Konkola Copper Mines plc

Konkola Mine
No. 3 Shaft
Rock Engineering Section

Date: 06/03/2012 **GT/2012**

Geotechnical considerations in the design of the PPCF Cut 2 /SLC hybrid mining method on 2340L, 4300mN-4500mN area at Konkola Mine No. 3 Shaft

Abstract

The performance of the Post Pillar Cut and Fill (PPCF) mining method has not been satisfactory to date due to the low total ore recoveries recorded ranging between 60-65% as a result of ore losses within the production stoping panels. There has been a cumulative loss of significant amount of ore over a number of mining blocks as mining progresses down dip. A trial run for a total extraction mining system has been devised to recover the last slice forming a closure area between 2270L and 2340L Cut 2 in the 4300mN-4500mN block. The hybrid mining system is expected to improve total ore recovery up to 80% with 10% as sub-grade material and therefore will enable the extraction of about 70% of ore without dilution. Against such background, it has been decided to review the mining aspects related to the proposed use of the current PPCF mining method in combination with SLC mining method to take the remaining ore forming a closure between 2340L PPCF Cut 2 and 2270L.

In order to successfully extract the ore, a geotechnical review of the hangingwall stability analysis in the 4300mN-4500mN stoping panel has been conducted. The aim is to highlight the inherent risks associated with the operations and implementation of remedial measures that will reduce or eliminate the risks. The rockmass in the area is in a fairly competent rockmass (ore) with Rockmass Rating (RMR) = 57 and Q-System = 4.44 and require systematic bolting.

Hangingwall stability analysis using the Mathews Stability Graph method in Figure 1 on page 7 indicate that the roof span is in the supported transition zone, implying that the excavation will only be stable with support, but as the hydraulic radius increases towards the extraction of the entire area between 4300mN-4500mN stoping panel, the stability of the roof span will fall rapidly into the caved zone. The primary mechanism of rockmass failure is that of buckling of laminated rock strata.

It is therefore recommended that the recovery of the remnant sill pillar (closure area) between the 2270L and 2340L Cut 2 in the 4300mN-4500mN mining block should continue with systematic primary support rehabilitation of 2.4m long permasets in the roof and sidewalls to be installed at 1.0m ring and bolt spacing on a square grid with spot-bolting where support is missing.

All standard mining procedures and operating guidelines should be strictly adhered to before and during the operations. However, it is not recommended to conduct any remnant sill pillar recovery activities beyond the 4300mN and 4500mN areas using the proposed PPCF/SLC hybrid mining method due to the severe rockmass deterioration and ground damage in the stoped out panels, resulting into excessively wide spans and slender pillars between the 4300mN–3600mN stoping panels.

Introduction

The report discusses the preliminary geotechnical criteria through empirical and observational rock engineering design principles that will be used to derive the mining standards for the PPCF/SLC hybrid mining method. The geotechnical risks are identified and taken into account to develop the stope design parameters, stoping sequences and support requirements. A monitoring programme to assess the ground conditions as mining progresses is also included.

Below are the areas that have been earmarked for the trial of PPCF/SLC hybrid mining method:

a) 2340L, 4400mN-4500mN Area

The stoping panel has been mined to cut 2 position. The stoping panel appears to be stable though with minor falls of ground in some portions from the roof and pillar sidewalls. Backfilling with waste rock should be done in order to confine the pillars sidewalls and the exposed updip sidewall in the footwall drive. Install timber packs in the pillar crosscuts position opposite the access crosscut to the slot raise to provide the required passive support and hence ensure guaranteed stability in the area once slot and stope rings blasting commence.

Spot-bolting using 2.4m long permasets should be done before stoping commences in order to control immediate rockmass deformation and preserve the integrity of the rockmass and immediately arrest the induced stresses and stress changes that will result from the stoping out operations.

b) 2340L, 4300mN-4400mN Area

The stoping panel has been mined to cut 2 position. The 2340L, 4300mN access crosscut to lode (cut 2) experienced both structurally controlled and stress related failure/damage at the FWC/Orebody contact. Due to the long stand-up time of the footwall and hangingwall drives in the area, ground conditions have significantly deteriorated, with increased severity of ground damage to excavations ranging from minor blocky falls at the weak GFW/unit 'A' contact to severe pillar sidewall spalling, hangingwall collapse and floor heaving in some places. This has resulted into excessive spans between pillars across the full width of the orebody. The ground damage can be attributed to delayed backfill placement in the 4100mN-4300mN stoping panel, allowing pillars to deteriorate and soften, and thereby shedding load to the adjacent stiff remnant 4300mN-4400mN stoping panel.

Backfilling with waste rock in the 4300mN-4400mN stoping panel should be done expeditiously in order to confine and stabilize the pillar sidewalls (increase the pillar load bearing capacity) and the exposed weak GFW/unit 'A' contact on the updip sidewall of the footwall drive.

Rockmass Condition in the Trial Area

The results of scanline mapping of the discontinuities in the area revealed the following information tabulated below:

Dip/Dip Direction	**Set Type**	**Description (Spacing/Condition)**
20˚/270˚	Bedding	Orebody is thinly to thickly bedded, with bedding planes being planar and smooth. Some beds are filled with kaolin and iron oxide infill. Average bedding spacing is 0.1-0.35m.
85˚/008˚	Joint Set 1	Average joint spacing is: 0.5m – 1.5m.Joint planes are planar and smooth with calcite infill. Aperture size: < 1mm
77˚/066˚	Joint Set 2	Average joint spacing is: 2.0m – 3.0m.Joint planes are planar and smooth with quartz and calcite infill. Aperture size: < 1mm

Table 1: Joint Survey of the rockmass condition in the 4300mN-4500mN block

a) The following data collection is a calculation of the **Geomechanics RockMass Rating (RMR)** system for mining:

Parameter	**Condition/Value**	**Rating**
UCS/Intact Rock Strength	120-200MPa	12
Rock Quality Designation (RQD)	60	13
Joint Spacing	Bedding = 0.1-0.35m. J1 = 0.5m – 1.5m J2 = 2.0m – 3.0m (Rated based on the most critical bedding plane affecting the stability of the stope hangingwall).	10
Condition of discontinuities	Unidirectional, smooth and planar	15
Ground water	Dry to slightly wet	7
Total Rating	**Fair rock**	**57**

Table 2: Rockmass Rating System (After Bieniawski 1989)

b) The following data collection is a calculation of the **Rock Tunnelling Quality Index, Q**

Parameter	Condition/Value
Rock Quality Designation (RQD)	60
Joint Set number (J_n)	Three joint sets = 9
Joint Roughness number (J_r)	Zones containing calcite/clay minerals thcick enough to prevent rock wall contact = 1
Joint Alteration number (J_a)	Silty, iron oxide or sandy–clay staining/coating = 3
Joint Water Reduction Factor (J_w)	Dry or moist excavation = 1
Stress Reduction Factor (SRF)	UCS/Insitu principal stress = 150MPa/0.027Z= 150/0.027*690 = 8.05 (High Stress and may be unfavourable to wall stability) with a rating of 0.5-2.0 SRF = 0.5
Total Rating	**Q = (RQD)/(J_n) * (J_r)/ (J_a) * (J_w)/ (SRF)** **Q = (60/9)*(1/3)*(1/0.5) = 4.44** **Fair rock**

Table 3: Classification of individual parameters used in the Rock Tunnelling Quality Index, Q (After Barton et al, 1974)

From the results of structural mapping and data from core logging of the rockmass, Q = 4.44 for fair rock. Based on the existing empirical data from case histories, the modified Q' assumes that both the Joint Water Reduction Factor (J_w) and the Stress Reduction Factor (SRF) are equal to one i.e. (J_w)=(SRF)=1. Therefore, **Q' = (RQD)/(J_n) * (J_r)/ (J_a) = (60/9)*(1/3)= <u>2.22</u>**

Hangingwall Stability Analysis

The Modified Stability Number, N', is given by:

N' = Q' * A * B* C

Factor A - Rock Stress Factor = UCS/σ_1 = 150/(0.027 *690)= 8.05MPa

For 2 < UCS/σ_1 < 10, A= 0.1125*(UCS/σ_1)-0.125

A = (0.1125*8.05)-0.125=<u>0.78</u>

Factor B – Rock Defect Orientation Factor- The most critical discontinuity controlling the stope surface stability in this design is the bedding plane separation with 20˚/270˚.

Using the joint/discontinuity orientation adjustment chart: **<u>B = 0.3</u>**

Factor C- Orientation of the design surface- The dominant failure mode of the stope design surface is gravity fall of ground due to slabbing and/or buckling of beds in the hangingwall.

C = 8-6 Cos θ = 8-6 Cos 20˚ = <u>2.36</u>, where θ is the dip of the bedding

Q' = <u>2.22</u>
N' = Q' * A * B* C = 2.22 * 0.78 * 0.3 * 2.36 = <u>1.23</u>

Stability Index/Hydraulic Radius = S= Area of Design surface/Perimeter of same Design surface = (20*100)/2(20+100) = <u>8.33</u> , where;

Stope design surface = 100m along strike (4400mN-4500mN)

Stope backlength = 20m

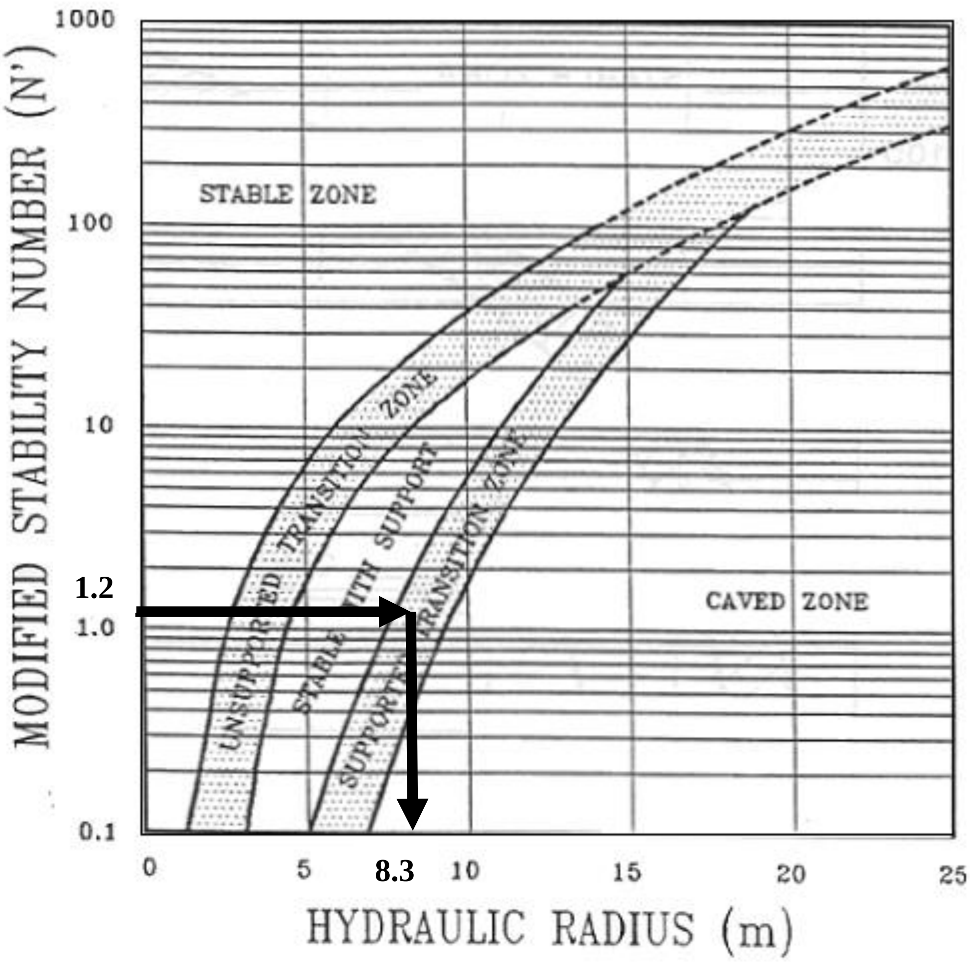

Figure1: Modified Stability graph of spans in the 4300mN-4500mN area

Mining Layout and Sequence of extraction

The conceptual plan layout of the method is shown below in Figure 2:

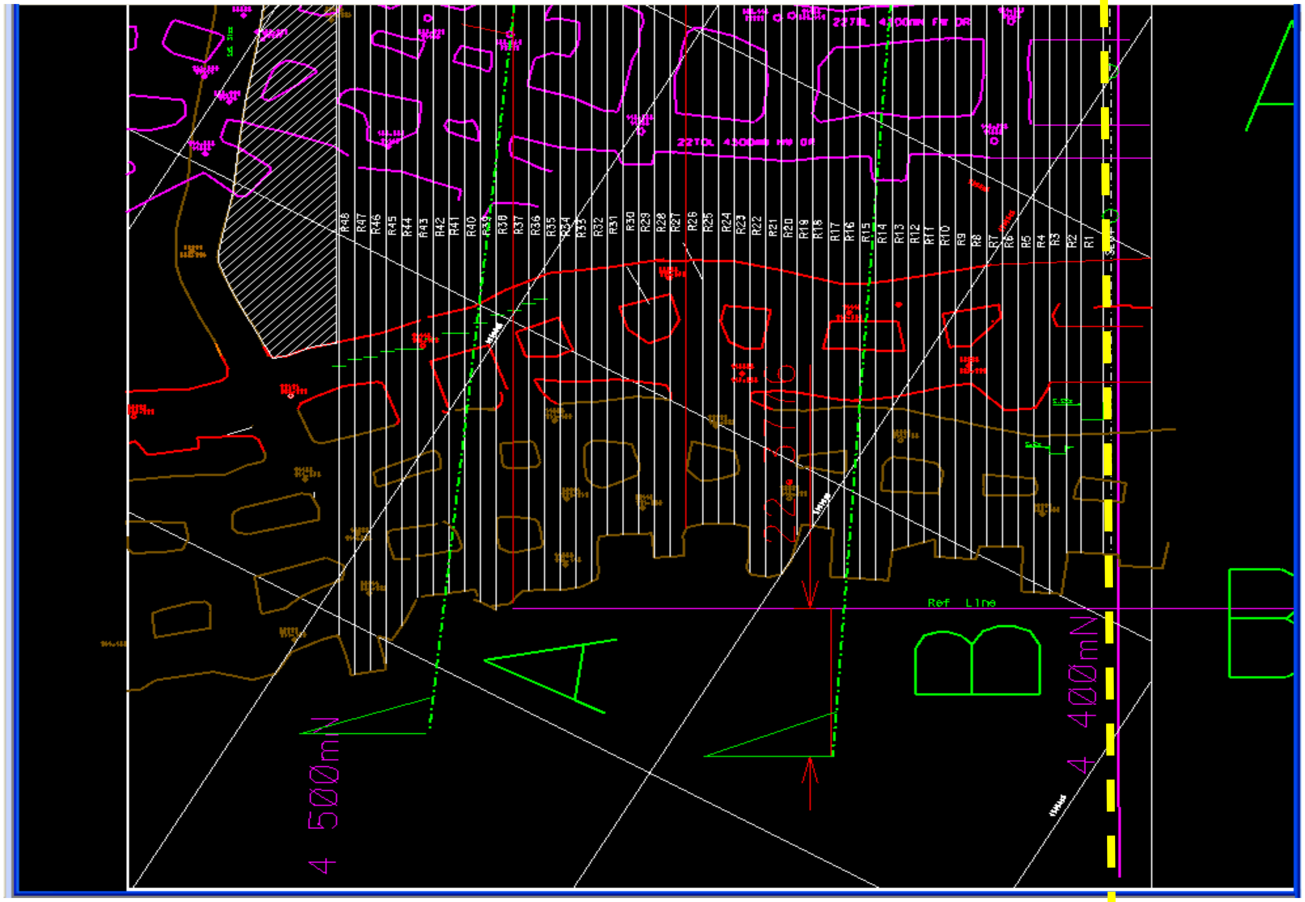

Figure 2-First phase of production: 2340L, 4400mN-4500mN Cut 2 PPCF/SLC hybrid layout

The PPCF/SLC hybrid mining method on trial run is an in-orebody technique of ore extraction. The 2270L and the 2340L, Cut 2 defines the top and bottom of the mining block respectively. In the first phase of ore extraction, drilling, blasting and extraction will be done on the compacted waste fill in the footwall drive along strike and advancing from the 4400mN towards the 4500mN in the northern direction. Production will be done via the 4500mN access crosscut to the orebody and through the extraction drive developed parallel to the strike in the footwall formation. In the second phase of ore extraction, production will advance along strike as before but heading southwards from the 4400mN towards the 4300mN area. Production will be done via the 4300mN access crosscut.

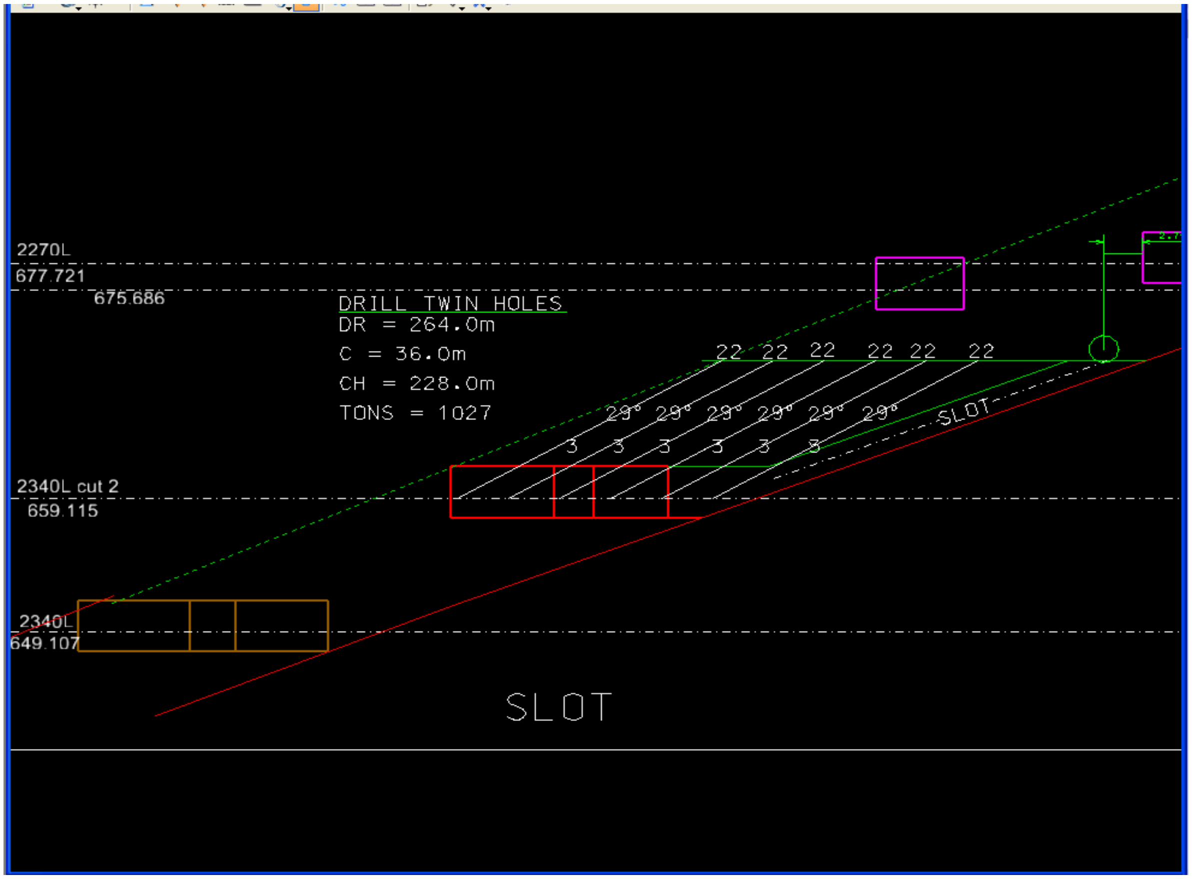

Figure 3-First phase of production: Section A-A, 2340L, 4400mN-4500mN Cut 2 PPCF/SLC hybrid layout

Mining of the slot raise is in progress and is almost 90% complete at the 4400mN position to create a free breaking face for the stope rings on either side of the planned stoping panels. The position of the access crosscut to the slot raise must be adequately barred down and re-supported with 2.4m long permasets at 1.0m ring and bolt spacing on a square grid.

Conclusions

- In the trial run of the PPCF Cut 2/SLC hybrid mining method, it is anticipated that immediate dilution following ring blast will not be possible as the caved waste rock behind the blasted ring, will confine the broken ore in a slot form and since the stope will always be kept full of blasted ore and waste rock material from the old stopes above, dilution due to H/W peeling will be minimized;
- It is also anticipated that the breaking of ground will always be by choke blast and this will significantly reduce ore loses within the stope;
- Geotechnical Engineering personnel will strictly monitor the production blasting of the stope rings in this trial run and advise accordingly.

Recommendations

- In an attempt to extract the ore at minimum dilution levels, a 4m wide crush pillar with a width to height ratio in the order of 2:1 must be left behind to hold-up the H/W beam in the tensile zone above the stope and hence prevent the caved waste rock material lying in the old stoped out areas above (2270L) from migrating downwards into the current stope (2340L Cut 2) immediately after ring blasting;
- When the crush pillar fails early enough behind the stope face, even the ore left in the old stopes above that may migrate downwards and be recovered, will be left in the stope as dead ore. Therefore, a monitoring programme to have a better understanding of the failure mechanism of the crush pillar and at what stage it will fail, will assist in assessing the behaviour of the immediate hangingwall in order to optimise roof spans and the support regime in future designs;
- The monitoring programme will also aid in establishing a long term hangingwall

stability monitoring strategy for safety purposes;

- ❖ Analyses with numerical modelling using Phase2 and/or Examine 2D will be incorporated in the designs at a later stage so that in-situ stresses before mining are compared with the induced stresses after mining, displacements and orientation of minor and major principal stresses are determined. DIPS programme incorporating Unwedge numerical modelling analysis to understand the stability of the presence of wedges will also be done at the same time.

J. KUNDA
SENIOR GEOTECHNICAL ENGINEER- 3 SHAFT CONVENTIONAL SLOS + PPCF

Distribution:
Mining Manager
Underground Manager-3 Shaft
Head of Geotechnical Services
Head of Safety-3Shaft
SOU Managers
Mine Captain– Conventional SLOS
Contractor Site Managers-Carmine, Bayete, Gilgle and Chiweme
Shift Bosses
File