 Equivalent dimension value gives category 4 for reinforcement categories which recommends support by systematic bolting, (and unreinforced shotcrete, 4-10cm).

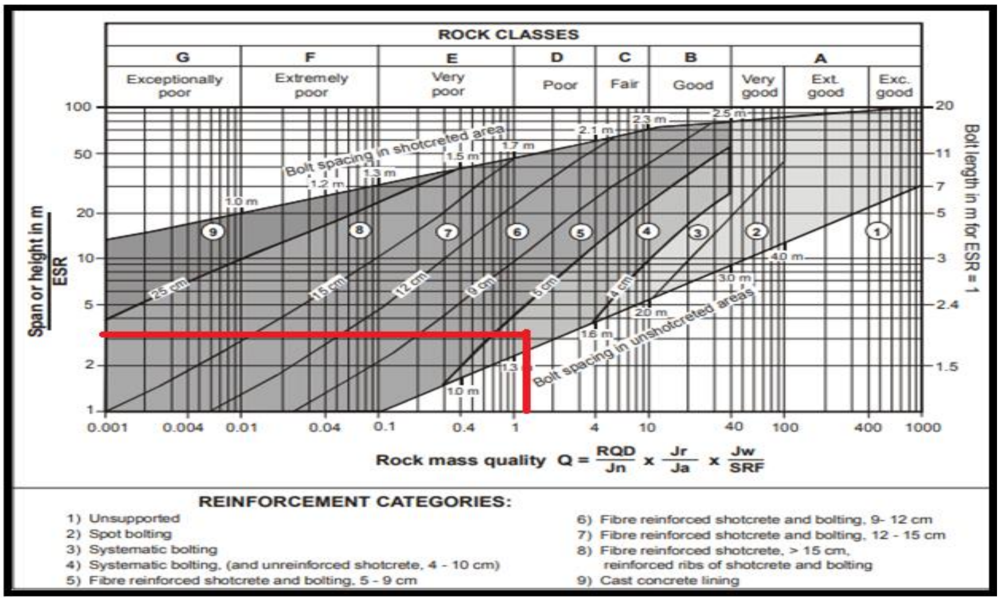

Figure 1: Q value plotted against the De value for support requirement (Grimstad and Barton, 1993)

The suggested support could be due to the fact that the carbon rich rock mass is rated as a poor rock mass and therefore it is expected that it has a low strength hence there is need to provide a lot of support.

The length of rock bolts that can be used is estimated from the excavation width (B) and excavation support ratio (ESR) by the formula.

$$L = \frac{2 + 0.5B}{ESR}$$

Width (B) or Diameter= 5m Hence: **L= 2.81m**

SOFTWARE ANALYSIS

RS2 Analysis

The contour plot of the major principal stress, sigma 1, orientated at 00 from horizontal is the generated and with applied necessary support, as shown in the fig below.

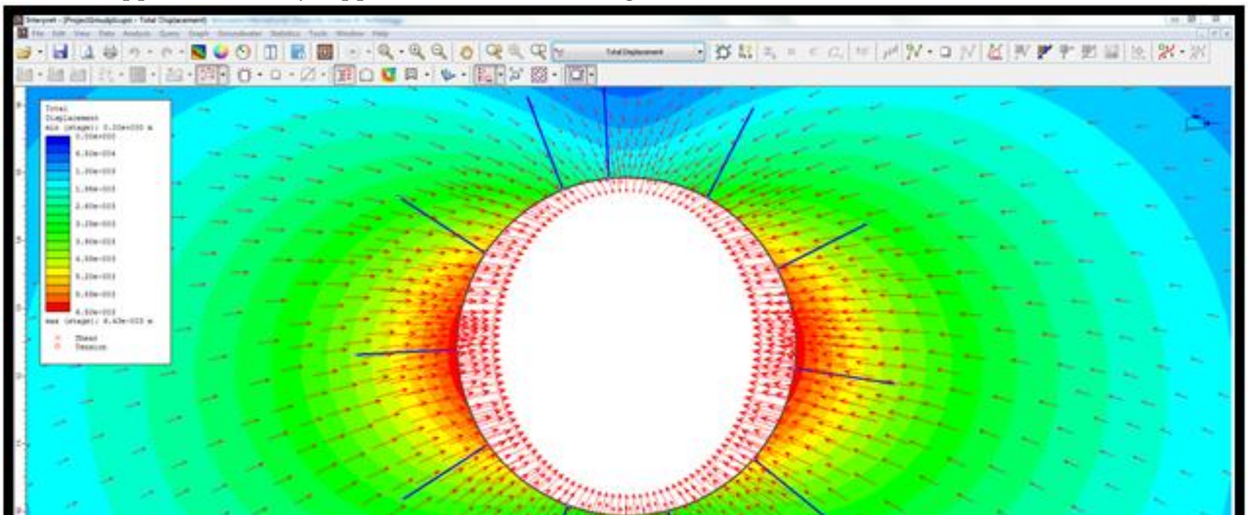

Figure 8: CONTOUR PLOT

The initial anticipated total displacement before support on is estimated to be 6.43mm and from the diagram above shear failure is likely to occur at the roof and floor side while the side walls are under tension. The rock mass strength at the floor and roof side is estimated to be 2.846 (i.e., (0.001) 0.5 x 90Mpa), and the sigma 1, which is taken as induced stress at that position was estimated to be 20.63 MPa, thus giving factor of safety of 0.14 (i.e., 2.846/20.64) at the boundary. However, the factor of safety at the side walls with estimated induced stress of nearly 7.93MPa gives factor of safety of 0.36 at the boundary. This indicate that sidewalls requires more stabilization than floor or roof top since the displacement is likely to occur along the sidewalls and fail under tension than compression at the roof top (See Figure 9).

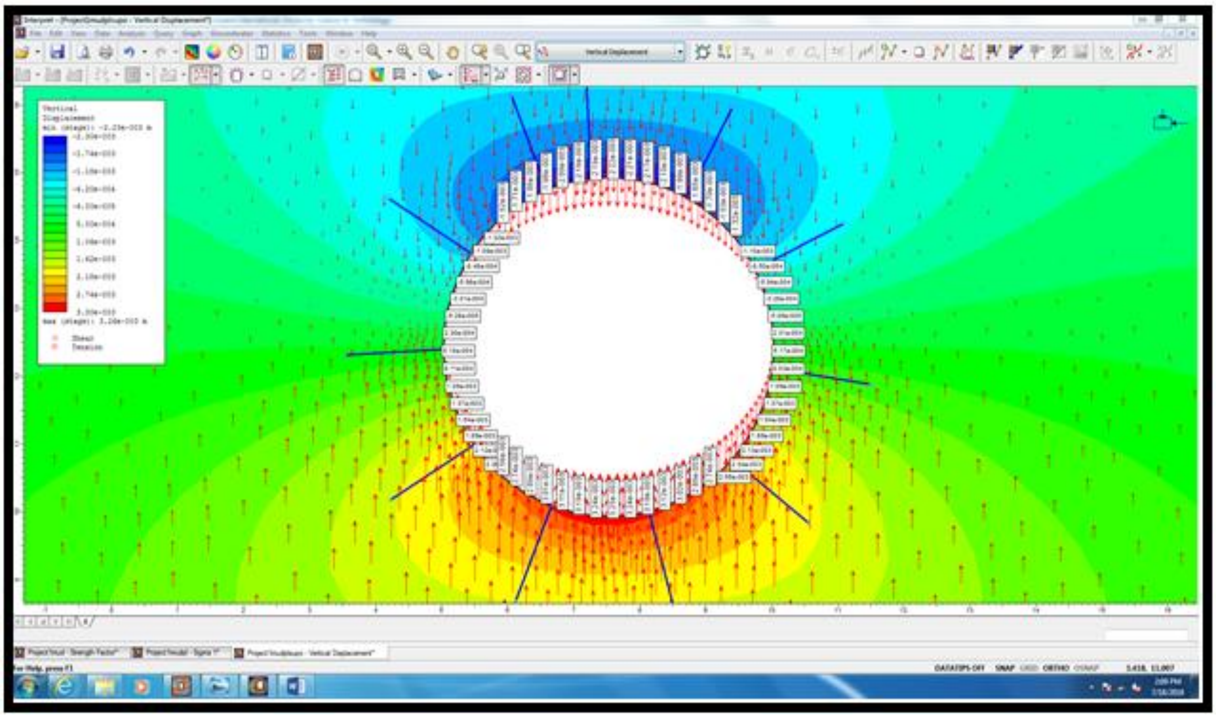

Fig.9: Additional sidewalls stabilization

CAR maximum displacement stands at 6.45mm with initial horizontal movement estimated to 6.42mm and reduced to 6.38mm after bolt installation. Vertical movement is anticipated to be 3.26mm and reduced to 2.1mm after bolt installation. And it can be seen that overall displacement will occur at the side walls. Failure is likely to occur on side walls as the area is highly tensioned compared to the roof top and floor

CONCLUSION

The empirical methods were used to evaluate rock mass quality and estimate the support element required for proposed ore drives and numerical methods were used to assess the response and stability of ore drives before and after support system installation for selection of optimum support systems that will allow safe and profitable extraction. The Carbon rich rock mass of Zone 5 was classified, and it is inferior quality rock.

The recommended support for RMR classification suggested systematic bolts 4-5m long, spaced 1-1.5m in crown and walls with wire mesh combined with Shotcrete 100-150mm in crown and 100mm sides and steel sets light ribs spaced 1.5m where required while that of Q-system suggested systematic bolting, (and unreinforced shotcrete, 4-10 cm).

The observations suggest that the use of two or more classification systems in design and rock engineering, will lead to better and more accurate results in terms of adequately classifying the rock mass and providing sufficient support. This is proved by the more intense support of (steel sets light ribs spaced 1.5m where required) that was suggested by using the RMR classification.

In conclusion the empirical methods show that extraction will be feasible with the selected mining method (Sub-level open stoping).

RECOMMENDATION

On the basis of the present study, the following recommendations are proposed to achieve safety and also maximize on profits:

1. The use two or more classification systems in order to account for the induced stresses subjected to the ore drives and consider the strength of rock mass at the same time.

2. Further investigation on the Carbon rich Units.
3. Suggestion of different mining method other than direct tunnelling on the CAR unit to access the ore.
4. Adoption of an intensive technological monitoring programme on the ore drives to ensure the safety of the workers.

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