Geotechnical considerations in the design of the MOCB mining method at Konkola No. 3 shaft

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Synopsis

Konkola No 3 shaft experienced severe mining-induced stress and time-related deterioration in ground conditions in an area where an over-cut and bench (OCB) mining method was used to extract a shallow dipping wide orebody. This caused an unacceptable risk for safe mining operations to sustain the required levels of production. After a review of the conditions it was decided to change the mining method to CASCADE to recover ore in the affected area and modify the OCB method and use a modified over-cut and bench method (MOCB) for new mining sections.

This paper describes the geotechnical criteria included to derive the mining standards for the MOCB. The geotechnical risks are identified and taken into account to develop stope design parameters, stoping sequences and support requirements. A monitoring programme to assess conditions as mining progresses is also discussed.

Introduction

Location

Konkola No. 3 Shaft is one of the two shafts at Konkola Mine operations. It is the most northerly shaft of the operations owned by Konkola Copper Mines Plc in the rich Copperbelt Province of Zambia. The Konkola mine is located in the town of Chililabombwe some 450 kilometres northwest of Lusaka.

Geological setting

The area lies on the nose of the Kafue Anticline, the dominant regional structural feature on the Zambian copperbelt trending south- east north-west. The stratigraphy ranges from the Achaean Basement complex consisting of granites and schists to the Late Precambrian Katanga System, a sedimentary series containing quartzites, conglomerates, sandstones, siltstones, dolomites and limestones.

The No. 3 shaft exploits the Kirilabombwe North orebody on the nose of the Kirilabombwe anticline. The ore is hosted in a shale package comprising units of thickly bedded siltstones and laminated micaceous siltstones with dolomite bands. The thickness ranges from 6 m on the flanks of the fold to 15 m in the fold nose area. The dip varies from a low 10 degrees in the nose area to about 60 degrees on the flanks. The orebody has a total strike length of over 6 kilometres and extends from the sub-outcrop and is open ended below the 4 500 level.

Geotechnical characteristics

The No. 3 shaft is an intermediate depth (less than 1 000 m) operation. *In situ* stress measurements indicated that the maximum principal stress aligns with the Kirilabombwe anticline axis and the Intermediate and Minor Principal Stresses have similar magnitudes and trend parallel to cross faults. The k-ratio is estimated at 0.85 (RMT 2001).

The rock mass condition in the stratigraphy ranges from very good in the footwall series and the immediate hanging wall to poor and generally fair in the orebody. The rock mass condition also shows variation from poor in the fold axis area to fair and good towards the flanks.

Brief mining history

Mining operations at Konkola No. 3 Shaft started in the late 1930s. The principal mining method was sub-level open stoping using scraper units to load ore into boxes to locomotive tramming systems. This method was used from the 300 Level to the 1660 Level. The method was generally expensive and of low productivity. Mechanized mining was introduced in the mid 90s in order to increase production in the substantial wide flat dipping resources. Post pillar cut and fill (PPCF) was used on the 1850 level advancing up dip to the 1660 level. The method was discontinued because of insufficient backfill to sustain the required production rates and was replaced by the over-cut and bench (OCB) in late 2000, initially without backfill but later backfill was introduced.

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Mining-induced stress-related damage

During mining of OCB stopes between 1660 and 1850 levels ground conditions deteriorated with increased severity of damage to excavations ranging from minor blocky falls at junctions in the early stages to severe pillar sidewall spalling, hangingwall collapse and floor heave in the later stages. The deterioration was attributed to (Naismith *et al.* 2004):

- increased stress levels in remnant areas as mining progressed up dip, leaving a progressively small remnant pillar towards 1660 level
- delayed backfill placement allowed pillars to deteriorate and soften, thereby shedding load to adjacent stiff remnant areas
- rapid oxidation and weathering of the orebody led to reduction of the rock mass strength.
- laminations and weak partings in the orebody created thin beams that break when left in the roof of the excavations.

Based on hazards identified, which include:

- out of sequence mining
- delays in installing secondary support
- delays in placing backfill
- weak ground that deteriorates rapidly with time and exposure to atmosphere
- large excavation spans.

It was decided that future mining methods and strategy should incorporate the following:

- reduce remnant areas by mining to strict sequence
- have rapid support installation and backfill placement potential
- rapid extraction rate to reduce exposure time
- avoid leaving thin beams in the immediate hangingwall
- site major access ways in competent footwall series rocks
- keep development dimensions as small as possible.

Modified over-cut and bench (MOCB) method

The modified over-cut and bench method (MOCB) appears to satisfy the above requirements and is being tried between 1850 L and 2200 L in a block 400 m long along strike between 2160 m W Section and 4275 m N Section. The trial is to be extended to a second block that is 500 m long along strike from 4050 m N Section to 3500 m N Section.

Mining layout and sequence

The major difference between the conventional OCB method and the MOCB is that the over-cuts are aligned parallel to an apparent dip of 7° instead of being flat and along strike. The conceptual layout is shown in Figure 1.

Initial access into the orebody is via an access cross-cut from a waste ramp. Once in the orebody an incline (ore access ramp) is developed against the hangingwall on apparent dip of 7°. From this access ramp, extraction drives are developed parallel to the strike at 100 m intervals. These drives define the top and bottom of the mining block. A block consists of a series of 4 m wide x 4 m high over-cut raises at 14 m centres mined parallel to the Access ramp leaving a 12 m wide longitudinal pillar between adjacent raises. A 10 m pillar is left between the access ramp and the first over-cut raise in the block to protect the ramp and also between the Extraction drive and the bottom of the stope panels. The final panel spans are 10 m, separated by 4 m longitudinal pillars with dip lengths of 80 m to 100 m.

Rockmass condition in the trial area

The sequence of stoping consists of developing the over-cuts. These are slyped to a span of 10 m. Once two adjacent over-cuts have been slyped, the first over-cut raise is benched to the footwall contact. Backfill is then placed immediately after the benching phase has been completed. After filling the first over-cut, the adjacent over-cut is benched and slyping in

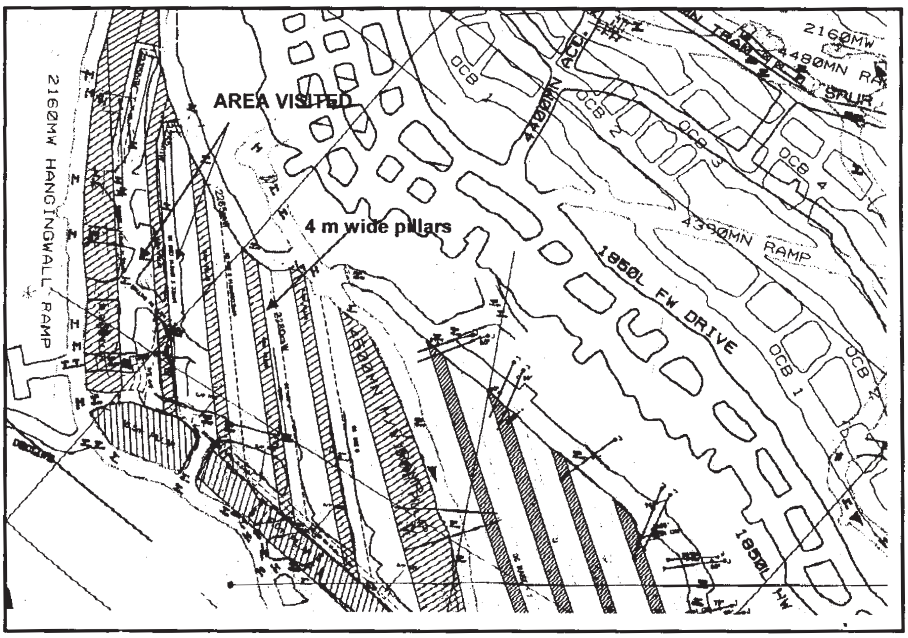


Figure 1—MOCB layout

next over-cut commences. The estimated stand-up time for the ledged over-cut before filling is 2 to 3 months. The trial area lies in the axial fissure zone and the rockmass is affected by joints associated with a major fold axis and some minor faults. Two prominent joint sets and two sets of random joints exist (Table I). The critical rock units are the hangingwall quartzite (roof spans) and the ore shale (bench sidewalls). From borehole information, the condition of the Hangingwall quartzite and ore shale vary, as indicated in Table II.

Geotechnical risks in the MOCB design

The geotechnical risks to be considered are as follows:

- stability of the roof of fully ledged over-cut (10 m wide) and full height sidewall against total collapse
- stability of bench sidewalls during benching and lashing operations
- localized wedge and slab failures
- regional stability with particular regard to out of sequence mining and security of main accesses
- pillar performance during the various stages of the stope life
- backfill placement—quality , scheduling and bulkhead design.

Stability of spans

Analysis

The stability of the roof span of the full width over-cut was analysed based on the hydraulic radius or shape factor and Mathews/ Potvin's stability number (Potvin *et al.* 1989). Q-rockmass rating parameters for the hangingwall quarzite were used because the roof of the over-cuts is designed to be along the geological hangingwall (GHW).

The hydraulic radius values are used to estimate unsupported apparent dip span limits for 10 m and 12 m wide stopes for a given rockmass condition. The results are shown in Table IV.

Conclusions

The proposed 10 m wide over-cuts are stable without support over 40 m apparent dip spans in good ground and 15 m spans in fair ground. This is confirmed by observations made in existing sub-level open stoping where stopes of similar dimensions were used successfully. Experience from existing OCBs indicates that the sidewalls are stable over the full thickness of the orebody and do not influence span stability.

Localized wedge and slab failures

Analysis

Occurrence of localized wedge and slab failures in the roof and sidewalls as a result of intersecting discontinuity planes was investigated using 'UNWEDGE', an equilibrium stability analysis computer package. The results are indicated in Table V.

Conclusions

The analysis indicates that kinematically feasible wedges occur both in the roof and sidewalls of the excavation. Two types of wedges are formed in the roof. The first type is

Table I

Discontinuity sets

Set	Dip	Dip direction	Spacing	Aperture	Infill	Surface condition
Bedding plane	15–20°	260°	0.15–0.20 m	<1 mm–5 mm	Kaolin FeOx	Smooth-planar
J1 J1c	60–80° 75°	010–025° 170–195°	0.5–1.25 m	1 mm–2 mm	Silt/clean	Rough planar– rough undulating
J2 J2c	70–90° 80°	070–080° 225–250°	0.5–1m	<1mm–3mm	Clean, FeOx and Silt	Rough planar
J3	75°	135°	1.5–2m	3mm–5mm	silt	Rough planar

Table II

Rockmass rating

Rock type		Thickness (m)	Rating			Notes
			Q-rating	RMR-rating	Description	
Hangingwall quartzite		> 20	4. 6 30 1.1		Fair Good to very good Poor	Massive quartzite, some kaolinization in bands Massive quartzite, minor kaolinization in bands Quatzite with kaolinized bedding planes
Ore Shale	Unit E	1.5		60	Fair to good	Micaceous siltstones interbeded with sandstone layers, kaolinized in places
	Unit C/D	3–4.5		40	Fair	Thinly bedded siltstones with dolomitic interbands, kaolinized to give weak parting planes
	Unit B	3		75	Good to very good	Thickly bedded to massive siltstone
	Unit A	0.5–1		10	Very poor to poor	Poorly consolidated sandy siltstone

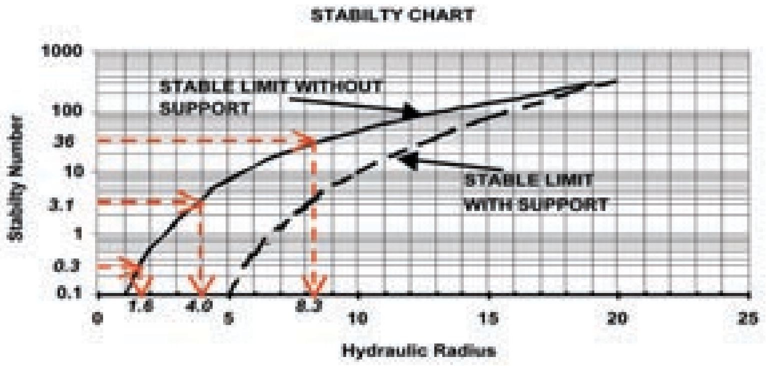


Figure 2—POTVIN stability graph

Table III

Stability numbers

Rockmass condition		Parameters			Stability number
Class	Q-rating	A	B	C	
Poor	1.1	0.70	0.20	2	0.3
Fair to good	4.6	0.75	0.30	3	3.1
Very good	30	0.80	0.30	5	36

Table IV

Unsupported span limits

Rockmass condition	Stability number	Hydraulic radius	Unsupported dip span	
			10 m wide over-cut	12 m Wide over-cut
Poor	0.3	1.6	< 10	< 10
Fair to good	3.1	4	15–40	10–24
Very good	36	8.3	>100	>100

formed by the intersection of the bedding plane with any two of the joint planes. These generally have a short apex length (< 3 m). The second type is formed by the intersection of the two major joint sets and the random joint set without the bedding plane. These wedges are very large and generally have large apex lengths (> 9 m). The failure mechanism is either falling under gravity or sliding along one or both of the joint planes. Wedges formed in the sidewall either topple or slide along the bedding plane. In addition, observation in existing over-cuts indicates that severe sloughing and unravelling occurs in Units C and D of the ore shale due to time dependent weathering, resulting in the formation of an overhang brow at the Unit E contact and a ledge at the Unit B contact. This leads to localized unstable conditions when the bench is taken.

Regional stability

Analysis

To establish an idea of the stress levels in the proposed MOCB mining and surrounding areas as a result of earlier mining, a mine scale model was set up using FLAC3D (ITASCA, 2004). This indicated that loading on the proposed MOCB areas is not ever more than 40 MPa. In addition, stress variation within the trial mining block as extraction of the stopes progressed was simulated using a 2-D finite element code PHASE2. More complex three-dimensional modelling is planned for the future as more insight is gained.

Model geometry

Figure 3 shows the model geometry for the PHASE 2 analysis.

Model results

The variation of stress as mining progresses across five stopes is indicated in Figure 4. The stress levels are about 20 MPa at the over-cut raise development stage. Five over-cut raises can be mined at the same time without significant rise in the stress level. The stress level rises to between 35 and 45 MPa in the 4 m pillars (P1 to P4) separating the 10 m wide over-cuts and stabilizes at about 30 MPa after the whole block has been mined and backfilled. The stress level in the10 m wide ramp pillar (RP) is constant at about 22 MPa throughout the mining stages. As expected, benching of the over-cuts does not increase stress level in the pillars. The pillar strength is, however, expected to reduce as the w/h ratio reduces. This is analysed in detail in the next section.

Pillar stability and performance

Analysis

To assess the pillar stability and performance in detail another two-dimensional model using a displacement discontinuity boundary element code (Malan 2004) was set-up.

Model geometry

The sequence of mining modelled was as follows (Figure 5)

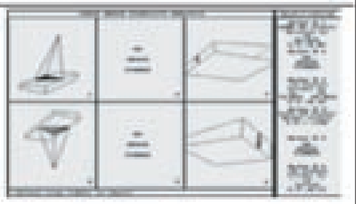
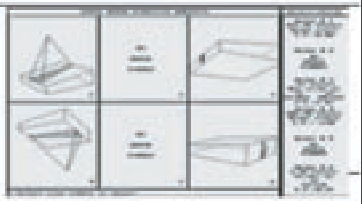
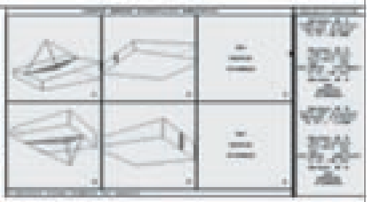
Step 1—over-cut raise stage—4.0 m wide and 4.0 m high excavations at 14 m centres, to define a 10 m wide 4 m high rib pillar/ rectangular pillar (strike length >/= 4 times apparent dip span)

Step 2—down-dip over-cut raise slyped to full over-cut width—10 m wide and 4.0 high opening down-dip and 4 m wide by 4 m high openings up-dip to define a 4 m wide and 4 m high rib/rectangular pillar

Step 3—both down-dip and up-dip over-cut raises slyped to full stope width—10 m wide excavations both down-dip and up-dip leaving a 4 m wide and 4 m high rib/rectangular pillar with 10 m wide excavations on either side

Table V

Wedges formed (Over-cut raise)

Discontinuity set Combination	Wedges formed				Remarks
		Roof	Up-dip sidewall	Down-dip sidewall	
Bedding, J1, J2	Face area	7 m^2	-	-	
	Apex height	2.6	-	-	
	Weight	8	-	-	
	Failure mode	Rotate on j1	-	-	
Bedding, J1,J3	Face area	15 m^2	15 m^2	10 m^2	
	Apex height	1.1	1.9	1.5	
	Weight	15	13	7	
	Failure mode	Fall under gravity	Rotate on J3	Slide on j1 and bedding	
Bedding, J2,J3	Face area	12 m^2	16 m^2	10 m^2	
	Apex height	0.9	2.1	1.7	
	Weight	62	17	9	
	Failure mode	Fall under gravity	Slide on j3	Slide on j2 and bedding	
J3, J2,J3	Face area	7 m^2	1 m^2	0.7 m^2	
	Apex height	10	0.4	0.3	
	Weight	62	0.3	0.2	
	Failure mode	Rotate on j2	Slide on j2 and j3	Slide on j1	

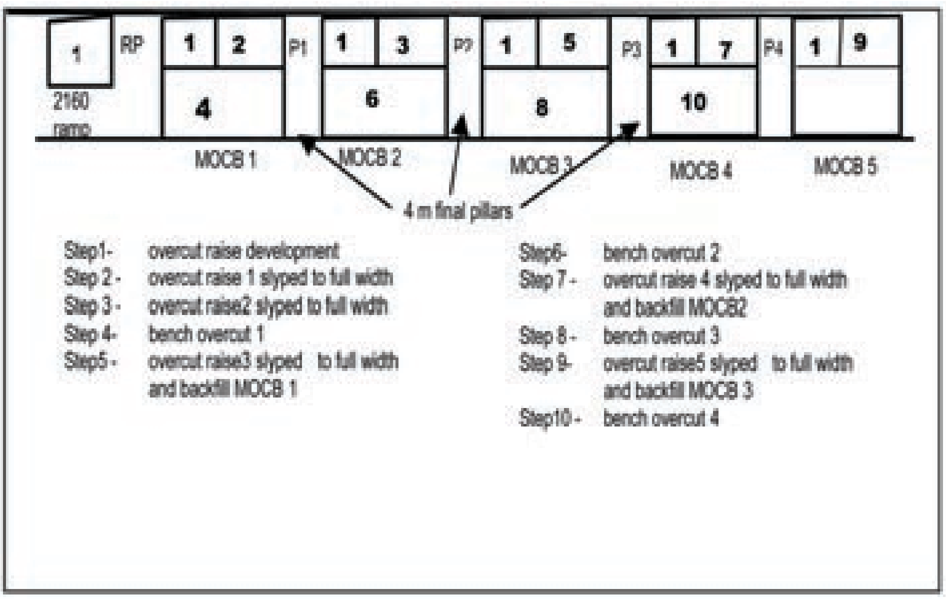


Figure 3—Model geometry for block stress variation simulation

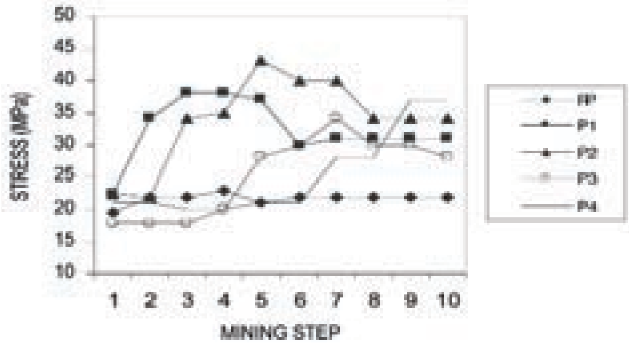


Figure 4—Principal stress variation as extraction progresses

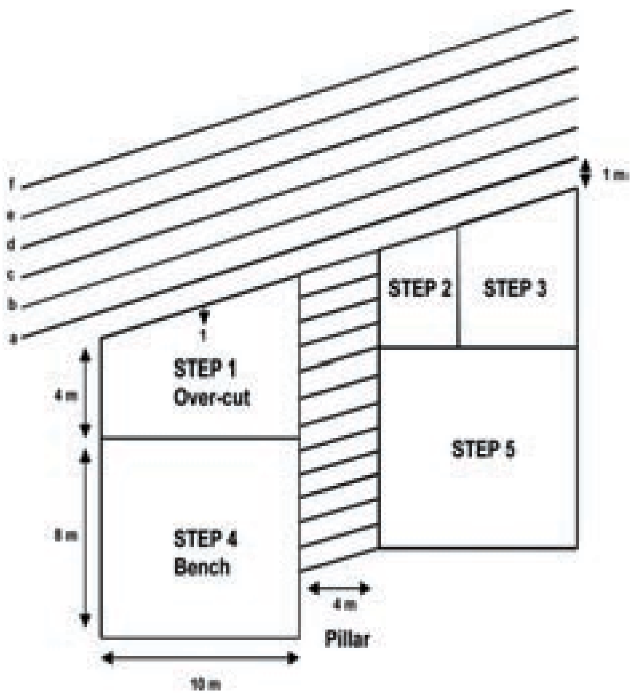


Figure 5—MOCB model geometry (after Malan, 2004)

Step 4—down-dip side benched—to leave a 4 m wide pillar which is 9 to 12 m high on down-dip and 4 m high on up-dip side
Step 5—bench up-dip over-cut raise -to leave a 4 m pillar that is 9 to 12 m high

Modelling results

The results are similar to the ones obtained in the PHASE2 model and are presented in Figure 6.

The maximum stress is 26 MPa at step 1, rising to a peak of 42 MPa in Step 4 and becoming uniform at 33 MPa after step 5.

To assess the stability of the individual pillar through the mining process, the load induced on the pillar (PL) is compared to the strength of the pillar (SP). The two are related by the factor of safety (FOS):

$$FOS = \frac{S_P}{P_L}$$

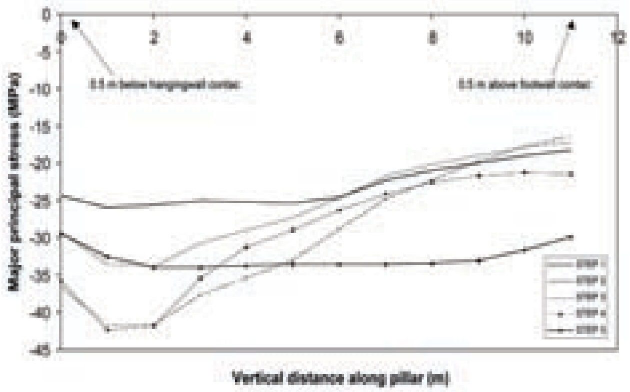


Figure 6—Maximum stress variation in the centre of 4 m pillar from H/W to F/W contact (after Malan, 2004)

$$S_P = k\frac{w^{0.5}}{h^{0.7}}$$

k = rockmass strength
w = pllar width
h = pillar height
for a rectangular pillar

$$w = w_{eff} = \frac{4A}{P}$$

where W_{eff} is effective width and A is pillar area and P its perimeter.

The rockmass strength is estimated from geotechnical mapping data computed using the GSI approach (Hoek 1998). It varies from 20 to 30 MPa for poor ground and climbs to 35 to 45 MPa in better ground (AMC, 2004).

Table VI shows the pillar strengths for an 80 m long longitudinal pillar in 25 MPa and 40 MPa strengths (average poor and fair to good ground conditions, respectively).

The load on the pillar is estimated from the model results plotted in Figure 6 and is compared to the pillar strengths in Table VI to obtain the factor of safety. Figure 7 shows the factor of safety for a range of rockmass strengths from 25 MPa (poor rockmass) to 45 MPa (good rockmass) at the various mining steps.

As can be seen from Figure 7, the pillars are stable at the over-cut raise development stage in all rockmass conditions but the factor of safety falls to below unity from step 2 onwards in the poor rockmass and from step 3 in the fair

Table VI

Pillar strength

Ht (m)	Pillar strength (MPa)			
	10 m (W_{eff} = 18) pillar		4 m(W_{eff} = 8) pillar	
	25 MPa rockmass	40 MPa rockmass	25 MPa rockmass	40 MPa rockmass
4	40	64	26	43
5	34	55	23	37
10	21	34	14	23
12	18	30	12	20

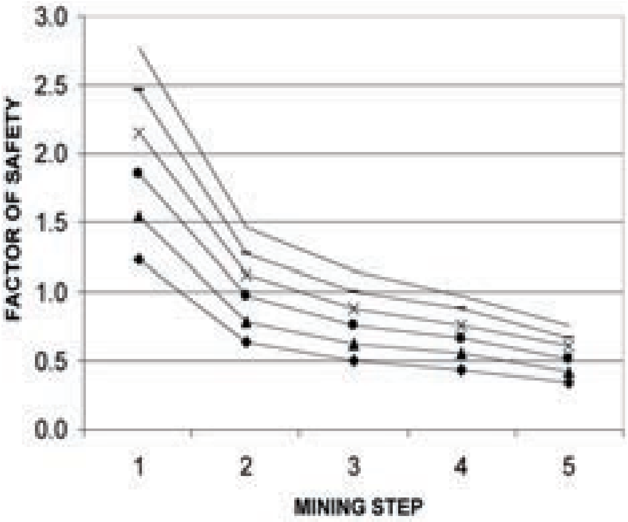


Figure 7—Pillar safety factor for a range of rockmass strengths at the various mining steps

rockmass. This implies that the pillars will fail. In practice however, the pillars are observed to be stable at steps 1, 2 and 3, except for minor sidewall spalling. This is attributable to the fact that the strength of the pillars is increased by sidewall support with permasets and mesh and straps that are installed during development and ledging of the over-cut raises. This emphasizes the requirement to install the support early after development and to place backfill soon after benching.

Backfill

From the previous section it is obvious that backfill is an essential component for the success or failure of this mining method. The role of the backfill is mainly to confine the pillar sidewalls (increase the pillar capacity to carry load) after stopes have been benched out. The critical issues include the backfill material balance, the backfill properties and the strength of bulkheads.

These aspects among others were analysed in detail during an audit supervised by SRK (2003) and are discussed by N. James and A. Naismith (2004). It was concluded that the backfill plant and reticulation systems have the capacity to produce backfill to meet the mining backfill requirements.

The average requirement for backfill in the MOCB stopes is between 26 000 m^3 to 30 000 m^3 per month. The plant has a production capacity of 170 m^3 per hour. This can be transferred to the pour areas at the rate of 90 m^3 per hour to give a monthly availability of 52 000 m^3 assuming 16 hours operations per day over 26 days.

It is essential that the backfill drains easily to reduce the risk of bulkhead failure as well as to attain the required shear strength quickly. An industry standard of 100 mm/hour percolation rate is currently being followed but slurry with lower percolation rates down to 80 mm/hour has been found to drain well.

After several trials, cable truss bulkheads are in use. These comprise a cemented tailings wall contained in geofabric material and a framework work of cable trusses. The bulkheads are designed on the assumption that the bulkhead will carry the full pressure head generated by cohesion less classified tailings. The concept is that the full pressure head is applied horizontally onto the bulkhead and is resisted by a combination of shear resistance developed by cemented tailings wall/rock interface and the total shear resistance developed by reinforcing tendons grouted into the rock. A safety factor of 1.2 is applied.

Conclusions

Lessons from OCB mining

Lessons learnt from OCB mining are as follows:

- weak planes in the C, D and E zones in the ore shale and contact with the hangingwall quartzite cause instability in wide roof spans
- small strike pillars between adjacent OCBs are only marginally stable and tend to fail during the stope life
- delayed support installation leads to hangingwall instability
- the rockmass deteriorates rapidly on exposure to atmosphere
- out of sequence mining creates unnecessary remnants and a generally high stress environment
- delayed backfill placement after benching allows pillars to soften by sidewall spalling and sloughing, thereby shedding load to adjacent stiff areas.

Findings from MOCB design analysis

Analyses carried-out to address the identified adverse mining conditions for the MOCB design indicate the following:

- 10 m wide roof spans are stable if located in the hangingwall quartzite and can stand without support over a 15 m dip length in poor ground and 40 m in good ground
- localized wedge and slab failures from the roof and sidewalls occur. These failures can be prevented by systematic roof bolting with 2.4 m tendons supplemented by 6 m cable anchors and mesh and tendon straps
- stress levels do not rise significantly in areas away from the stoping panels. A systematic layout of abutment pillars can allow production to take place from several faces at the same time, thereby minimizing the exposure time of excavations
- pillar stresses at the over-cut raise development stage are low so that several over-cut raises can be developed at the same time without compromising pillar stability. Pillar stresses rise to deleterious levels when adjacent over-cut raises are sylped to full over-cut width. The pillar stability becomes adversely affected. The effect becomes more significant after benching as the pillar with ratios are reduced, thereby reducing the strength of the pillars. Pillar damage can be prevented by sidewall support and confinement with backfill
- the backfill plant has the capacity to produce sufficient good quality slurry and the surface storage, transportation and underground reticulation systems are adequate to satisfy backfill placement needs. Bulkheads of sufficient strength can be designed and constructed.

Table VII

Secondary support

Excavation type	Secondary support	Spacing	Compliance
Over-cut raise	6.0 cable anchors	3 m dip x 4 m strike	30 m behind the face
	mesh and straps	None	Not applicable
Ledge	6.0 cable anchors	3 m dip x 4 m strike	6 m behind the face
	mesh and straps	full cover in roof and sidewalls to the floor	10 m behind the face
Extraction drive	6.0 cable anchors	3 m dip x 3 m strike	30 m behind the face
	mesh and straps	full cover in roof and sidewalls to 2.0 above floor	30 m behind the face
Drilling drive	6.0 cable anchors	3 m dip x 3 m strike	at over-cut junctions
	mesh and straps	None	not applicable
Draw point x/cut	6.0 cable anchors	3 m dip x 3 m strike	
	mesh and straps	full cover	
Bench s/wall	6.0 cable anchors	none	not applicable
	mesh and straps	none	not applicable

Trial mining with the MOCB is in progress and one stope is 90% complete. Based on the analysis, the following operating guidelines and procedures are in place:

- roof of excavations should be located along the contact between the bedded ore shale and the massive hangingwall (GHW) to avoid leaving thin weak bands in the roof
- roof spans in the full over-cuts are to be maintained at 10 m width and all the other excavations to be less than 5 m wide and 5 m high quartzite
- several over-cut raises can be developed at the same time. Two adjacent over-cuts can be ledged immediately one after the other. The slyping should be carried out carefully so as not to damage the resulting 4 m pillar sidewalls. The pillar sidewalls should be bolted. Once the slyping is completed, the first over-cut should be benched and backfilled to confine the pillars. Slyping of the third over-cut is to be carried-out concurrently with the backfill placement in the first benched-out stope
- support with 2.4 m long permasets at 1.0 m to 1.2 m spacing on a square pattern is to be installed concurrently with development in all the excavations. Secondary support comprising 6.0 m long fully tensioned and grouted and tendon straps are to be installed at some later stage as detailed in Table VII
- bulkheads to be designed by geotechnical personnel and signed off by civil engineering. Construction to be carried out by backfill section and signed off by geotechnical personnel. Good drainage system to be maintained at the bulkhead
- planning, survey, geology and geotechnical personnel to carry out regular inspections to ensure the above guidelines are adhered to.

Monitoring programme

In order to have a better understanding of the rockmass deformation mechanisms, a monitoring programme is planned and is in the initial stages of being implemented.

Objectives

The major objectives of the programme are:

- assess and monitor pillar performance to optimize future pillar design
- assess and monitor the behaviour of the immediate hangingwall to optimize roof spans and support regime in future designs
- establish a long-term hangingwall stability monitoring strategy for safety purposes.

Monitoring instrumentation

In the initial stage the instrumentation will comprise:

- regular survey of cable anchor holes using a borehole probe to locate possible open bedding planes
- laser reflector closure pegs in the sidewall and hangingwall to measure changes in excavation dimensions and pillar dilation
- wire extensometers in the hangingwall to measure bedding plane separation
- rod extensometers in the pillar sidewalls to monitor pillar dilation
- backfill load cells to assess load dilation in the backfill.

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