Geotechnical assessment of stope design for the SF3 Orebody at Rosh Pinah zinc mine

G.P. Shali[1,2] and A. Vakili[3]

[1]University of the Witwatersrand
[2]Rosh Pinah Zinc Corporation, Namibia
[3]Mining One Consultants, Australia

Rosh Pinah zinc mine is one of the few active underground mines in Namibia and operates at a depth of less than 1 000 m below surface. Overall, a sub-level open stoping mining method with a top-down sequence is employed at the mine.

In order to make informed decisions on the extraction sequence of the SF3 orebody, a geotechnical feasibility study was conducted using traditional empirical methods as well as numerical modelling using non-linear codes. A new approach for non-linear numerical modelling was carried out using StopeX which is a FLAC3D plug-in using the unified constitutive model (UCM) proposed by Vakili (2016). Stope design empirical methods were compared with numerical modelling to assess the dilution at the end of mining. The results of both empirical and numerical modelling suggest that there will be considerable dilution when mining the lower levels with the current total extraction top-down sequence.

INTRODUCTION

The Rosh Pinah Zinc Corporation mine has been in continuous operation since 1969 in the southern part of Namibia, some 18 km from the Orange River and is located at Long.16 46′, Lat. 25 57′ (See Figure 1). The mine produces zinc and lead concentrates as well as silver, copper and gold as by-products and with a mill capacity of 2 000 tonnes per day. A sub-level open stoping mining method with a top-down sequence is employed to mine the orebody that is 500 m below surface.

The orebody is hosted between two faults, namely, the SF3 and the Northern fault. The dominant ore types are carbonate and Arkose/Breccia. The carbonate ore is more dominant in the upper levels and the ore becomes more Arkosic in lower levels. SF3 is mostly a thin orebody with a number of discrete lenses.

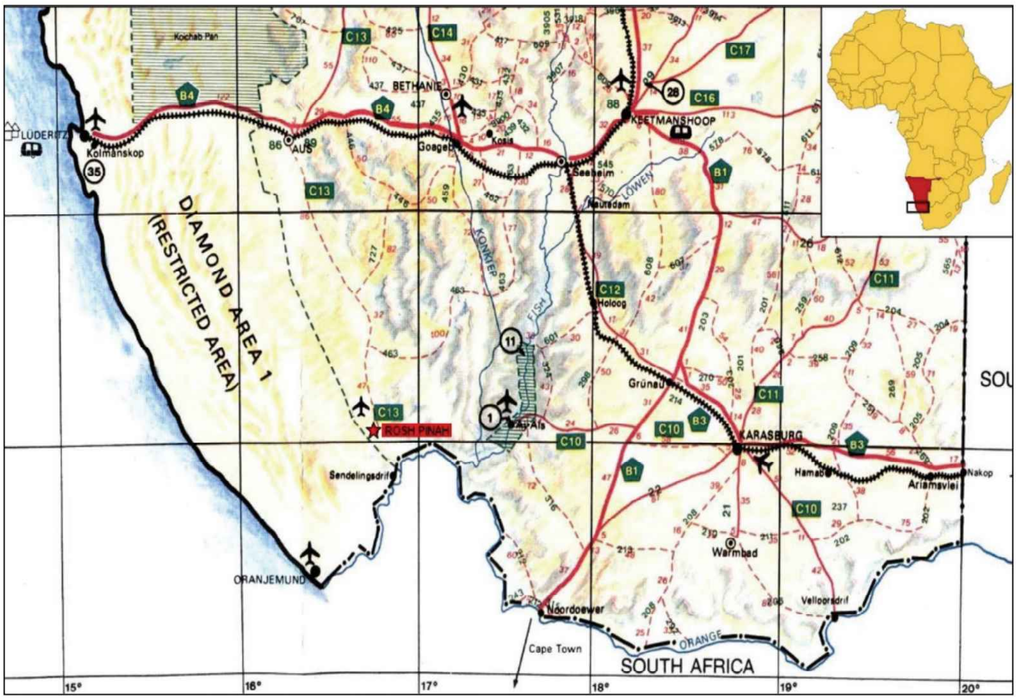

Figure 1. Locality map for Rosh Pinah Zinc Corporation

The aim of the study is to carry out a geotechnical investigation of the SF3 ore body at Rosh Pinah Zinc Corporation mine. A detailed geotechnical investigation of the SF3 ore body will include underground mapping within the infrastructure and sill drives on all three levels, namely the 20, 50 and 110 levels. The data obtained from field mapping will then be used for stope designs as well as further analysis to be done to obtain parameters that will be used for numerical modelling.

Furthermore, a review of the upper mined out levels of SF3 will be back-analysed to provide assumptions on the stability of the proposed stope to be mined at the lower levels.

GEOLOGICAL MODEL

The geometry of the SF3 deposit is shown in Figure 2. The SF3 ore body is hosted in Arkose rock with carbonates as the rock type that contains the orebody. The orebody has a strike length of 170 m in the NE in the upper levels and reduces to 90 m at the lower levels of the ore body. The width of the ore body is in the range of 7 m on the upper levels and slightly enlarges on the lower levels to an average of 25 m. The upper levels were mined at the depth of 270 m and the lower level in situ ore will be mined at the depth of 495 m.

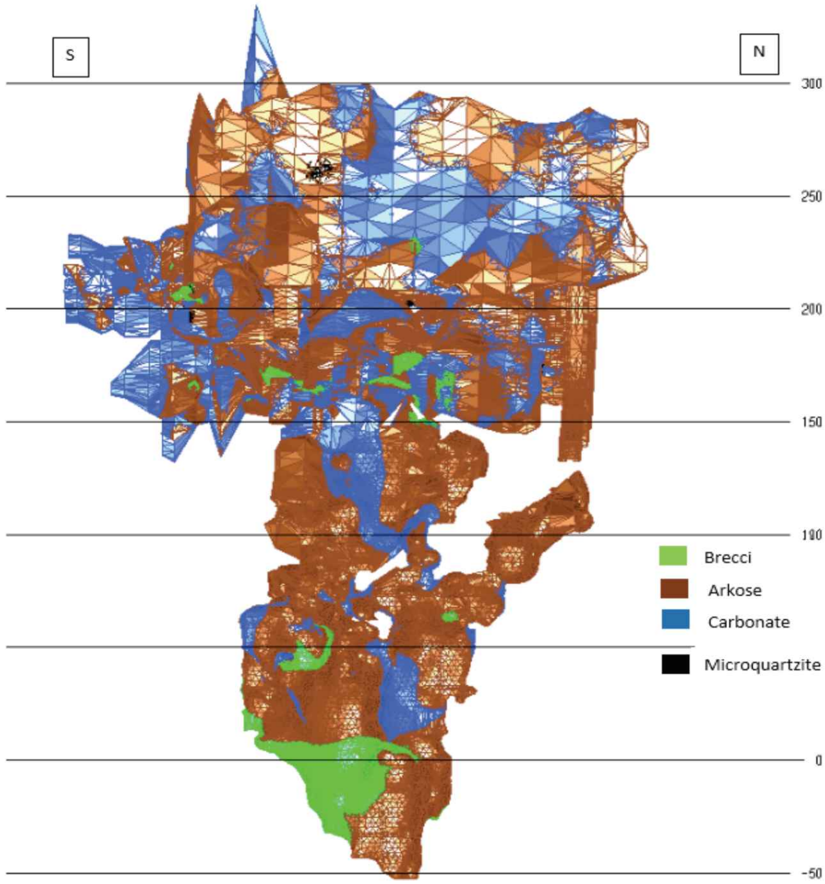

Figure 2. NS section illustrating the geology of the SF3 orebody of Rosh Pinah zinc mine

ROCK MASS MODEL

A geotechnical rock mass model was constructed. The 3D geological modelling software programme Leapfrog, was used, using geotechnically logged boreholes for the purpose of understanding the geotechnical domains of the SF3 orebody. The rock mass model consists of geometry of the orebody and large-scale geological structure.

In situ stress

The in situ stresses at Rosh Pinah zinc mine have been estimated using the CSIR hollow inclusion (HI cell) method. The test was carried out in the western ore field at -30 level at a depth of 495 m below surface. The site for testing was far away from mined-out stopes with an excavation of intact back and side-wall and the hole is in virgin ground conditions. According to Handley and Piper (2014), an ideal location for stress measurement should be located in good, homogenous, fine grained, isotropic rock mass that is located away from geological structures, sheared and further away from mined-out stopes. Figure 3 indicates the location where stress measurement took place at WF3 -030.

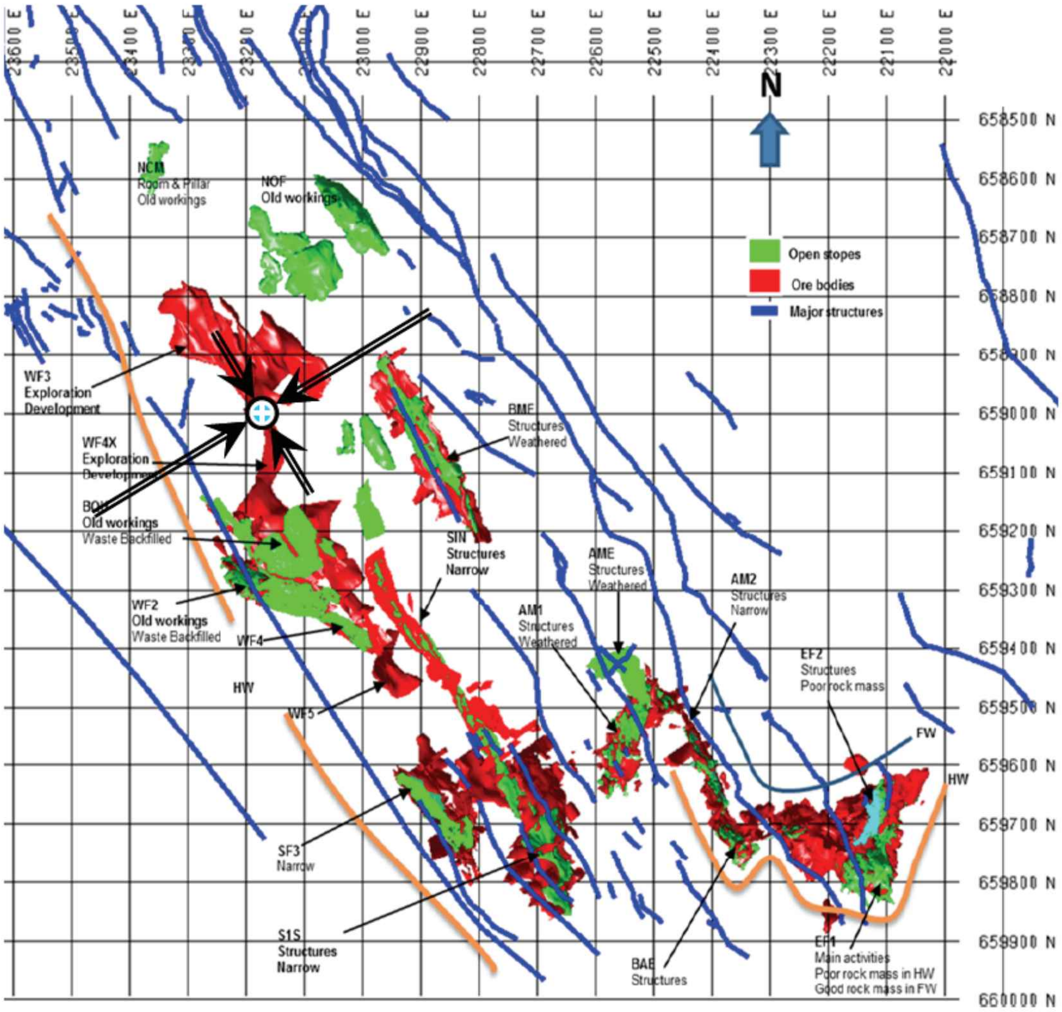

Figure 3. Plan of the mine showing location and detail of the second stress measurement showing major geological trends, stoping and orebodies. After Handley and Piper (2014)

The orientation of the borehole has a dip of 60°and dip direction of 135°. Table I shows a summary of the in situ stress conditions for Rosh Pinah zinc mine.

The results from the two stress measurements show a large variation of all three principal stresses. The major principle stress for stress one measurement is twice greater than that of stress measurement two. According to Handley and Piper (2014), both measurements are considered to be of good quality and reliable despite the fact they differ in magnitude. Stress measurement two was used because its virgin stress corresponded to that of the expected virgin stress of 13 MPa. According to SRK (2012), the bearing of stress measurement two also corresponded to that of sigma one stress field that was estimated in the SRK report to be 142.6 °- as derived from the Okiep measurements near Black Mountain in the Western Cape of South Africa and carried out by Wesseloo and Stacey (1998). Underground observations on site indicate a low stress environment at current depths for the SF3 orebody and no signs of stress damage were reported.

Table I. In situ stress regime for Rosh Pinah zinc mine

Stress Measurement One			
Stress components	Gradient (MPa)	Plunge (°)	Trend (°)
σ_1	0.117	34	342
σ_2	0.069	39	106
σ_3	0.026	32	227
σ_v	0.073		
Stress Measurement Two			
Stress components	Gradient (MPa)	Plunge (°)	Trend (°)
σ_1	0.044	14	135
σ_2	0.029	59	21
σ_3	0.018	27	232
σ_v	0.028		

Mine scale structures

The SF3 ore body is situated between the northern fault as well as the SF3 fault. The hangingwall is bounded by the SF3 fault whilst the footwall is bounded by the northern fault. Geological mapping data as well as borehole data were used to create the 3D structures in Leapfrog. More refined structures with the aid of level maps from the geologists provided an improved structural model for the SF3 ore body. The structures were mapped and interpreted over at least one mine level to determine the continuity of the structures.

CONCEPTUAL STOPE DESIGN

The main focus of the study is to review the stability of each stope and assess the possibility of dilution. The assessment utilised the Matthews stability number to determine the spans at which the stopes will be stable and safe to mine. Finally, the stopes were evaluated using a non-linear numerical modelling model to determine the global stability.

Figure 4. Drill core showing lithological units of the SF3 ore body

Rock mass classification

A rock mass classification was carried out using borehole data as well as mapping data. All the infill holes drilled from the geological drilling programme are geotechnically logged by the geotechnical engineers at the mine. Table II shows the summary of the rock mass classification parameters derived from geotechnically logged bore holes and used in the designs.

Table II. Lithological domains and geotechnical rock mass parameters

Domain	Ja	Jr	RQD	GSI
Arkose	5	2.5	80	67
Carbonate	1	3	100	89
Micro-quartzite	1	2	100	84
Breccia	1	3	100	89

Table II indicates that the host rock which is Arkose and is massive has an RQD value of 80 and a GSI of 67. Rock strength tests were carried out by Roclab. The UCS of the host rock mass was estimated to be 223 MPa. Figure 4 shows different lithological units comprising the SF3 ore body. The carbonate ore was estimated to have a UCS of 150 MPa which indicated a competent rock. More competent ground conditions can be found at the bottom stopes from level 20 to -040 comprising of Arkose and Breccia.

Figure 5. Whisker plots of Uniaxial Compression Test values

Empirical stope design methods

Empirical stope design method used at Rosh Pinah mine follows the extended Matthews method after Mawdesley to assist in determining the stability of open stopes. The applicability of empirical stope designs was confirmed during mining of the western ore field Three. However, these methods have not been tested in the SF3 ore body.

The following equations are used to define the hydraulic radius, HR, and the stability number, N, for the stability number (after Potvin 1988).

$$HR\ (m) = (\text{stope face area}) / (\text{stope face perimeter}) \quad [1]$$

$$N' = Q' \times A \times B \times C \quad [2]$$

Where Q′ is defined by:

$$Q' = (RQD/Jn)\ (Jr/Ja) \quad [3]$$

Table III. Stope stability in put parameters for SF3 ore body

SHAPE AND GEOMETRY									
segment	**H/W dip**	**strike length (m)**	**true H/W height (m)**	**H/W perimeter (m)**	**H/W surface area (m^2)**	**HR**	**n'**	**stability**	**mining levels**
HW	60	60	21	162	1260	7.8	1.1	Yes	Level 110
SW	90	7	21	56	147	2.6	2.9	Yes	
Backs	0	60	7	134	420	3.1	0.7	Yes	
HW	60	20	24	88	480	5.5	6.8	Yes	Level 110-2
SW	89	9	24	66	216	3.3	14.4	Yes	
Backs	19	20	9	58	180	3.1	2.8	Yes	
HW	57	62	54	231	3323	14.4	8.2	No	Level 080
SW	90	57	6	126	342	2.7	16.3	Yes	
Backs	0	62	7	138	434	3.1	4.1	Yes	
HW	67	62	67	258	4154	16.1	9.2	No	Level 050
SW	90	62	6	136	372	2.7	16.6	Yes	
Backs	0	6	67	146	402	2.8	4.6	Yes	
HW	57	55	127	364	6985	19.2	11.3	No	Level 020-1
SW	90	27	127	308	3429	11.1	30	Yes	
Backs	0	55	27	164	1485	9.1	7.5	Yes	
HW	57	31	60	182	1860	10.2	16.2	Yes	Level 020-2
SW	90	6	60	132	360	2.7	27.4	Yes	
Backs	0	35	6	82	210	2.6	18.3	Yes	
HW	57	80	166	492	13280	27	16.2	No	Level -010
SW	90	8	166	348	1328	3.8	27.4	Yes	
Backs	0	80	8	176	640	3.6	18.3	Yes	

The rock quality designation (RQD), the joint set number (Jn), the joint roughness (Jr)and the joint alteration number (Ja) are shown in Table 3. Table 3 shows the input parameters used in the stability graph method.

The N stability number is derived from factor A, which is the stress factor, B which is the joint adjustment factor and C which is the gravity factor. Table IV shows the values of A, B and C constants estimated for the SF3 orebody. The stress factor A was determined by obtaining the maximum induced stress acting at the centre of the excavation after excavating the respective mining step and replicating the actual mining sequence in FLAC3D.The ratio of the strength of the rock and induced stress is then calculated and extrapolated on the graph to obtain a value for the A factor using the strength of the host rock.

The ease of block fallout is calculated using the stereo net for factor B. Consideration is given to the most critical joint set angle respective to the stope face angle. The gravity adjustment factor C was calculated taking into consideration the mechanism of failure around the stope walls. The most dominant failure mode was identified to be gravity driven and slabbing. Table IV shows the values of the adjustment factors used to estimate the stability of the SF3 orebody.

Table IV. Stope stability in put parameters for SF3 orebody

Stope wall	A		B		C	
	Min	Max	Min	Max	Min	Max
HW	0.1	1.0	0.2	0.5	2	6
Sidewalls	0.1	1.0	0.2	0.4	4	8
Backs	0.1	1.0	0.2	0.6	2	8

According to Villaseca (2014), the stability graph method is used as a design tool however the system has a number of limitations that must be understood be understood in order to assess its applicability in any particular geotechnical environment. The HR values of the designed stopes were obtained by accumulating the stope dimensions as mining descends down wards. The HR was found to be somewhat difficult to measure accurately as a result of the irregular ore body geometry but was nonetheless estimated with Minesight

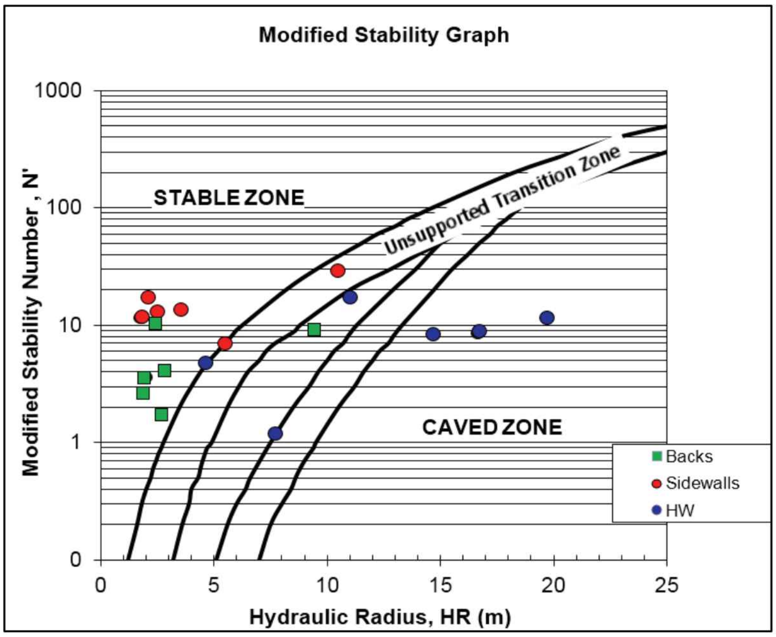

Figure 6. Stability graph for unmined stopes. After Nickson (1992), from Potvin and Hadjigeorgiou (2001)

Mining has taken place at the upper stopes from level 230 down to level 140. Hence the stability graph in Figure 6 only indicates stope from level 110 to level -010 with level -040 hangingwall stability completely beyond the graphs caved-zone district. The initial mining steps taking place at 110 level are expected to be stable before the mining of the levels beneath however support can be assessed in terms of cable anchors to support the hanging walls. According to Nickson (1992), the graph cannot be used to design cable bolt hangingwall spans were the cables may be installed from drill drives. Mining level 110 stope will be undercut by level 080 which will result to a higher HR of 14.4 hence this stope plotted in the caved zone as shown in Figure 6 above for the hanging wall. Furthermore, mining level -010 will result to a very high hydraulic radius which plots further within the caved zone. It is recommended that a stope scan using the cavity monitoring system be carried out after mining level 110 to determine the overbreak of the stope and validate the design assumptions

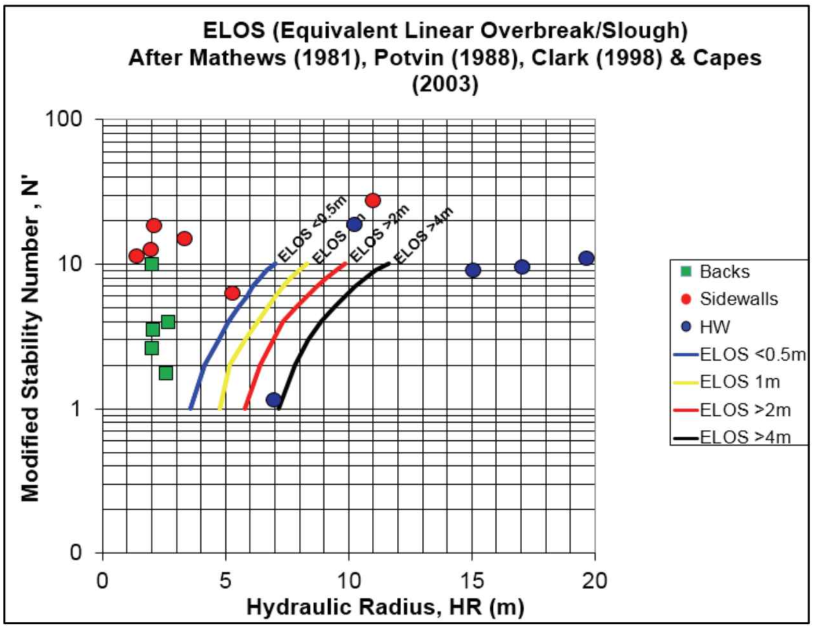

Figure 7. Graph showing the equivalent linear overbreak slough (ELOS). After Mathews (1981), Potvin (1988), Clark (1998) and Capes (2003)

The ELOS proposed by Clark and Pakalnis (1997), was used to assess the dilution of stopes. Figure 7 indicates that the mining level 110 is below <0.5 m, which is within the limits of acceptable overbreak. Mining level 80, which undercuts Level 110, results to an ELOS of greater than 4 m. An ELOS greater than 4 m indicates a major overbreak is likely to occur and in most cases an overbreak beyond an ELOS of 4 m does not justify to design a stope of such dimension.

The following categories have been used to determine stope performance:

- Stable – depth of failure 0.0 to 2.5 m.
- Transition – depth of failure 2.5 to 4.0 m.
- Unstable – depth of failure > 4 m.
-

The results shown in Figure 7, indicate that an ELOS of greater than 4 m may be expected when mining level 80 with a stope height of 30 m cumulatively, as mining progresses downwards. Further representation of stopes mined to level 20 with a cumulative height of 90 m shows a much larger ELOS, greater than 4 m. Consideration will have to be made to leaving a pillar at level 50 to reduce the mining span.

Numerical modelling

Numerical modelling was conducted using FLAC3D which is a 3D finite difference software from ITASCA Consulting Engineers. The SF3 Orebody was modelled using the improved unified constitutive model (IUCM) which can better and more accurately predict the stress/strain relationships of the rock mass or intact rock samples in continuum numerical models than conventional constitutive models, Vakili (2016).

Constitutive model

The SF3 ore body was modelled using the IUCM proposed by Vakili (2016).According to Vakili (2017), the IUCM is a result of collating the most notable recent research works in the area of rock mechanics and extensive back-analysis of mining case histories together into a unified material model. Vakili also emphasised that in developing this model, the use of any new theory or technique was avoided where possible, and all the adopted processes were based on well-accepted and widely applied rock mechanics

techniques and theories. Table 4 shows the derived rock strength input parameters for the WF3 orebody situated 300m NW of SF3 orebody. No rock strength tests were done for the SF3 ore body.
The key input parameters used for anisotropic rocks are:

- Elastic modulus of intact rock.
- UCS of the intact rock.
- Anisotropic factor.
- mi $_{min.}$
- mi $_{max.}$
- GSI.
-

The assumptions used are such that the cohesion is zero as well as the tensile values at residual state. The frictional angle is assumed to be 45°. A detailed and transparent description of the IUCM components are presented by Vakili (2016). This includes step-by-step instructions on the procedures that were implemented in each of the model's algorithms. Vakili (2017) showed comparisons between the IUCM model results and other conventional models (i.e. Elastic and Mohr- Coulomb) for several well documented case histories.

Table V. Input parameters used for forward analysis for the SF3 ore body

Rock Strength Properties									
Rock Unit	**Density (t/m3)**	**E_i (GPa)**	**UCS (MPa)**	**mi_{Max}**	**mi_{Min}**	**Anisotropy Factor**	**Dip (°)**	**Dip Direction (°)**	**GSI**
Arkose	2.69	72	292	15	7.5	-	-	-	67
Breccia	3	72	232	15	7.5	-	-	-	89
Carbonate	3.83	62	139	20	10	-	-	-	89
Micro-quartzite	2.76	58	163	12	6	1.4	75	235	84

MODELLING CONSTRUCTION AND APPROACH

The geometry of the model indicating the topography as well as the ore body geometry is indicated in Figure 7. The topographic surface, geotechnical domains and orebody geometries were included in the model. A mesh of 3 m × 3 m×3 m was used to envelope the ore body. This was used in order to obtain more satisfactory results whilst increasing the mesh the mesh size beyond the 3 m. The far field zone size of 48 m was used which is the largest distance from the outer boundaries of the model. These parameters were considered to minimise the run time of the model whilst taking into consideration the level of accuracy that is required to run the model.

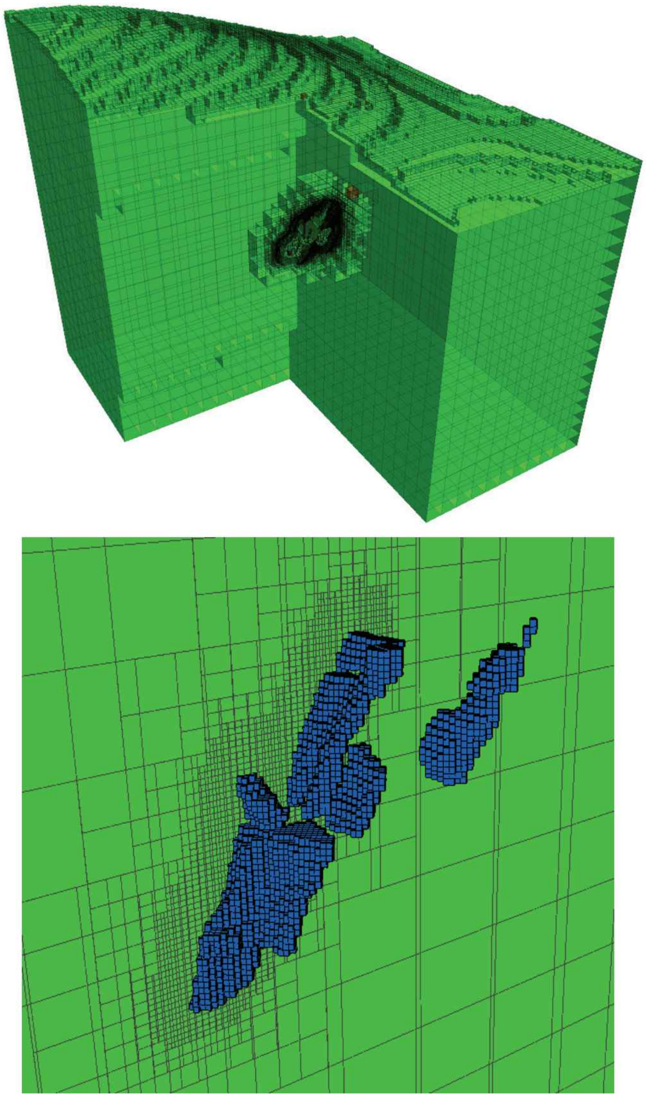

Figure 8. Geometry of model indicating the topography as well as the orebody geometry

Numerical modelling was undertaken in 10 different phases as shown in Figure 8. The model sequence used is top to bottom approach as used at the mine. Mining has taken place at the upper levels from 230 to 140. A similar mining sequence approach is planned for mining level 110 to - 010. No backfill was previously used at the mine and there are no plans for using backfill in the future since mining is carried out as a total extraction and a natural pillar can be left between levels to reduce stope span. The ore body geometry together with the main lithologies such as Arkose, Micro-quartzite and Breccia were included in the model with Micro-quartzite allocated anisotropic properties. Carbonate ore is regarded as the host rock mass.

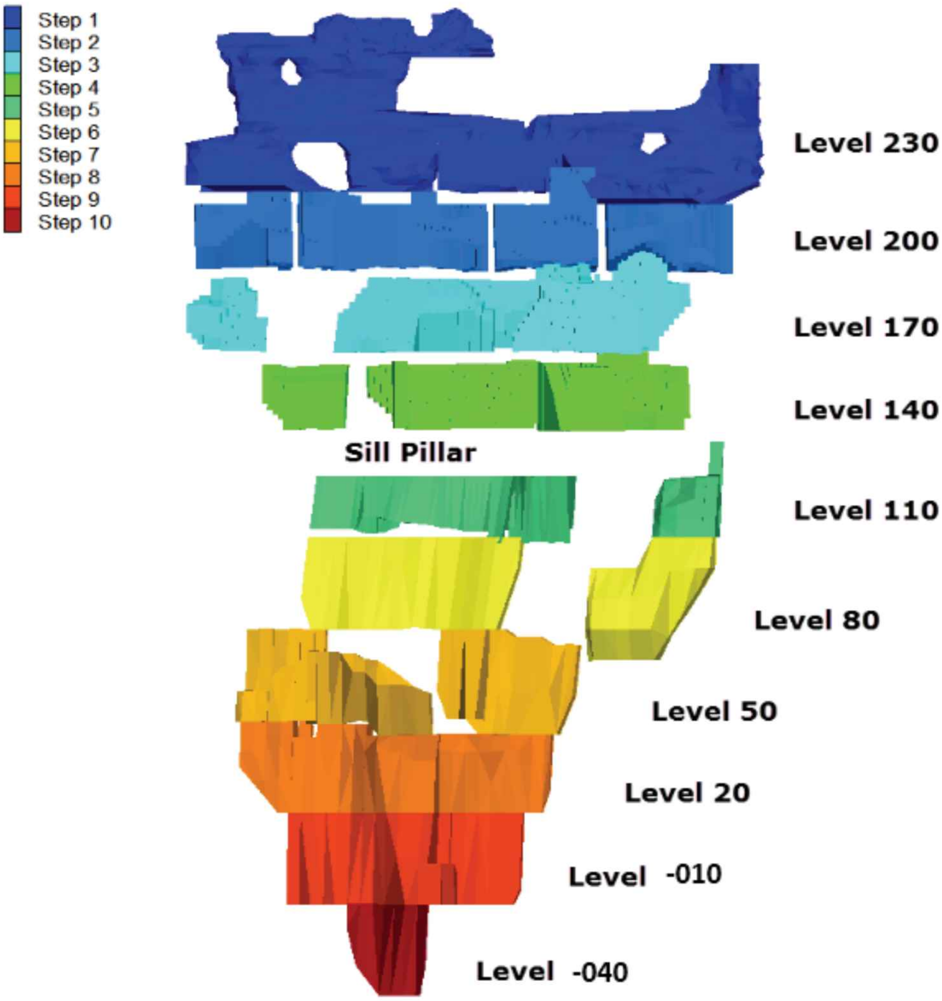

Figure 9. Front view showing the model sequence

NUMERICAL MODELLING RESULTS AND INTERPRETATION

The model output was examined using volumetric strain. By definition, volumetric strain is the change in volume due to a deformation (ΔV/V) which can be calculated by summing minor, intermediate and major principle stains. According to Watson et al (2015), experience in numerical back analysis at several mining operations showed that a model volumetric strain of between 1% to 3% often generates similar or close to overbreak/breakout volumes to those obtained from actual underground excavations. The volumetric stain of 0.5 %, 1% and 3 % were calculated as shown in Figure 10. It was found that more dilution was expected for 0.5 % volumetric strain with about 20.6 % dilution. The lowest dilution was expected with a 3 % volumetric strain of about 12.3 % dilution. According to Le Roux (2015), excess dilution of greater than 10 % can reduce the recovered grades of the ore mined.

Figure 10 shows the isosurface volumetric strain of 0.5 % which has resulted to the most dilution of 20.9 %. The figure below also indicates that most of the dilution will be occurring at the lower levels of the orebody towards the end of mining the SF3 020 as well as SF3 -010 blocks.

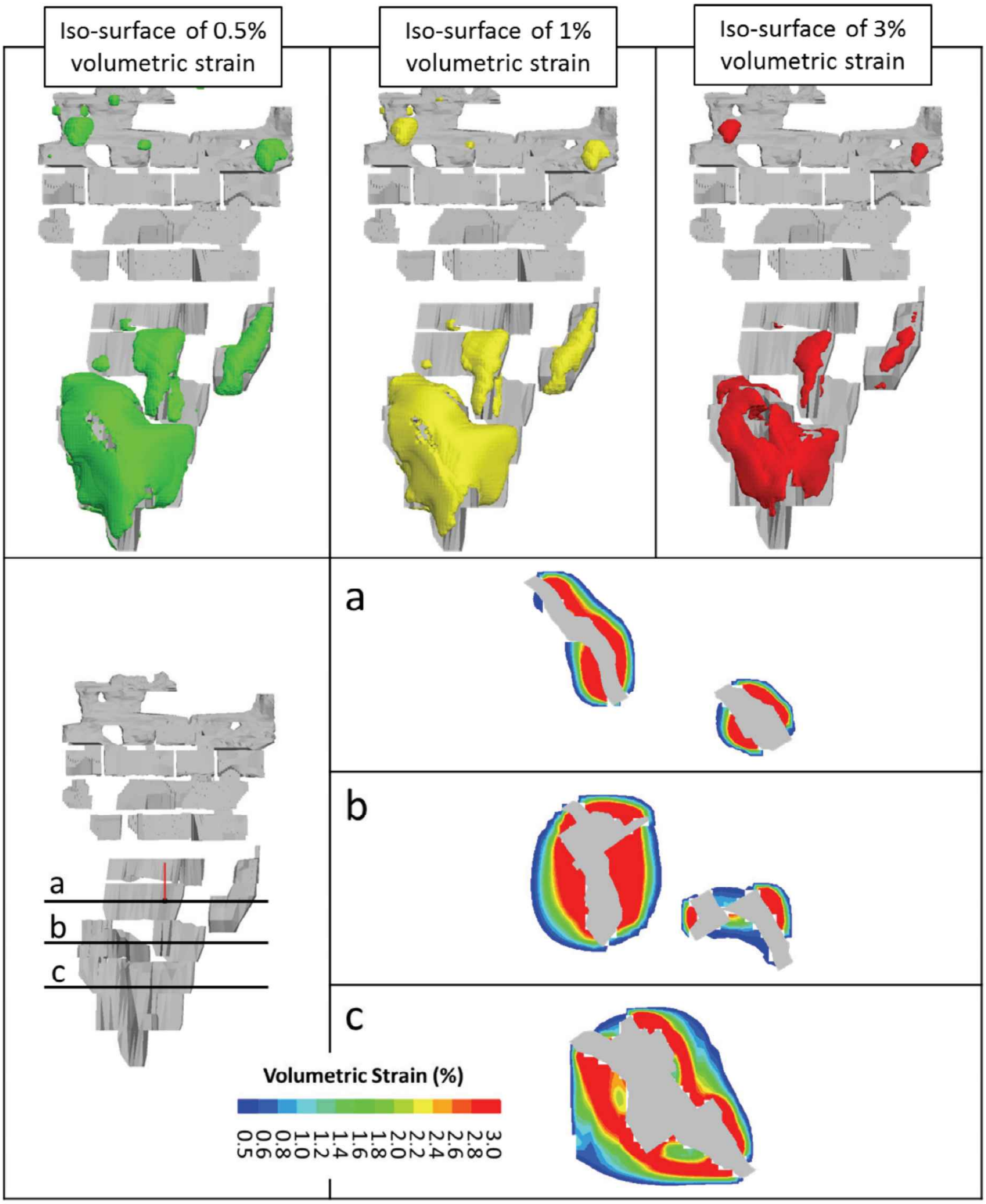

Figure 10. Isosurface of volumetric strain

Another parameter that was used to assess the stability of the stopes was the minor principle stress. As the minor principle stress approach zero the stope instability becomes prone to gravity since the rock mass is free to dilate. According to Martin et al (1999), confining stress expressed as sigma three, may be a good indicator for predicting the amount of dilution. The zone surrounding the mined stope walls is close to zero means that there are no confining stresses and rock may fall due to gravity. Figure 11 show the possibility of high dilution both as shown vertically and horizontal in cross sections.

Once the stopes are extracted, a CMS can be used to back analyse and calibrate the model. The CMS can be used to calibrate the model with the volumetric strain by accordingly adjusting the percentage of volumetric stain and see the correlation

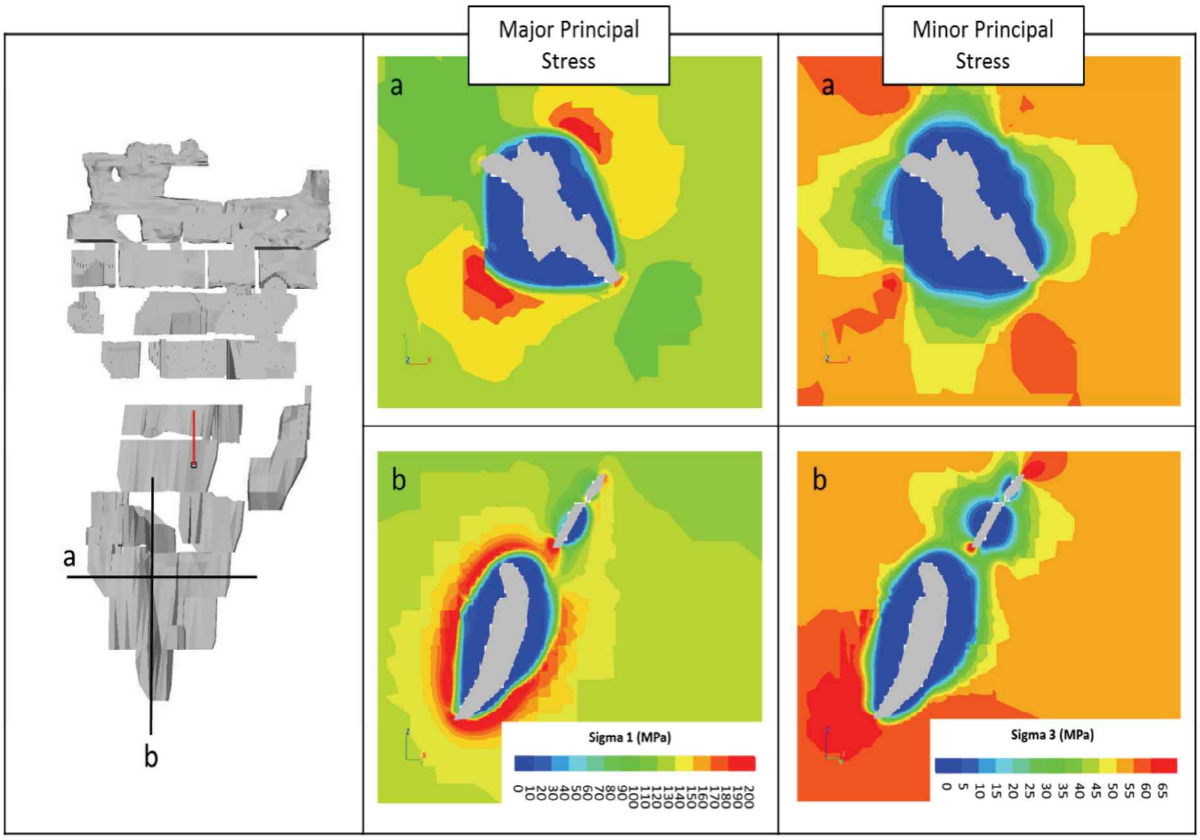

Figure 11. Visual representation of the minor and major principal stress

CONCLUSION

The rock mass of the ore body of SF3 ore field is of good quality and little dilution has been encountered at the upper levels of the ore body between 140 and 280 levels. The use of empirical modelling shows that the lower portion of the ore body can be mined with a maximum HR of 13 comprised of stope width of 58 m and height of 30 m. However, according to the numerical modelling results, it shows that the stopes are likely to be stable and no pillars will be required to partition the stopes into more stable sizes. The use of numerical modelling is justified in the upper stopes that were mined previously with minor dilution of stope overbreak with the no major dilution predicted.

It is recommended that various methods for dilution minimization be employed as soon as mining takes place. Cable bolting can be an alternative to minimise dilution. Furthermore, instruments such as extensometers can be installed into the hangingwalls to measure deformation as well as calibrate numerical models. Regular CMS scans should also be carried out to verify both empirical and numerical models.

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George Shali

Rock Mechanics Engineer
Rosh Pinah

George Shali has a GDE in Rock Mechanics as well as a Strata Control certificate and is currently completing his MS in Mining Rock Mechanics in University of the Witwatersrand. He is currently employed by Rosh Pinah zinc mine in Namibia as a rock mechanics engineer. Since 2015 he has been involved in various geotechnical roles in both open pit and underground , including at Dundee Precious Metals, Chelopech, Bulgaria, Otjihase Mine, Weatherly Namibia, as well as Swakop Uranium, Husab mine.