A GEOTECHNICAL REVIEW AND CONSIDERATIONS IN THE DESIGN OF A NEW MINING METHOD AT KONKOLA MINE

By

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Declaration

I, Webby Mutambo, declare that this work is entirely my own with the exception of citation of work of other people both published and unpublished, which I have duly referenced to and acknowledged.

To the best of my knowledge and belief this dissertation contains no material previously published for the award of any other degree or diploma in any university.

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This dissertation by Webby Mutambo has been approved as fulfilling the requirement for the award of Degree of the Master of Mineral Sciences by the University of Zambia.

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Dedication

To the almighty God, the giver of my life and to whom I look up to.

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Abstract

Konkola Mine of Konkola Copper Mines Plc comprises of Number 1 and 3 Shafts located on the northern most part of the Zambian Copperbelt. The mining licence area is characterised by the Kirilabombwe anticline dividing the West and North Limbs striking generally in the northwest to southeast directions. Average dip angles are 60° in the Central region, very flat (about 10°) towards the 'nose' area and 60° in the North. The Number 1 Shaft is located in southern part of the ore body and lies between 0mN to 3000mN survey mark positions and extends to 800mS. The current development is below 3150 feet level. The Number 3 Shaft is located on the North Limb and the lowest tramming level is at 1850 feet level. Production is currently below the 2270 feet level.

As mining is focused below 2270 feet level at Number 3 Shaft and below 3150 feet at Number 1 Shaft, there is need to determine a suitable mining method. This must take into consideration the geotechnical parameters of the host rock and the ore body especially in the nose area where several mining methods have been applied. The dip and the depth from surface influenced the selection of a suitable mining method.

This research involved collection of scan line mapped data. The data collected includes joint spacing, joint orientation, joint condition and rock samples to determine the uni-axial compressive strength. The collection of data at the mine site was done in two parts: the first part involved collection of the existing mapped data while the second part involved underground mapping. The collected data was analysed using a computer based spreadsheet to determine the rock mass rating and the information was subjected to evaluation using specialised geotechnical Phase2D software.

A review of the design and implementation of the proposed mining method has been made with the aid of a computer model using Phase2D. Ground support requirements for the mining method have also been defined. This selected mining method has been analysed based on the productivity and safe extraction way of taking out the ore body in the nose area of Konkola. The results show the options that involve mining of primary and secondary stopes with different stope spans and will require cemented backfill in primary stopes.

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Abbreviations and Acronyms

ASCu	Acid Soluble Copper
Dir	Direction
DR	Democratic Republic
ESR	Excavation Support Ratio
FW	Footwall
GFW	Geological Footwall
GHW	Geological Hangingwall
HW	Hangingwall
HWQ	Hangingwall Quartizite
LRO	Longitudinal Retreat Open
mH	Metres Height
mN	Metres North
MOCB	Modified Overcut and Bench
mW	Metres West
NE	North East
NW	North West
P_s	Pillar Strength
PPCF	Post Pillar Cut and Fill
Q_s	Pillar Stress
RMR	Rock Mass Rating
RQD	Rock Quality Designation
SF	Strength Factor
SRF	Stress Reduction Factor
TCu	Total Copper
ZCCM	Zambia Consolidated Copper Mines

CHAPTER 1

INTRODUCTION

1.1 LOCATION OF THE RESEARCH PROJECT

Konkola Mine is located about 450 km northwest of Lusaka , the capital city, of Zambia. It is in the most northerly of the Zambian Copperbelt mines. The mine is located in the town of Chililabombwe and lies 25 km north of Chingola (Nchanga Mine) and 12 km from the Democratic Republic of Congo border Figure 1.1.

Chililabombwe is situated at an elevation of 1 360 metres above sea level on the Central African Plateau. Low-lying hills occupy the central area of the town and a low range of hills is located along the international border with Democratic Republic of Congo . The topography between the hills is gently undulating with deeply weathered red lateritic soils. The top soils are generally sandy but with a heavier textured subsoil. Most of the lateritic soils are much leached as a result of the high rainfall and thus tend to be acidic and relatively infertile [7].

Zambia being a landlocked country is dependent on road/rail links to ports in neighboring countries for the importation of goods and the export of its metal products. Currently, the major exit and entry port for Zambia’s exports and imports is Dar-es-Salaam in Tanzania which is linked to the Copperbelt by road and railway. Air connection is made via Johannesburg or Lusaka with regional daily flights in and out of Ndola and Kitwe airports.

Mineral exports from Zambia have traditionally a major country’s foreign exchange earnings.

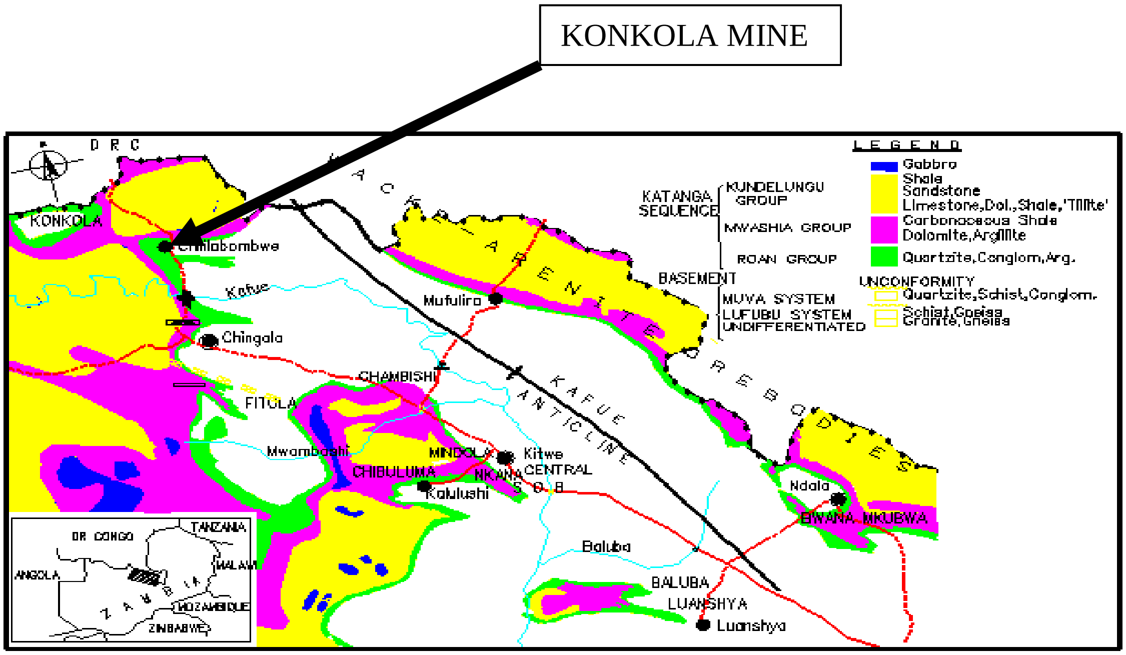


Figure 1.1: Location of the Konkola mine on Geological Map of the Zambian Copperbelt (Reproduced with permission from KCM Plc)

1.2 STATEMENT OF THE PROBLEM

There have been increased development requirements and ore losses in the current in-ore body mining methods resulting into increased lead-time between stope development and commencement of stope exploitation. This is costly considering the small tonnage recovered from the ore body at Number 3 Shaft. Ore recoveries of 60-65% are also deemed to be low from the current mining methods, unless they are above 80% tenable with other mining methods.

Against such background, it is required that a suitable mining method, that will, apart from maximizing ore recovery, minimise premature hanging wall failure, improve local ground stability, enhance the safety of operating personnel and reduce on stope development requirements, be studied and tried.

1.3 OBJECTIVES OF THE RESEARCH PROJECT

The main objectives of the study/ research involve the:

- Suggesting of the suitable mining method taking into consideration geotechnical factors expected to prevail at depth such as stress conditions, standup time and influence of discontinuities on rock strength.
- Analysing of the current mining methods versus the proposed.

- Analyse the inherent geotechnical conditions of the area and associated risks of the current mining method.
- Analyse the rock mass environment of the proposed mining method in order to successfully extract the ore.

1.4 HISTORICAL BACKGROUND

The greater Konkola area ($230km^2$) contains five ore body deposits. These are the Konkola, Kirilabombwe, Sadle Lode, Kakosa and Fitwaola ore bodies. A gemcom model of the greater Konkola area has been approximated to support a billion tonnes at a copper grade of 3.11% TCu. Hence it represents a potential for high output mining operations in the long term.

Konkola ore body was exploited for a brief spell in 1957/58 but this was discontinued due to unfavourable economic factors. The Kirilabombwe ore has been exploited since 1957 through the Number 1 Shaft and since 1963 through the Number 3 Shaft. These have been viewed as separate entities due to a barren gap in the upper section dividing the two. However, current exploration information indicates that at lower depths, the ore body is continuous. Studies to optimise production from Konkola Mine have been done by Australian Mining Consultants [7].These include mechanisation and increasing the number of production faces.

Number 3 Shaft exploits the northern and eastern ore bodies of the Kirilabombwe ore body. Mining before the 1990s was by Sub Level Open Stoping with scraping due to the low dips. Number 1 Shaft exploits the southern ore body and mining was by Sub Level Open Stoping using gravity [7].

1.5 ORE RESERVES

The Konkola Deep Mine Project has been recognised as one of the promising future copper mining projects in Zambia. The Konkola ore body is predominantly sedimentary ore shale with sulphide mineralisation, though oxide ores exist due to weathering effects. Exploration boreholes have indicated that the ore body is continuous below 2270feet level at Number 3 Shaft and below 3150feet level depth at Number 1 Shaft. It has been explored by surface drill holes down to a depth of 5958 feet level in the southern part and 3607feet level in the northern part . The average thickness of 11.5m true thickness of the ore body which has extensive length of about 11km. Sixty Seven percent of the ore body has a dip of less than 45^o .This gives rise to substantial amount of ore per length with grade varying from 2.4% to 5.7%.

1.6 PRODUCTION

At Number 1 Shaft, the levels above 2200 feet level have been mined using Sub Level Open Stoping with scraping in the north extension area. Applications of any particular mining method have been dependent on the characteristics of the ore body and the host rock. Mining methods that have been implemented after the year 2000 have been influenced by the need to increase production through mechanisation.

The highest copper production at Number 3 Shaft was recorded in the 2002 financial year during which 945,000 tonnes of copper at a grade of 3.13% TCu / 0.28% AsCu was produced. Average yearly production since inception has been 690,000 tonnes of copper at 3.00% Tcu but has shown steady improvement to 1,080,000 tonnes of copper. [7]

Ore extraction at Number 3 Shaft above 1660 feet level has predominantly been by Sub Level Open Stoping with gravity draw in the steeper sections (dip $>45^o$) in the western and northern ore bodies. In the shallow dipping nose area, mechanical draw using scrapers has been used. Ore has been trammed by locomotives in drives located in the footwall, with transfer from stopes being done through ore passes and loading boxes.

The Scraper assisted Sub Level Open Stoping method was labour intensive, despite the advantage of a high extraction ratio of about 75%. In the 1990s, in-ore mining methods were introduced from the 1660feet level towards the 1850feet level with the aim of increasing productivity and also reducing high cost per tonne. These included the Post Pillar Cut and Fill (PPCF) and Room and Pillar, introduced in 2000. A modification of the Room and Pillar, Overcut and Benching method (OCB) was introduced in the late 1990’s in the thick, flat dipping areas. In this the area the ore body from 1660 feet level changed its dip from about 60^o to about 10^0 to 20^o.

Below the 1850 feet Level, a modification of the OCB called Modified Over cut and Bench (MOCB) has been used, with steeper sections planned for PPCF and Longitudinal

Retreat Open (LRO) stoping methods. Mining conditions between the 1660feet Level and the 1850feet Level abutment zone deteriorated in the 2nd quarter of the 2003 financial year, due to non-availability of hydraulic fill.

1.7 GEOLOGY

1.7.1 Stratigraphy

Rock formations in the Konkola Mining licence area are of sedimentary deposition and some meta-metamorphosed rocks categorised in the Katanga system. The Number 1 Shaft ore body, lying on the southern flank of the Kirilabombwe anticline has an average thickness of 9m and dips between 35^{o} and 70^{o}. The dip decreases to the north. The structure has been traced to a depth of 2 km. Above the 720m level, a barren zone, nearly 1.5 km wide separates the Number 1 Shaft from the Number 3 Shaft.

The Number 3 Shaft ore body lies across the axis of the Kilirabombwe anticline and has an average thickness of about 13 m. The dips are shallow (10^{o}) at the nose (axis) of the anticline and increases to the south to 35^{o} and to the east to 60^{o}. The predominant ore minerals are chalocite, chalcopyrite, bornite and carrollite. The geological setting of Konkola Mine as a whole is shown in the stratigraphic column shown in Figure 1.2.

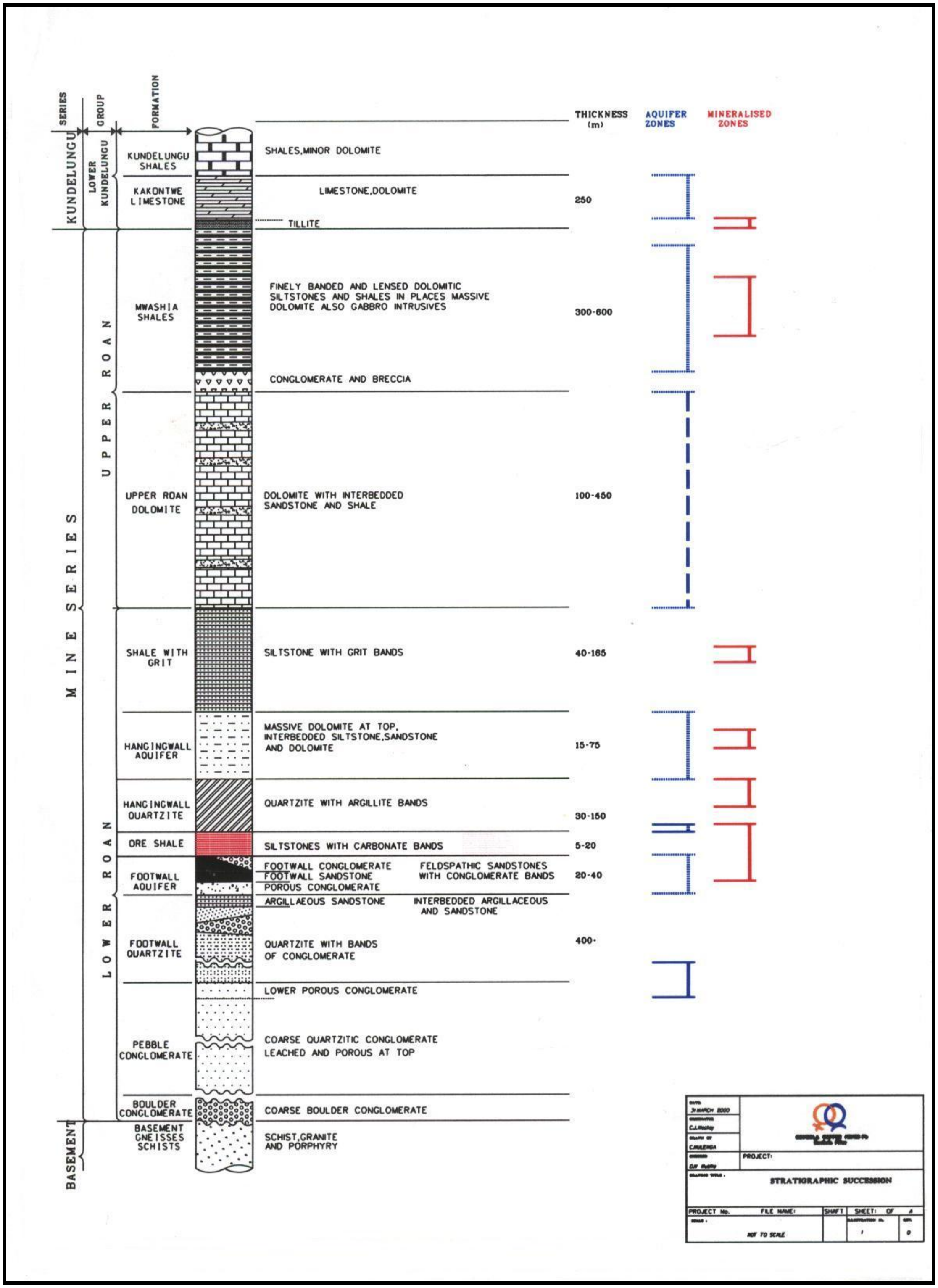


Figure.1.2 Stratigraphic column of Konkola Mine (Reproduced with permission from KCM Plc)

1.7.2 Ore Horizon

The ore shale, at a true thickness varying between 5-20m, is a siltstone with carbonaceous bands. It is divided into five ore horizons (A, B, C, D and E) that vary in strength with composition of the siltstone and carbonaceous bands. Table 1.1 below shows the composition of the ore horizons in the Konkola Mine licence area [7, 18]. The table shows the thickness of various ore horizons, their texture, sandstone contents and the calcareous content.

Table 1.1: Konkola Mine Licence Ore Horizon [18]

HORIZON	A	B	C	D	E
THICKNESS (m)	0.15- 1.15	1.4 - 2.0	0.9 – 2.0	1.0 – 1.8	0.6 – 1.5
TEXTURE	Fine to medium grained	Fine grained	Fine grained	Fine to medium grained	Medium grained
SANDSTONE CONTENT	Fair	Low	Low	Fair	High
CALCAREOUS CONTENT	Fair	Lowest	High	High	Fair

Unit A, at 0.15m to 1.15m true thickness is finely bedded and frequently weathered to brown micaceous clay. It is the weakest in the ore horizon. It has low compressive and tensile strength. It greatly contributes to dilution. The hanging wall of Unit A is generally the assay footwall. Copper in the form of chalcocite is primarily concentrated in the sandy laminae. Certain areas in the mine are leached with copper grades being extremely high.

Unit B is characterized by its massive appearance and is essentially a quartz-feldspar-mica siltstone with fine incipient bedding. Fine grained, its delicate coloured banding is enhanced where it is more weathered. It consistently carries good copper grades primarily in disseminated sulphide form.

Unit C comprises of 1.1m alternating bands of grey siltstone and pink dolomitic sandstone bands (pink due to the presence of iron and magnesium). Shear cleavage is present in this unit.

The contact between units C and D is gradational; D being of thinly bedded light grey siltstone containing few layers and lenses of pink calcareous sandstone. In some places the calcareous sandstone has been leached and the cavities are partially or completely filled by secondary copper and manganese minerals. The major copper minerals are carbonates and oxides, which occur sporadically throughout the unit.

Unit E constitutes the top of the ore body and consists of closely bedded dark grey, sandy siltstone with numerous inter-bedding of pink to white dolomitic sandstone, which progressively becomes numerous towards the top.

1.7.3 Hanging wall

The hanging wall is generally competent and comprises siltstones, sandstone and shales with inter-bedded dolomite and gabbro intrusions. The hanging wall quartzite (HWQ) formation (with argillite bands) lying immediately above the ore shale degrades to a weak and in places kaolinised band towards the ore shale contact. The assay-hanging wall in most cases lies 1.0m below the geological contact. Stability of the roof (back) of excavations and stopes is influenced by the characteristics of the rock mass remaining in the immediate roof. Propensity of bedding separation occurring due to the tensile zone is higher for thinly bedded, laminated ore shale than the more massive quartzite formation. Assessment of rock fall risk cost of roof support and revenue gain/loss should determine the location of the excavation/stope roof.

1.7.4 Foot wall

Foot wall rock formations which include foot wall Quartzite, Argillaceous Sandstone, Porous Conglomerate, foot wall sandstone and foot wall Conglomerate are generally competent. Permanent structures positioned in the same rock formation such as haulages are location in the foot wall Quartzite. Weak zones however exist on the contacts when crossing the foot wall rock formation towards the ore body. All these are underlain by the basement gneisses and schist formations.[18]

The porous conglomerate, which varies in thickness from 9m-20m is the most continuous of the main footwall aquifer and rests on the argillaceous sandstone. It is a fairly well sorted conglomerate, pebbles ranging from 2.5cm-7.5cm, with a zone of scattered boulders near the base. The pebbles are predominantly granite, but also quartzite, feldspar, mica and chlorite are present. The cementing matrix varies considerably in composition in the mine. It is highly calcareous in the folded section south of the Number 1 shaft and becomes more siliceous towards the north.

Resting on the porous conglomerate is the foot wall sandstone comprising of fairly even grained sandstone, with occasional cross bedding, gritty lenses and shale bands. It ranges in thickness from 3m in the north to 30m in the south.

The foot wall conglomerate underlies the ore formations and is indistinguishable from the porous conglomerate. In some locations in the mine, the footwall conglomerate is sporadic and is not present south of the ore body. The rocks of this formation are extremely porous, but not excessively permeable. In the folded area the calcareous matrix is almost completely leached out and individual pebbles are poorly cemented, producing bad ground conditions.

Below the main foot wall aquifer is the argillaceous sandstone, which comprises of inter-beds of argillites (i.e. clay texture with particle size of less than 0.06mm) and sandstones. This is an aquiclude (i.e. where there is exchange of water) and is hence suitable for the location of drain drives provided a competent horizon within the formation is located. Five horizons can be identified in the north limb of the Konkola ore body and the most suitable is unit 3 (see Table 1.2).

Table 1.2: North limb Argillaceous Sandstone Horizons [18]

HORIZON	THICKNESS (m)	ROCKMASS CONDITION	WEATHERING	RATING
1A (Underlie Porous Conglomerate)	4-7	Thinly bedded and wet	Heavily leached	Class IV Poor rock
1B	3-6	Massive to slightly massive with some jointing.	Partially leached bands	Class III Fair Rock
2	3-4	Closely bedded and presence of cavities along bedding plane.	Leached and kaolinised in some places	Class III-IV Fair to poor rock
3	5-9	Massive and well-cemented bands, slight jointing.	Minor leaching	Class II Good rock
4	7-12	Closely bedded and well cemented.	Minor leaching	Class III Fair rock
5 (Overlaying Footwall Quartzite)	3-8	Thinly bedded and well cemented. Minor jointing.	Leaching prominent in some places.	Class III Fair rock

Below the argillaceous sandstone is the footwall quartzite, which comprises light to dark grey medium grained quartzites. The quartzites are fairly well-bedded and are highly siliceous apart from the arkosic and gritty zones.

The basal conglomerate group lie on top of the argillaceous sandstone. The lower members are thickly bedded to massive fine-grained sandstones. Higher in the succession, they gradually become coarser-grained and ferruginous, and several poorly sorted conglomerate and grits are encountered in the upper members.

1.8 STRUCTURAL

1.8.1 Faults

There are two major faults within the Konkola Mine licence area [7]. These are the Lubengele and the Luansobe faults. Minor faults occur in some areas of the mine, with a maximum throw of about 15.0m. In the far north area at Number 1 Shaft these are on the 3000mN area striking NW/SE and around the nose area striking NE/SW. Around the nose area the ore body is flatter and the minor faults are associated with increased intensity of the jointing and leaching.

1.8.2 Bedding

The ore body is extremely well bedded and individual planes are prominent structural features. True bedding plane spacing varies from a few centimetres to a metre. The combination of bedding, together with cross jointing and faulting affect the block size giving rise to various stability problems within the hanging wall and stope pillars.

1.8.3 Jointing

Two categories of joints are present in the Number 3 Shaft area namely the oblique joints (JA) and cross-joints (JB). The oblique joints strike normal to the bedding and dip opposite to it. The cross- joints strike normal or sub-normal to bedding and dip steeply. The frequency of the oblique jointing on both the west and north limbs is higher in comparison to the cross-joints due to tectonic activity related to the main fold. However, in certain locations with mini folding (4130-4150mN area) where the localised fold axis trends from Northwest to Southeast, cross jointing is more intense. Both joint types show shear as well as tensional movements.

Structural discontinuities derived from the dip and dip directions of mapped data are summarised in the Number 3 Shaft geological environment in section 1.10.

1.8.4 Leaching

The area on the west limb, Number 3 Shaft area, is associated with the effects of leaching which is mainly as a result of faulting and joint intensity. This has resulted in intense weathering characterized by oxidation of sulphide minerals to malachite and chrysocolla, together with some kaolinisation and extensive limonite staining on joints and bedding. During development of excavations, ravelling, slaking and swelling occur resulting in minimal stand-up time.

1.9 HYDROGEOLOGY

Konkola Mine is among the wettest mines in the world pumping about 300,000 m^3 per day equivalent to a water to ore ratio of 58:1.There are three water bearing horizons at the Konkola Mine property namely; the hanging wall aquifer, foot wall aquifer and the lower porous conglomerate [7].

The hanging wall aquifer is a massive dolomite, interbedded by siltstone, sandstone and dolomite. It provides about 35% of the mine's water release (140 000 m^3 per day) and 70% of the Number 3 Shaft's water due to its proximity to the anticlinal structure and the cross-fault at 3850mN location.

It is essential that the water table be lowered below the cave line (65°) before production can commence. This is achieved by drilling dewatering holes.

The footwall aquifer comprises of the footwall conglomerate, footwall sandstone and porous conglomerate. It accounts for 40% of the mine's water release for both Number 1 and 3 Shafts. It is recharged by underground water systems. The footwall aquifer can also be drained off due to its high porosity and permeability. Pilot holes are drilled into the drain drive, which is located at a lower elevation than the footwall haulage. Footwall haulages are mined well before production, in some cases about two years in advance thus providing effective means of advance dewatering. The pebble conglomerate is not connected to the two aquifers and thus does not require dewatering.

1.10 NUMBER 3 SHAFT GEOLOGICAL ENVIRONMENT

1.10.1 Eastern Area A (160mW-470mW)

This zone has been affected by tensional cracking due to the presence of a fault at the 970mW location representing weak ground and expected to be wet. Ore shale is a massive grey inter-bedded sandstone and shale, steeply dipping (+70°) with well developed NW-SE jointing. The ore body ranges in thickness between 2.0m and 4.0m and passes eastwards into coarse sandstone. Assay foot wall is above geological foot wall [18]. The summary of the mapped data for the eastern area is shown in table 1.3 below.

Table 1.3: Summary of Bedding and Joint orientations

SET	DIP	DIP DIR	SPACING (m)	APERTURE	INFILL	SURFACE CONDITION	RQD
JBEDDING	34°	353-000°	0.15-0.30	<3mm	Soft to medium sand fill.	Rough undulating with moderate weathering	40
JA	72-80°	221°	0.70	1cm	Medium to soft shearing gouge.	Slickensided, indicates shearing displacement on joint	40
JB	76°	120°	0.50m	<1mm	Clean to Quartz coarse grained infill	Rough gouge stronger than wall rock	40

1.10.2 Western Area B (470mW-1300mW)

Ore shale consists of alternating bands of grey, finely bedded siltstone and white calcareous sandstone, with abundant mica along the bedding planes. Average dips are 35°- 40° between 400mW to 850mW and 60° - 80° between 850mW to 1300mW. Average thickness is 4- 6m and the copper grade is 3.5% at both locations. Assay footwall is below the geological footwall by about 0.5m.

As a result of the laminated nature of the ore shale, peeling of drives often occurs. Collapse of hanging wall due to kaolinisation of joints and bedding in the hanging wall quartzite occurs locally during stoping [18]. The summary of the mapped data for the western area is shown in table 1.4 below.

Table 1.4: Summary of Bedding and Joint orientations

SET	DIP	DIP DIR	SPACING (m)	APERTURE	INFILL	SURFACE CONDITION	RQD
JBEDDING	65-70°	022-028°	0.05-0.10	Tight to <1mm	Clean to fine grained soft shearing infill (mica)	Rough to smooth undulating.	60
JA	60-60°	141-185°	1.0-1.5m	Tight to <1mm	FeOx stain	Rough to smooth undulating.	60
JB	85-87°	283-292°	0.70-1.0	Tight to <1mm	Coarse grained, medium to hard quartz fill	Very rough	60
JC	75	253	>3m	Tight <1mm	Clean	Smooth undulating	60

1.10.3 Northern Area C (1300mW-3860mN): -

Ore is at dip of 5° to 15° faulted and heavily jointed. Calcareous units within the ore shale are leached and koalinised resulting in extensive need to support the hanging wall. Chalcopyrite is the dominant copper mineral, with minor Carrolite and Cuprite. Ore body thickness is between 5.0- 7.0m thick. Large blocks and slabs commonly break away from hanging wall as a result of jointing and faulting associated with weak bedding planes. Transfer of water from hanging wall to foot wall along joints and fault planes has resulted in an accumulation of residual water in the footwall rocks. Presence of water also reduces frictional resistance of the discontinuity planes to slip [18]. The summary of the mapped data for the northern area is shown in table 1.5 below.

Table 1.5: Summary of Bedding and Joint orientations

SET	DIP	DIP DIR	SPACING (m)	APERTURE (mm)	INFILL	SURFACE CONDITION	RQD
JBEDDING	15°-20°	260°	0.5-0.20	<1mm to 5mm	Kaolin FeOx	Smooth-planar	35
J1	60-80°	010-025°	0.5-1.25m	1mm-2mm	Silt/clean	Rough planar– rough undulating	35
J1c	75°	170-195°					35
2	70-90°	070-080°	0.5 – 1m	<1mm-3mm	Clean, FeOx and Silt	Rough planar	35
J2c	80°	225-250°					
J3	75°	135°	1.5 - 2m	3mm-5mm	silt	Rough planar	35

1.10.4 Central Area D (3860mN-3200mN)

Generally the ore body is of low grade (1.2% TCU).The assay foot wall lies about 0.5m above geological footwall at the top of unit ‘A’. Consequently, foot wall drives which are mined along g the geological foot wall have to be supported to prevent peeling. Unit ‘A’ is the weakest zone of the ore body and causes the breach of crown and rib pillars and closure of drill holes before charging. The average dip is at 20°[18]. The summary of the mapped data for the central area is shown in table 1.6 below.

Table 1.6: Summary of Bedding and Joint orientations

SET	DIP	DIP DIR	SPACING (m)	APERTURE (mm)	INFILL	SURFACE CONDITION	RQD
JBEDDING	14°-30°	132-234°	0.06-0.60	<1mm to 5mm	Clean to soft shearing	Smooth to rough planar	60
JA	76°	040°	0.06-2.0	1-5mm	Clean, soft shearing <5mm to hard gouge <5mm	Smooth to rough planar	60
JB	30-69°	010-040°	0.2-2.0	0.1-1.0mm	None	Smooth	60
JC	78°	099°	0.20-2.0	0.1-1.0mm	Clean to hard shearing gouge <5mm	Smooth planar to very rough	60

Experience at Konkola Mine has shown that the geology of the mining licence area is generally consistent down dip. The conditions described in the Zones above are projected to the lower elevations. Variations at a smaller scale are anticipated especially the intensity of jointing, but not so much as to affect the overall mine plan for the zone.

Surface boreholes drilled previously and geological mapping of exposed areas have been used to create the model used in the mine plans.

CHAPTER 2

LITERATURE REVIEW

2.1 STRESS ENVIRONMENT

In-situ stress measurements at Konkola Number 3 Shaft were conducted in 2001. [26] Of the three measurements as in table 2.1, the first two were near mined out stopes and

are considered to have been influenced by induced stresses. The third measurement on the west limb has been accepted as virgin stress for the mine due to its distance away from mine activities. The sites and summary of the measurements are shown in Figure 2.1 and Table 2.1.

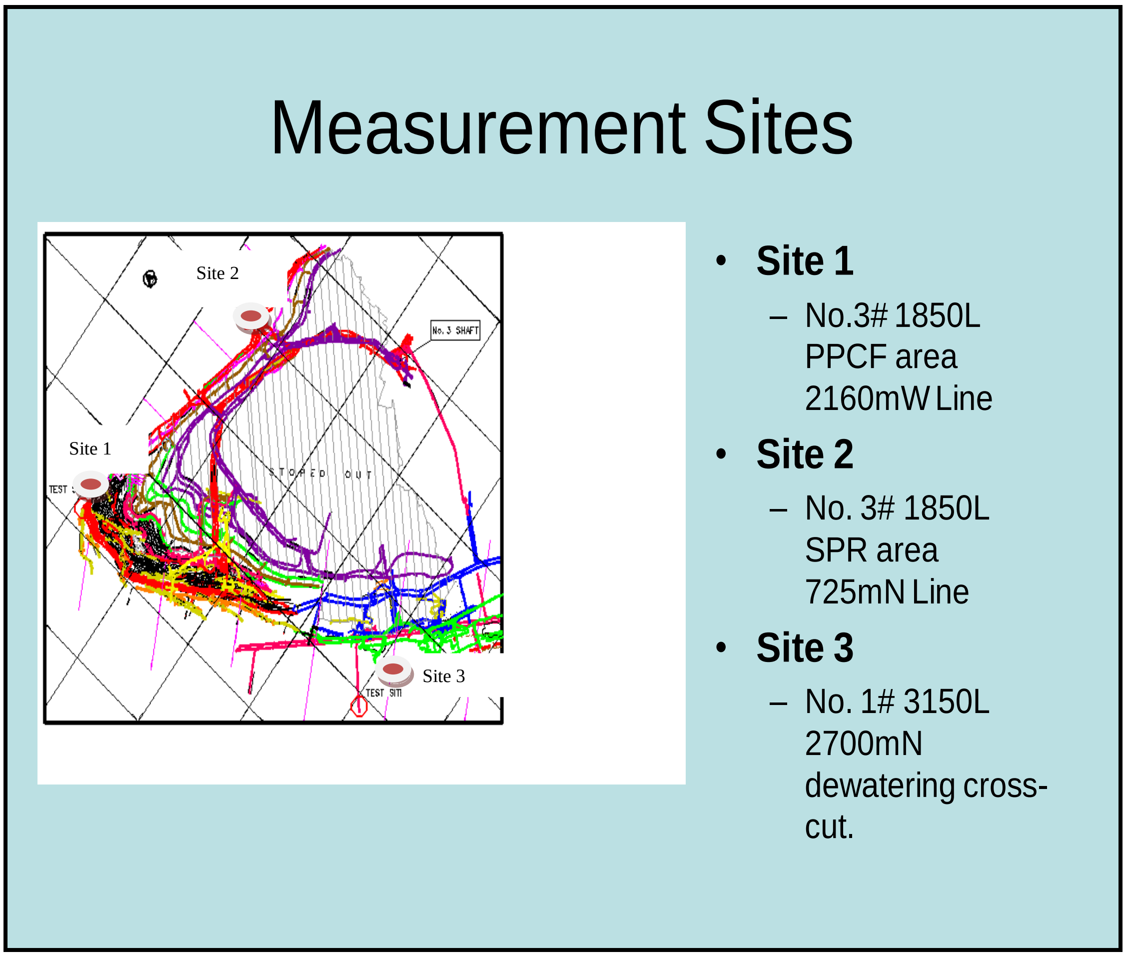


Figure 2.1: Location of in-situ Measurement Sites

Table 2.1: Summary of in-situ Stress Measurements

Stress Component	Magnitude	Bearing	Inclination
σ_1	39MPa	121^0	51^0
σ_2	18MPa	9^0	17^0
σ_3	15MPa	267^0	34^0

Maximum stress values that were recorded using the numerical stress analysis program (Phase 2D) before the in-situ measurements were done were 34MPa, 16MPa and 12MPa for σ_1, σ_2 and σ_3 respectively. These indicated stress level at different places before disturbing the ground by mining excavations and subsequent stoping out.

On the mine wide scale, the principal stress direction is aligned to the anticline axis and the axial fissure. Intermediate and minor stresses have a similar numerical magnitude and they have similar strike to the cross-faults intersected on the west limb between 2000mN and 3000mN.

2.2 ROCK MASS PROPERTIES

A large extent of the Number 3 Shaft rock mass falls within fair to poor ground conditions with the average Rock Mass Rating ranging from 30 and 55. A summary of the Ore shale rock mass properties for the geological zones at Number 3 Shaft, obtained from face mapping and borehole logging, are given in Table 2.2. Tables 2.3 and 2.4 show the rock mass ratings for the hanging wall quartzite and the footwall formations. The roof of the stopes is mostly located in the hanging wall quartzite for the open stoping and room and pillar mine layouts. Where the Post Pillar Cut and Fill is used, the roof of the sequential lifts is located in the ore horizon. Rock mass properties used in stability models for Number 3 Shaft formations are summarised in Table 2.5 [18].

Table 2.2: Summary of Ore Shale Rock Mass Ratings [18, 19]

ZONE	**DIP**	**THICKNESS (m)**	**RATING**		**DESCRIPTION**
			RMR	Q Value	
A EASTERN SECTION	70°	2-4	57	4.15	Fair
B WESTERN SECTION	60-80°	4-6	55	3.34	Fair to Good Rock
C NORTERN SECTION	10-15°	10-15	43	0.90	Poor to Fair Rock
D CENTRAL SECTION	25-35°	7-9	63.0	8.03	Good Rock

Refer to section 1.10 for raw data that was used to calculate rock mass rating.

Table 2.3: Summary of Hanging wall Quartzite Rock Mass Ratings [18, 19]

Rock Type	**Thickness (m)**	**Rating**			**Notes**
		Q-rating	RMR-rating	Description	
Hanging wall Quartzite	> 20	4. 6	58	Fair	Massive quartzite, some kaolinisation in bands
		30	75	Good to Very good	Massive quartzite, minor kaolinisation in bands
		1.1	45	Poor	Quartzite with kaolinised bedding planes

Q= (RQD÷Jn)×(Jr/Ja)×(Jw÷SRF) after Barton [3, 15, 23]

Where RQD is the Rock Quality Designation

Jn is the joint set number

Jr is the joint roughness number

Ja is the joint alteration number

Jw is the joint water condition reduction factor

SRF is the stress reduction factor

Refer to APPENDIX 3B for the explanation of the classification of Q-system.

RMR= ∑ of factors (UCS+RQD+ Joint spacing+ Joint condition+ Water condition) after Bieniawski [5, 15, 23]. The Rock Mass Rating System is presented in APPENDIX 3A

Table 2.4: Summary of Footwall Formations Rock Mass Ratings

ROCK TYPE	STRONG GROUND	MEDIUM GROUND	WEAK GROUND
	RMR	**RMR**	**RMR**
FOOTWALL SANDSTONE	59	65	54
POROUS CONGLOMERATE	52	48	41
ARGILLACEOUS SANDSTONE			
UNIT 1A	54	60	63
UNIT 1B	43	40	44
UNIT 2	69	67	75
UNIT 3	54	47	46
UNIT 4	61	62	62
FOOTWALL QUARTZITE	83	85	83
PEBBLE CONGLOMERATE	-	94	-

2.3 STABILITY OF SPANS

2.3.1 Hang wall Quartzite Roof Spans

Stability of the roof spans in the stopes is determined using the Mathews Stability Number (N') $=Q' \times A \times B \times C$ [15]

Where:

Q' is Modified Rock Tunneling Quality Index

A is a measure of the ratio of intact rock strength to induced stress. As the maximum compressive stress acting parallel to a free stope face approaches the uni-axial strength of the rock, factor A degrades to reflect the related instability due to rock yield. (APPENDIX 4A)

B is a measure of excavation surface. Joint which form a shallow oblique angle ($10\text{-}30^0$) with the free face are likely to become unstable (i.e. to slip or separate). Joints which are perpendicular to the face are assumed to have the least influence on stability. (APPENDIX 4B)

C is a measure of the influence of gravity on the stability of the face being considered. (APPENDIX 4C)

Table 2.5: Summary of Hanging wall Quartzite (HWQ) Roof Stability [18]

Rock mass Condition		**Parameters**			**Stability Number**
Description	Q-rating	A	B	C	
Poor	1.1	0.70	0.20	2	0.3
Fair to good	4.6	0.75	0.3	3	3.1
Very good	30	0.80	0.30	5	36

The stability of the roof span of the full width overcut was analysed based on the hydraulic radius or shape factor and Mathews/ Potvin's stability number (Potvin et al 1989)[15]. The hydraulic radius values are used to estimate unsupported apparent dip span limits for 10m and 12m wide stopes for a given rock mass condition. The results are shown in Figure 2.2 and Table 2.6 [22].

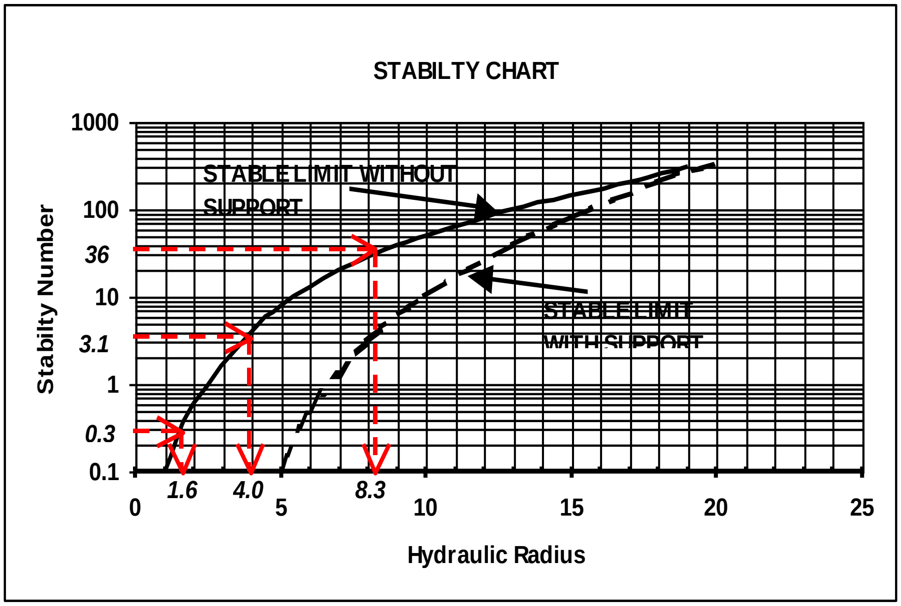


Figure 2.2: POTVIN Stability Graph [15]

Table 2.6: Unsupported HWQ Roof Span Limits

Rock mass condition	Stability Number	Hydraulic radius	Unsupported dip span	
			10m wide overcut	12m wide overcut
Poor	0.3	1.6	< 10	< 10
Fair to Good	3.1	4	15 - 40	10 - 24
Very good	36	8.3	>100	>100

Figure 2.2 and the Table 2.6 above show the limit of the unsupported span with different stability numbers of 0.3, 3.1 and 8.3. It showed that the unsupported span for 0.3 is less than 10m and 15m to 24m for 3.1 stability number. For the 8.3 stability number the unsupported span is greater than 100m [22].

2.3.2 Stability of Ore Shale Roof

From the mapped data in section 1.10, the stability of the ore shale in the roof is shown in the table below:

Table 2.7: Stability of Ore Shale Roof

ZONE	Q'	A	B	C	N	STABLE SHAPE FACTOR	CAVE SHAPE FACTOR
A EASTERN SECTION	4.56	0.58	0.95	5.60	14.07	6.1	10.9
B WESTERN SECTION	6.91	0.58	0.8-1.0	4.5-6.8	27.3	7.7	12.1
C NORTERN SECTION	10.0	0.44	0.3	1.42	1.87	3.1	7.2
D CENTRAL SECTION	5.6	0.72	0.3	1.93	2.33	3.5	7.7

Stable limit without support in the roof of the ore shale was also analysed in eastern, western, northern and central areas as from the data mapped in 1.10 section. The calculations showed the stable shape to be 6.1m, 7.7m, 3.1m and 3.5m respectively. The cave shape was calculated to be 10.9m, 12.1m, 7.2m and 7.7 m respectively.

2.3.3 Stability of Ore Shale Sidewalls

From the mapped from section 1.10, the stability of the sidewall is shown in the table below:

Table 2.8: Stability of Ore shale Sidewalls

ZONE	Q'	A	B	C	N	STABLE SHAPE FACTOR	CAVE SHAPE FACTOR
A EASTERN SECTION	4.56	0.58	0.3	5.6	4.44	4.1	8.5
B WESTERN SECTION	6.91	0.58	0.2	6.8	5.45	4.3	8.9
C NORTHERN SECTION	10	0.44	0.95	1.42	5.94	4.5	9
D CENTRAL SECTION	5.6	0.72	0.8	1.93	6.23	4.6	9.2

Stable limit without support in the sidewalls of the all shale was also analysed in eastern, western, northern and central areas as from the data mapped in 1.10 section. The calculations showed the stable shape to be 4.1m, 4.3m, 4.5m and 4.6m respectively. The cave shape was calculated to be 8.5m, 8.9m, 9.0m and 9.2 m respectively.

2.4 FAILURE MECHANISMS

2.4.1 Spalling and Sloughing of sidewalls

Generally, rock failures at Number 3 Shaft are initiated from the bedded weak units 'C' and 'D' of the ore shale. This weakness has been due to percolation of water through weak bedding planes especially in the nose area that has caused leaching and weathering. Some sections of the mine however, have competent units 'C' and 'D' e.g. the area between 4390mN and 4480mN.

In the Over Cut and Bench (OCB) mining method, instability mostly occurs at the benching stage when the width to height ratio is below 2:5 [22]. Spalling and sloughing in units 'C' and 'D' occur leaving a ledge at the bottom of the pillar and a brow at the roof contact. This failure is exacerbated by the geological discontinuities present in the rock mass. [18]

Unit 'A' of the ore horizon forms the assay footwall and the geological footwall contact. It is 0.15m to 1.15m thick consisting of finely inter-laminated grey siltstone to pink/brown dolomite. In most places it is deeply weathered to an incompetent micaceous sandy mud. It therefore plays a role in stability and dilution of excavations at Konkola Mine. It has a low tensile of about 2MPa and compressive strength of about 10MPa and easily sloughs out on exposure. When left in place, without exposure, it acts as a stress relief valve contributing to the stability of excavations.

2.4.2 Structural controlled failures

The current depth of the Number 3 Shaft Mine falls in the intermediate depths between 1850 feet Level and 2450 feet Level. This means geological discontinuity patterns have a significant role in the failure mechanisms and constitute an integral function in rock mechanisms design. Free fall and slide/rotation of wedges in the roof and sidewall do occur at joint intersections with particular orientation characteristics. Where the centerline of the wedge formed falls within the baseline of the wedge, free fall may occur. Where falls outside of the baseline and the plane/line of intersection of the joints is steeper than the angle of friction of these joints, slide/rotation may occur.

Wedges occurring in the roof are in a tensile zone, which, initially was in compression before the onset of mining the excavation. This stress change and the magnitude of the tensile stresses cause a deterioration of joint strengths resulting in further increase in the propensity for rock falls.

2.4.3 Stress loading in abutment areas

In abutment (remnant) areas like the 1660 feet level and the 3800mN OCB6/5, stress loading as benching progressed led to floor heaving as the footwall was exposed. The mechanism was bulking of the bedding planes in the units 'C' and 'D' mostly in the rooms and cross-cuts that were less confined. Shear displacement on bedding and joints extending over 10cm distances were observed in excavations within this locality.

2.5 MINING METHOD SELECTION

Mining method selection is based on magnitude of ground displacement and extraction voids and pillars. There are so many factors that are considered in mining method selection system and these include strength of the ore body and country rock, dip of the ore body, thickness of the ore body e.t.c. Mineral recovery from sub surface rocks involves:

- Development of physical accesses to mineralized zone;
- Liberation of the ore from the enclosed hoist rocks; and
- Transport of this mineral to the mine surface.

2.5.1 Classes of Mining Methods

The mining methods fall into three broad categories i.e. unsupported, supported and caving mining systems [3].

Unsupported mining methods are employed in sufficiently competent ore rock in which blocks of ore (low grade mostly) or barren zones are left un-mined (or are possibly recovered later) to provide local and regional support. This class includes room and pillar, stope and pillar, sublevel open stoping, vertical crater retreat and shrinkage methods [3, 5].

In supported mining methods, production openings will not remain standing during their active life and major caving or subsidence to the surface is not tolerated. These systems are employed in ore/rock that is incompetent to moderately competent. This includes cut and fill mining methods [3].

2.5.2 Mining Method Selection Criteria

Broad selection considerations for underground mining methods are listed in Table 2.9 while Table 2.10 gives specific geological and geotechnical criteria.

Table 2.9 Underground Mining Method Selection Considerations

EVALUATION PARAMETER	CONSIDERATIONS	CRITERIA	SUITABLE METHOD
Geotechnical	Lithology Ground water Geophysics Ore genesis	E.G. -Ore Strength-Strong/ Moderate -Waste Strength- Strong/ Moderate	E.G. Room and Pillar
Mineral occurrence	Continuity of ore zones within ore zone (geologic grade)		
Ore body configuration	Dip Plunge Size Shape	E.G. -Beds-Thick/ Thin -Ore Dip-Flat/ Moderate	E.G. Room and Pillar
Safety/regulatory	Labour intensity of method Degree of mechanisation Ventilation requirements Refrigeration requirements Ground support requirements Dust, noise & gas controls		
Environmental	Subsidence potential Groundwater contamination Noise controls Air quality controls		
Economic	Mineable ore tonnes Ore body grade Mineral value Capital costs Operating costs		
Labour/political	Costs, influences		

Table 2.10: Geotechnical and Geological Mining Methods Selection Criteria

MINING METHOD	ORE STRENGTH			WASTE STRENGTH			BEDS.		VEINS			ORE DIP		
	WEAK	MODERATE	STRONG	WEAK	MODERATE	STRONG	THICK	THIN	NARROW	WIDE	MASSIVE	FLAT	MODERATE	STEEP
Room & Pillar		**x**	**x**		**x**	**x**	**x**	**x**				**x**	**x**	
Sublevel Stoping		**x**	**x**			**x**			**x**	**x**	**x**			**x**
Shrinkage stoping		**x**	**x**	**x**	**x**				**x**	**x**			**x**	**x**
Cut & Fill		**x**	**x**	**x**	**x**				**x**	**x**	**x**		**x**	**x**
Square set	**x**			**x**	**x**				**x**	**x**	**x**		**x**	**x**
Block caving	**x**	**x**		**x**	**x**					**x**	**x**			**x**
Sub level caving		**x**	**x**	**x**	**x**					**x**	**x**			**x**
Long wall	**x**	**x**		**x**				**x**				**x**	**x**	

Where **x** denotes the suitability of the selected mining method in terms of ore strength, waste strength, beds, veins and ore dip. E.G. Room and Pillar mining method should have the following:

- Ore Strength must be strong or moderate;
- Waste Strength must be strong or moderate;
- Beds must be thin or thick ; and
- Ore dip must be flat or moderate

2.6 NUMBER 3 SHAFT EXISTING MINING METHODS

The current existing Mining Methods at Number 3 Shaft are:

- Overcut and Bench;
- Post Pillar Cut and Fill;
- Cascade;

- Modified Overcut and Bench;
- Drift and Fill; and
- Sub Level Open Stoping.

2.7 CURRENT MINING METHODS

2.7.1 Over cut and Bench – OCB

This is a variant of Room and Pillar mining method suitable for flat dipping ore bodies extending on a large coverage. It was used in levels above the 1850 feet level. The dip of the ore body in the Number 3 Shaft 'nose' area is about 10°. This mining method is mechanized with self-propelled single boom drilling jumbos and large capacity trucks and loading equipment. Most of the major developments are mined within the ore body.

Inclined ramps with dimensions of 4.0m high by 4.0m width are mined at 7^o to 9^o gradients on the hanging wall and footwall at pre-determined intervals up dip of 100.0m. From the footwall ramp, breakaways at 8^o or suitable angles between 90^o to 75^o, depending on ground conditions, are mined off the ramp along strike and at a suitable gradient to intersect the hanging wall. These are then developed on a flat up to the hanging wall ramp position forming the limit of the panel. This establishes the over cut and the dimensions generally are 7.0m width × 2.7m height (down dip side) though dependent on ground conditions.

Ore is recovered in the roof by using a boomer machine which drills holes up to the hanging wall contact with the aid of the mine lay out. The holes are then blasted with explosives at the breakaway position. Ore on the footwall is recovered by benching operation. This operation continues from the breakaway position retreating to the hanging wall ramp position to form strike rooms (dimensions of 7.0m width, 10.0m height, from the over cut floor level to geological footwall). Service access to the drilling bench is mainly through the hanging wall ramp. As mining progresses, dip room cross cuts (4.0m width ×3.5m height) are mined at predetermined intervals dependent on pillar sizes.

On completion of the bench, ideally the dip rooms should be widened to dimensions of 9.0m w ×10.0m height of the ore body to maximize the recovery of ore. However, ground conditions in some areas deteriorate to levels not safe for equipment and personnel to work in.

2.7.2 Post Pillar Cut and Fill

Post pillar cut and fill (PPCF) mining method is a form of inclined Room and Pillar mining which advances up dip in a series of lifts, each at the height of a development drift. Waste or tailings are used as backfill material. The fill provides the working floor for the subsequent lift. [7]

Vertical Post Pillars are created from footwall to hanging wall to stabilize the hanging wall in the immediate mining area.

Depending on the dip and thickness of the ore body, the pillars can be of differing height. They can range from 3.5m in the hanging wall drive to 6.0m in the foot wall drive. The height to width ratio of these pillars is normally over 1.5 as a result the pillars crush with time. Placement of backfill material around these pillars confines them and allows the maintenance of considerable residual strength.

PPCF is also a variant of Room and Pillar mining method. The disadvantage of this method apart from men working below the laminated ore shale is the costly operation of re-supporting the working environment. This method is also associated with low productivity due to constraints in opening up more mining faces.

2.7.3 Cascade Mining

The method is suitable for shallow dipping ore bodies where regular drilling patterns and safe mechanized extraction are required. The method requires a drilling drive positioned on the footwall between draw point crosscuts. Stope rings are drilled and blasted from this drive. The blasted ore gravitates to the bottom of the stope and is loaded into tips using LHD's. The Rib pillar is drilled from the pillar crosscut while the crown pillar is drilled and blasted from the old chamber hanging wall drive or crosscut. Careful grade control is required to prevent excessive dilution.

2.7.4 Modified Overcut and Bench

Modified Overcut and Bench mining method is an in ore body mining method suitable for ore bodies with dips ranging from flat to 30^{o} and thick ore bodies . A combination of steeper dipping (25^{o} – 30^{o}) and thick (>12m) ore body is not favourable, as very high rooms will be created rendering the method unsafe.

This mining method at Number 3 Shaft was introduced in areas where an over-cut and bench (OCB) mining method was designed above 1850 feet level. The shaft experienced severe mining induced stresses (>35MPa) and time related deterioration of ground conditions in the areas where OCB mining method was used to extract the shallow dipping wide ore body. After a review of the conditions, it was decided to change the mining method to Cascade mining to recover ore in the affected area and modify the OCB method and use a Modified Overcut and Bench method (MOCB) for new mining sections below 1850 feet level.

The MOCB mining method requires developing a regional pillar at the top and bottom of the stope. The extraction drives at the bottom and top of the stope are the main arteries between which development of the stopes takes place. Each panel consists of a rib pillar and a stope. The stope is extracted by overcutting and benching. An overcut raise is developed over the entire back length of the stope and later widened to the full size of the stope of about 10.0m [22].

2.7.5 Drift and Fill

This mining method is suitable for flat dipping thin ore bodies (about 6m). The drift is mined from the footwall ramp as a single pass along strike up to the end of the panel. Support of the hanging wall is done progressively with advance of mining. A continuous strike pillar (about 3m) is left. Backfilling of the stoped drift is conducted before the next up-dip drift is excavated.

2.7.6 Sub Level Open Stoping (SLOS)

As the name implies, the ore body is sub divided into sublevels up dip with an extracting haulage below. Each stope is designed with a rib pillar of about 4.0m and crush crown pillar of about 4.0m and the stope has rings which are drilled conventionally with a long hole drifter machine. The method is applied to both steeply and flat dipping ore bodies. Number 1 Shaft uses SLOS where ore fall by gravity after blasting due to steep dips while Number 3 Shaft uses both SLOS by scraping in shallow dips and SLOS by gravity in steep dips.

Under certain conditions, the ore in the pillars can be recovered after the stope rings are blasted or pillars can be left to crush on their own.

Mining in sublevel open stoping is carried out from horizontal sub levels, at determined vertical intervals that range between 15.0m to 30.0m vertical height. The sub levels are mined within the ore body at elevations between the two main levels. Ore is broken by drilling and blasting from the sub level drifts.

The blasted rings cut a vertical slice of ore which breaks up and falls to the chamber (bottom) of the stope, from where it is recovered at the main level.

CHAPTER 3

METHODOLOGY

3.1 Preamble

The research involved collection of scan line mapped data. The collection of data at the mine site was done in two parts. The first part involved collection of the existing mapped data and the second part involved underground visits for underground mapping. The collected data was analysed and the information was subjected to evaluation using specialised geotechnical softwares which included Phase 2D. The information was used to build and develop the computer based spreadsheet to select and rank the suitable mining method for geotechnical design. Finally, the top three mining methods were further analysed based on productivity, recovery and dilution.

3.2 Data Collection

Data collection on site was by scan -line mapping which involved underground visits and collection of the mapped data. A procedure in scan line mapping is very important as this helps in achieving the desirable results.

To achieve this, a balance should be achieved between scan-lines situated in the strike direction and those in the dip direction i.e. a good balance of drives and cross-cuts. This should not deviate from a 60:40 ratio in this regard. This provides enough information for data processing. The following were obtained during scan-line mapping: joint spacing, joint orientation, joint condition, rock samples for testing the uni-axial compressive strength.

The scan-line must not cross a main structural feature i.e. fold axis or major fault. Separate scan-lines should be placed on either side of such features. Since average values can be misleading and the weakest zones may determine the response of the whole rock mass, these zones must be rated on their own. Narrow and weak geological features that are continuous within and beyond the stope or pillar must be rated separately.

Scan-lines must be positioned so that they are horizontal, except where measurements are taken on a ramp, and situated at a height of 1.5 m off the floor in full size excavations.

When 'window mapping' i.e. pre-selecting portions of development that are defined by readily identifiable features, care must be taken to ensure that they are representative of the general rock mass. Where, significant exposure within the same area or rock type is found i.e. 500 m of footwall or ore body drives in the same direction, then window mapping may be practiced. In this case a 30 m exposure in 100 m will be sufficient.

When mapping an excavation in only one direction in a specific area, the following situations were evaluated to obtain a representative fracture frequency [18]:

i. If all the discontinuities are present in the sidewalls, it was established whether they all intersect the horizontal scan line;
ii. If they all do not intersect the horizontal line, it was measured on a vertical line as well;
iii. If a set is parallel to the side-wall, measurements were taken on a line in the hanging wall that is at right angles to the sidewall.

This conflicting situation of different sampling procedures can be resolved if the sum of the measurements is divided by a factor to arrive at the average frequency.

3.3 Interpretation of Collected data

The research then highlights geotechnical considerations in the design of the selected mining method using the raw data information obtained in section 3.2. It brings out the optimal design parameters that can make for economical and safe extraction bearing in mind the prevailing ground conditions. This is possible to arrive at by modelling using numerical methods. Support design for the selected mining method must be ascertained by empirical method (NGI Q system). Drilling and blasting operations also have to conform to the prescribed sizes of the designed stopes and blast design respectively.

CHAPTER 4

GEOTECHNICAL CHARACTERISTICS OF THE INSITU ROCK AND MINING METHODS RANKING

4.1 ROCK MASS RATING

Underground mapping was conducted in the footwall, ore shale and hanging wall rock formations in the nose area of the ore body. For each measurement, the following were noted: discontinuity type (bedding, joint or foliation), rock type, discontinuity spacing, dip and dip directions, condition of the discontinuity (fill type, roughness and weathering), Rock Quality Designation (RQD), Uni-axial Compressive Strength (UCS), and Ground water condition. Table 3.1 shows the major discontinuities that where mapped in the nose area. [19]

Table 4.1: Rock Mass Rating for the Ore shale

Set	Dip	Dip Direction	Spacing (m)	Aperture (m)	Infill	Surface Condition	RQD	RMR
Bedding Plane	15-20°	260°	0.15-0.20	<1– 5	Kaolin FeOx	Smooth /Planar	35%	45
J1	60-	010-025° 170-195°	0.5-1.25	1-2	Silt/clean	Rough /Undulating	40%	46
JR	70-	070-080°	0.5 – 1	<1-3	Clean, FeOx	Rough /Undulating	42%	46

The UCS of the Ore shale was found to be 80MPa to 120MPa established in laboratory using point load testing machine from the samples that were collected underground. The UCS and RQD of other rock units are summarised Appendix 1A and 1B.

The Rock Mass Rating (RMR) for the hanging wall quartzite was also determined by the mapped data as in Table 4.2

Table 4.2: Rock Mass Rating for the Hanging wall Quartzite

Set	Dip	Dip Direction	Spacing (m)	Aperture (mm)	Infill	Surface condition	RQD	RMR
J1	550	3300	2-6	Tight	Nil	Straight/ Rough	75%	76
J2	800	030°	4	1	FeOx	Very Rough / Undulating	82%	79
J3	75°	240°	Nil	<1-2	Clean and FeOx	Rough /Planar	80%	71

The UCS of the hanging wall quartzite was found to be 220MPa established in the laboratory using the point load testing machine from the sample that were collected underground. For UCS and RQD of other rock units are given in Appendices 1A and 1B.

The Rock Mass Rating (RMR) for the Footwall Conglomerate was also determined by the mapped data as in Table 4.3.

Table 4.3: Rock Mass Rating for the Footwall Conglomerate

Set	Dip	Dip Direction	Spacing (m)	Aperture (mm)	Infill	Surface Condition	RQD	RMR
J1	750	3000	1m	0.1-1mm	Kaolin	Straight/ Rough	43%	61
J2	600	030°	0.3m	1mm	FeOx/Silt	Very Rough / Undulating	65%	60
J3	45°	2000	1m	<1-3mm	FeOx	Rough/ Planar	75%	69
JR	800	0750	Nil	1-5mm	FeOx	Rough Planar	45	69

The Uniaxial Compressive Strength of the footwall conglomerate was found to be 150MPa established in the laboratory using the point load testing machine. For UCS and RQD of other rocks unit are given in Appendices 1A and 1B.

The above information aided in ranging of the suitable mining methods using the University of British Columbia (UBC) mining selection tool. The condition of the ore zone, hanging wall and footwall is very important in this tool.

4.2 MINING METHODS RANKING SYSTEM

The initial analysis to rank the possible mining method using University of British Columbia (UBC) numerical ranking criteria was done taking into account the Geometry and Grade Distribution, Rock Mass Rating and Rock Substance Strength of the mining environment.

The initial analysis in the search for the most appropriate mining method(s) started with the Numerical Ranking criteria by Nicholas.[18] Though his method is viewed as not being accurately representative, it still sets an initial step towards the direction in which appropriate mining methods should follow.

To consolidate the Nicholas approach, an analysis was conducted based on the physical and spatial characteristics of the ore body and country rocks in relation to each mining method, taking into consideration the stoping requirements, the sizes of openings as well as the support requirements to mention but a few (dip of the ore body, strength, ore body size e.t.c) as in Table 4.9.

These UBC and Nicholas criteria together give a standard approach in selecting the appropriate mining method.

4.2.1 Numerical Approach by Nicholas (1981)

This selection criterion is neither definitive nor quantitative because economic, technological and environmental considerations are not taken into consideration. Only geologic and physical characteristics are considered.

The optimal properties of each mining method are compared with those of the ore body in question and based on how close they match or agree with each other, points are allocated out of 4 with -49 being the minimum. The method with the minimum points allocated cannot be employment and is eliminated.

The properties considered include the geospatial and rock mechanics properties of the ore body, hangingwall and footwall as well as the grade. The allocation of points or grading is done according to the Table 4.4:

Table 4.4: Classes, Points and Grades of Ranking

CLASS	**1**	**2**	**3**	**4**	**5**
POINTS	**4**	**3**	**2**	**1**	**-49**
GRADE	**Very Good**	**Good**	**Fair**	**Poor**	**Very Poor**
COMMENT	Condition fully met	Condition satisfactorily met (acceptable)	Condition met with average satisfaction	Condition met with little satisfaction	Condition not met at all

An example is that of sublevel open stoping, which requires a deposit which is fairly steep (>45^0-50^0) or greater than 70° for good recovery of ore. At the same time, the Number 3 Shaft in the 2120feet Level ore body has an average dip of about 70°, thus this condition is fully met by sublevel open stoping and is graded as very good and four points are allocated.

The points scored by each mining method for each of the given properties are added and ranked chronologically with the highest at the top of the list or table.

4.2.1.1 Properties considered for ranking

4.2.1.1.1Ore body geometry

i. General shape/width

The general shape of the ore body can be categorised as:

Equi-dimensional - all dimensions are equal in all directions

Platy tabular - two dimensions are many times the thickness which does not usually exceed 100m.

Irregular - dimension vary over very short distances

ii. Ore thickness

This is the measure of the perpendicular distance from the hanging wall to the footwall of the ore body. It is categorised as shown in the Table 4.5.

Table 4.5: Ore body thickness

Category	Thickness(m)
Narrow	**< 10**
Intermediate	**10 -30**
Thick	**30 -100**
Very thick	**>100**

iii. Dip

This is the measure of the deviation of the ore body from the vertical axis. The categorisation is shown in the Table 4.6:

Table 4.6: Ore body dip

Category	Dip
Flat	**< 20°**
Intermediate	**20 - 55°**
Steep	**> 55°**

4.2.1.1.2 Grade distribution

This is the measure of the variation of grade within the ore body. It is defined as follows:

i. Uniform – The grade at any point in the deposit does not significantly vary from the mean grade of the deposit.
ii. Gradational – Grade varies within different zones of the ore body.
iii. Erratic – Grade varies drastically within short distances and has no characteristic pattern of change.

4.2.1.1.3 Rock Mechanics Characteristics

i. Rock substance strength

This is given by the ratio of uni-axial strength and overburden pressure.

Its categorisation is shown in the Table 4.7:

Table 4.7: Ore body Strength

Category	Rock Substance Strength
Weak	< 8 MPa
Moderate	8 – 15 MPa
Strong	> 15 MPa

ii. Fracture frequency

This is the measure of the extent of fracturing in a rock by monitoring the occurrence of fractures within a distance of one metre. This shown in the Table 4.8:

Table 4.8: Fracture Frequency

Category	No. Of Fractures	RQD
Very close	> 16	0 -20
Close	10 - 16	20 - 40
Wide	3 -10	40 – 70
Very wide	<3	70 - 100

iii. Fracture shear strength

This measures the strength of the joints by measuring their resistance to shearing. This is categorised as follows:

(i) Weak – Where the joints are clean with a smooth surface or are filled with material with strength less than rock substance strength

(ii) Moderate – The joints are clean with a rough surface

(iii) Strong – Where the joints are filled with a material that is equal to or is stronger than the rock substance strength.

4.2.2 Ranking-Mining Method Selection Tool

The mining method selection tool is based on a version of the Nicholas approach [11], modified by the Rock Mechanics Group of the University of British Columbia (UBC) for selection of mining methods based on ore body characteristics. Selection involves summation and ranking of numerical values associated with ore body characteristics that reflect the suitability of a particular method. This interactive presentation of the selection process allows for the investigation of the influence of ore body characteristics on the selection of appropriate mining methods.

UBC mining method selection is a modified on-line version of the Nicholas approach for selection of mining methods based on ore body characteristics. [11] Selection involves summation and ranking of numerical values associated with ore body characteristics that reflect the suitability of a particular method. This interactive presentation of the selection process allows you to investigate the influence of ore body characteristics on the selection of appropriate mining methods. Table 4.9 shows an on-line mining method selection tool used in selecting mining methods most suitable for the ore body under study. [11] The parameters used are:

(i) Geometry and Grade Distribution

(ii) Rock Mass Rating

(iii) Rock Substance Strength

Table 4.9 UBC Mining Method Selection Tool

Selection Basis	Ore body Characteristics	Mining Method Rankings
Geometry and Grade Distribution	**General Shape:** Massive **Ore Thickness:** Intermediate (10-30m) **Ore Plunge:** Flat (less than 20deg) **Grade Distribution:** Gradational **Depth:** Deep (more than 600m)	(best) **Sublevel Stoping (37)** **Cut and Fill Stoping (27)** **Room and Pillar (26)**
Rock Mass Rating (after Bieniawski 1973)	**Ore Zone:** Medium (40-60) **Hanging Wall:** Strong (60-80) **Footwall:** Strong (60-80)	**Sublevel Caving (22)** **Block Caving (20)** **Top Slicing (17)** **Square Set Stoping (9)** **Open Pit (-14)** **Shrinkage Stoping (-25)** **Long wall Mining (-30)**
Rock Substance Strength (unconfined compressive strength / principal stress)	**Ore Zone:** Medium (10-15) **Hanging Wall:** Strong (more than 15) **Footwall:** Strong (more than 15)	(worst)

Sublevel Stoping, cut and Fill Stoping and Room and Pillar are the top three mining methods that proved to be the most suitable and are selected for further analysis.

4 .2.3 Analysing the top three mining methods

The three selected mining methods were compared in terms of the advantages and disadvantages. Ore recovery, productivity and dilution are the many key factors at Konkola that can influence the selection of the mining method at the nose area of the ore body.

4.2.3.1 Advantages of the three mining methods

The selected mining methods have to be compared in terms of their advantages. Table 4.10 shows the advantages of the selected mining methods.

Table 4.10: Selected Mining methods Advantages

Sublevel Stoping	Cut and Fill	Room and Pillar
Moderate to high productivity per employee shift	Moderate to high productivity per employee shift	Moderate to high productivity per employee shift
Moderate mining costs	Moderate production rates	Moderate mining costs
Moderate to high production rate	Good selectivity and sorting; can use waste as fill	Moderate to high production rates
Readily Mechanised	Low development costs	High flexibility; method can be modified and can operate on multiple levels simultaneously
Not labour intensive	Moderate capital investment; adaptable to mechanisation	Readily mechanised; suitable for large mobile equipment
Low breakage costs; fairly low handling cost	Versatile, flexible and adaptable	Not labour intensive; extensive skill not required
Non entry method so little exposure to un safe conditions	Surface waste(tailing e.t.c) can be deposited as fill underground	Selective method; allows waste or low grade to be left in place
Easily Ventilated	Subsidence is not allowed	Limited development requirements

Unit operations can be carried on simultaneously		Multiple working areas can be operated simultaneously
Recovery 75%	Recovery 90- 100%	Recovery 60% to 80% without pillar
Dilution 20%	Low dilution 5% to 10%	Dilution 10% to 20% [26]

4.2.3.2 Disadvantages of the Sublevel Stoping, cut and Fill Stoping and Room and Pillar mining methods

The selected mining methods have to be compared in terms of their advantages. Table 4.11 below shows disadvantages of the selected mining methods.

Table 3.11 Selected Mining methods Disadvantages

Sublevel Stoping	**Cut and Fill**	**Room and Pillar**
Fairly expensive, development costs are high	Fairly high mining costs	Expensive and continual ground control maintenance of back required if hanging wall and ore not competent
Non selective, low flexibility	Costly handling of waste which maybe up to 50% of mining costs if not used as backfill	Large capital expenditures for extensive mechanisation
Long hole drilling requires careful alignment(<2% deviation)	Filling complicates cycle, causing discontinuous operation	Some ore loss in pillars

Large blasts may cause excessive vibration, air blast and structural damage	Must provide stope access for mechanised equipment	Good ventilation is difficult due to low air velocities in large opens
Labour intensive	Labour intensive, requiring skilled miners and close supervision	Orebody depth should be moderate
Lower productivity per man shift	Consolidation of fill may result in some ground settlement and instability [26]	

4.3 FURTHER ANALYSIS OF THE TOP THREE MINING METHODS

The mining method to be selected for further analysis is to provide high recovery with backfill material and low dilution. It should also provide high safety standards and must be profitable. The mining method with high recovery has to be subjected to geotechnical analysis.

From the top three mining methods, although Sublevel Open Stoping is among the top three mining methods, it is suitable for steeply dipping ore bodies. The development costs in SLOS are quite high. This leaves two mining methods for further analysis. Room and Pillar had a recovery of 60% to 80% and dilution ranged from 10% to 20% whilst Cut and Fill had a recovery ranging between 90% to 100% and dilution ranged from 5% to 10%. It is against this analysis that Cut and Fill mining method with high recovery and low dilution was chosen for geotechnical analysis.

CHAPTER 5

DATA ANALYSIS AND DISCUSSIONS

5.0 PREAMBLE

The proposed mining method is Cut and Fill mining method. This mining method is intended to replace modified over cut and bench (MOCB) and Post Pillar Cut and Fill (PPCF) in the Nose area of the ore body. The objective is to improve ore recovery and eliminate risks associated with the current MOCB and PPCF methods. Primary and secondary sequence of extraction with backfill is going to be applied in this Cut and Fill mining method. The intended mining method was arrived at by modelling using Numerical stress analysis program (Phase2), and empirical design (Potvin/ Mathew's Stability Graph). Rock properties estimates in Numerical modeling Phase2 are given Appendix 5.

5.1 BASIC CHALLENGES ASSOCIATED WITH THE CURRENT MOCB AND PPCF:

The current mining methods in the nose area of the ore body have got challenges associated with them. The following are the challenges faced with the MOCB and the PPCF mining methods:

- Long stope preparation period of about 6 months (MOCB);
- Relatively high development requirement (PPCF);
- Intensive support requirement (MOCB);
- Risk of working within large span open Stopes of about 10.m wide by 12.0m High (MOCB);
- Risk of over mining pillars (PPCF);
- Lower productivity (PPCF);
- Relatively low ore recovery (65-70%) (MOCB, PPCF); and
- Several number of bulkheads points for back filling (PPCF).

5.2 STOPE DEVELOPMENT

Figures 5.1 and 5.2 below show respectively the plan and section of the existing mining method (MOCB) and the proposed Cut and Fill mining method with primary and secondary stopes.

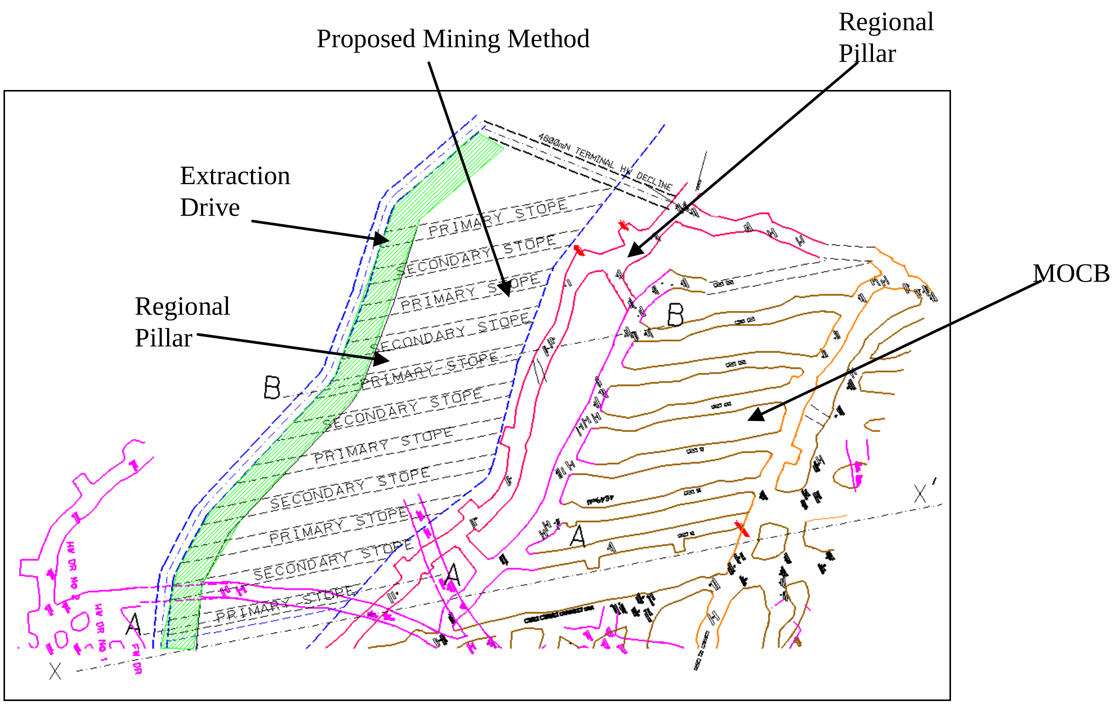


Figure 5.1 Plan of existing MOCB and the proposed method

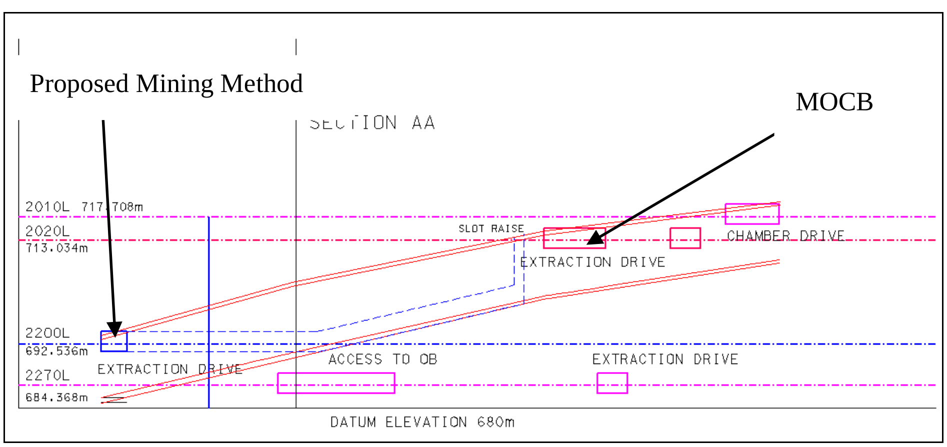


Figure 5 .2 Section of the proposed mining method.

5.2.1 Extraction Drive

The extraction drive is going to be mined at the geological hanging wall (GHW) because of the competence of the hanging wall quartzite. The extraction drive will serve as a permanent access, tramming route and fresh air intake. The size of the drive is planned to be 5mW × 4mH so as to accommodate both LHD's and dump trucks.

5.2.2 Drilling Raises (Crosscuts)

The drilling raises are to be mined on the footwall at an apparent dip so that the ore can be left in the roof for stoping (see Figures 5.1 showing the section of the mining method). The over cut raise is planned to hole through to the upper extraction drive before stoping the ore in the roof. The upper extraction drive serves as the collector of exhaust ventilated air from stoping activities and provides access for backfilling if need be.

5.2.3 Foot wall /Extraction Drive

Where it will be required, the development will be mined in the lower part of the ore body and will be used for drilling and extraction of ore. The drive to be 4mW × 4mH to accommodate the LHDs. Because of the wider size of the lower part of the pillar, stress levels around the drive are expected to be relatively low, as indicated in Phase2D model results Figures 5.3 and 5.4.Ground deterioration around the development may not be that serious with some levels of support. For improved stability pillar recovery should not lag too much behind secondary stope extraction.

5.2.4 Hanging wall Chamber Drive

A hanging wall chamber drive will not be mined in the proposed Cut and Fill mining method as is the case in the current MOCB.

5.2.5 Access Ramp and Crosscut

The access crosscut is mined from footwall drive/ramp through the ore body to connect to the hanging wall drive. In the current MOCB access ramps are positioned within the ore body in the hanging wall. The stress analysis distribution along the access crosscut ranged from 16MPa and 20MPa before extraction and around 7MPa after stoping out as shown the Figures 5.3 and Figure 5.4.This shows changes in stress levels before and after stoping out.

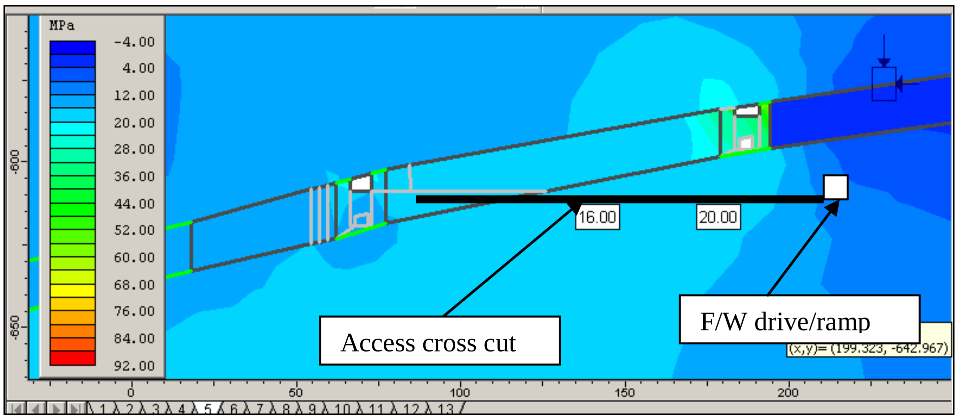


Figure 5.3 Section showing stress levels around stope development openings including Access cross cut before Extraction

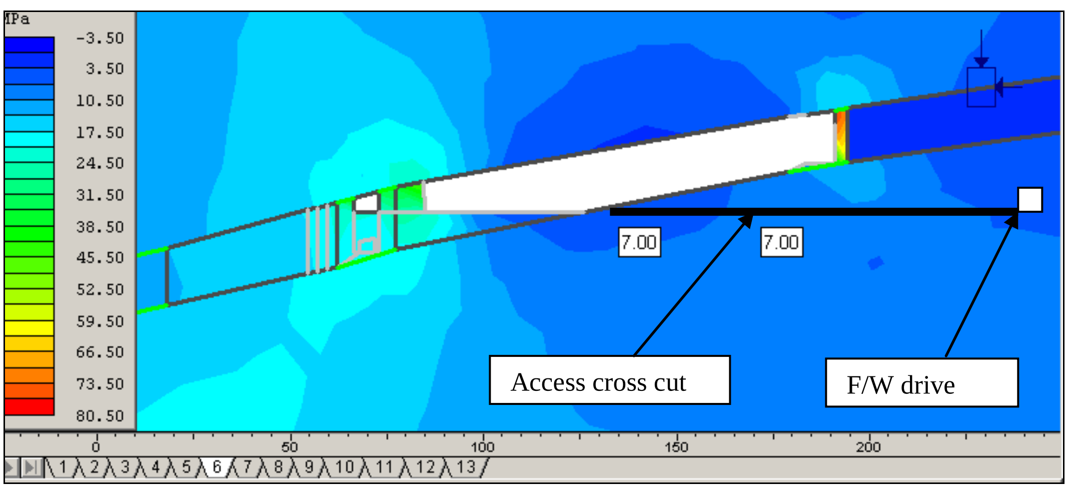


Figure 5.4 Section showing stress levels after Extraction around the Access Crosscut

5.2.6 Access Crosscut position

The access crosscut to the stopes (extraction drive) will have to be mined towards the Geological Footwall. That is where the footwall drive will lie below the ore body before extraction. The footwall drive will be mined up to the current MOCB which has been backfilled with hydraulic fill. The location and size of the footwall drive and the size of the pillar is shown in Figure 5.5a. The figure shows the location of the drive on the footwall side before stoping out.

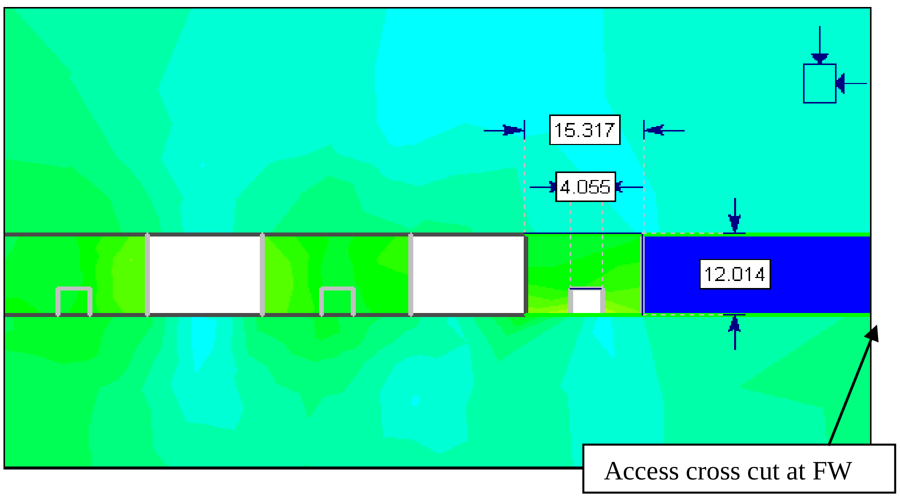


Figure 5.5a Long section showing Access cross at the footwall position

An analysis has to be carried out to compare stress levels if the drive is to be located on the hanging side. The Figure in 5.5b shows the location of the drive on the hanging wall side before stoping out.

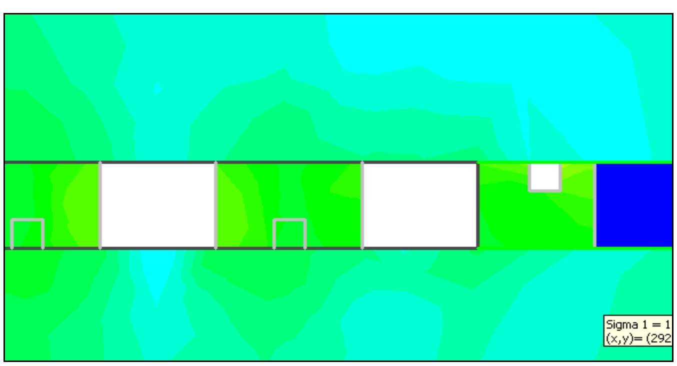
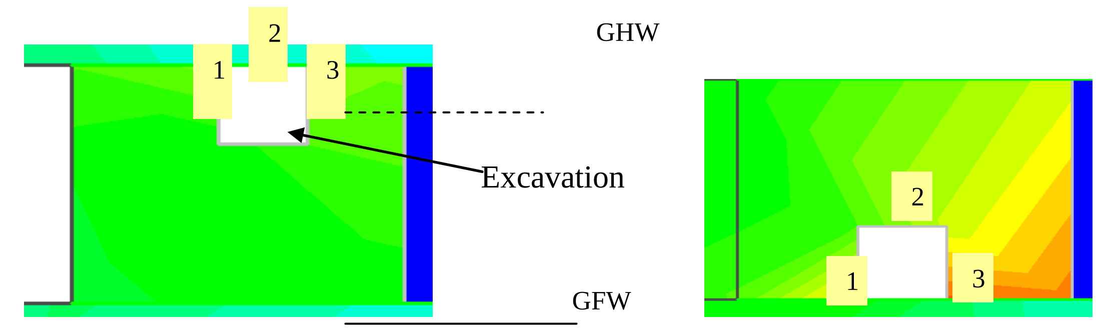


Figure 5.5b Long section showing Access cross at the hanging wall position

5.3 STRESS LEVELS AROUND ACCESS CROSS CUT

Stress analysis for the location of the drive on the hanging wall side and foot wall side was done as shown in Figure 5.6. Analysis was also done to know the stress changes, percentage build up in terms of stresses and the change in strength factor in both cases when mining towards and away from the access crosscut.



Where: 1, 2 and 3 are the measurements point

Figure 5.6 Location of maximum stress levels around access ramp/cross cut positioned in hanging and footwall part of the ore body

a) Mining away from the access crosscut – maximum stress values are recorded on upper part of sidewall of the crosscut mined at hanging wall, and on the lower part of the sidewall of crosscut mined at footwall position. (see Table 4.1).

Table 5.1 Mining away from the access crosscut

Development	Cross cut at HW position			Cross cut at F/W position		
Location	1 (upper)	2 (crown)	3 (upper)	1 (lower)	2 (crown)	3 (lower)
Max stress levels (MPa), before & after 100% block extraction	16.4-34	12.4-15.9	19.4-33.6	14.8-35.0	16.0-28.2	19.0-33.6
Stress change (absolute values)	17.6	3.5	14.2	20.2	12.2	14.6
Percentage stress buildup	107	28	73	136	76	76
Strength Factor change	5.3 - 1.9	7.3 - 7.6	3.4 - 1.9	4.8 - 1.6	6.5 - 2.0	3.9 - 2.1

Higher strength factors (low stress change) in the crown of crosscut positioned at hanging wall may be an indication of stability.

b) Mining from centre outward towards access cross cut – maximum stress values are recorded on upper part of sidewall for cross cut mined at hanging wall, and on lower parts of the sidewall for cross cut mined at footwall position.(see Table 5.2).

Table 5.2 Mining towards from the access crosscut

Development	Cross cut at HW position			Cross cut at F/W position		
Location	1 (upper)	2 (crown)	3 (upper)	1 (lower)	2 (crown)	3 (lower)
Max stress levels (MPa), before & after 100% block extraction	27.1-45.7	17.1-19.6	32.2-43.2	27.7-45.6	23.4-37.7	32.1-42.8
Stress change (absolute values)	18	2.5	11	17.9	14.3	10.7
Percentage stress buildup	64	15	34	65	61	33
Strength Factor change	2.2 - 1.4	2.9 - 3.1	1.7 - 1.4	2.4 - 1.3	2.7 - 1.5	1.9 - 1.5

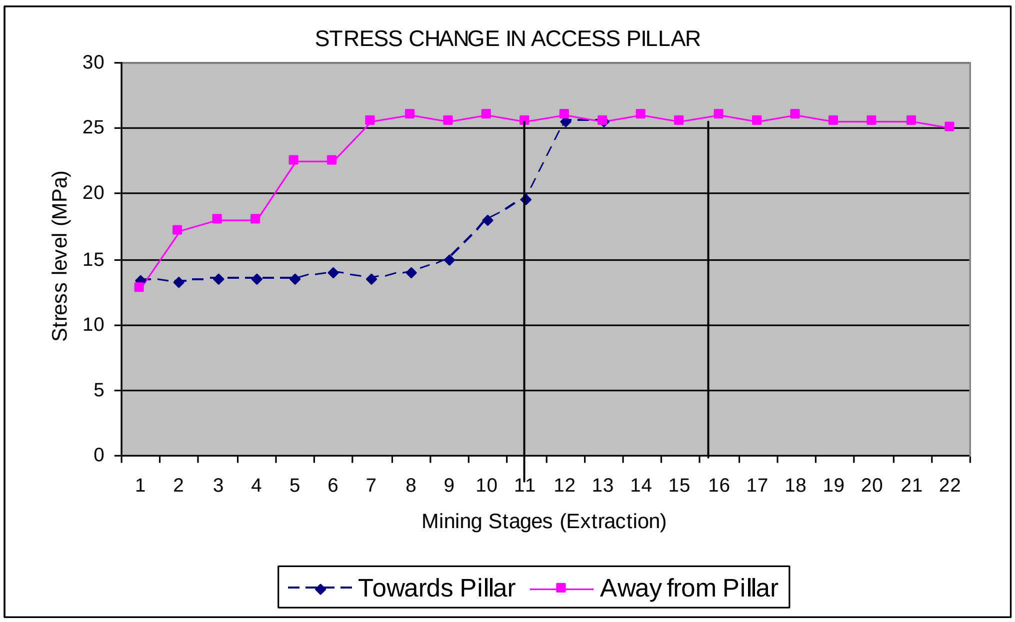


Figure 5.7 Plot of pillar stress versus extraction levels (mining stages)

Figure 5.7 shows that stress levels reach a maximum value much earlier when extraction takes place away from the pillar than toward the pillar. Subsequently, deterioration of the ground around the development may set in much earlier when mining away from the pillar. This may happen even earlier before block extraction is complete.

5.4 STOPE DEVELOPMENT - MOCB VS PROPOSED CUT AND FILL MINING METHOD

The proposed cut and fill mining method is basically a form of inverted MOCB, except that the open stope will be non entry to personnel apart from entry for remote loaders. To prevent broken ground from being thrown after blasting far inside the stope, improvement should be made to the blasting system, as is the case in open pit blasting.

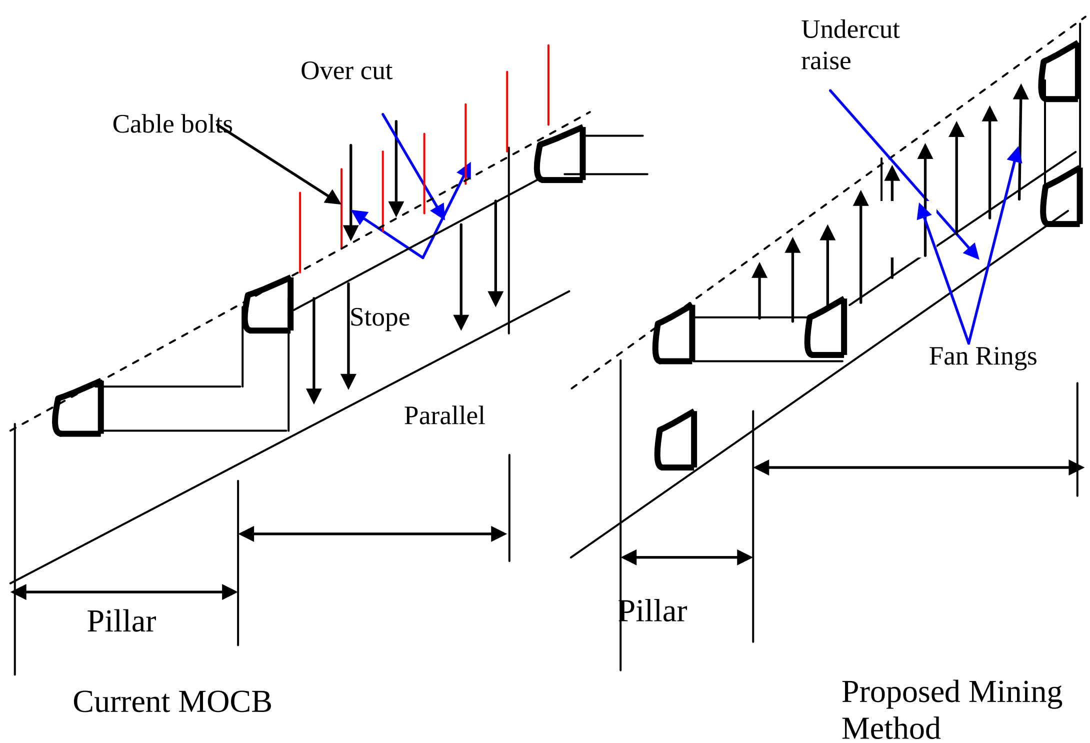


Figure 4.8 Schematic section showing development layout of the two methods

In MOCB the strike pillar is left unmined, whereas in the proposed method the pillar will be mined. For the new mining method, it should be desirable to extract the pillar together with the Primary stope, and then the foot wall drive may not be required. The loaders will have to extract the ore from the crosscut extended into the pillar. Should the foot wall drive be required (to avoid loader getting into the open stope), then it will be economical to bench up to the limit of secondary stope position to have the final stope.

In the proposed mining method a breakaway crosscut will be mined perpendicular to the hanging wall drive (no sharp corners) prior to mining on apparent dip, hence, positioning the bulkhead in solid abutment for stability.

5.5 ORE BODY CONFIGURATION AND ASSOCIATED ORE LOSS

In the proposed primary/secondary extraction method, the length of the crosscut before reaching the geological footwall (GFW) will depend on both the dip and thickness of the ore body. See Figure 5.9 below.

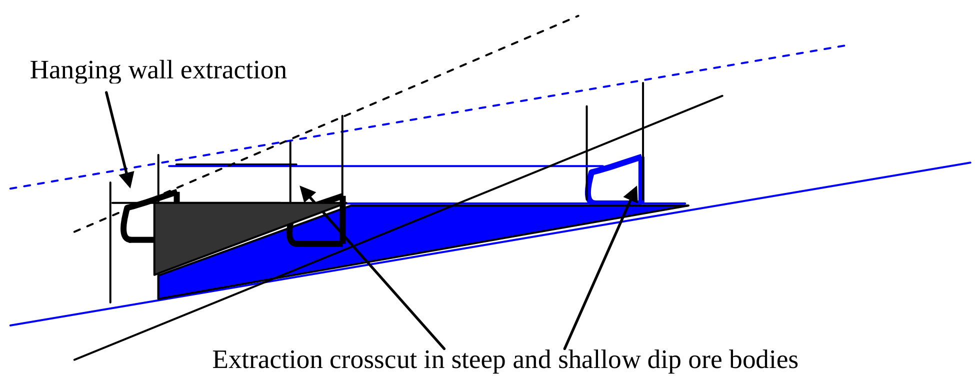


Figure 5.9 Section showing length of access cross cut in relation to ore body dip

Where the dip of the ore body is relatively steep or greater the 20^0, the crosscut reaches the GFW over a shorter horizontal distance. This will result into the strike pillar being relatively small in width of about 4.0m, with a lower width to height (W/H) ratio of about 1.1, and consequently less ore loss.

On the other hand, where the dip is shallower or less than 20^0, the cross cut reaches the GFW over a longer horizontal distance, resulting into the strike pillar being excessively large, with a high W/H ratio of more than 1.6 with much more ore loss as shown in Figure 5.9 . The ore loss associated with longer access crosscuts can be minimized by mining a diving crosscut (steeper drive of about 8^0 from the hanging wall to the footwall). The blue line in Figure 5.9 shows a shallower ore body and the black one shows a relatively steeper one.

5.6 REGIONAL PILLARS AND STOPE DEVELOPMENT

5.6.1 Preamble

A regional pillar is the block of ore left between mining levels for the purpose of ground stability. This is normally left along strike to assist in the stabilization of the mining blocks. (See Figure 5.10)

Regional pillars are designed to reduce stress levels in working areas that are below and above the mining block to improve ground conditions and as protection for critical access developments.

Considering that the benched pillar may be filled with cemented hydraulic backfill, in addition to primary stopes, the need for regional ore pillars may not arise, as stress transmission from hanging wall to footwall may occur as closure takes place across cemented backfill in the levels up-dip. A regional pillar is left is normally 10.0m in thickness.

5.6.2 Regional Pillar Intervals

The current regional pillar will not be mined out, as it separates the uncemented backfilled MOCB stopes above from the new stopes below. When indications of excessive stress levels occur in the lower working blocks then the need for regional pillars will arise.

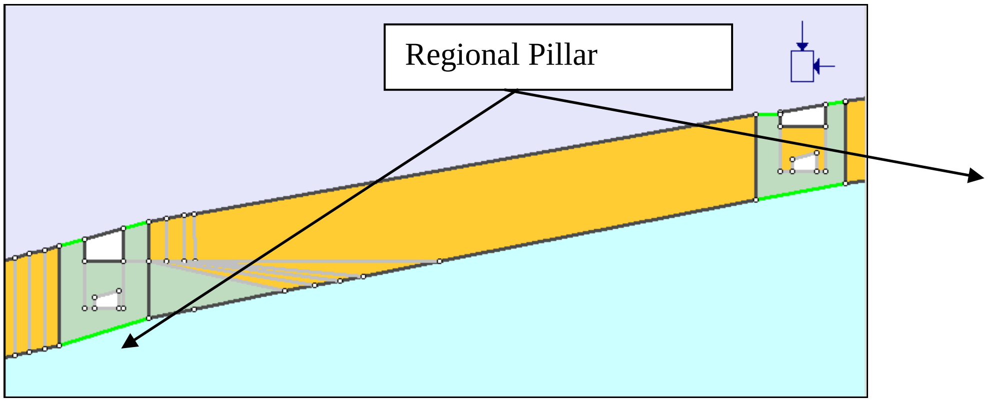


Figure 5.10 Regional pillars

5.7 PROPOSED MINING METHOD, STOPE DEVELOPMENT AND DRILLING LAYOUT

Figure 5.11 shows a typical mining layout for the proposed mining method. The stope rings will be fanned from the drilling drive to the assay hanging of the ore body and retreating level below.

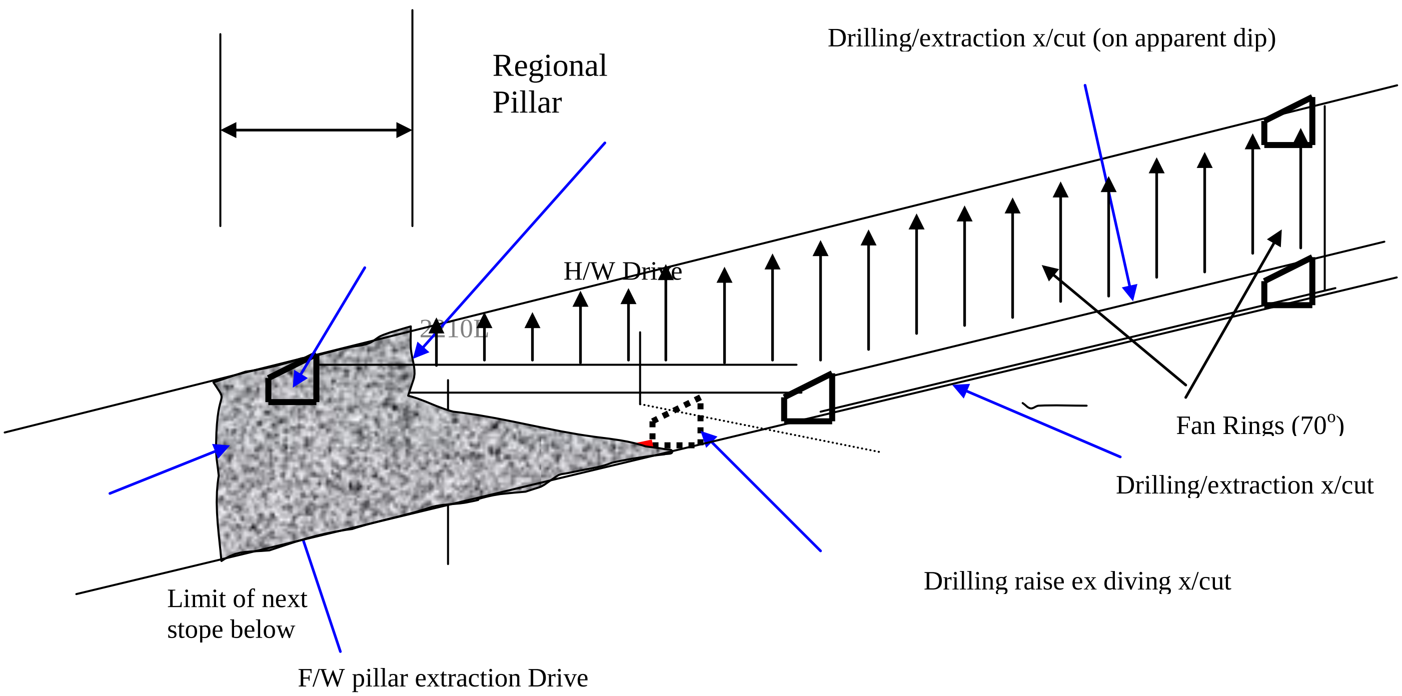


Figure 5.11 Section showing development and stope drilling layout

5.7.1 Two Development Options

There are two options that are to be considered in this cut and fill mining method. This will depend on whether to leave a pillar or not.

1. Mine hanging wall drive only (as access to stopes above and for back filling stopes below) and leave unrecovered pillar.
2. Mine hanging wall and foot wall drives to enable recovery of the pillar after primary and secondary stope extraction. Primary stopes are the initial stopes mined out and secondary stopes are the subsequent stopes mined after backfilling of the primary stopes with cemented fill.

OPTION1. Mine hanging wall drive only and leave strike(regional) pillar in situ

- As access to the stopes above –(for stope drilling and cleaning) and
- As access for back filling future down dip stopes

Observations:

- Results in relatively reduced stress levels in working areas below;
- Significant ore loss due to unmined ore pillar; and
- Possibility of using benched portion of pillar as slot for the lower Stope.

OPTION2. Mine hanging wall and footwall drives to enable recovery of the pillar after primary stoping.

- The footwall drive within the pillar to be used as:

1. A drilling and gathering level for the benched pillar ore (safely outside the Stope).
2. Development of the F/W extraction drive to be delayed (minimize time dependent deterioration)
3. The F/W drive to connect to secondary stope cross cut as access

Observations:

- Increased stope tonnage due to pillar recovery
- Middling(ore left) between hanging wall and footwall drives should be ≥ 6m for stability purposes
- Footwall drive to be mined smaller (4width x4 height) than extraction drive above (5width x4height) for stability purposes and adequately supported (located in stress shadow)

5.7.2 Diving Cross Cut

In order to minimize in situ ore loss, in the case of shallow dip or wide ore bodies, due to longer crosscuts, it is essential that the stopes are created by mining the crosscut to reach the GFW earlier over a short distance. In shallow dip or wide ore bodies shortening the cross cut can be achieved by developing a diving crosscut (at an angle) as opposed to horizontal access crosscut prior to establishing the main drilling crosscut on apparent dip to the ore body dip.

5.8 GROUND SUPPORT REQUIREMENTS

The Q-system is used to assess the rock mass quality around the area where the proposed mining method is going to start from. Q-system is the tunneling quality index that classifies rock masses with respect to in situ parameters including rock quality, joint condition and stress state. Support design for the extraction drives and footwall raise is determined based the quality of the rock mass. The Q system of rock mass classification relates rock mass quality with respect to block size and joint condition. Q ratings are given by[11]:

$$Q = \frac{RQD}{Jn} \times \frac{Jr}{Ja} \times \frac{Jw}{SRF}$$

Where;

- RQD is the Rock Quality Designation ;
- J_n =number of joint sets;
- J_r =Joint roughness number;
- J_a =Joint alteration or filling;
- J_w =Water condition; and
- SRF =Stress Reduction Factor.

Geotechnical mapping of the ore shale in the MOCB area is shown below:

- Bedding planes are thinly bedded at average spacing of 0.03m;
- Moist to damp in some places;
- Slightly weathered;
- UCS =110MPa;
- Dip/Dip Direction 20°/260, 80°/250°;
- Joint condition – Slight Iron oxide staining, Rough and undulating, tight joint (<1mm thick);
- Random joint at greater than 10m spacing at dip and dip direction 90°/080°;and
- RQD = 50% .

Q- Ratings

J_n = 6 (2 joint set + random);

J_r = 3 (Rough undulating);

J_a = 3(Low friction coating <1mm);

J_w = 1.0 (Moist); and

SRF = 2.

Substituting the values into the equation, we have:

$Q = (50 \div 6) \times (1 \div 3) \times (1 \div 2) = 1.3$

The larger the number the more good is the ground condition. Q is also used to determine the stability of excavations.

The Excavation Support Ratio (ESR) is a factor providing a level of safety depending on the designed usage and service life of the excavation. Using the Excavation Support ratio of 1.6 for permanent openings, Equivalent Dimension (D_e) is the calculated as:

D_e =Span or Height in m/ ESR = 5/1.6=3.1

Using the equivalent dimension (D_e) and the Q-system value as calculated above, the support design is ascertained by plotting these on Figure 5.12. From Figure 5.12, it can be seen that the required support is rock bolts with shotcrete at 40mm to 75mm. For other support requirements refer to Appendix 2.

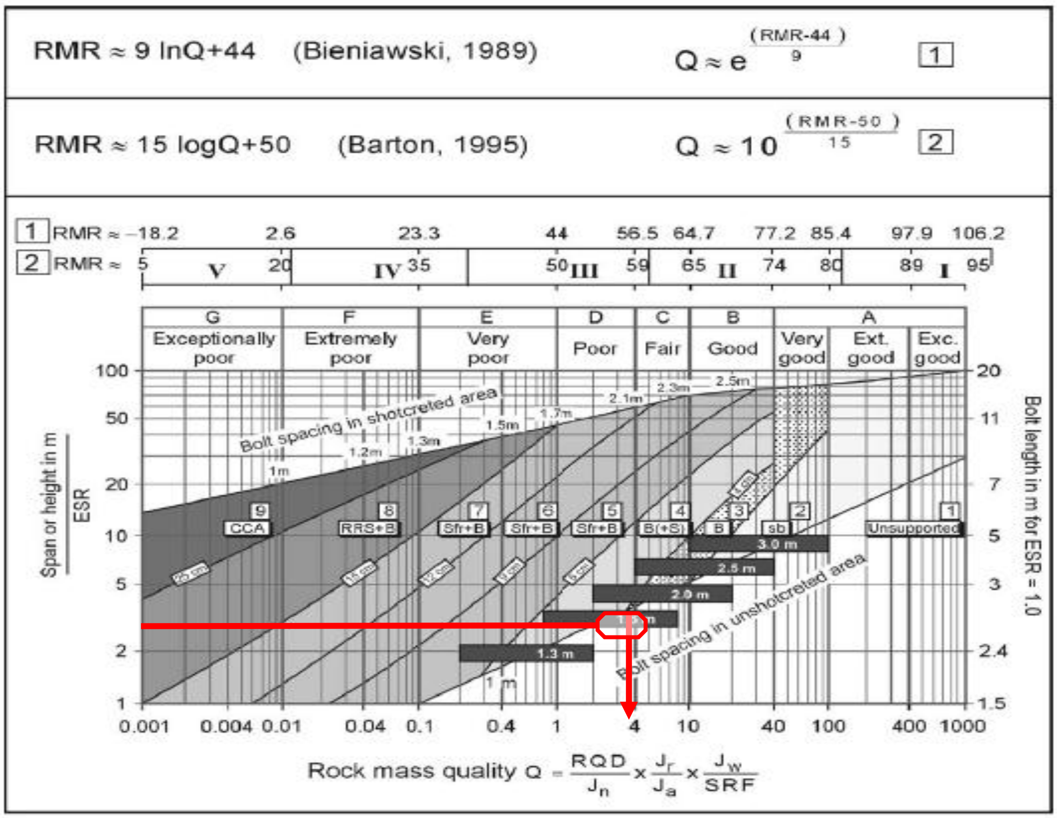


Figure 5.12 Q System graph [15]

The final support design for the foot wall and hanging wall extraction drives and foot wall raise (in secondary stopes) are as shown in Table 5.3.

Table 5.3: Support Requirements

Support Element	Support Standard
Rock bolt/Permasets	2.4m long at 1m ring and bolt spacing following square pattern
Diamond Wire mesh & Tendon Straps	Diamond mesh and tendon straps (straps at 2.0m spacing)
Cable bolts	6.0m long, 25-40 tonne cable bolts at 2.0-3.0m spacing
Shotcrete	50mm thick shotcrete

Shotcrete application will not be required in primary stope development as stress levels required to cause ground deterioration are expected to be low. In secondary stopes, shotcrete will be applied as a result of ground deterioration caused by stress levels.

5.9 HANGING WALL STABILITY OF STOPES PRIOR TO BACKFILLING

Though stopes will be non-entry, stability of the hanging wall in both primary and secondary stopes is critical, in the sense that any hanging wall caving close to the advancing draw point will cause dilution, and subsequently poor filling of the stope.

Hanging wall stability of stopes is best determined by relating the geotechnical condition of the strata lying immediately above the hanging wall surface established as the **N'-Number**, and the span of the excavation, which is expressed in terms of **hydraulic radius** (Area/ Perimeter). Once the rock condition is determined, then the span can be estimated.

The parameters associated with the Hanging wall Quartzite rock formation are:

- RQD = 75%
- Moist to damp in some places
- Fresh – Slightly weathered

- UCS = 220 MPa
- DIP/DIP DIR 55°/330°, 80°/030°
- Joint condition – Slight Iron staining, Rough Undulating, tight joint (Tightly heeled)

Using the Mathews/Potvin method, [12] design parameters are calculated as shown below:

N' – Stability Number = Q'× A × B × C

Modified Q'= (RQD÷Jn) × (Jr÷Ja)

= (75÷3) × (1.5÷0.75)

Q'= 50.0

A - Rock Stress Factor = 1 (UCS/σ1> 12)

B - Rock Defect Orientation Factor = 0.3

C- Design Surface Orientation Factor = 2.4

Q' -Modified NGI Rock Mass Rating = 50

N' = 50×1×0.3×2.4 = 36.0

Rock Stress Factor (A) estimation

This factor is determined by computing the ratio of maximum compressive stress induced to the uniaxial compressive strength of the rock material in the immediate hanging wall of the excavation.

Table 5.4 Range of Maximum stress levels

	Primary stopes				**Secondary stopes**			
Stope width	8	10	12	15	10	12	14	19
Range of Maximum compressive stress, σ_1	12.9-14.7	12.2-13.5	12.5-13.7	10.6-12.5	5.1-6.3	4.2-5.1	5.1-6.8	3.9-5.1
	13.8	12.9	13.1	11.6	5.7	4.7	5.9	4.5

Increasing tension

Increasing tension

Table 5.4 shows range of stress levels taking into account different stope sizes of the stope spans. In general, the wider the stope span, the more the roof of the stope is subjected to tension and the more it becomes susceptible to failure. In all cases the hanging wall will be subjected to some degree of tension, giving a stress factor, A=1, as the ratio, UCS/$\sigma 1$>> 12. The roofs of secondary stopes are more in tension than the roofs of primary stopes. This is because the modulus of elasticity of the backfill on either side of the secondary stope is far much less than that of the rock mass and has to deform a lot before it can carry any significant load of more than 12MPa.[12]

To determine the stability of the stope, hydraulic radius is determined. It is given by the following equation:

Hydraulic radius (h) = AREA/PERIMETER = (S × L)/(2S + 2L)

Where: h = hydraulic radius (m);

S = stope span (m); and

L = stope length (m).

Hydraulic radius more accurately accounts for combined influence of size and shape on excavation stability. It is useful to become familiar with the range of “spans” for the given hydraulic radius. This will provide a means of comparison with other designs methods which do not use hydraulic radius.

Assuming the length of the stopes to be 150m, for various suggested stope spans, using the N – Number, one can determine which unsupported stope span will be stable and which one will not be. Suggested standard stope spans are 8m, 10m, 12m, 15m and 19m, for both primary and secondary stopes.

Calculated hydraulic radii, and plot of N’, h.

Table 5.5 shows the comparison for different span sizes:

Table 5.5: Range of hydraulic radius

Height (H)	12	12	12	12	12
Span(S) or width (W)	8	10	12	15	19
W/H ratio (secondary)	0.7	0.8	1.0	1.3	1.6
Hydraulic Radius, h	3.8	4.7	5.6	6.8	8.4
Stability Number, N’	36	36	36	36	36
Plot of N’ and h	Very stable	Very stable	Stable	Stable	Just stable

As earlier mentioned, the secondary stope hanging walls will be more susceptible to tensile stresses than primary stopes.

Using the modified Q’ and N’ values that are determined, it can be shown that unsupported hanging wall will be stable for stope widths up to 12m as shown in figure 5.13. The method used to determine stability of stope spans applies well to primary rather than secondary stopes. This is the reason why primary stopes should be sufficiently large and with strength to support the roof of the secondary stope.

Since stopes will be non-entry, the stability of secondary stope may be analysed on the basis of stability of the drilling cross cut mined within the secondary stope which also depends on the overall pillar strength in relation to stress levels generated within the rock mass. The stopes will be non entry and should any hanging wall failure occur, it will be restricted deep inside the open stope, behind the advancing stope face and hence only 5% to 10% dilution may occur. However, caving in both primary and secondary stopes is an undesirable incident as it results in the stope not been tight filled due to the voids created by the collapsed ground.

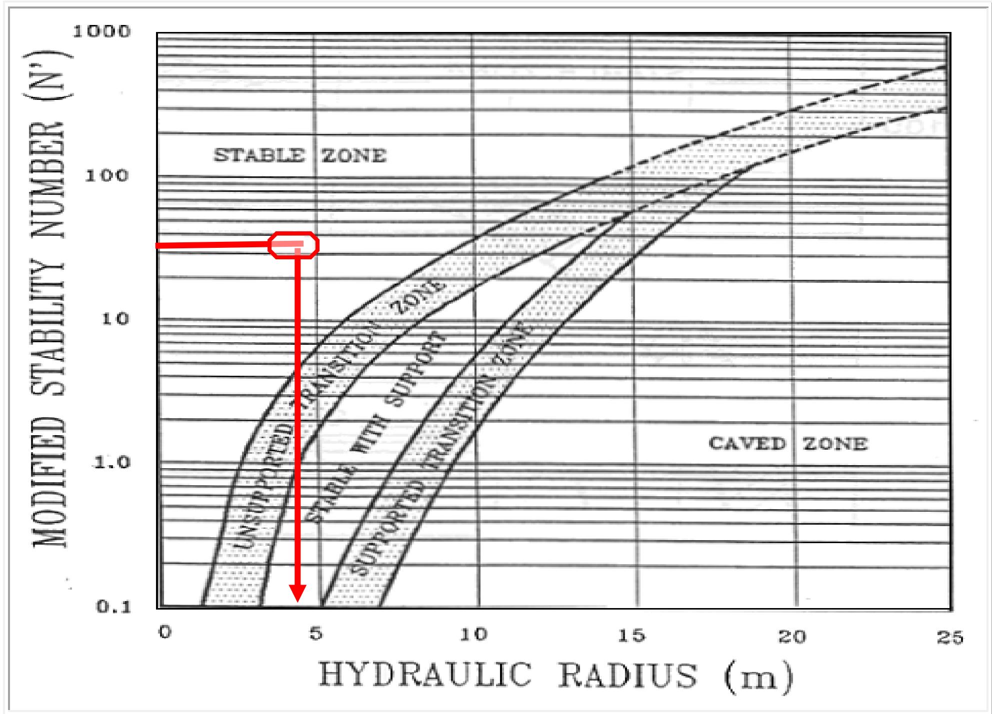


Figure 5.13 12m unsupported stable span in rock mass of N'=36. [11]

5.10 REGIONAL PILLAR (STRIKE PILLAR) STABILITY ANALYSIS

At constant depth and rock material strength, stability of unconfined pillar is a function of width to height ratio (regional pillar). [5] For the pillar failure occurs when the width to height ratio is less than 1.6. Stability of confined pillar (Secondary stopes) is enhanced by the confinement provided by the backfill material. The back fill material must be tight filled.

The stability of the regional pillar is also critical where the hanging wall and foot wall drives are required as access for backfilling the down dip stopes and extracting the pillar, respectively, at a later stage. Pillar as a natural support should be designed in such a way that as little displacement as possible should take place within the periphery of the excavation mined within it. The pillar should not fail.

5.10.1 Stope Design Considerations

The pillar stability in the proposed mining method has to be considered. Different pillar sizes were taken into consideration for strength factor determination. The 15.0m, 20.0m, 25.0m and the 30.0m pillars were taken into consideration as shown in Figure 5.14. The drive was located at the centre of the pillars in the hanging wall. The stress levels around the drive were recorded in the roof, sidewalls and the floor for all the pillar sizes as indicated in the Figure 5.14 for the 15.0m pillar.

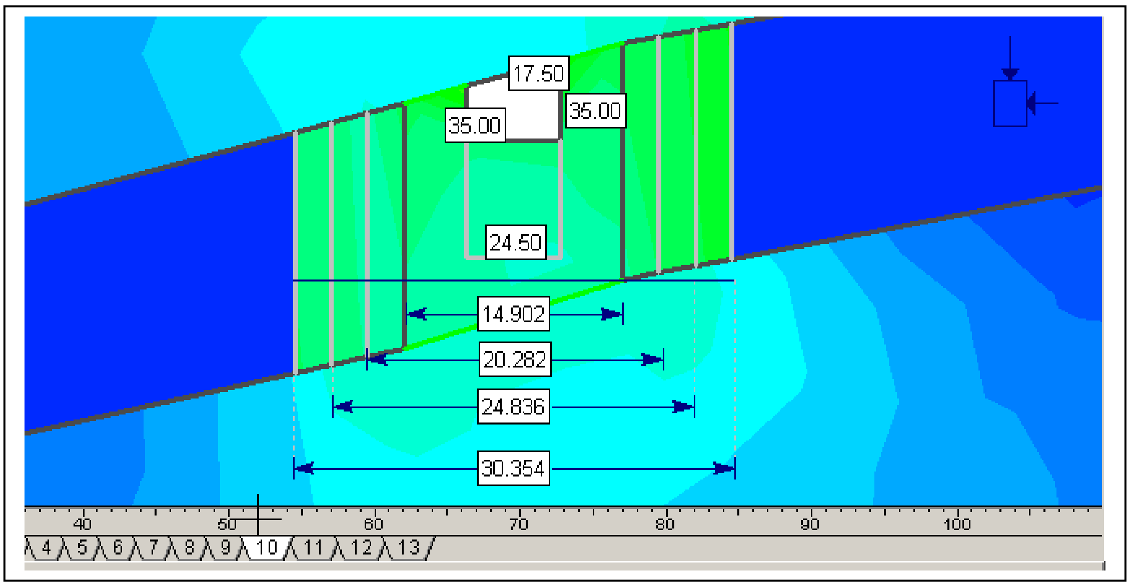


Figure 5.14 Pillar structure in an inclined ore body

The pillar strength and the stress levels in all the pillar sizes under consideration are summarised in Table 5.6. The width to height ratio was also taken into consideration. The stress levers and strength factors were achieved by using numerical modeling.

Table 5.6: Pillar strength and the stress levels in all the pillar sizes

No.	W_{pillar}	H_{pav}	W/H ratio	Parameter	Location (with H/W Dr)			
					1	**2**	**3**	**4**
1	**30**	14	2.1	σ_{1max}	35.0	17.5	35.0	24.5
				Strength Factor, SF	2.35	5.74	2.61	3.39
2	**25**	14	1.8	σ_{1max}	38.5	21.0	38.5	28.0
				Strength Factor, SF	2.09	4.96	2.35	3.13
3	**20**	14	1.4	σ_{1max}	42.0	21.0	42.0	31.5
				Strength Factor, SF	2.09	3.39	2.09	2.61
4	**15**	14	1.1	σ_{1max}	45.5	21.0	45.5	35.0
				Strength Factor, SF	**1.57**	**2.09**	**1.57**	**2.09**

It is noted that there were increased stress levels after mining down dip stope. The smaller the pillars the higher the stress levels as shown in table 5.6.

The pillar will be stable for all pillar widths under consideration, but the 15m pillar, with a W/H = 1.1, may not be stable because width to height ratio is almost the same.. Also it is important to note that inclined ore bodies tend to slide down dip. The possibility of putting the access pillars at 90^{o} to the ore body is also practically not feasible. Width to height ratio is also very important in designing of the required pillar. See table 5.7 below.

Table 5.7: Pillar Stress Vs Pillar width (F/W drive)

No.	W_{pillar}	H_{pav}	W/H ratio	Parameter	**Location** (with F/W Drive)						
					1	**2**	**3**	**4**	**5**	**6**	**7**
1	30	14	2.1	σ_{1max}	30.3	26.2	25.5	25.7	26.9	31.6	36
2	25	14	1.8	σ_{1max}		30.82	27.6	27.9	29.5	36.7	
3	20	14	1.4	σ_{1max}		32.7	30.8	30.9	32.6	36.8	
4	15	14	1.1	σ_{1max}			34.7	34.9	36.8		

5.10.2 Current pillar sizes (Modified Over Cut and Bench)

In the current mining method of Modified Over Cut and Bench (MOCB), the 5.0m pillar is left between the extraction drive and the stope with a 4.0m height of the pillar. The stope is then backfilled with uncemented classified tailings to provide support against the stope walls, and consequently provide a global stability of the rock mass. Unlike cemented fill, uncemented fill will not significantly transit stresses but will just confine and stabilise the mined out areas.

5.10.3 Pillar Stability

Pillar size will have an influence on the stability of the extraction drive located at the middle. The bigger the pillar the more stable the extraction drive experiences. Ground disturbance around the drive will occur when the lower block is stoped out depending on the size of the pillar. Taking the worst case of angle of break of 65^{o}, the 15.0m pillar shows that it will affect the stability of the extraction drive and the effect reduces as the pillar is increased as can be seen in the break lines in Figure 5.15.

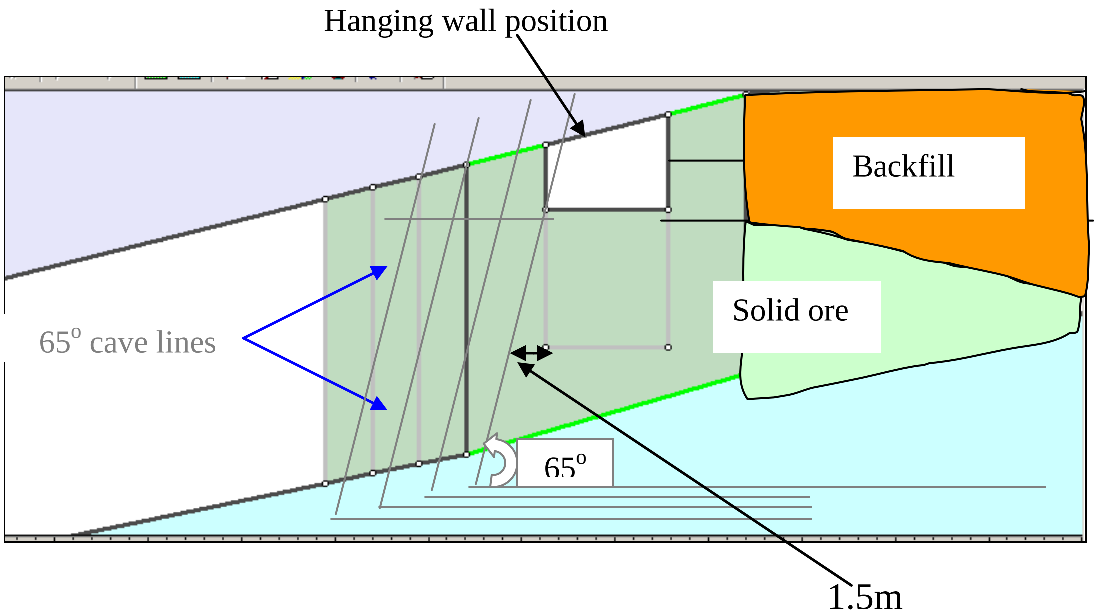


Figure 5.15 Effect of cave line on stability of down dip side of pillar

From the pillars under consideration, the distance covered from the floor shows that the 25m and 30m pillars will not have effects on the extraction drive and the drive can remain stable. The 15.0m pillar has a cover distance of 1.5m from the floor of the drive and can affect the stability of the drive. The 20.0m pillar can also have some effect on the stability of the drive that may require massive support. (Table 5.8)

Table 5.8: Effect of cave line on stability of down dip side of pillar

Pillar width (m)	Cave angle (deg)	Cover Distance on floor(m)	Remarks
15	65	1.5	Drive will cave due to smaller cover length
20	65	4	Drive unstable due to reduced pillar in hanging wall position
25	65	6	Stable due to enough cover length both in the footwall and hanging wall positions
30	65	8.5	Stable due to enough cover length both in the footwall and hanging wall positions

5.11 ACTUAL PILLAR - STABILITY ANALYSIS PRIOR TO BENCHING

Prior to benching of the extraction drive stress analysis of the four width parameters that are under consideration was done. Numerical modeling was used as shown in Figure 5.16

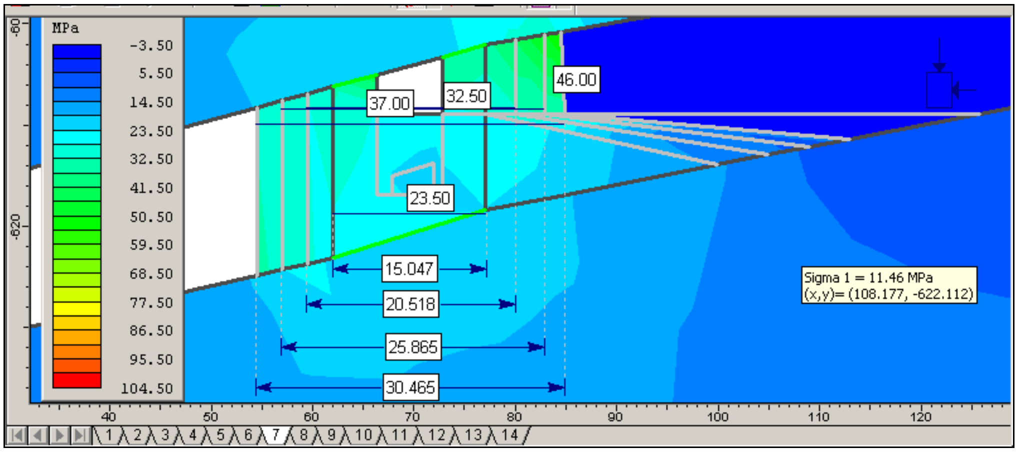


Figure 5.16 Real Pillar W/H ratio vs. Pillar strength

Strength factors and stress levels are shown in Table 5.9. The strength factors indicates whether the pillar can stand or not prior to benching. The 15.0m pillar gave a strength factor of 1.3 and showed that the pillar can collapse. For the rest of the pillar widths the strength factors show that the pillars will remain stable.

Table 5.9: Stress levels and strength factors

No	W pillar	P_{height}		H_{pav}	W/H ratio	Parameter	Location points				Remarks
		Down dip	Up dip				1	2	3	4	
1	30	14	8.16	11.0	2.7	σ_{1max}	37.0	32.5	46.0	23.0	
						Strength Factor, SF	2.1	2.4	2.6	4.2	Stable
2	25	14	7.81	10.9	2.3	σ_{1max}	41.5	37.0	50.5	23.5	
						Strength Factor, SF	2.1	2.1	2.1	3.9	Stable

3	20	14	7.26	10.6	1.9	σ_{1max}	46.0	41.5	56.0	28.0	
						Strength Factor, SF	1.8	1.8	1.8	2.9	Stable
4	15	14	6.77	10.4	1.4	σ_{1max}	50.5	56.0	64.0	32.5	
						Strength Factor, SF	1.6	1.6	1.6	2.6	Stable
5	**15**	**14**	**6.77**	**10.4**	**1.4**	σ_{1max}	50.5	55	64	-	
						Strength Factor, SF	1.6	1.3	1.3	-	Unstable

5.12 STABILITY ANALYSIS IN CASE OF PERMANENT PILLAR STRUCTURES

The analysis is carried out on the assumptions that the pillar is between the current mining method and the proposed mining method. The current mining method is stoped out and backfilled with uncemented backfill, hence, the need to leave a permanent pillar. The proposed stope will be the inverted MOCB and must be filled with cemented backfill and without a natural pillar but with cemented sidewall. Other pillar sizes where considered for pillar strength and width to height ratio as shown in Figure 5.17 and Table 5.10.

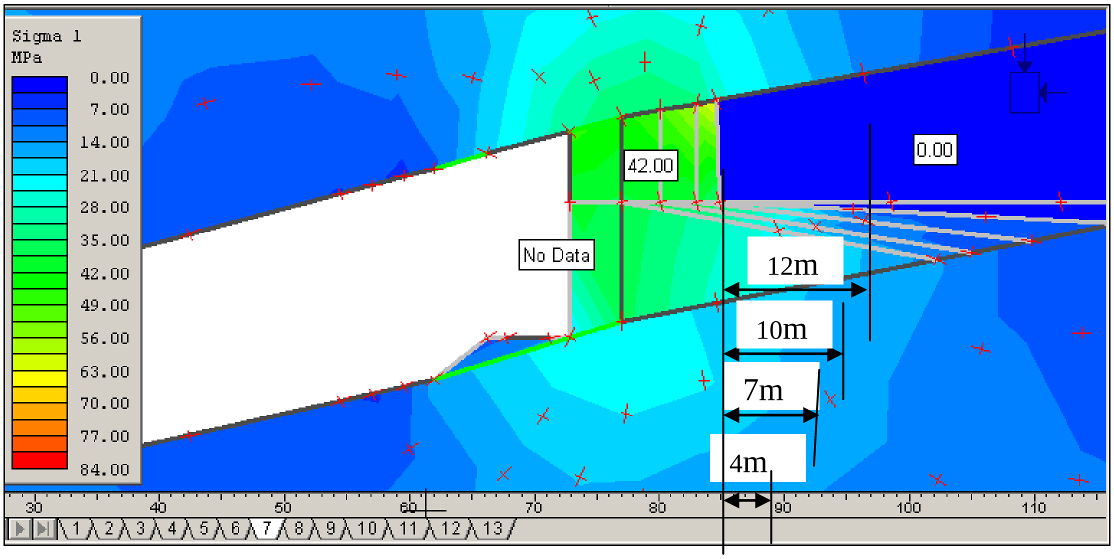


Figure 5.17 Variation in Pillar widths (at secondary stope position)

Table 5.10 Pillars Stress Vs Pillar width

Width (m)	**Av. Height (m)**	**Ratio, W/H**	**Max Pillar stress σ_1 (MPa)**	**Pillar Strength $Ps_{(MPa)}$**	**SF ($Ps/\sigma s$)**	**Comments**
12.0	11.1	1.1	42.0	88.2	2.1	Very stable
10.0	10.9	0.8	45.5	81.9	1.8	Very stable
7.0	10.6	0.7	52.5	84	1.6	Stable
4.0	10.4	0.4	70.0	71	1.3	Unstable

With the long axis of the stope on apparent dip, the excavation cuts the prominent bedding planes at an acute angle (not parallel), and this enhances pillar stability. Even a 4m wide pillar with a W/H << 1.0, have a short stand up time.

5.13 EXTRACTION SEQUENCE

The basic requirement for safe and economical extraction of ore is that mining operations should progress from mined out areas, or areas of weak ore shale, toward solid ground or stronger ore shale. Mine closures should at all costs be avoided.

5.13.1 Stope-Pillar Extraction Arrangement

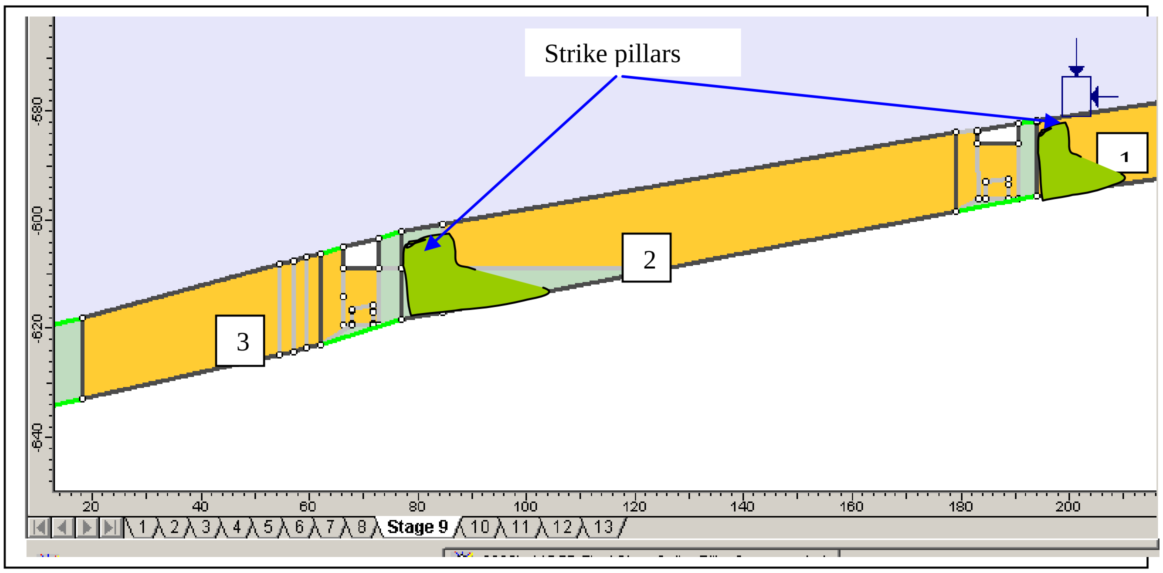
5.13.1.1 Down dip extraction sequence

(1) Mine leaving strike pillar (in cemented and uncemented up dip stopes); and

(2) Mine without strike pillar, up against the plug (cross cuts in all up dip stopes to be cemented – plug).

Figure 5.18 shows down dip extraction sequence and leaving the strike pillars on the proposed mining method and the current MOCB mining method.

Depending on the rate of development, ore extraction can take place on 2 to 3 mining Levels, on a 45^{o} mining echelon in order to increase the number of working stopes.



3 = Proposed Mining

2 = Proposed Mining Method

1 = Current MOCB

Figure 5.18 Down dip extraction sequence

4.13.1.2 Stope / Pillar Extraction sequence along strike

The sequence comprises of **primary and secondary** extraction, with cemented and uncemented hydraulic backfill respectively**.** The strike pillar, carrying the hanging wall and footwall drives will have to be partially or fully recovered. The sequences of extraction are shown in Figures 5.19 and 5.20. Numerals (i, ii, iii, etc) in the figures show the different stages of mining.

Figure 5.19 shows mining of the strike pillar at a later stage behind the primary and secondary stopes

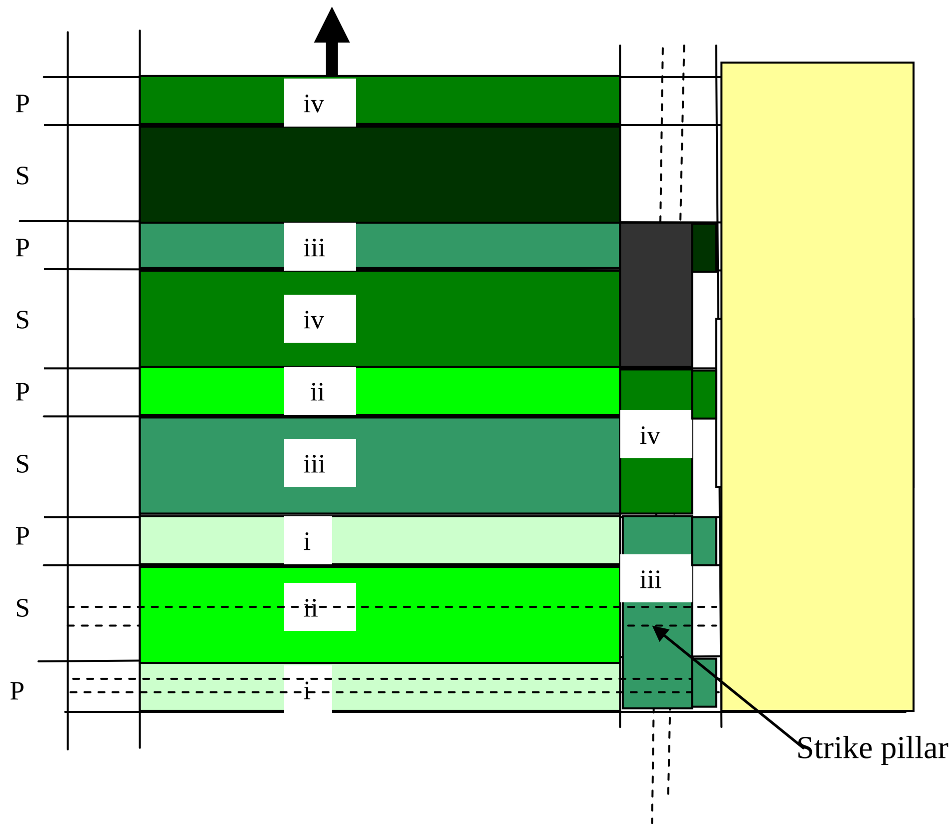


Figure 5.19 Primary – secondary extraction sequence with delayed pillar

Where P is a primary stope; and

S is a secondary stope.

Pillar Recovery can be done either by benching to cemented backfill wall at the primary stope positions or by leaving a 4m skin against an uncemented backfill at secondary stope positions as illustrated in figure 5.20 below. The ground condition in both pillar recovery is stressed and requires heavy support like in secondary stopes. In cemented backfill pillar recovery is maximized, hence over 95% ore body recovery. Where secondary stopes are plugged pillar recovery is maximized, hence 95% ore body recovery.

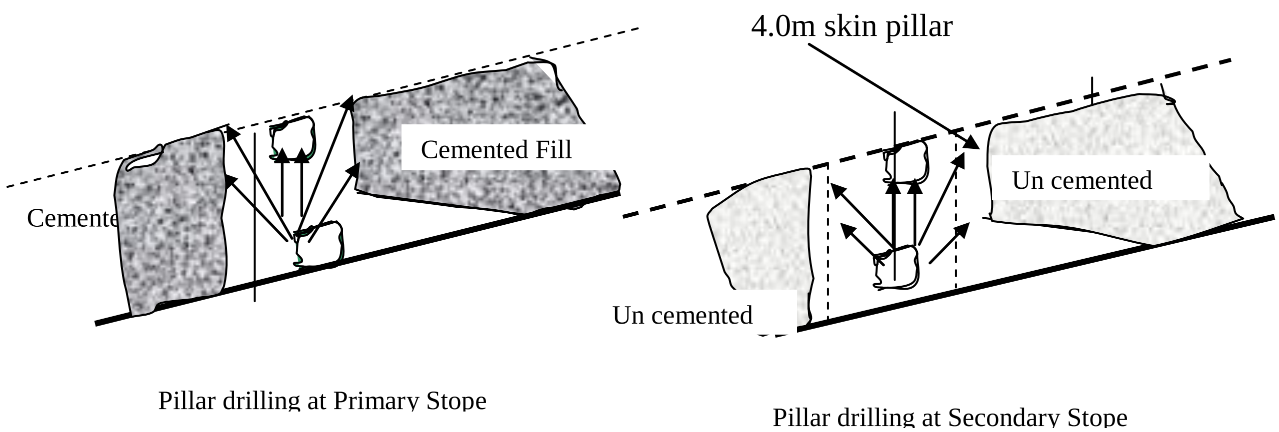


Figure 5.20 Pillar Recovery drilling pattern

Figure 5.21 show mining the stope and then recover pillar as one stope (by extending stopes to the H/W drive).

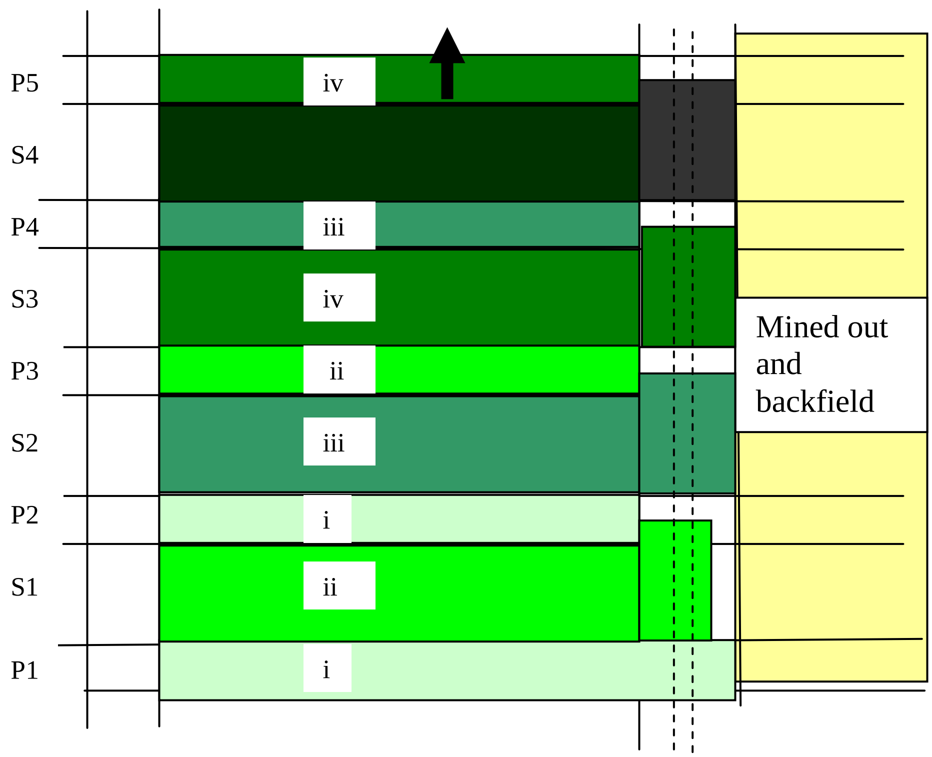


Figure 5.21 Primary – secondary extraction sequence (together with pillar) in plan view

It should be noted that the primary and secondary sequence still achieved but a 4m pillar will remain between hanging wall drive and up dip stopes (should cemented backfill be absent). Ground condition will be better in this case as there is no pillar to extract separately.

The extraction sequence will be as follows:

i. Mine P1 (including pillar) and P2 (up to edge of pillar) and backfill
ii. Mine P3 (up to pillar edge) and S1 (including S1pillar and part of P2 pillar) and back fill, alternatively, extraction of P1 and S1 can be staggered to allow for further curing of backfill in S1.
iii. Mine P4 (up to edge of pillar) and S2 (including S2 pillar and part of P3 pillar) and backfill.
iv. Repeat the sequence.

5.13.2 Mining with small pillar

In a proposed mining method, leaving a 4.0m pillar will not increase or improve total ore recovery due to pillars left. In the current mining method, a 4m pillar is left hence the recovery will be the same if a 4m pillar is left in the proposed mining method.

4.13.3 Continuous extraction with cemented backfill

The continuous extraction will be achieved by using cemented backfill which can maximize ore recovery. If stopes are backfilled with cemented fill, extraction can extend against the walls of the backfill. Only one stope will be extracted on one level at any one time unless more than one block is fully developed. Production faces can be increased by advancing two retreats outward. One stope can be in production while others are in stope ring drilling stage and development of the drives (Figure 5.22). The method will be costly due to cement requirement but with high recovery.

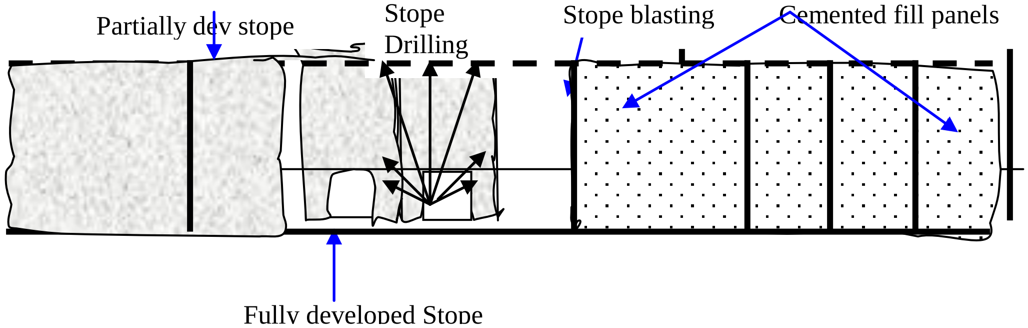


Figure 5.22 Continuous extraction with cemented backfill

5.14 BACKFILL

Backfill is generally described as any material used to refill an excavated area. In underground mining operations, backfill is material or a mixture of materials used to fill underground stopes. The following are fundamental reasons for use of backfill material:

- Ore pillar recovery in open stoping methods. This is achieved by way of giving structural support to the roof and sides of the stopes for improved production and safety
- Ensuring long-term regional stability
- Limiting excavation exposure
- Working platform in particular mining methods such as Cut and Fill.
- Environmental protection and waste disposal. Economic use of surface land has demanded use of underground waste as backfill material.
- Prevention of inundation (water flooding) in some underground mines
- Storage of very explosive methane gas (CH_4) in Coal mines, and
- Prevention of surface subsidence.

5.14.1 Viability of the proposed mining method

The success of this mining method will depends on having cemented fill which will have adequate strength backfilled in the primary stopes. This will be necessary so that the exposed fill faces exhibit sufficient strength to remain free standing during the process of secondary stope extraction. The backfill will also have to provide confinement to the secondary stopes and thus minimize sloughing and reduce rock movements.

It is noted that the modulus of fill is much less than that of rock (about 100 times less) and as such stress cannot be taken up by the fill and therefore it cannot be used to modify regional stress distribution. Since the cemented backfill in the primary stopes will have to be exposed in the secondary stopes, the challenge will be on the control of dilution from the backfill. Secondary stopes are planned to be placed with uncemented fill. The required uniaxial compressive strength (UCS) of the backfill to be placed in the primary stopes is estimated using the formula below which is based on stope dimensions. [27]

$$UCS = \gamma H / (1+H/L)$$

Where,

γ = Unit weight of the backfill, KN/m^3;

H=total height of exposed to fill the stope, m; and

L= strike length (width) of the backfilled stope, m.

5.14.2 Backfill application

As the proposed mining method will depend on the quality of backfill and the timely adherence to mining and backfill placement sequencing, the following conditions should apply:

- For the backfill method to work, the mining block has to be relatively dry to be able to minimize uncontrolled flow of water;
- Bulkhead design should be based on the expected hydraulic pressure generated by the wet fill;

- Cemented fill masses should have an aspect ratio of not less than 1.1, i.e. the minimum strike length should not be less than the ore body width. Exposed slender fill mass (with w/h < 1.1) may experience deep mass failures;
- In normal circumstances mining with backfill is applied in up-dip mining scenarios. Where it is applied in down-dip scenario, the ore body has to be shallow dipping, such as in the case of the MOCB area, or cemented fill is used in the base of the secondary stopes, to facilitate the removal of the ore in the block immediately below (similar to extracting the pillar above);
- For economic reasons primary stopes should be smaller than secondary stopes; and
- With sufficient backfill strength in primary stopes that can remain stable, mining of the secondary stope can take place.

5.14.3 Bulkhead position

All bulkheads must be designed to withstand the hydraulic pressure from the backfill material. Unlike in the current MOCB mining where the draw point cross-cut is mined at an acute angle to the extraction drive, the proposed method will have the cross-cut mined at right angle to the extraction drive. This to provide anchorage for the bulkhead that is not disturbed.

5.15 STRESS ANALYSIS OF PRIMARY & SECONDARY STOPES

The stability of the proposed mining method has to be analysed in terms of stress effects. The positioning and sizes of the primary and secondary stopes will have a great influence on the stress generated around the excavation and the damage to the rock mass. The following have to be considered in the design of the new mining method:

- Primary stopes will always be mined against solid rock in strike abutment and hence rock conditions will be better than secondary stopes;

- Lower stress levels in secondary stope rock mass, prior to stoping, is critical in ensuring rock stability during crosscut mining and blasting operations;

- The best location of the crosscut within the secondary stope is in the centre of the stope where stress levels are lower than toward the edges;

- Mining of the crosscut through the pillar redistributes stresses leading to increased stress concentration on the sides; and

- Though in general large secondary stopes offer better working conditions, and are more economical in terms of cement usage, issues of roof instability and ore losses arise.

Stress distributions differ with respect to the dip of the ore body. The steeper ore bodies have stress concentrations inclined towards the up dip and the stress concentration will tend to increase towards the centre in the flatter ore bodies (see Figure 5.23). The proposed mining method is in the relatively flatter ore body in the nose area of the Konkola Mine

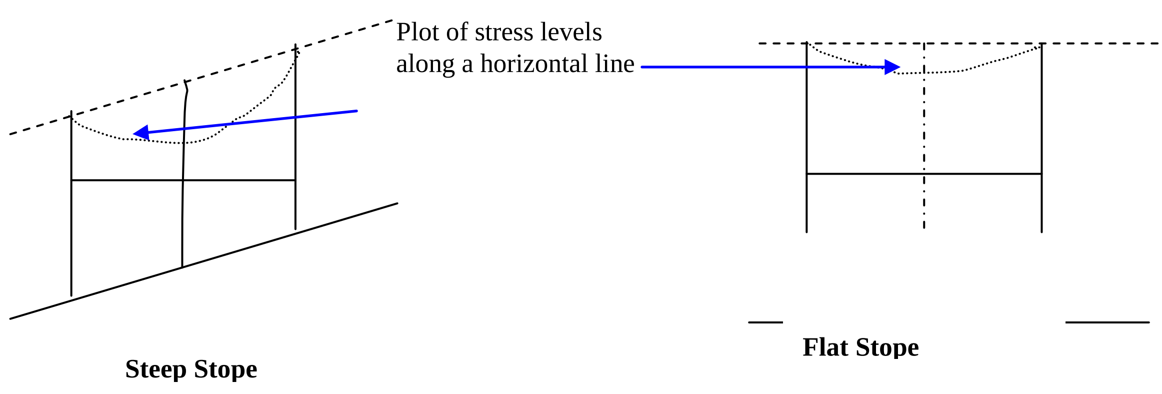


Figure 5.23 Pillar Stress distribution in steep and flat ore bodies

A systematic way of operation has to be enforced in the proposed mining method in order to achieve a desirable ore recovery. Primary and secondary stopes have to be taken as in Figure 5.24. P and S are primary and secondary respectively.

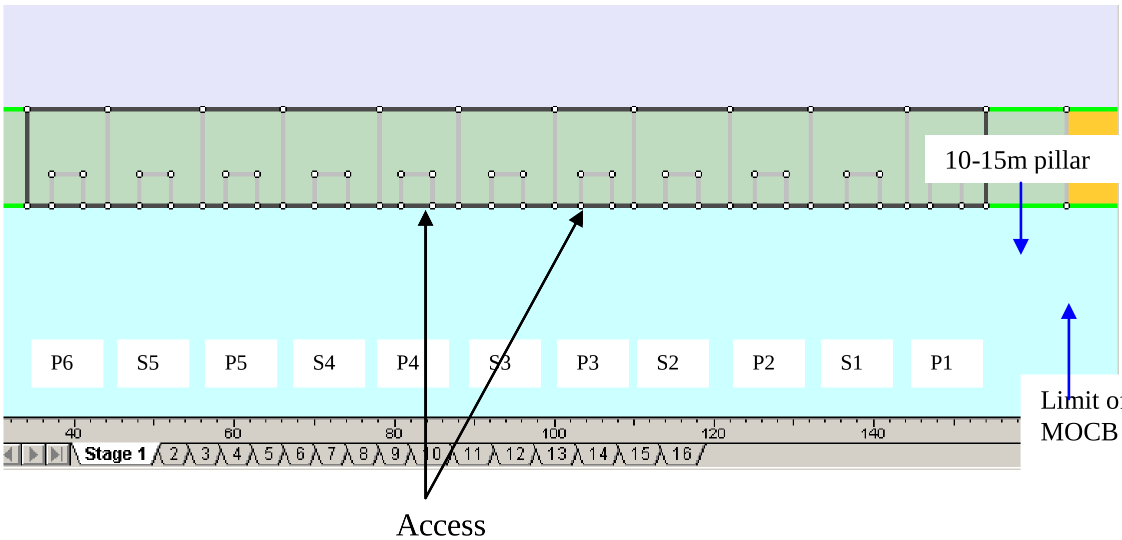


Figure 5.24 Schematic arrangement of Primary and Secondary stopes with access crosscuts

5.16 PILLAR STRENGTH ANALYSIS

Minimum width/height ratio – (Pillar Strength vs. Pillar Stress = Strength Factor)

From development stage, through extraction and back filling of the stope, the width to height ratios and strength factors provide the necessary information for pillar stability. A 15.0m regional pillar was taken into consideration in terms of the strength factors (S.F) around the excavation using numerical modeling as shown in Figures 5.25 to Figure 5.28. The pillar is intended to remain essentially intact and elastic during the life of production, hence the pillar should not yield.

The strength factors surrounding the drive mined in the regional pillar and the stoped out drive are shown in Figure 5.25.

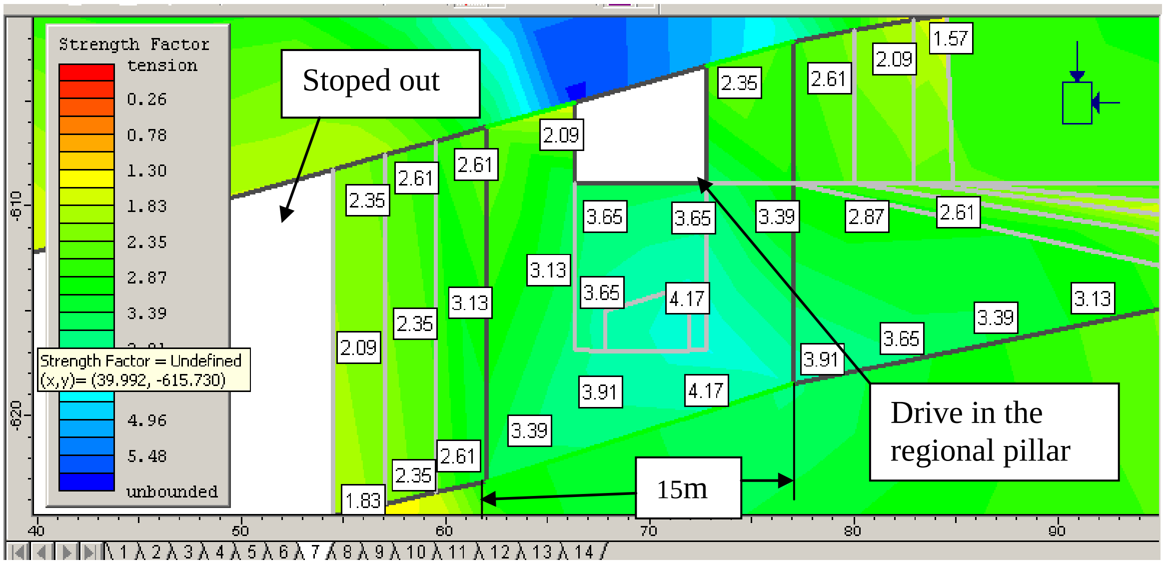


Figure 5.25 Pillar size of 15.0m – Strength Factor distribution in pillar

Leaving a 15.0m regional pillar gives the strength factor as shown in Figure 5.25 with the lowest being 1.83 around the stoped out down dip stope.

The stoped out area is then backfilled and strength factor analysed around the drive mined in the regional pillar and around the backfilled stope as shown in Figure 5.26. Strength factors do not change when area is backfilled with hydraulic fill.

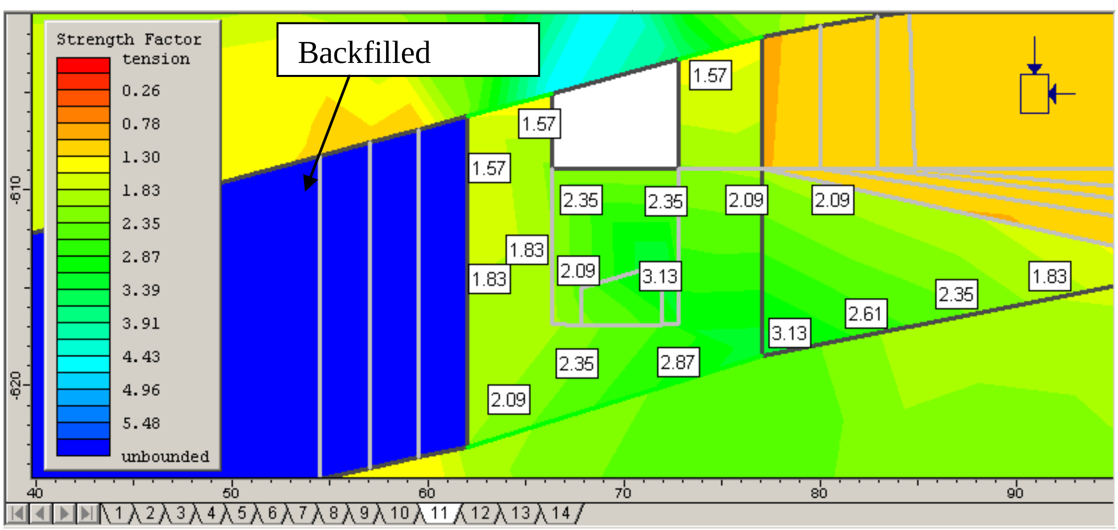


Figure 5.26 Strength Factors distribution after backfilling the stope

Further analysis of strength factors was done after benching out the drive and the strength factor distribution recorded as in Figure 5.27. Benching may extend to the pillar limits on both up and down dip at primary stope positions if backfilled. Strength factors do not change when area is backfilled with hydraulic fill.

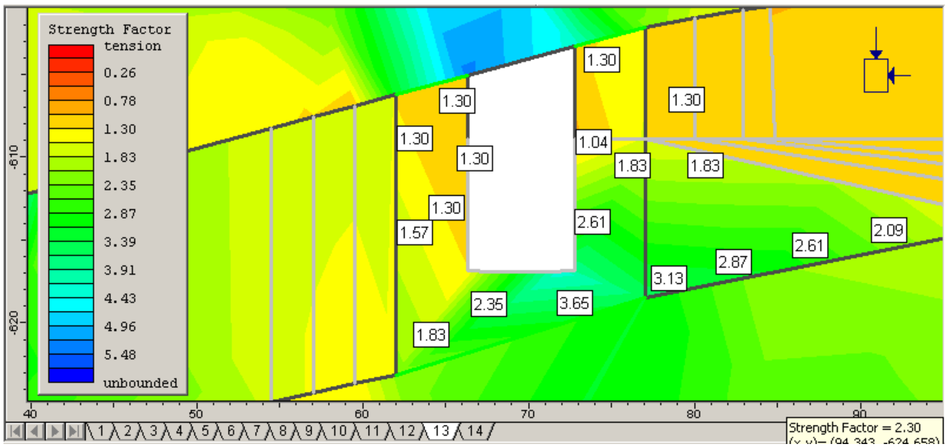


Figure 5.27 SF distribution in the 15m pillar with a drive benched

The drive was then analysed by partially filling the backfill material as shown in Figure 5.28 below which shows that strength factors do not change with hydraulic fill. Therefore, hydraulic fill just confines the area to maintain stability.

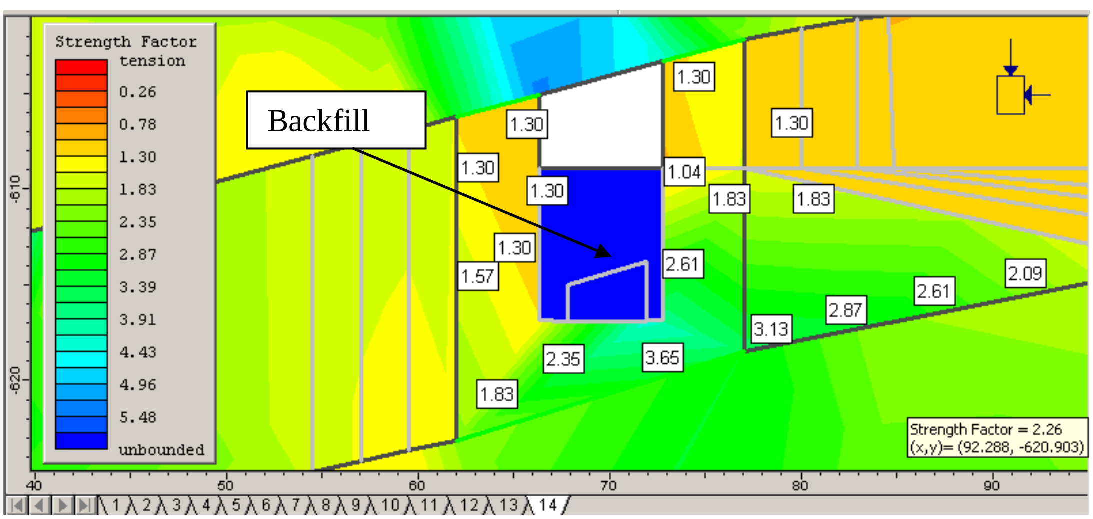


Figure 5.28 Strength Factors distribution in the 15m pillar after backfilling

5.17 DRILLING AND BLASTING

For proper ground control and fragmentation purposes the blast design must be such that the throw is minimised and the break confined within the stope limit. For this reason the following must be adhered to:

- Drilling and blasting operations have to conform to the prescribed drilling sizes and blast design respectively; and
- Blast rings should be inclined at 70^{o} toward the open stope to minimize ground throw distance. This will allow ore to be thrown where lashing will not be directly in the stope.

5.18 ORE RECOVERY ANALYSIS

In Section 5.9 for hanging wall stability of stopes, different sizes of stope span were suggested: 8m, 10m, 12m, 15m and 19m for both primary and secondary stopes. All except 19m span showed stability of the roof as calculated from the modified stability number and hydraulic radius plot.

Ore recovery in the stable spans will take preference in the selection of the proposed mining method. Ore recovery calculations were done in all the suggested stope spans as shown in the Figure 5.29. The wider the span the more ore will be left in the stope. With a cone angle of 53^{0}considering the dip of the ore body, 8.0m stope span recorded the highest ore recovery. A 5.0mW X 4.0mH drive is to be mined before stoping out. A sample of calculations on 8m and 19m ore recovery at a 50.0m stope length and 12.0m stope height are shown below:

- **8m stope span**

$$\text{Ore Recovery} = \frac{\text{Planned Ore} - \text{Ore left in the stope}}{\text{Planned Ore}}$$

$$= \frac{(8 \times 12 \times 50) - (1/2 \times 2 \times 2 \times 2/\tan 53 \times 50)}{(8 \times 12 \times 50)}$$

$$= \frac{4800-150}{4800}$$

$= 0.97$

$= 97\%$

- **19m stope span**

$$\text{Ore Recovery} = \frac{\text{Planned Ore} - \text{Ore left in the stope}}{\text{Planned Ore}}$$

$$= \frac{(19 \times 12 \times 50) - (1/2 \times 9 \times 2 \times 9/\tan 53 \times 50)}{(19 \times 12 \times 50)}$$

$$= \frac{11400-3150}{11400}$$

$= 0.72$

$= 72\%$

A 8.0m span is thus suitable for the proposed mining method without considering dilution.

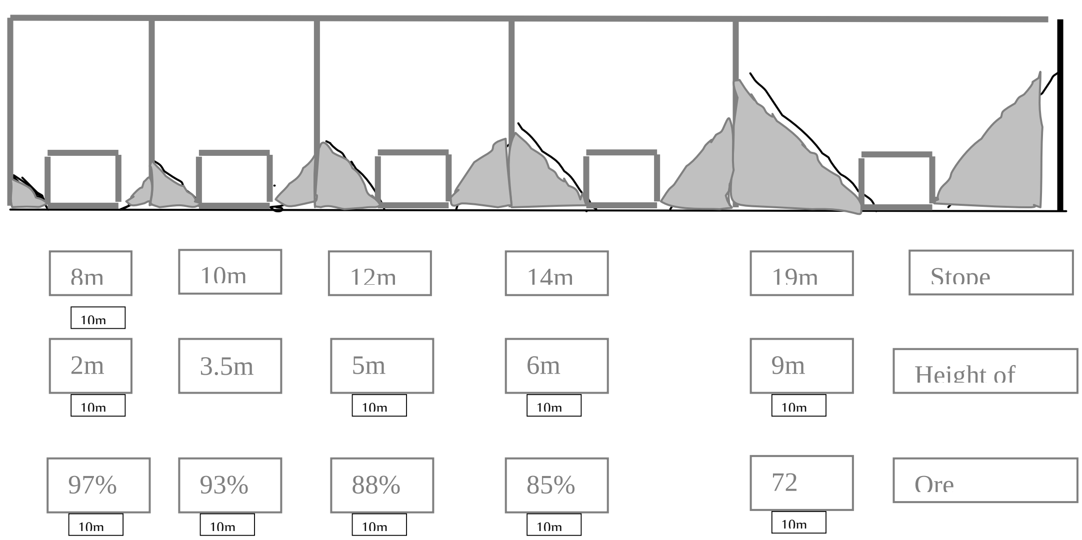


Figure 5.29 Ore Recovery in different stope spans

CHAPTER 6

CONCLUSIONS AND RECOMMENDATIONS

6.1 CONCLUSIONS

- In the nose area the thickness of the ore body ranges from 8m to 14m. Essentially the proposed method is an inverted version of the MOCB, but using primary-secondary sequence of ore extraction to maximize ore recovery. The method offers more than one stope in production at any one time, hence, high productivity. The success of any mining method is among other factors heavily dependent on bridging the gap between the design stage and the execution stage. It is imperative, therefore, that the design parameters and working procedures are followed as far as practically possible.

- While either of the options highlighted in Section 5.9 can be chosen i.e. the 8.0m, 10.0m, 12,m, 14.0m and 19.0m spans, advantages of ore recovery and risks are not the same. Thus option involving the mining of 8.0m primary and 8.0m secondary stopes with cemented backfill will be ideal. Unlike in MOCB, absence of a chamber drive in this method will enhance the stability of the hanging wall extraction drive as creation of pillars is eliminated.

- Another option of having 10m stopes with 4m pillar may arise if the economic issues regarding the use of cemented backfill are considered, as is in a case with the current MOCB design. Using this means of extraction implies that mining will progress in solid ground without the need of mining in secondary stopes. In that case, only uncemented hydraulic fill will be used thus doing away with the cost of cement. The possibility of the 4m pillar collapsing will not be so much of a concern as these stopes will be non-entry.

- The choice of the size of the 8.0m primary stopes depends on the stability of unsupported stope spans during extraction. The smaller the stope span the more stable it is, but smaller cemented primary stopes (aspect ratio < 1.1) may not be strong enough to support the open secondary stopes, as the fill may fail due to slenderness, unless more cement is used. On the other hand, if the primary stope is larger than the secondary stope, more cement will be used per a tonne of ore extracted. In general terms, the wider the primary stope the more unstable it becomes in the hanging wall, but the more stable it is in the secondary stope as a result of the wider fill in the primary stope. Wider stopes also lead to higher ore loses.

- In all the options, for safe and maximum productivity to be realized, access development should be put in place in advance to provide more than one production face on a level through commencement of production from the centre of the block toward the access ramps. Available hydraulic backfill plants with stone grinding plant provide an avenue for increased production from current levels in the PPCF and MOCB areas.

6.2 RECOMMENDATIONS

- For long stability with the proposed mining method of Cut and Fill it is necessary and desirable to position the access ramp in the hanging wall part of the ore body with the geological hanging wall (GHW) forming the back of the stope. Experience has shown that any skin of ore left below the GHW creates instability in the back of the development. This is because the skin is weak.

- Where possible, the unit ‘A’ band (weak portion of ore) should not be exposed during the mining of drilling crosscuts on apparent dip. Mining should be directed by survey pegs and not by geological information. Survey lines will also ensure the development is mined in the middle of the stope where, like in secondary stopes, induced stress levels will be low.

- To maximize collection of broken materials by the loader, it may be desirable to blast the stope in a form of a trough where all the broken rock would collect.

However, this would not only allow some ore loss in situ, but it will also enhance secondary stope stability by the reduced height of the adjacent backfill.

- For stability purposes, it is recommended that primary stopes should be designed with the fill aspect ratio of not less than 1.1, as slender fill mass will not support secondary stopes. Secondary stopes should be designed with W/H ratio of not less that 1.0 or SF of not less than 1.5.

- Where the thickness of the ore body is too small to accommodate both the hanging wall extraction drive and the footwall pillar extraction drive, then the footwall drive should not be mined but instead the pillar should be taken as part of the stope.

- Considering the average height of the stopes to be 9-12m, ii is recommended that cemented fill tests results based on laboratory results should be carried out to establish backfill strength characteristics. For economic purposes, it is desirable to carry out cement zoning during fill placement. This means that in primary stopes, the cement content should vary, decreasing in percentage up dip, as highest loads are at the lower parts of the stope.

- The support of primary stope development will be lighter as mining will take place in almost virgin ground and towards the ore body that is not mined, while support of the secondary development will attract additional secondary support to ensure stability for the life of the stope. It is recommended that secondary development should not remain standing for a long time prior to commencement of stoping activities as deterioration is time dependent.

- The regional pillar between the proposed mining method and the MOCB should not be mined out, as it separates the uncemented backfilled MOCB stopes above from the new stopes below. The proposed Cut and fill mining method with cemented fill will be a new mining method at Konkola Cooper Mines Plc. Thus, it is essential to carry out a trial operation using the method in one of the mining blocks. This will provide a learning curve for personnel involved in operations.

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APPENDICES

APPENDIX 1A

PONIT LOAD TESTING DATA

FORMATION	RANGE (MPa)	AVERAGE
Footwall Quartzite	148 to 551	330
Argillaceous Sandstone Unit 1	24 to 403	181
Argillaceous Sandstone Unit 2	21 to 265	99
Argillaceous Sandstone Unit 3	44 to 372	175
Argillaceous Sandstone Unit 4	48 to 409	130
Argillaceous Sandstone Unit 5	15 to 312	180
Porous Conglomerate	12 to 240	74
Footwall Sandstone	61 to 478	200
Footwall Conglomerate	14 to 353	164
Ore Shale Unit A	4 to 6	10
Ore Shale	20 to 399	190
Hanging wall Quartzite	59 to 307	168

(After Mutambo, KCM, 2011)

APPENDIX 1B

AVERAGE RQD FOR ROCKS - KONKOLA MINE

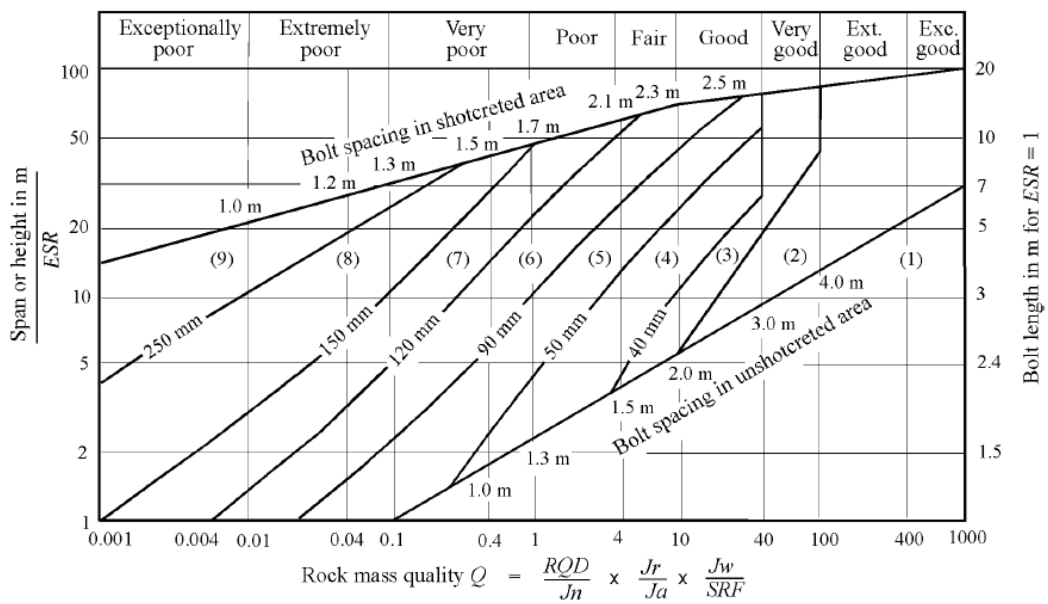
BASED ON CORE LOGGING AND UNDERGROUND MAPPING

FORMATION	RANGE (%)	AVERAGE (%)
Footwall Quartzite	80 to 95	88
Argillaceous Sandstone	50 to 80	65
Porous Conglomerate	50 to 70	60
Footwall Sandstone	75 to 90	73
Footwall Conglomerate	45 to 70	58
Ore Shale	25 to 60	43
Hanging wall Quartzite	80 to 90	85

(After Mutambo, KCM, 2011)

APPENDIX 2

Other estimated reinforcements categories based on tunneling Quality index Q



REINFORCEMENT CATEGORIES

1) Unsupported
2) Spot bolting
3) Systematic bolting
4) Systematic bolting with 40-100 mm unreinforced shotcrete
5) Fibre reinforced shotcrete, 50 - 90 mm, and bolting
6) Fibre reinforced shotcrete, 90 - 120 mm, and bolting
7) Fibre reinforced shotcrete, 120 - 150 mm, and bolting
8) Fibre reinforced shotcrete, > 150 mm, with reinforced ribs of shotcrete and bolting
9) Cast concrete lining

(After Grimstand and barton 1993)

APPENDIX 3A

Rock Mass Rating

A. CLASSIFICATION PARAMETERS AND THEIR RATINGS

	Parameter		Range 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 comp. strength	>250 MPa	100 - 250 MPa	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 discontinuities		> 2 m	0.6 - 2 . m	200 - 600 mm	60 - 200 mm	< 60 mm		
	Rating		20	15	10	8	5		
4	Condition of discontinuities (See E)		Very rough surfaces Not continuous No separation Unweathered wall rock	Slightly rough surfaces Separation < 1 mm Slightly weathered walls	Slightly rough surfaces Separation < 1 mm Highly weathered walls	Slickensided surfaces or Gouge < 5 mm thick or Separation 1-5 mm Continuous	Soft gouge >5 mm thick or Separation > 5 mm Continuous		
	Rating		30	25	20	10	0		
5	Ground water	Inflow per 10 m tunnel length (l/m)	None	< 10	10 - 25	25 - 125	> 125		
		(Joint water press)/(Major principal σ)	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		

B. RATING ADJUSTMENT FOR DISCONTINUITY ORIENTATIONS (See F)

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

C. ROCK MASS CLASSES DETERMINED FROM TOTAL RATINGS

Rating	100 ← 81	80 ← 61	60 ← 41	40 ← 21	< 21
Class number	I	II	III	IV	V
Description	Very good rock	Good rock	Fair rock	Poor rock	Very poor rock

D. MEANING OF ROCK CLASSES

Class number	I	II	III	IV	V
Average stand-up time	20 yrs for 15 m span	1 year for 10 m span	1 week for 5 m span	10 hrs for 2.5 m span	30 min for 1 m span
Cohesion of rock mass (kPa)	> 400	300 - 400	200 - 300	100 - 200	< 100
Friction angle of rock mass (deg)	> 45	35 - 45	25 - 35	15 - 25	< 15

E. GUIDELINES FOR CLASSIFICATION OF DISCONTINUITY conditions

Discontinuity length (persistence) Rating	< 1 m 6	1 - 3 m 4	3 - 10 m 2	10 - 20 m 1	> 20 m 0
Separation (aperture) Rating	None 6	< 0.1 mm 5	0.1 - 1.0 mm 4	1 - 5 mm 1	> 5 mm 0
Roughness Rating	Very rough 6	Rough 5	Slightly rough 3	Smooth 1	Slickensided 0
Infilling (gouge) Rating	None 6	Hard filling < 5 mm 4	Hard filling > 5 mm 2	Soft filling < 5 mm 2	Soft filling > 5 mm 0
Weathering Ratings	Unweathered 6	Slightly weathered 5	Moderately weathered 3	Highly weathered 1	Decomposed 0

F. EFFECT OF DISCONTINUITY STRIKE AND DIP ORIENTATION IN TUNNELLING**

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

(After Bianiawski, 1989)

APPENDIX 3B

Rock Mass Quality based on the evaluation of Q.

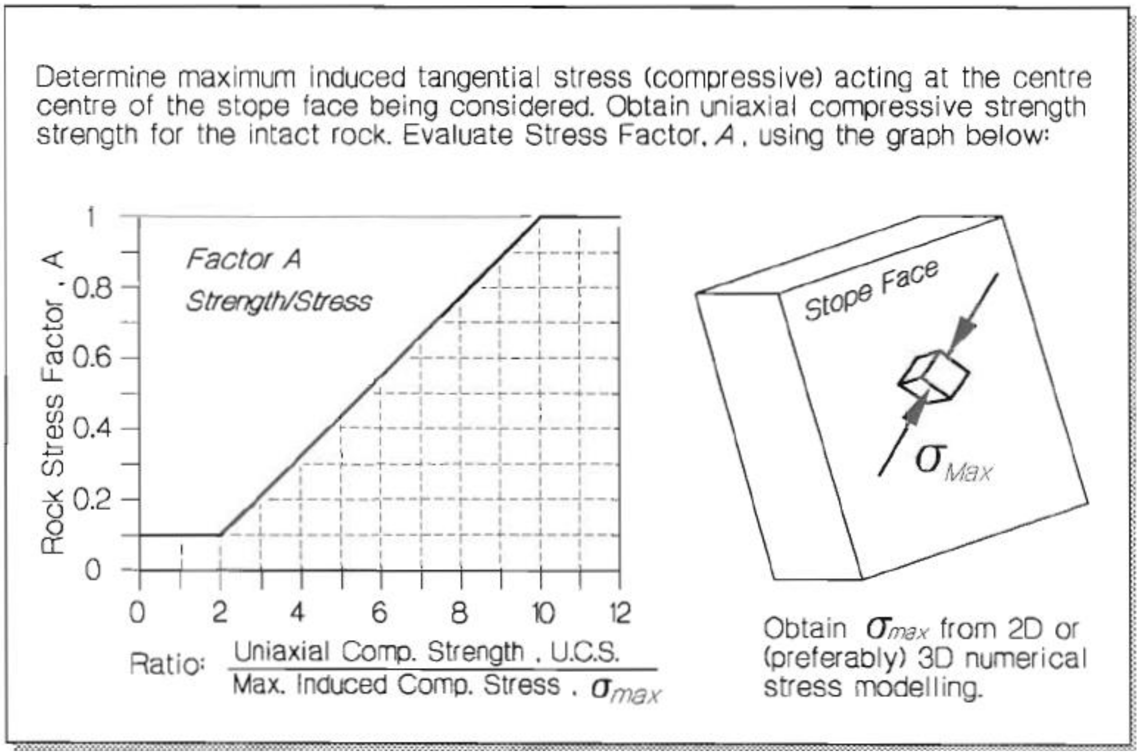
Tunnelling Quality Index $Q = \frac{RQD}{Jn} x \frac{Jr}{Ja} x \frac{Jw}{SRF}$	Rock Mass Description
0.001 - 0.01	Exceptionally Poor
0.01 - 0.1	Extremely Poor
0.1 - 1	Very Poor
1 - 4	Poor
4 - 10	Fair
10 - 40	Good
40 - 100	Very Good
100 - 400	Extremely Good
400 - 1000	Exceptionally Good

(After Barton et al. 1974)

APPENDIX 4A

Determination of stress reduction factor A

Rock Stress Factor A

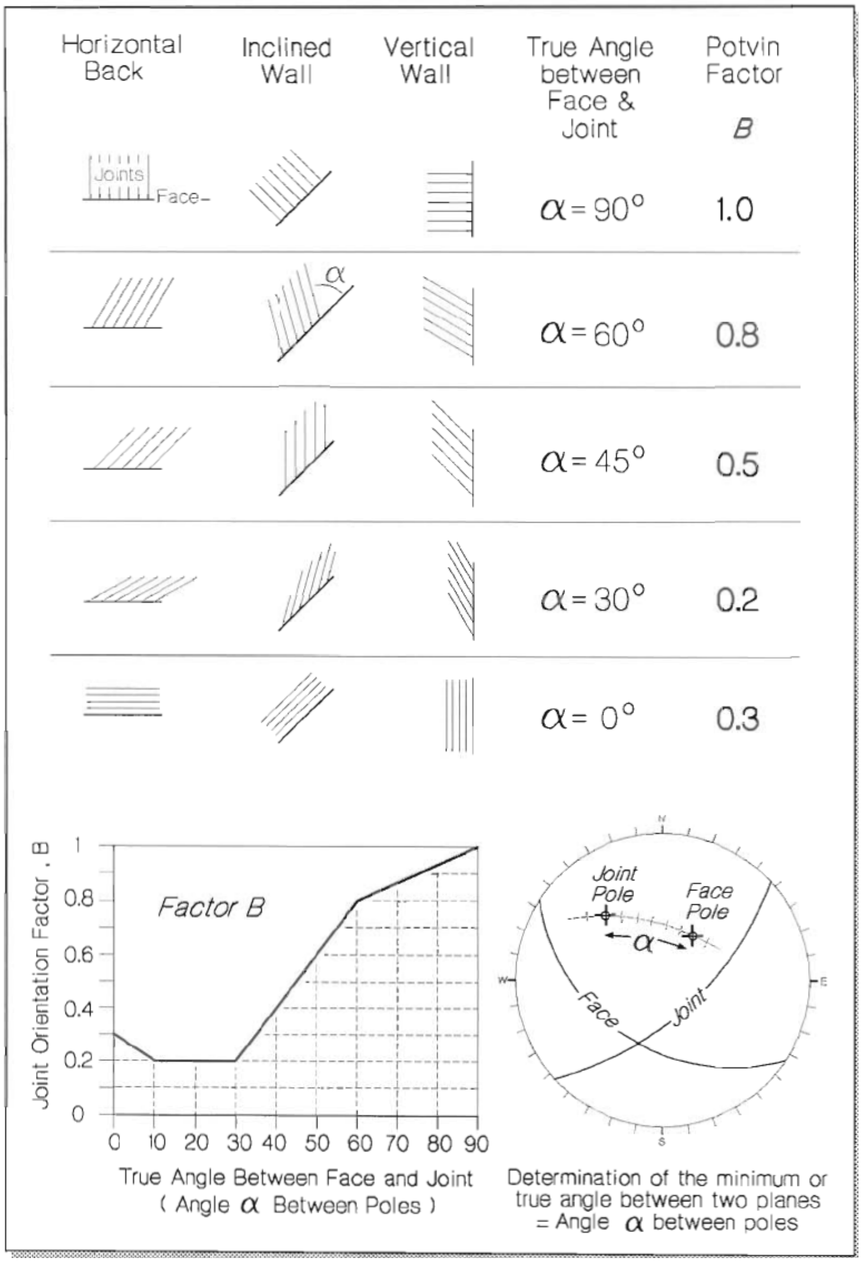


Rock Stress Factor, A, (After Potvin, 1988) reference number 15

APPENDIX 4B

Determination of orientation factor B

Joint Orientation Factor, B

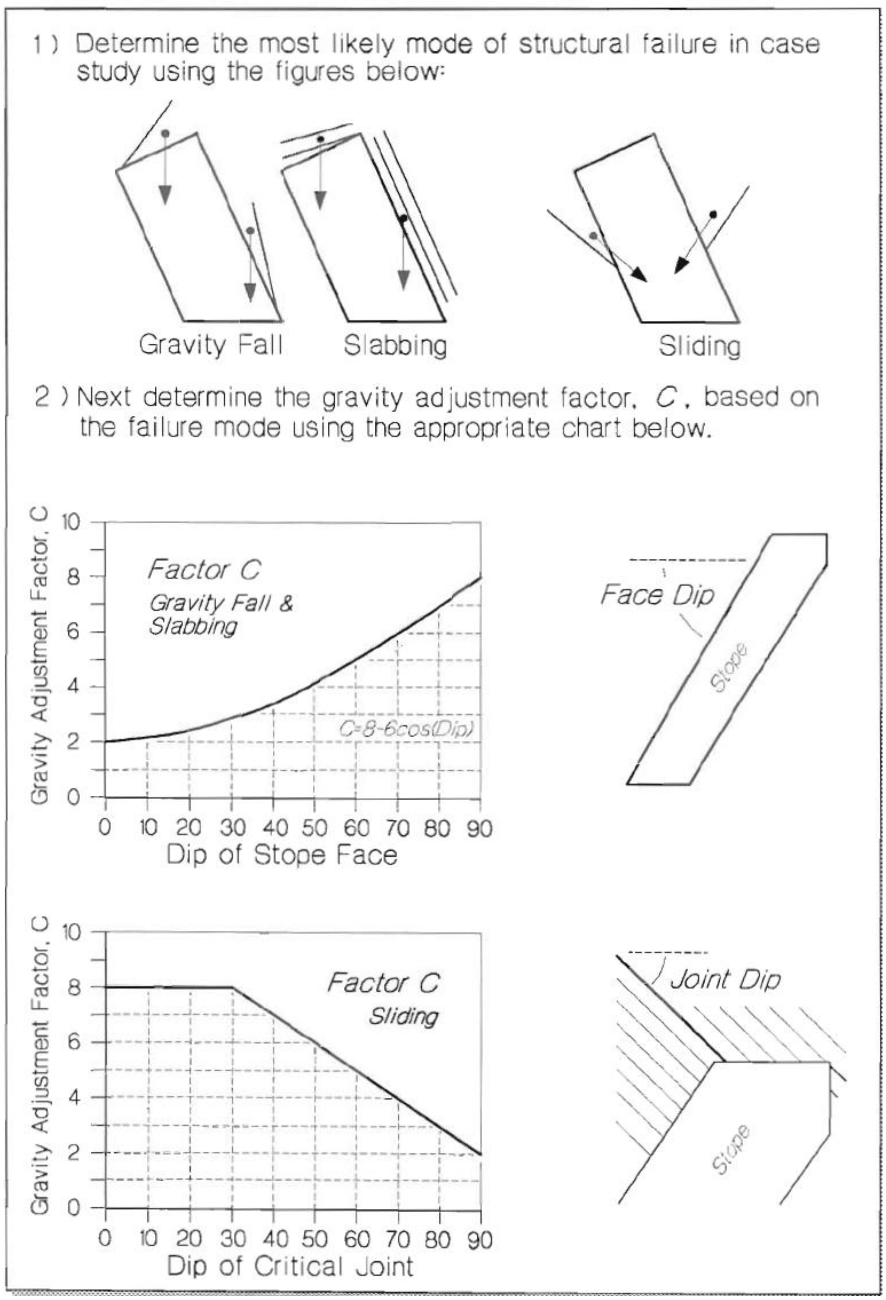


Determination of Joint Orientation factor, B, (After Potvin, 1988) reference number 15

APPENDIX 4C

Determination of Gravity Orientation Factor C

Gravity Adjustment Factor, C



Determination of the Gravity Adjustment factor, C, (After Potvin 1988) reference number 15

APPENDIX 5

KONKOLA COPPER MINES PLc
KONKOLA MINE
Geological Services Dept.
Geotechnical Section
Rock Property Estimates

	Hangingwall Quartizite	Ore shale	Unit A	Footwall Conglomerate
Strong rock mass				
RMR	**75**	**75**	**45**	**85**
UCS	**220**	**200**	**90**	**220**
mb(rock mass value)	**7.8**	**5.7**	**2.0**	**12.9**
s (rock mass value)	**0.06**	**0.06**	**0.002**	**0.19**
Young's Modulus	**42**	**42**	**7.5**	**75**
Poison's Ratio	**0.2**	**0.2**	**0.3**	**0.2**
Cohesion	**9.3**	**8.6**	**1.3**	**12.8**
Friction Angle	**54**	**53**	**40**	**58**
Average rock mass				**75**
				170
RMR	**65**	**60**	**25**	**9.0**
UCS	**150**	**150**	**30**	**0.062**
mb(rock mass value)	**5.4**	**3.4**	**1.0**	**42**
s (rock mass value)	**0.02**	**0.012**	**0.0002**	**0.2**
Young's Modulus	**24**	**18**	**2.4**	**6.7**
Poison's Ratio	**0.2**	**0.2**	**0.3**	**57**
Cohesion	**4.3**	**3.5**	**0.8**	
Friction Angle	**52**	**51**	**30**	
Weak rock mass				
RMR	**50**	**40**	**5**	**60**
UCS	**110**	**110**	**5**	**130**
mb(rock mass value)	**3.2**	**1.6**	**0.5**	**5.3**
s (rock mass value)	**0.004**	**0.001**	**0.00003**	**0.012**
Young's Modulus	**10**	**6**	**1**	**18**
Poison's Ratio	**0.2**	**0.2**	**0.3**	**0.2**
Cohesion	**2.1**	**1.7**	**0.3**	**3.1**
Friction Angle	**47**	**45**	**13**	**54**

(After SRK, 2002)