# Production Potential Of Nchanga Underground Mine's Collapsed Blocks

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**Abstract:** the main purpose of this study is to recommend modification to block caving at Nchanga, ensure that it meets anticipated production levels and address the adverse ground conditions, of the intensely fractured orebody. Excavations of current methods are driven close to the incompetent orebody. Determination of the appropriate method based on criteria of selection techniques, together with the analysis of operating costs and safety. Reclamation of ore in the collapsed blocks entirely depended on maximizing revenue, recovery of the mineral and safe working environment for equipment and personnel. On recommendation of a suitable method, extent of the collapsed blocks was another aspect considered. The proposed methods of extraction were variants of block caving, further shortlisted based on the extent of collapse. Economic appraisal of both the recommended and current mining methods employed included extraction, recovery, development, reclamation costs, revenue estimation and revenue raised from finished copper.

**Index Terms:** Abutments, block caving, bolting, fault zone, mining method selection, MRMR, Nchanga Mine, scrapper drift, UCC.

## INTRODUCTION

Nchanga mine lies on the Zambian Copperbelt. It has 3 superimposed strataform orebodies in a basal part of a thick succession of sediments of pre-cambrian age. Sediments lie on a Basement complex of granites, schist and gneisses, these are;

- Lower orebody (LOB)
- Intermediate orebody (IOB)
- Upper orebody (UOB)

## GEOLOGICAL SETTING

Nchanga Underground Mine is located on the southern limb of the Nchanga Main Syncline. The syncline is asymmetric, plunging to the northwest with a 20° to 30° gently dipping South Limb and a steep overturned North Limb. The rocks are mainly the Achean basement complex consisting of granites, gneisses and schists and the late Precambrian Katanga system, a sedimentary series containing quartzites, argillites, arenites, siltstones, dolomites and limestone[1]. The major ore bodies are the LOB hosted in argillaceous shale locally known as the Lower Banded Shale (LBS) and the UOB in a feldspathic quartzite (TFQ). The Nchanga Underground Mine extracts the LOB. Towards the east of the main syncline, the rocks are closer to surface and mining is carried out from the Nchanga Open Pit where mainly the UOB is mined [2].

## MINING METHOD

The LOB from the Nchanga underground mine is extracted by continuous advancing long-wall block caving mining method. The method involves undercutting a competent Arkose that is broken by blasting subsequent to which an incompetent overlying Transitional Arkose/Shale layer and the LBS cave due to tensile forces developed in the undercrown after the blasted Arkose rock has been drawn[2]. The development layout consists of a Trough Drive on the undercut level located 1.8m the TGMG (Top of Good Mining Ground) the maximum height excavations can reach before exposing the intensely fractured ore zone, below the Assay Footwall (AFW) and Scraper Drifts 6m below the Trough Drive on the extraction level. Both the Trough Drive and Scraper drifts are oriented parallel to strike.

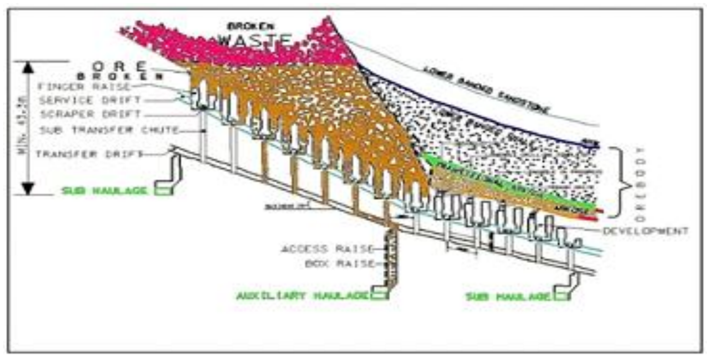


***Fig. 1.*** *Block caving*

Caved ore drops from the undercut level to the Scraper drifts via a series of finger raises developed from the Scraper Drifts to the Trough Drive. The ore is then scraped along the Scraper Drifts into a sub transfer chute to a Transfer Drift located about 20m below the AFW, well in competent footwall rocks and from there to the main tramming level. Access to the Scraper Drifts is through a Service Drift developed along the dip of the ore body. A typical block is 120m long along strike and 80 to 100m along dip and is serviced by a single Service Drift located in the centre of the block. One Service Drift caters for several blocks along dip and is extended as mining progresses down dip [2]. Undercutting and hence caving in a single block is started from two positions along a single trough drive and progresses down dip at the rate of 4 pairs of drives per year, blocks along strike can be caved at the same time.

## STRESS ENVIRONMENT

In-situ stress levels in the mining areas are generally low due to shallow depth of the operations. No stress measurements have been carried-out, but reasonably assumed that the stress tensor is similar in terms of alignment comparative to the

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bedding plane as the stress tensor measured at Konkola Mine. The major principal stress is usually sub-vertical ($45-60^{\circ}$) and sub normal to the bedding plane ($60-90^{\circ}$) [2]. The intermediate and minor principal stresses are almost equal and oriented parallel to the bedding plane. Major, intermediate and minor principal stress gradients of 0.041 MPa/m, 0.018 MPa/m and 0.016 MPa/m respectively, K-ratio is 0.850. The initiation of caving from multiple positions along a drift and opening up several blocks along strike leads to creation of small abutments in which high-induced stresses occur. Other zones of relatively high-induced stresses are the down dip side of the caving block. In some cases, production requirements caused opening up of several faces along dip as well thus creating more zones of high stress at the caving block boundaries [2]. High mining induced stress is a major problem, has led to collapse of certain blocks resulting in temporary and permanent losses of scraper and service drifts (Fig. 2a & 2b), inaccessible Ore. Prime causes for the induced stress build-up are:

- Geological estimation of the orebody orientation, estimation enables excavations not to be too close/too far from the ore zone. Poor estimation, leads to excavations being driven too close, up-setting the standard 1.8m to TGMG, at times driven in the intensively fractured ore zone.
- Proximity of excavations to the orebody. Over excavations, a small middling fraction is left between the ore body and the excavations. Trough drives are driven close to the ore zone beyond the standard 1.8m TGMG. Reduces the stand-up time for the excavation, tend to collapse before support installation is set.
- The orebody lies in the fault zone, the fault and shear zone affect the LOB, the main fault zone trends 320° and down throws the orebody south westwards, affects areas like 2720 3W and 4W. A prominent fissure system trends South East and North West, levels affected include 1820 feet level**.** Jointing and discontinuities associated with fault zone act as water conduits causing weathering of the rock mass, resulting in poor quality and thinly bedded rock mass.
- Excessive vibrations from blasting, greatly affects the ground stability, due to an increased use of powder factor.
- Failure of current mining methods i.e. UCC uses many drill holes and high amount of explosives, excavations are mined close to the intensely fractured assay.
- Mining induced stress, as depth of excavations increases, vertical stresses increases causing varying stress conditions to occur at points within mining blocks as mining progresses down dip,

$$\textit{Vertical rock pressure} \quad \delta v \approx 27 Mpa/km \qquad (1)$$

$$\textit{Horizontal pressure} \quad \delta_h \sim (0.5-3)\delta_v \qquad (2)$$

***Fig. 2a**. Collapsed scraper drift*

**Fig. 2b.** Blocked crosscut

## EXCAVATION DAMAGE

Excavation damage is mainly caused by high mining induced stresses that are generated in small remnant pillars, closure positions and areas in the caving front abutment. Damage occurs mainly in the Scraper and Service Drifts (Fig. 2a). Due to the caving sequence, scraper drift is subjected to cycles of very high loading, when caving of drifts up dip takes place and suddenly become de-stressed, when the cave front advances down dip. Further loading and unloading cycles generated as the finger raises put off draw during extraction of the caved rock. Other types of damage are sidewall spalling, damage of pillars left between finger raises, widening of the peak of the finger raises and slabbing in the roof because of high horizontal stresses generated in the middling between the Trough drive and Scraper drift if this middling is too small [2].

## MINING METHOD SELECTION

The ultimate goals of mining method selection are to maximize company profit, maximize recovery of the mineral resources and provide a safe environment for the miners by selecting the method with the least problems among the feasible alternatives [4]. Characteristics that have a major impact on the determination of the mining method are physical and geological characteristics of the deposit (Table 1), ground conditions of the host rock and ore zone (Table 2 &3), mining and capital costs, production rate, labour, environmental considerations and safety [5]. There is no single appropriate mining method for a deposit, usually two or more viable methods. Each method entails some inherent problems. Consequently, the optimum method is one offers the least problems. To determine which mining method is feasible, we need to compare the characteristics of the deposit with those required for each mining method; the method(s) that best matches should be the one(s) considered technically feasible, and then be evaluated economically [4].

***Table. 1.*** *Orebody characteristics*

| Ore strength | Weak and fractured |
|---|---|
| Shape | tabular |
| Grade | moderate |
| Thickness | 0.5-45 |
| Depth | Intermediate-deep |
| Dip | 20-30° |
| Rock strength | Weak Hangingwall, moderate to strong footwall |
| Uniformity | Not uniform |

***Table. 2.*** *Rock Mass Rating of Host Rock Bieniawski (1989)*

| | Hangingwall (BSSL) | | Footwall (Arkose) | |
|---|---|---|---|---|
| **Parameters** | **Description** | **Rating** | **Description** | **Rating** |
| UCS(MPa) | 1-10 | 2 | 100-300 | 12 |
| RQD (%) | 16 | 3 | 25-30 | 8 |
| Joint spacing (mm) | 1.5 | 5 | 0.5-1.0 | 15 |
| Joint condition | Rough | 5 | Smooth, no infill | 20 |
| Ground water condition (l/m) | Wet | 7 | Wet | 7 |
| Total Rock Mass Rating | | 22 | | 62 |
| Adjustment due to orientation of joints | | -5 | | 0 |
| RMR | | 17 | | 62 |
| Rock mass classification | Very poor rock | < 20 | Good rock | 80←61 |
| Class | V ( stand up time 10min /0.5m span) | | II (stand up time 6months/ 4m span) | |

***Table. 3.*** *Rock Mass Rating of Ore zone [Bieniawski (1989)]*

| Parameter | Description | Rating |
|---|---|---|
| UCS(MPa) | 1-35 | 2 |
| RQD (%) | 25 | 3 |
| Joint Spacing(mm) | 0.5-1.0 | 15 |
| Joint Conditions | Smooth, no infill | 20 |
| Ground water Conditions(l/m) | Wet | 7 |
| Total Rock Mass Rating | | 47 |
| Adjustment due to orientation of joints | | -5 |
| RMR | | 42 |
| Rock Mass Classification | Fair Rock | 60←41 |
| Class | III ( stand up time 1week / 3m span) | |

## SELECTION TECHNIQUES

Several methodologies have been developed in the past to evaluate suitable mining methods for an ore deposit, based on the physical and mechanical characteristics and geotechnical properties of the rock [5]. Techniques for evaluation; University of British Columbia (UBC), Hartman (1987), Morrison (1976), Nicholas (1981), Boshkov and Wright (1973), Laubscher (1981) and Analytical Hierarchy Process (AHP)

### 1. UBC method

An online computer based version of the Nicholas approach technique based on ore body characteristic. Involves summation and ranking of numerical values associated with orebody characteristics. The selection technique shows viable mining methods (Table. 4 & 5). It does not account for other factors i.e. status of underground water.

***Table. 4 .****Method selection, deposit depth (+600m)*

| Selection Basis | Orebody Characteristics | Mining Method Rankings |
|---|---|---|
| Geometry and Grade Distribution | General shape: Platy-Tabular | (best)<br><br>Longwall Mining (30)<br>Cut and Fill (30)<br>Sublevel Caving (27)<br>Block Caving (26)<br>Square Set Stoping (22)<br>Shrinkage Stoping (17)<br>Top Slicing (16)<br>Open pit (-18)<br>Sublevel Stoping (-25)<br>Room and Pillar (-37)<br><br>(worst) |
| | Ore Thickness: Intermediate(10-30m) | |
| | Ore Plunge: Intermediate(20-55deg) | |
| | Grade Distribution: Gradational | |
| | Depth: Deep (more than 600m) | |
| Rock Mass Rating (after Bieniawski 1973) | Ore Zone: Medium (40-60) | |
| | Hanging Wall: Very Weak (0-20) | |
| | Footwall: Strong (60-80) | |
| Rock Substance Strength | Ore Zone: Very Weak (less than 5) | |
| | Hanging wall: Very Weak (less than 5) | |
| | Footwall: Medium (10-15) | |

***Table. 5.*** *UBC method selection, deposit depth (-600m)*

| Selection Basis | Orebody Characteristics | Mining Method Rankings |
|---|---|---|
| Geometry and Grade Distribution | General shape: Platy-Tabular | (best)<br><br>Open pit (31)<br>Longwall Mining (29)<br>Cut and Fill (29)<br>Sublevel Caving (27)<br>Block Caving (26)<br>Square Set Stoping (21)<br>Shrinkage Stoping (18)<br>Top Slicing (16)<br>Sublevel Stoping (-23)<br>Room and Pillar (-36)<br><br>(worst) |
| | Ore Thickness: Intermediate(10-30m) | |
| | Ore Plunge: Intermediate(20-55deg) | |
| | Grade Distribution: Gradational | |
| | Depth: Intermediate (100-600m) | |
| Rock Mass Rating (after Bieniawski 1973) | Ore Zone: Medium (40-60) | |
| | Hanging Wall: Very Weak (0-20) | |
| | Footwall: Strong (60-80) | |
| Rock Substance Strength | Ore Zone: Very Weak (less than 5) | |
| | Hanging wall: Very Weak (less than 5) | |
| | Footwall: Medium (10-15) | |

The technique is based on the ranking parameters which explains the mineral deposit status the demerits of this approach is the limitation in the number of criteria and the selection alternatives in this technique, although the depth and the rock mass rating scores are added this limitation still binds[6]. Criteria such as deposit dimension, thickness changes or its uniformity, availability of skilled personnel in extraction, recovery in any mining method, subsidence effects and underground water status are neglected, this limitation also exist in the choice and alternative of selection. Deposit depth of the Lower orebody is from about 300 m to 700 m therefore, the deposit depth spans two options. This can be more easily countered by doing the selection twice, once for the intermediate part of the orebody and one time for the deep part of the orebody [7].

## 2. Hartman (1987)

This technique uses a flow chart selection process based for defining a mining method based on the geometry of the deposit and ground condition of the ore zone. The system is aimed at more specific mining methods similar to that proposed by Boshkov and Wright; the method is qualitative and includes surface and underground mining methods [4]. The flow chart proposed four methods in relation to the Nchanga ore body characteristics (Table. 6) the methods are in two classes, supported and caving methods, these include;

- Block caving
- Square Set Stopping
- Cut and Fill

The technique faces limitation, the approach for the selection of a suitable mining method is neither enough nor complete. It is easy to design a methodology that will automatically choose a mining method for the orebody in question.

***Table. 6.*** *Hartman's chart (Modified after Hartman, 1987)*

| Locale | Ore, Rock Strength | Class | Geometry | Method |
|---|---|---|---|---|
| **Underground** | Moderate to strong, competent | Unsupported | Tabular, flat, thin, large size | Room and Pillar |
| | | | Tabular, flat, thick, large size | Stope and Pillar Mining |
| | | | Tabular, steep, thin, any size | Shrinkage Stoping |
| | | | Tabular, steep, thick, any size | Sublevel Stoping |
| | **Moderate to weak, incompetent** | **Supported** | **Variable shape, thin ,any size** | **Cut and Fill** |
| | | | Tabular ,steep, thin, small size | Stull Stoping |
| | | | **Any shape, any dip, thick, any size** | **Square Set Stoping** |
| | **Moderate to weak, cavable** | **Caving** | **Tabular, flat, thin, large size** | **Longwall Mining** |
| | | | **Tabular or massive, steep ,thick, large size** | **Sublevel Caving** |
| | | | **Massive, steep, thick ,large size** | **Block caving** |

## 3. Morrison (1976)

This classification system divides underground mining methods into 3 basic groups

- Rigid pillar support
- Controlled subsidence
- Caving

General definitions of ore width, support type and strain energy accumulation are used as a criterion for determining mining method in this classification. This system helps in choosing one method over another based on various combinations of ground conditions, the conditions are evaluated to determine the type of support required [4]. The Morrison method of selection deals with the accumulation of strain energy, as the strain energy increases in the ore body caving methods are employed. The Morrison method of selection would classify this deposit as "invariably wide." Based on this, the only methods applicable would be caving methods (Fig. 3). The methods shaded i.e. Sublevel caving, Block caving.


***Fig. 3.*** *Morrison's chart (Modified after Morrison, 1976)*

## 4. Nicholas (1981)

The classification determines feasible mining methods by numerical ranking and is quantitative it uses the ore geometry, grade distribution, rock mechanics characteristic such as Rock substance strength, fracture spacing, fracture shear strength of the ore zone, hanging and footwalls[4] (Table. 7).

***Table. 7.*** *Characteristic Values Multiplied by Weighting Factors (After Nicholas, 1981)*

| Mining methods | Geometry/Grade Distribution | Rock Mechanics Characteristics | | | | Grand Total |
|---|---|---|---|---|---|---|
| | | | HW | FW | Total | |
| Block caving | 11 | 8 | 4.8 | 3.5 | 16.3 | 27.3 |

| | | | | | | |
|---|---|---|---|---|---|---|
| Top slicing | 8 | 8 | 5.6 | 5.0 | 18.6 | 28.6 |
| Sublevel caving | 13 | 7 | 4.8 | 1.5 | 13.3 | 26.3 |
| Square set stoping | 8 | 8 | 5.6 | 5.0 | 18.6 | 26.6 |
| Sublevel stoping | 11 | 5 | 5.6 | 1.0 | 11.6 | 26.6 |
| Longwall mining | -37 | 8 | 4.0 | 3.0 | 15.0 | 22.0 |
| Shrinkage stoping | 11 | 6 | 4.8 | 4.0 | 14.8 | 25.8 |
| Cut and fill stoping | 7 | 8 | 5.6 | 5.0 | 18.6 | 25.6 |
| Room & pillar | -38 | 7 | 6.4 | 1.5 | 14.9 | -23.0 |

## 5. Boshkov and Wright (1973)

This classification uses the general description of the ore thickness, ore dip, strength of the ore, and strength of the walls to identify common methods that have been applied in similar conditions [4], the result of this classification provide up to three methods that may be applicable.

***Table.8.*** *Applications of Underground Mining Methods (Modified after Boshkov and Wright, 1973)*

| Type of Ore body | Dip | Strength of Ore | Strength of Walls | Commonly Applied Methods of Mining |
|---|---|---|---|---|
| **Thin beds** | Flat | Strong | Strong | Open stopes with casual pillars |
| | | | | Room and pillar |
| | | | | Longwall |
| | | **Weak or strong** | **Weak** | **Longwall** |
| **Thick beds** | **Flat** | Strong | Strong | Open stopes with casual pillars |
| | | | | Room and pillar |
| | | **Weak or strong** | **Weak** | **Top slicing** |
| | | | | **Sublevel caving** |
| | | Weak or strong | Strong | Underground glory hole |

Boshkov and Wright's method would classify this deposit as either "thick beds" or "thin beds" with a "weak/strong" ore and a" weak" wall rock (Table. 8) above, since there are two areas of concern, the thin rich and the collapsed areas. Based on this classification, feasible mining methods include Top slicing, Sublevel caving and Block caving.

## 6. Laubscher (1981)

The selection process is based on rock mass classification system which adjust for expected mining effects on the rock mass strength, this system is aimed at mass mining methods, primarily block caving vs. stopping, the main emphasis is on cavability, the two parameters that determine a caving method is used over a stopping method are the degree of fracturing, RQD, joint spacing and the joint rating which describe the character of the joint i.e. discontinuity, filling and water conditions, the system puts emphasis on the jointing as the only control determining cavability[4].

***Table. 9.*** *Laubscher's (1981) RMR classification system*

| Rock mass Parameter | Value | Rating |
|---|---|---|
| **UCS (MPa)** | 1-35 | 4 |
| **RQD (%)** | 25 | 4 |
| **Joint Spacing(mm)** | 0.5-1.0 | 5 |
| **Joint Condition** | Smooth, no infill | 21.31 |
| | Total | 34.31 |

Laubscher's system requires more information than that provided, but a guess mate can be made from the data given. In actuality, one would have looked at the drill core and could therefore make the necessary measurements. Using Laubscher's (1981) RMR classification system, the rating of the rock is 34.31 (Table 9) above:

$$RQD + Joint\ Spacing\ ; 4 + 5 = 9 \quad (3)$$

$$Joint\ Rating\ = 21.31 \quad (4)$$

The values are then plotted (RQD +Joint Spacing against Joint Rating) (Fig. 4).

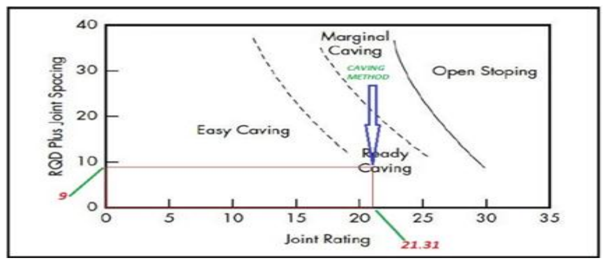


***Fig.4.*** *Laubscher's (1981) classifications for cavability evaluation*

Using his first method selection, which is based primarily on jointing, the ground would be considered either "easy caving" or "ready caving" (Fig. 4). The newer selection scheme, which uses the total mass rating and the hydraulic radius, indicates that a hydraulic radius of 28 is required for the deposit to cave (Fig. 5). A hydraulic radius of 28 is equivalent to a square area of (112m2) or an area of (100 by 125 m).

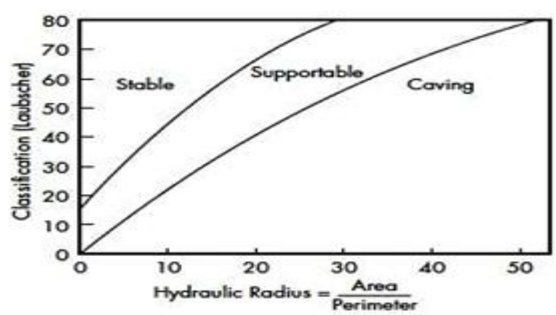


***Fig.5.*** *Laubscher's cavability based on hydraulic radius and classification*

Laubscher's (1990) mining rock mass rating (**MRMR**) classification system is one of the three classification system used, the other two are Geo-mechanics classification system **(**Bieniawski, 1973) and the Norwegian Geotechnical Institute's Q-System (Barton *et al*, 1974). The **MRMR** system involves the use of in-situ rating to a rock mass based on the measure of the geological parameters the parameters are weighed according to their relative importance the total rating is 100, values range between 0-100 with five rock mass classes comprises 20 per class from very poor to very good which are the result of the relative strengths of the rock mass [8]. The limitation of the **MRMR** system is its inability to adequately address the influence of the fractures/veins and cemented joints on the competency of the rock mass. Laubscher and Jakubec introduced the **IRMR** classification in 2000 to address the concerns about the application of the **MRMR** system to a jointed rock mass, recognizing the fact that the competence of a jointed rock mass is a function of the nature, orientation and continuity of the discontinuities[9],[10]. The revised **MRMR** system termed the in-situ rock mass rating classification system it has the following concept;

- Rock block strength (RBS)
- Cemented joint adjustment
- Joint condition(Jc) adjustment modification
- Water adjustment parameters.

## 7. Analytical Hierarchy Process (AHP)

Multi-attributed decision making (MADM) technique developed by Thomas L. Saaty it's a tool that combines qualitative and quantitative factors in the selection of a process and is used for setting priorities in a complex unanticipated, multi-criteria problematic situation. Provides a flexible and easy to understand way of analysing complicated problems [11]. The model has found numerous and diverse applications and is practised successfully, this methodology has been applied to numerous decision problems such as software selection sourcing decisions, the main merit of the AHP is its ability to handle complex and ill structured problems which cannot usually be handled by rigorous mathematical models, in addition to simplicity, ease of use, flexibility and intuitive appeal, the ability to mix qualitative and quantitative criteria [11]. Features of AHP differentiate it from other decision making approach:

- Ability to handle both tangible and intangible attributes
- Ability to structure the problem in a hierarchical manner to gain insights into decision making process
- Ability to monitor the consistency with which a decision maker uses in his judgement [11].

The solution process consists of three stages, namely

- Determination of relative importance of the attributes.
- Determination of relative importance of each of the alternative with respect to each attribute.
- Overall priority weight determination of each of these alternatives.

The AHP approach with 8 criteria is used to develop suitable mining method, comparison matrices are created then relative weights are derived for the various elements (Table 10) this was done for ore body thickness (Table 11), further matrices were created and computed in the same way for the rest of the attributes such as dip, depth, safety, shape and operating costs (Table 12 & 13) [11].

***Table. 10.****Pairwise comparison scale*

| Comparison index | Score |
|---|---|
| Extremely preferred | 1 |
| Very strongly preferred | 3 |
| Strongly preferred | 5 |
| Moderately preferred | 7 |
| Equal | 9 |
| Intermediate values between the two adjacent judgements | 2,4,6,8 |

***Table. 11.****Comparison of methods with reference to thickness*

| | **BC** | **TS** | **SSS** | **SC** | **LW** | **Weight** |
|---|---|---|---|---|---|---|
| **BC** | 1 | 3 | 3 | 3 | 9 | 0.4036 |
| **TS** | 1/3 | 1 | 1 | 1 | 9 | 0.1792 |
| **SSS** | 1/3 | 1 | 1 | 1 | 9 | 0.0359 |
| **SC** | 1/3 | 1 | 1 | 1 | 9 | 0.1792 |
| **LW** | 1/9 | 1/9 | 1/9 | 1/9 | 1 | 0.023 |

***Table. 12.****Reference to RMR of Hangingwall. Depth and Grade*

| Deposit Dip | Weight | RMR of Hangingwall | Weight | Depth | Weight | Grade | Weight |
|---|---|---|---|---|---|---|---|
| **Block Caving** | 0.1957 | Block Caving | 0.0753 | Block Caving | 0.1667 | Block Caving | 0.0399 |
| **Top Slicing** | 0.1582 | Top Slicing | 0.139 | Top Slicing | 0.1667 | Top Slicing | 0.3263 |
| **Square Set Stoping** | 0.1957 | Square Set Stoping | 0.0298 | Square Set Stoping | 0.1667 | Square Set Stoping | 0.3263 |
| **Sublevel Caving** | 0.1957 | Sublevel Caving | 0.1585 | Sublevel Caving | 0.1667 | Sublevel Caving | 0.1038 |
| **Longwall stoping** | 0.0597 | Longwall stoping | 0.0468 | Longwall stoping | 0.1667 | Longwall stoping | 0.0581 |

***Table. 13.****Reference to Operating Costs, Shape and Overall Rating*

| Safety | Weight | Operating Costs | Weight | Shape | Weight | Overall Rating | |
|---|---|---|---|---|---|---|---|
| **Block Caving** | 0.0541 | Block Caving | 0.4493 | Block Caving | 0.1667 | Block Caving | 0.1939 |
| **Top** | 0.0541 | Top | 0.0422 | Top | 0.1667 | Top | 0.1541 |

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| **Slicing** | | Slicing | | Slicing | | Slicing | |
| **Square Set Stoping** | 0.3249 | Square Set Stoping | 0.0422 | Square Set Stoping | 0.1667 | Square Set Stoping | 0.161 |
| **Sublevel Caving** | 0.1083 | Sublevel Caving | 0.1011 | Sublevel Caving | 0.1667 | Sublevel Caving | 0.147 |
| **Longwall stoping** | 0.1004 | Longwall stoping | 0.2327 | Longwall stoping | 0.1667 | Longwall stoping | 0.107 |

Block caving with a rating of 0.1939 is the most preferred then Square Set Stoping, Top Slicing mining methods (Table 13). Block Caving was the most appropriate on consideration of the 8 factors in the mining method selection process. Unlike the traditional approach to mining method selection, AHP makes it possible to select the best method in a more scientific manner that preserves integrity and objectivity.

***Table. 14.****Summary of techniques used for mining method selection*

| | Hartmann | Nicholas | UBC | Laubscher | Boshkov & Wright | Morrison | AHP |
|---|---|---|---|---|---|---|---|
| Longwall | √ | | 1 | | √ | | 5 |
| Sublevel caving | √ | 4(26.3) | 3 | | √ | √ | 4 |
| Block caving | √ | 2(27.3) | 4 | √ | | √ | 1 |
| Top slicing | | 1(28,6) | 7 | | √ | √ | 3 |
| Square set | √ | 3(26.6) | 5 | | √ | | 2 |
| Cut & fill | √ | 9(25.6) | 2 | | | | |
| Sublevel open stopping | | 5(26.6) | 8 | | | | |
| Shrinkage mining | | 7(25.8) | 6 | | | | |
| Room and pillar | | | 9 | | | | |

Summary and outcome (Table 14), scores for each mining method (Table 15). The methods with the higher scores are Square set, Block caving, Sublevel caving and Top slicing.

***Table. 15.****Mining Methods and score outcomes*

| **Method** | **Score** |
|---|---|
| **Long wall mining** | 4 |
| **Sublevel caving** | 6 |
| **Block caving** | 6 |
| **Sublevel open stopping** | 2 |
| **Cut and fill** | 3 |
| **Top slicing** | 5 |
| **Shrinkage mining** | 2 |
| **Room and pillar** | 1 |
| **Square set** | 5 |

## SELECTION BASED ON COST ANALYSIS

Although the mining methods resulting from the selection process are all technically feasible, their mining costs may be significantly different.

***Table. 16.****Ranking of mining methods based on Relative operating costs*

| **Hartman's Ranking** | | **Morrison's Ranking** | |
|---|---|---|---|
| Methods | Operating cost estimations (%) | Methods | Operating cost estimations (%) |
| Long wall mining | 20 | Block caving | Lowest |
| Block caving | 20 | Sublevel open stopping | |
| Room and pillar | 30 | Sublevel caving | |
| Sublevel open stopping | 40 | Long wall mining | |
| Sublevel caving | 50 | Room and pillar | |
| Shrinkage mining | 50 | Shrinkage mining | |
| Cut and fill | 60 | Cut and fill | |
| Top slicing | 70 | Top slicing | |
| Square set | 100 | Square set | Highest |

Based on the Hartman's and Morrison's (Table 16) relative rankings of mining costs, those methods with the potentially lowest operating costs can be identified, however these cost rankings represent averages and the estimated cost provided the method is appropriate for the ground conditions[12]. The alternative mining method should not be any costlier than Block caving, it is however evident that in the supported methods there is a notable increase in cost due to support structure, by Howard. L. Hartman's cost estimations (Table 16). Block caving relative to Square set stopping is three times cheaper. Relative operating cost estimations (%) for Block caving is 20% and for Square set stopping is 100% [12], Top slicing is 70%, increase in costs is as a result of

- Longer production cycles to allow for supports.
- Cost for support material.
- Larger labour force required for support installation and maintenance.

Alternative mining methods applicable in collapsed blocks based on cost analysis are Block Caving and Sublevel caving, next is to determine if the deposit can be mined safely and with a high production rate.

## Top slicing

Top slicing was implemented by an American Mining Engineer W.E. Romeg in 1937. It was experimented on a stope and ore was extracted but it was not adopted due to high cost of extraction [13]. The main demerits were;

- Low tonnage output per stope.
- Large number of men required.
- High timber requirement with higher risk of fire.
- Ventilation difficulties.
- Difficulties of slicing through soft transition beds.

## Square Set Stoping

This is the least used of all supported mining methods. Small blocks of ore are systematically extracted and replaced by a prismatic skeleton of timber sets, framed into an integrated support structure and backfilled floor-by-floor [14]. However, it faces disadvantages which include;

- Very low productivity
- Very low production rate
- High mining cost
- Labour intensive requires trained labour
- High timber requirements
- Fire hazards

Square set stoping uses the same timber support and involves working within the ore body, faces the same demerits as Top slicing, hence not applicable, appropriate methods will be caving methods. However not all collapsed areas completely collapsed, only the scraper and service drifts collapsed in such a case there is no need of implementing Sublevel caving. Redesigning the current method (variant) is preferred to reduce the cost of development. The applicable method in collapsed areas will involve working in the ore body. Safety is a very cardinal issue in method selection, analysis above Square set method proves to be safe than caving but faces many demerits. Block caving is a suitable and relatively unsafe hence requires support i.e. steel arch sets, shotcreting. Access to collapsed areas when setting up supports, suitable and safest way is by employing remote control automation.

## GROUND CONTROL STRATEGY

RQD and jointing determines a means of predicting cavability the values in (Table 2) for the hanging wall is suitable for caving, therefore Block caving is applicable, and however, implementing a modification of since the current method has proved to be unsuccessful. The RMR for the Hanging wall lies under "very poor rock" and type of support recommendation for the mine drifts at Nchanga mine, using the RMR system guidelines for excavation and support in rock tunnels[15] (Table 17).

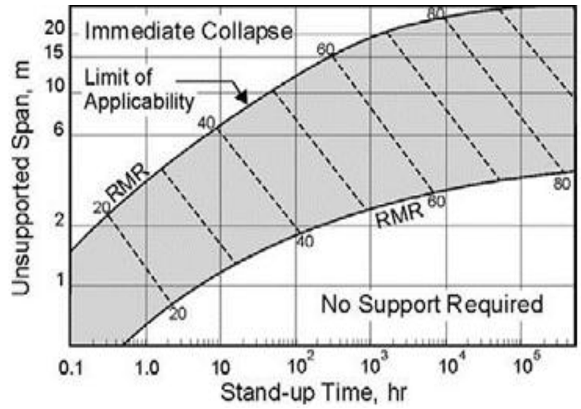


***Fig. 6.*** *Stand-up time chart (Barton and Bieniawski, 2008)*

***Table. 17.****The RMR system guidelines for excavation and support in rock tunnels*

| | SUPPORT | | |
|---|---|---|---|
| **Rock mass class** | **Rock bolts (20mm diameter, fully grouted)** | **Shotcrete** | **Steel sets** |
| Very Good rock I RMR; 81-100 | Generally, no support required except for occasional spot bolting. | Generally, no support required except for occasional spot bolting. | Generally, no support required except for occasional spot bolting. |
| Good rock II RMR; 61-80 | Systematic bolts 5-6m long spaced 1-1.5m in crown and walls with wire mesh, bolt invert. | 50mm in crown, where required. | None |
| Fair rock III RMR; 41-60 | Systematic bolts 4m long spaced 1.5-2m in crown and walls with wire mesh in crown. | 50-100mm in crown, 30mm in sides. | None |
| Poor rock IV RMR; 21-40 | Systematic bolts 4-5m long spaced 1-1.5m in crown and walls with wire mesh. | 100-150mm in crown, 100mm in sides. | Light to Medium ribs spaced 1.5m where required. |
| Very poor rock V RMR <20 | Systematic bolts 5-6m long spaced 1-1.5m in crown and walls with wire mesh, bolt invert. | 150-200mm in crown, 150mm in sides and 50mm on face. | Medium to heavy ribs spaced 0.75m with steel lagging and fore poling if required close invert. |

Categories of support in use at Nchanga underground mine, is based on the stage in life of an excavation, the support in use and type of support [2];

- Primary support; the rock left in situ as pillars, barrier pillars, boundary pillars and shaft pillars.
- Temporary support; this is mainly used to support newly exposed roof until the roof has been permanently supported during making safe. Extension of centre lines and drill holes for permanent support e.g. mechanical props and timber poles with wire

mesh.

- Permanent support; these are other methods of supporting, such as timber sets, steel sets, rock bolts, fibrecrete.

Focus is on permanent support systems being used to support scrapper drifts in the Block caving areas (Table 18).

***Table. 18.****Support required at Nchanga Mine (LOB & Block A)*

| Ground conditions | RMR | Ground control problems | Support required |
|---|---|---|---|
| **Poor quality rock mass and stress environment in the main fault zone** | 0-25 | Total closure of excavation, buckling of steel support | Use of elliptical steel sets and frequent support rehabilitation |
| **Poor quality rock mass in fissured zone at LOB** | 5-30 | Self-caving and collapse of excavations | Steel arch set support or even rail square sets with concrete |
| **Poor quality rock mass at fringe areas at LOB and Block A** | 30-50 | Blocky and wedge* failures by sliding and dislabbing | Reinforcing concrete lining, shotcrete, lacing and mesh, steel arch sets and fibrecrete |
| **High stress zones in closure areas at LOB** | 60-75 | Fracturing of rock mass and collapse of excavations | Reinforcing concrete lining, shotcrete, lacing and mesh, steel arch sets and fibrecrete |
| **Good quality rock mass areas in thick ore body areas at LOB** | 65-80 | Blocky falls of ground | Spot bolting |

Collapsed blocks occur in the poor quality rock mass in the fault and fissured zones in both LOB and Block A. Hence, the appropriate mining method will require Steel arch sets, rail square sets with concrete and occasional shotcrete, to address the ground conditions [2].

## BLOCK CAVING MODIFICATIONS

Variants of Block caving are applicable in collapsed areas with application of permanent supports. According to the RMR system guidelines for excavation and support in rock tunnels, the hangingwall requires.

- Steel sets; Medium to heavy ribs spaced 0.75m with steel lagging and fore-poling and close invert.
- Shotcrete ;150-200mm in crown, 150mm in sides and 50mm on face.
- Rock bolts; Systematic bolts 5-6m long spaced 1-1.5m in crown and walls with wire mesh and bolt invert, (Table 18).

## Undercut by conning with steel sets

Undercut by conning (UCC) this variant is already in use hence a modification of, will be implemented i.e. with steel sets supports. Starts by barring down of loose rock, clearing of debris then supporting the roof and sidewalls with steel sets. Reinforcing concrete lining, shotcrete can be done where applicable. An extensive service drift rehabilitation programme will be done. This variant requires skilled manpower, its expensive and time consuming, all scraper drifts that are badly damaged and collapsed are supported with Steel sets. In areas of adverse ground condition, this method has been implemented. Then UCC method is applied upon completion of supports and extract ore through finger crosscut.

## Under-cave

In this method, similar ground rehabilitation will be applied as in UCC, support of roof and sidewall with steel sets and shotcrete. Skilled manpower is also needed, usually less expensive support system such as rock bolts. Reinforcing concrete lining lacing can be used to support the scraper drifts (Fig. 7). The scraper drifts can be trough drives where drilling is done. Blasting is done by retreating from the boundary towards the undercut.

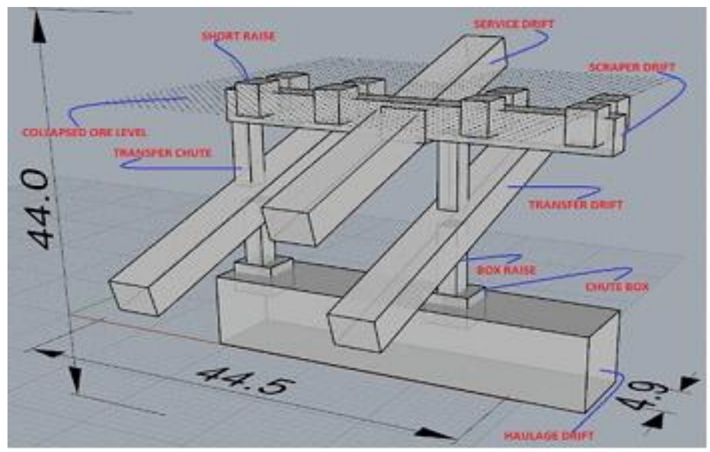


***Fig. 7.*** *Under-cave layout*

## Modified Under-cave

In this variant all support work at the service drift elevation are abandoned, new scraper drifts are mined at Transfer drift elevation and claim the ore through finger crosscuts (Fig. 8) parallel to original scraper drifts.

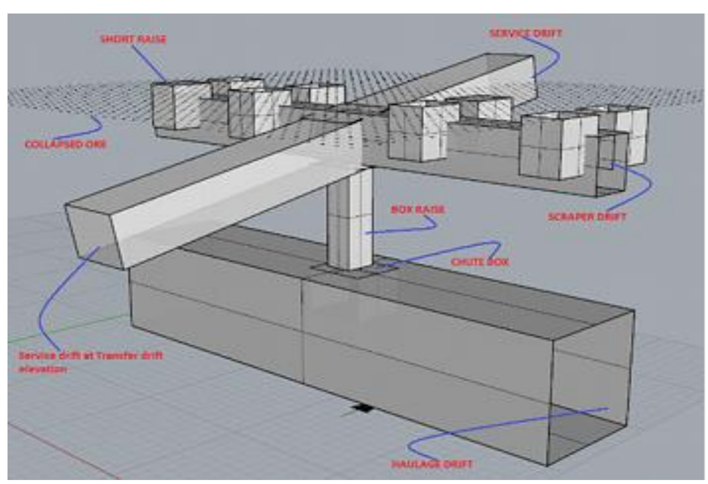


***Fig. 8.*** *Modified Under-cave layout*

This method imposes less restriction in terms of production, safety and faces moderate dilution and risk of hang-ups, due to intact raises at scraper drift elevation, new drifts are mined. Taking into account of the safety aspect and its implications in the collapsed areas UCC with steel sets is viable. It is safe as compared to other variants (Table 19), best applied in blocks with intact scraper and service drifts. The Modified under-cave variant is applicable in collapsed blocks, though not as safe as UCC but it is less restricted, the variant has to employ permanent supports.

***Table. 19.*** *Comparison of characteristics of proposed to standard mining method*

| Mining method | Support | Cost | Recovery | Dilution | Extra development | Safety |
|---|---|---|---|---|---|---|
| **UCC** | Bolting and Reinforcing concrete lining shotcrete | high | good | low | No | safe |
| **UCC with Steel sets** | Steel sets | high | good | low | No | safe |
| **Under-cave** | Bolts | low | moderate | high | Yes | Not safe |
| **Modified under-cave** | Bolting and Reinforcing concrete lining, shotcrete | moderate | | | Yes | Not safe |

## 1970 4WB
Access to service drift collapsed (blocked). Extent unknown, considered that the entire service and scraper drifts collapsed, transfer drift is still intact. The only way to extract ore is by employment of the Modified Under-cave method. New scraper drifts will be driven at the transfer drifts level.

## 1500 15WB
Unmined, full development required. Transfer drifts, service drift, Scraper drifts, raises with cones will have to be driven if UCC with steel sets method is to be employed. The Modified Under-cave method will require the driving of short raises from the scraper drifts, this block will require both variants.

## 2720 6EB & 2720 7EB
Entire service drift & scraper drifts assumed to have completely collapsed. In these blocks, driving of new scraper drifts at transfer drift elevation with short raises is required, hence the Modified Under-cave method is applicable.

## ECONOMIC ANALYSIS
Determine the economic viability of the recommended mining method in the collapsed areas (Table 20). There is excessive dilution from the hangingwall hence dilution of 100% and makes an extraction factor of 125%, metallurgical recoveries for concentrating, leaching, smelting and refinery are 80%, 77%, 94% and 99% respectively. Divisional overheads, corporate overheads, development, mining, metallurgical and realisation costs based on the 2010 budget unit costs (Table 21). Copper price based on 2010 LTEP at US$ 3308/ton of Cu.

***Table. 20.*** *Collapsed areas and their mineralization*

| Areas | % | % | Mineable | TCu | ASCu | AICu |
|---|---|---|---|---|---|---|
| **Blocks** | **TCu** | **ASCu** | **Ore** | | | |
| **1970 4WB** | 3.995 | 1.93 | 67345 | 2663 | 1308 | 1355 |
| **1970 6WB** | 2.55 | 0.65 | 478366 | 12190 | 3077 | 9112 |
| **1970 6WD** | 3.51 | 0.67 | 312466 | 10979 | 1879 | 9100 |
| **1970 7WD** | 2.60 | 0.13 | 166308 | 4323 | 213 | 4110 |
| **1500 3WB** | 3.30 | 2.08 | 204414 | 6752 | 4216 | 2536 |
| **1500 15WB** | 2.23 | 0.16 | 111312 | 2483 | 181 | 2302 |
| **2720 6EB** | 3.39 | 1.82 | 153289 | 5212 | 2787 | 2425 |
| **2720 7EB** | 4.89 | 2.02 | 105856 | 5352 | 1834 | 3518 |

***Table. 21.*** *Costs based on the 2010 budget unit costs*

| Activity | | Cost(US$) |
|---|---|---|
| **Secondary Development Cost** | secondary cost | 262/metre |
| **Capital / Primary Development Cost** | capital cost | 1100/metre |
| **Extraction Cost** | extraction cost | 3.61/tonne of ore hoisted |
| **Hoisting Cost** | Hoisting Cost | 2.22/tonne of ore hoisted |
| **Tramming Cost** | Tramming Cost | 2.45/tonne of ore hoisted |
| **Pumping / Dewatering Cost** | Pumping Cost | 1.34/tonne of ore hoisted |
| **Mining Overheads** | Overheads cost | 3.90/tonne of ore hoisted |
| **Steel sett support cost** | Support cost | 1857.50/metre of support |
| **Concentrator cost** | Concentrator cost | 2.84/tonne of ore milled |
| **TLP Costs** | TLP Costs | 8.95/tonne of ore milled |
| **Refinery cost /tonne of Refinery finished Cu** | Refinery cost | 0.04/pound of finished Cu |
| **Smelter cost/tonne of Smelter finished Cu** | Smelter cost | 0.21/pound of finished Cu |
| **Realisation Cost** | Realisation Cost | 0.071 /pound of copper sold |
| **Divisional Overheads** | Div cost | 5.20 /tonne of ore hoisted |
| **Corporate Overheads** | Corporate cost | 1.49 /tonne of ore hoisted |
| **Selling Price** | LTEP | 1.540 /pound of finished Cu sold |
| **Cathode Recovery** | Cathode Recovery | 65 % /tonne of ASCu in ore hoisted |
| **Anode Recovery** | Anode Recovery | 68 % /tonne of ASCu |
| **Rail line track laying** | | 244.88 /9 metre |
| **wire meshing service drift** | | 69.79 /metre |
| **Rock bolts** | 1.5,1.8,2.1 | 35.56,39.06,42.56 /metre |
| **Reinforcing concrete lining in scraper drifts** | | 279.35 /metre |
| **Selling Price** | | 3308.07/tonne of Cu sold |

***Table. 22.*** *Revenue raised from finished copper*

| Areas Blocks | Ore | Anode Recovery | Cathode Recovery | Finished copper | Revenue ($) |
|---|---|---|---|---|---|
| **1970 4WB** | 67345 | 942.053 | 800.200 | 1742.253 | 5,914,826.977 |
| **1970 6WB** | 478366 | 5692.742 | 2000.100 | 7962.842 | 27,033,291.190 |
| **1970 6WD** | 312466 | 5877.163 | 1221.350 | 7098.513 | 24,098,954.740 |
| **1970 7WD** | 166308 | 2613.030 | 138.450 | 2751.480 | 9,341,081.996 |
| **1500 3WB** | 204414 | 1878.228 | 2740.400 | 4618.628 | 15,679,918.760 |

| | | | | | |
|---|---|---|---|---|---|
| **1500 15WB** | 111312 | 1467.568 | 117.650 | 1585.218 | 5,381,704.145 |
| **2720 6EB** | 153289 | 1652.963 | 1811.550 | 3464.513 | 11,761,779.120 |
| **2720 7EB** | 105856 | 2344.177 | 1192.100 | 3536.277 | 12,005,412.880 |

### 2720 7EB

The service drift collapsed completely and as such the whole block has been sealed off. Scraper drifts 1B – 16B left un-mined, with a total of about 105,856 tonnes of in-situ ore at an average grade of 4.89% TCu. Costs of reclaiming the ore and fixing the block will include piping, track and trolley line laying (Figure 23). Employing the modified Under-cave method, costs will be as follows;

16 drifts * (undercaving costs per drift) = $790,400
Piping costs = $46.75/m (installations and repairs)
Truck laying = $105.66/m (installations and repairs)
Trolley line installation = $72.30/m (installations and repairs)

***Table. 23.****Cost of Undercaving (2720 7EB)*

| Reclamation work | Cost per meter($) | Distance(m) | Total costs($) |
|---|---|---|---|
| **Undercaving** | | | 790,400.00 |
| **Track laying** | 105.66 | 275 | 28,782.71 |
| **Trolley line** | 72.30 | 275 | 19,882.50 |
| **Piping** | 46.75 | 275 | 12,856.25 |
| **Loading box** | 50,000.00 /box | | 50,000.00 |
| | Total | | 901,921.46 |

About 3,500 tonnes of finished Cu, will be recovered from the employment of Modified Under-Cave Method in 2720 7EB. The revenue raised should be more than the mining and other costs/at breakeven, for the blocks to be economical to reclaim. Income is the difference between revenue and total production costs, (Table 24) revenue estimation is based on the current Block caving method.

***Table. 24.*** *Revenue raised, income and viability*

| Blocks | Revenue ($) | Mining costs ($) | Production costs ($) | Income ($) | Viable |
|---|---|---|---|---|---|
| **1970 4WB** | 5,914,826.977 | 919,770.9 | 3,218,837.532 | 2,695,989.445 | yes |
| **1970 6WB** | 27,033,291.190 | 10,045,528.92 | 25,135,112.4 | 1,898,178.79 | yes |
| **1970 6WD** | 24,098,954.740 | 6,408,202.92 | 17,741,925.63 | 6,357,029.11 | yes |
| **1970 7WD** | 9,341,081.996 | 3,301,122.96 | 8,893,862.432 | 447,219.564 | yes |
| **1500 3WB** | 15,679,918.760 | 5,708,970.68 | 12,041,799.75 | 3638119.01 | yes |
| **1500 15WB** | 5,381,704.145 | 4,473,101.44 | 8,021,194.943 | --2,639,490.79 | no |
| **2720 6EB** | 11,761,779.120 | 6,237,832.18 | 11,121,693.08 | 640,086.04 | yes |
| **2720 7EB** | 12,005,412.880 | 6,461,982.68 | 10,676,470.82 | 1,328,942.06 | yes |
| **Total** | 111,216,969.8 | 37,095,176.00 | 96,850,896.8 | 14,366,073.00 | |

The income raised and the costs incurred from the rehabilitation and reclamation of the collapsed blocks by use of proposed mining method (Table 25).

***Table. 25.*** *Revenue raised, income and Rehabilitation costs using the proposed mining methods*

| Blocks | Revenue ($) | Mining costs($) | Income ($) | viable |
|---|---|---|---|---|
| **1970 4WB** | 5,914,826.977 | 329,685.71 | 5,585,141.267 | yes |
| **1970 6WB** | 27,033,291.190 | 901,921.46 | 26,131,369.73 | yes |
| **1970 6WD** | 24,098,954.740 | 605,521.46 | 23,493,433.28 | yes |
| **1970 7WD** | 9,341,081.996 | 358,521.46 | 8,982,560.536 | yes |
| **1500 3WB** | 15,679,918.760 | 645,921.46 | 15,033,997.3 | yes |
| **1500 15WB** | 5,381,704.145 | 605,521.46 | 4,776,182.685 | yes |
| **2720 6EB** | 11,761,779.120 | 711,388.58 | 11,050,390.54 | yes |
| **2720 7EB** | 12,005,412.880 | 901,921.46 | 11,103,491.42 | yes |
| **Total** | 111,216,969.8 | 5,060,403.05 | 106,156,566.2 | |

Employing the Modified Under-cave methods incur less mining costs as compared to conventional block caving, the costs are almost less than half the costs incurred by the current block caving. Income obtained from mining = Total revenue – mining costs less the mining overheads

$$\$111{,}216{,}969.8 - \$37{,}095{,}176.00 = \$74{,}121{,}792.8$$

Contribution to the mine using the conventional block caving is the total income less the total production costs;

$$\text{Contribution to the mine} = \text{Total revenue} - \text{Total production costs}$$
$$\$111{,}216{,}969.8 - \$96{,}850{,}896.8 = \$14{,}366{,}073.00$$

Contribution to the mine (using proposed mining methods) = Total revenue – (Total Rehabilitation costs + Mining Overheads + Total metallurgical and other costs)

$$= \$111{,}216{,}969.8 - (47{,}056{,}895.59 + 6{,}237{,}488.4 + 5{,}060{,}403.05) = \$\,52{,}862{,}182.76$$

Income obtained from mining; = $111,216,969.8 – (Total Rehabilitation costs) = $106,156,566.2

Hence the Modified Under-cave method gives a higher income turnout, total net profit of about $52,862,182.76, five times more than $14,366,073.00, obtained from the current mining method.

## Discussion

During selection, there were usually two or more appropriate mining methods for the collapsed blocks. According to the UBC and Sir Nicholas techniques, applicable methods were, Longwall, Cut and Fill etc. The techniques didn't account for other factors i.e. underground water and skilled labour. Hartman technique presented a flow chart which defined the method being based on the deposit geometry and ore zone ground conditions. The system aimed at specific mining methods i.e. Block caving. The Hartman selection technique was similar to that proposed by Boshkov and Wright. This classification used the general description i.e. thickness, dip and strength of the ore body. The result of the classification provided best mining methods; Top slicing, Sublevel caving and Longwall stoping. Morrison classification system divided underground methods in three basic groups i.e. rigid pillar support, Controlled subsidence and Caving. The strain energy accumulation, ore width and support type were used as a criterion for the selection of a mining method. The area under study had a high strain accumulation, thus caving methods were applicable. Laubscher selection process is based on rock mass classification, the system is aimed at mass mining methods (Block caving) vs. Stoping methods with an emphasis on cavability, the Laubscher caving rules are based on a modified RMR, the MRMR [16]. Parameters that determined selection of caving method over stoping methods are Degree of fracturing, RQD, Joint spacing and Joint condition [17]. The selection techniques i.e. UBC and Sir Nicholas have limitations, the techniques are centred on ranking parameters that emphasised on the mineral deposit status. The techniques have limitations in the number of criteria i.e. deposit dimension, thickness changes, uniformity, subsidence effect and underground water status [6]. The limitation of the Laubscher selection process was its inability to address the influence of the fractures/veins and cemented joints on the competency of the rock mass [16], [17]. None of the selection techniques dealt with in situ stress, the techniques accounted for the vertical stress via depth, but none of them discussed how a high horizontal stress affects the selection of the mining method [6]. AHP, the decision making technique made it possible to select the best method scientifically, from the 8 alternatives that were studied, Block Caving and Sublevel caving were the most appropriate methods. Although the mining methods from the selection process were all technically feasible, their mining costs were significantly different. Based on the Hartman's relative rankings of mining costs, the methods with potentially lowest costs were Block caving and Sublevel caving. Block caving was the most appropriate on consideration of the six selection techniques and relative ranking of mining costs. The method involved working in the orebody, thus safety was the most important factor. On the other hand, collapsed areas occur in poor quality rock mass in the fissured and main fault zones in both LOB and Block A, where support and safety is very cardinal. Variants of block caving were shortlisted;

- Undercut by Conning with steel sets
- Under-cave
- Modified Under-cave

Under-cave methods proved viable than all other variants. Selecting the appropriate Under-cave method was determined by the extent of collapse. 1970 4WB, 1500 15WB, 2720 6EB and 2720 7EB Blocks collapsed up to Transfer drift elevation, the Modified Under-cave is the applicable method. It imposes less restriction in terms of production and safety, moderate dilution, recovery and risk of hang ups due to intact raises at scraper drift elevation. It is cheaper than the current methods employed.

## CONCLUSIONS

Based on conducted study it has been established that, collapsed blocks such as 1970 4WB, 1500 15WB, 2720 6EB and 2720 7EB cannot be exploited by any other, but variants of block caving with the application of permanent supports. Type of Steel sets support would include Medium to heavy ribs spaced 0.75m with steel lagging and fore-poling. Shotcrete would be 150-200mm in crown, 150mm in sides and 50mm on face and Rock bolts would be Systematic bolts 5-6m long spaced 1-1.5m in crown and walls with wire mesh. On analysis, extracting/reclaiming ore in collapsed blocks will require the employment of the Modified Under-cave, economically viable, relatively safe with moderate support cost and recovery.

## RECOMMENDATIONS

Appropriate variant to be employed in collapsed blocks is the Modified Under-cave method:

- Driving of new scraper drifts at transfer drift elevation
- Parallel to the initial scraper drifts

Supports requirement for the proposed mining method;

- Steel arch sets
- Rail square sets with concrete
- Bolting and occasional shotcrete

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