

KONKOLA DEEP MINING PROJECT: CONCEPT AND PLANNING

by

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INTRODUCTION

Konkola Division is the Northernmost mining division on the Zambian Copperbelt, 12 km from Zaire border. Greater Konkola (Fig. 1) which is an area of 23 km x 10 km contains a number of orebodies viz. Konkola, Kirila Bomwe, Saddle Lode, Kakosa and Fitwaola exploration and mining activities to date have revealed a good knowledge of the behaviour of these orebodies. The greater Konkola area can support a geological model of about a billion tonnes of ore resources at 3.1 1% total copper. Hence this area represents the potential for high output mining operations in the long term.

The Konkola orebody was exploited for a brief spell in 1957/58 by No 2 Shaft. Mining was discontinued due to economic reasons. Investigations in 1973 and 1988 both concluded that re-opening would be uneconomic at the metal prices then prevailing.

The Kirila Bomwe orebody (refer to Fig. 2) was divided into Kirila Bomwe South and North sections due to the existence of a large barren gap on upper horizons. These are being exploited by No 1 and No 3 Shafts respectively. The orebody is continuous below 700m depth. It has been explored by surface boreholes down to a depth of 1900 metres in the southern part and 1150 metres in the northern part.

The ore reserves which can be exploited by the two shafts are now limited and at the current rates and efficiency of mining, economic production will start declining by the year 2000 at No 3 Shaft and 2005 at No 1 Shaft. For continued copper production it was necessary to plan for mining at depth in this orebody. Hence, the idea of the Konkola Deep Mining Project was conceived.

A number of studies have been undertaken to assess the feasibility of mining at depth and to draw up a most suitable plan consistent with the company's corporate objectives. Each preceding study helped as a stepping stone for the following study by effecting improvements and incorporating new ideas. The latest of these studies was completed in April 1992. This represents ZCCM's latest plans for Konkola Division.

This paper attempts to put forward the philosophy and concepts which formed the basis of this plan.

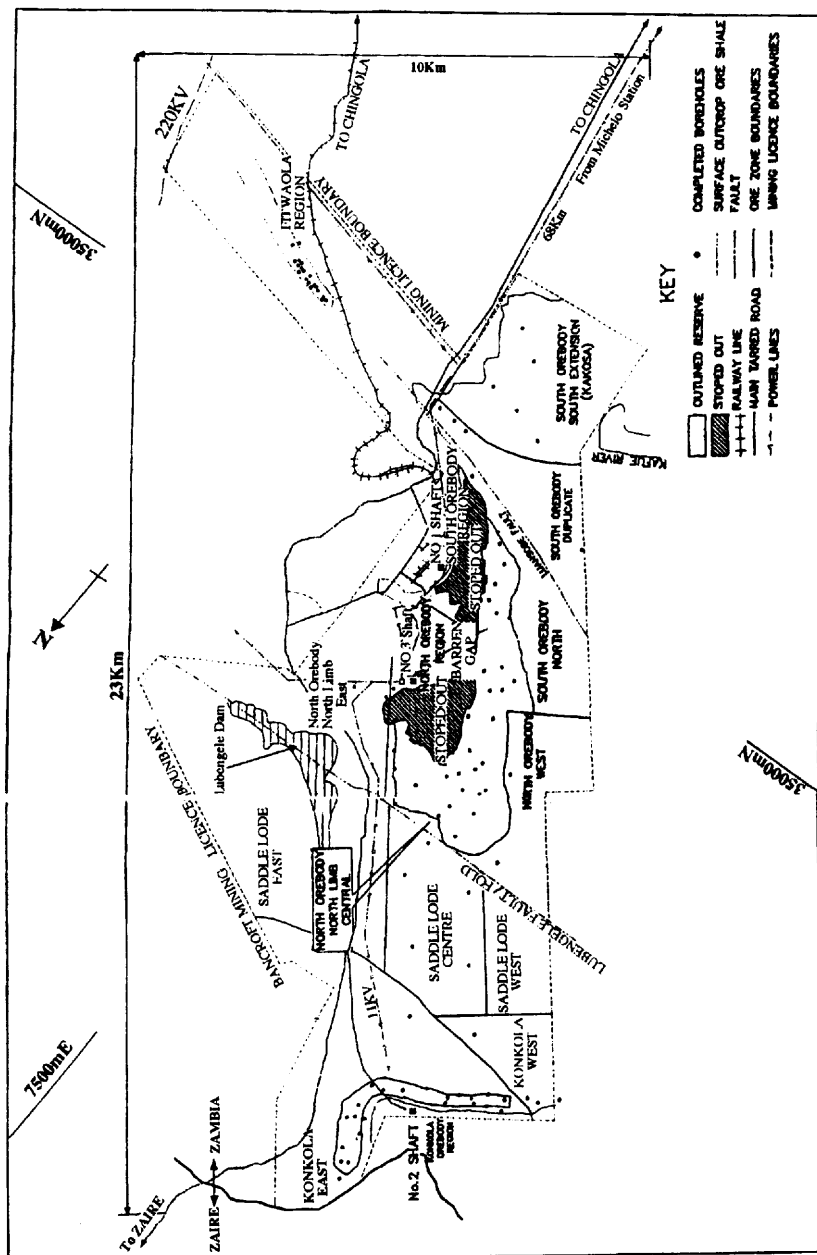


Figure 1. Map of Greater Konkola Area

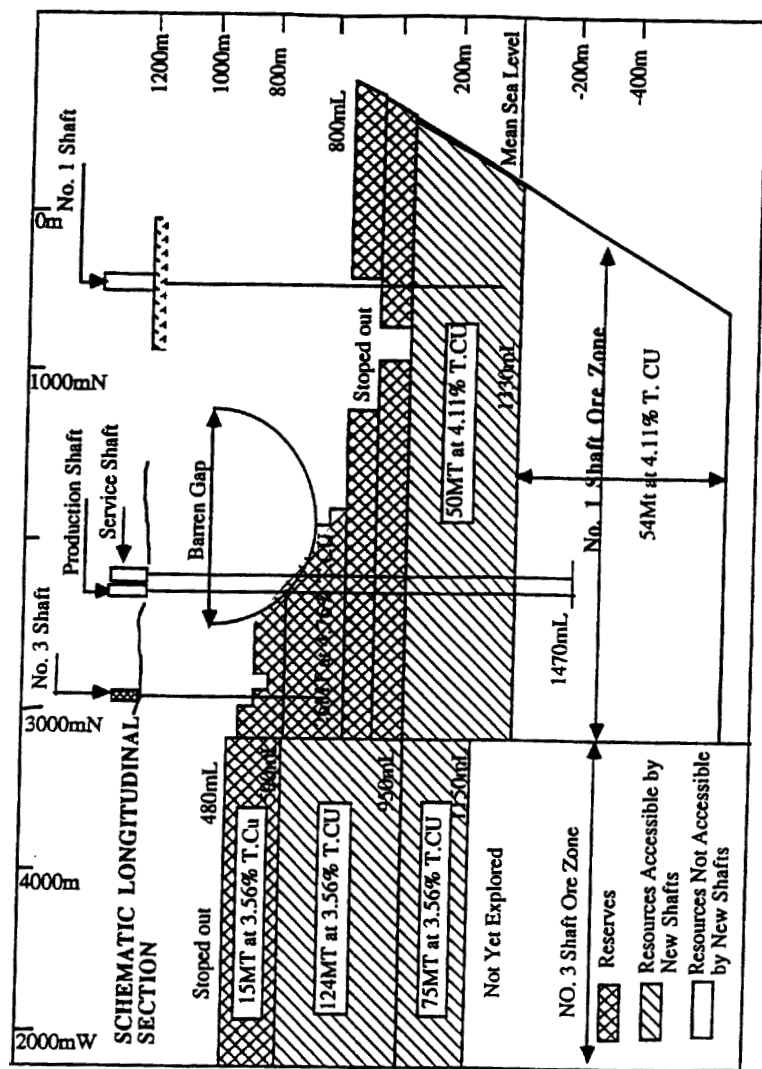


Figure 2. Konkola Mine Reserves And Resources

KIRILA BOMWE OREBODY

Because of the existence of the barren gap, the two ore zones, Kirila Bomwe South and North have been treated in the past as two separate orebodies and two separate mines. Their growth in all aspects from exploration to ore production and future planning took place as two separate entities. Because of various reasons such as the dip and thickness of reserves, rates of mining, start of mining etc., the two mines ended up in the configuration that they are today. Down-dip advance of mining in the South orebody is far more (950mL) than in the North orebody (590mL).

Initial attempts at long term planning for Konkola saw the two mines as separate, independent entities. When it was realised that below 700m level, the orebody is continuous the thinking changed. The orebody was viewed as one and planning to exploit it to its full potential started taking shape.

ZCCM's LONG TERM PRODUCTION PLAN

ZCCM's current production level is about 450 000 tonnes of primary copper per annum. At the turn of the century the Nchanga Open Pit currently producing over 30% of Zambia's finished copper will be exhausted. This coupled with a gradual decline of production from most of the current ore sources will cause a decline in ZCCM's total copper production.

In order to arrest the downfall, Kirila Bomwe orebody of Konkola Division is the most promising ore source on the copperbelt. Hence the logical step would be to maximise the production from this orebody though this alone will not compensate for the total decline in production.

This gave an indication of what is expected of the Kirila Bomwe orebody: exploitation to its fullest potential for a long term.

ASSESSMENT OF POTENTIAL

The moderate average thickness (11.5m true thickness) of this orebody has been compensated by its extensive strike length (10 km) and moderate dip of bulk of the ore reserves (67% of the reserves have a dip of less than 45 degrees Refer to Fig 3). Because of these features there is a substantial amount of ore per metre depth.

These geological features of the orebody make it amenable to a high volume operation whereby multiple and concurrent working places along strike and at depth can provide a flexible high volume production situation.

It was estimated that a 200 metre long panel with the application of fully mechanised cut and fill or bench and fill mining methods with primary and secondary phase mining could produce a modest amount of 600 tonnes of ore per day consistently including the filling time by alternating half the panel under mining while the other half is under filling. If approximately 10 panels can be kept in production on one main level and three main levels in production at any time, then this configuration could yield around 6 million tonnes of ore per year.

An earlier detailed study completed by Consultants in April 1990 had estimated the potential of North Orebody to be 3.2 million tonnes per year by applying semi-mechanised delayed filling methods where a lot of waiting time was involved for curing of the fill before mining next to it. The production from the South Orebody was maintained at the current level of 1.1 million tonnes per year. No attempt was made to change the mining method or improve the production rate. This had put the total production potential of the orebody at 4.3 million tonnes per year.

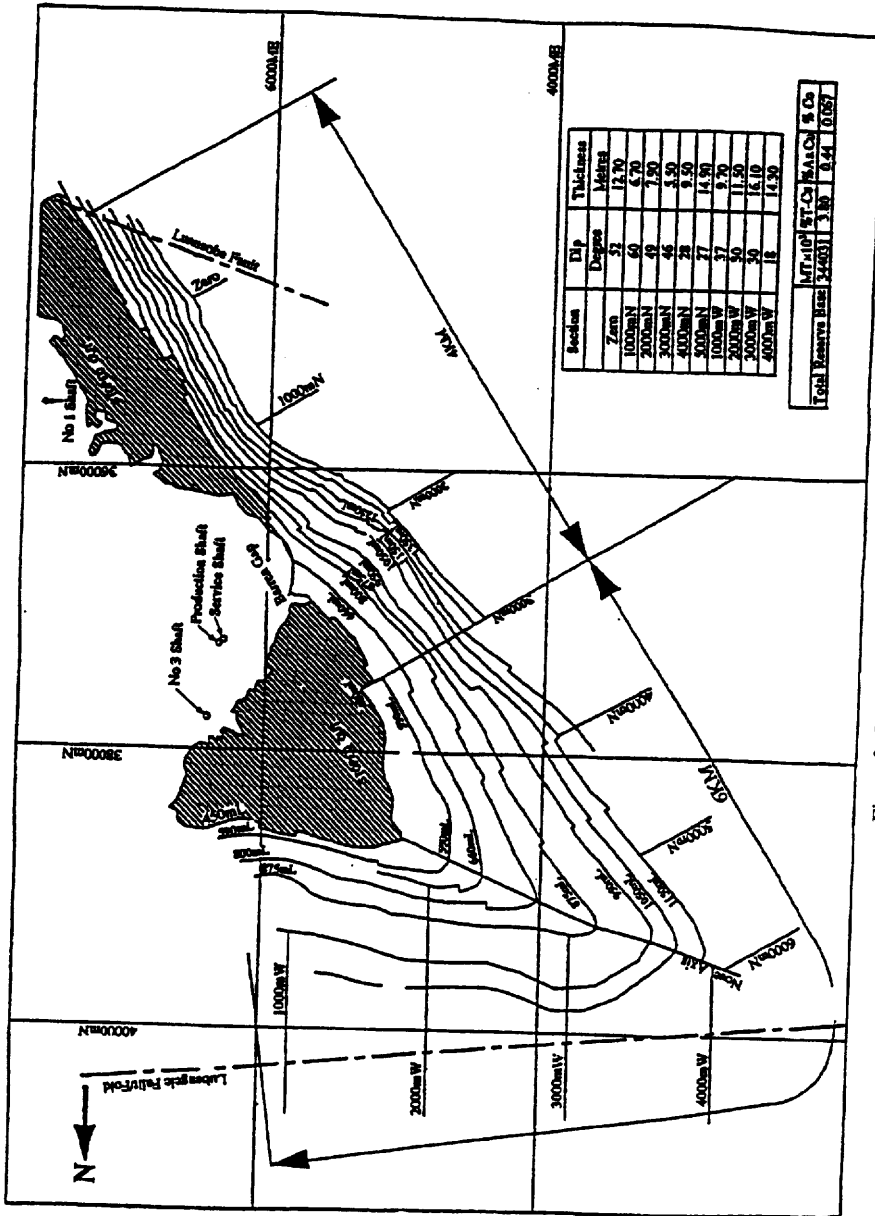


Figure 3. Composite Orebody Plan

In light of this it appeared reasonable to effect about 40% increase in production by affecting the improvements in the relevant areas.

Two main constraints to the build up of production and sustaining it at the desired level are:

- (1) Timely development of new areas at the required rate
- (2) Timely dewatering of the footwall and hangingwall aquifers for mining the extraction development and for stoping respectively

These were checked by detailed development scheduling and by predicted draw down rates of the aquifers using computerised hydrology model.

Detailed scheduling indicated that a basic production plan of 6 million tonnes per year is readily sustainable over a long term, based on currently identified resources. Production expansion potential exists in several areas of additional sources of future ore reserves such as:

- (1) Reserves outside of current planning
- (2) Additional unexplored reserves at depth
- (3) Adjacent reserves like the saddle lode
- (4) Additional general Konkola area reserves

Hence the ultimate production potential is estimated to be 9 million tonnes per year.

MINE PLANNING CRITERIA

The aim of the latest study was to prepare a General Mine Plan to achieve the following objectives:

- (1) To optimise economics
- (2) To provide maximum utilisation of the resource
- (3) To design a mine that would be competitive with world copper producers well into the 21st century

During the evaluation process, the following criteria for mine planning were developed after reviewing the previous studies:

- (1) Large identified ore resources and potential for additional resources suggested the inherent possibility for high volume, low cost-mining operations
- (2) Economics should be improved by optimising costs, providing quick production start-up and build-up to a high volume mine
- (3) Contemporary methodology, technology and structural organisation should provide effective and efficient mining results
- (4) Electrification should be utilised wherever the potential exists, and fossil fuelled equipment should be used only where essential
- (5) Provide an economically viable stand alone project

The following concepts were developed for the various aspects of mining and processing after a careful evaluation of the alternatives:

The ore resources are capable of sustaining a production rate of 9 million tonnes of ore per year. However, this rate can only be achieved in later years due to the necessity to develop a suitable mining echelon, along with adequate dewatering of the aquifers, whereas 6 million tonnes a year is achievable in year 6 of the project.

Because of this, it was decided that the project will be evaluated on a production rate of 6 million tonnes a year.

The basic infrastructure will be designed for the incremental expansion.

Two new vertical shafts should be developed from surface down to 1470m level and positioned between No 1 and No 3 shafts.

The new shafts will be capable of exploiting the majority of the 344 million tonnes of drilled ore reserves and resources and a large portion of the potential resources below the currently drilled horizons.

In order to fully exploit the resources by recovering 92% of the insitu copper at a dilution not exceeding 10% of the mined ore, backfill stoping will be practised. This method also reduces the environmental impact of a large tonnage mining operation.

The mine will be totally trackless; Mobile equipment will be diesel powered, drilling will use electro-hydraulic equipment and the movement of ore and waste from the internal passes to the shaft hoisting systems will be by conveyors.

A new concentrator will be built adjacent to the new shaft complex. This concentrator will utilise Semi-Autogenous Grinding (SAG) mills, large volume cells with differential location to produce copper and cobalt concentrates, and will be highly automated.

MINING INFRASTRUCTURE

Mining infrastructure consists of all excavations and installations for the movement of men, material, mine air and mine water as well as services and utilities.

The infrastructure planning was based on the following criteria:

- (1) It should be designed to cater for the needs of a 6 million tonnes per year operation with potential expansion to 9 million tonnes per year
- (2) It should be amenable to the extension of mining operations at depth and laterally as the orebody is open ended in these directions
- (3) It should be of extendable nature so that it can be progressively mined as and when necessary to defer the expenditure
- (4) It should facilitate optimum utilisation of men, materials and equipment to provide optimum operating cost
- (5) It should provide optimum systems flexibility to avoid dependence on any one item

This philosophy was used to design the shafts, material handling mine rock dewatering, drainage and pumping and ventilation infrastructure.

NEW SHAFTS COMPLEX

Various options for hoisting facilities had been considered to mine at depth. These included deepening of the existing shafts, conveyor systems -internal and from surface, sub-inclines, sub- vertical shafts, converting existing ventilation shafts into production shafts, a new shaft from surface etc. All were evaluated and sub-vertical shafts option was chosen for the initial study completed by a Consultant. This had some obvious disadvantages such as:

- (1) Maintaining and operating a large number of hoisting, loading, unloading and transfer facilities and shafts
- (2) Limitation on capacity dictated by the existing shafts from surface
- (3) Possibilities of delays in the project work due to dependence on the existing operation

In addition, the overall economics of the Consultants study were marginal. To improve this the concept of a new surface shaft to exploit both ore zones at depth with increased production rate was revived for consideration. The concept was that of a very large 9 metre diameter shaft incorporating service and production facilities in a more central location from surface to 1 550m level and was fondly referred to as the "Supershaft" .

Investigations revealed that this concept was technically and economically more viable than the sub-vertical shafts.

The latest study analysed and developed the concept further and it took a form of two smaller sized shafts, one totally dedicated to production and the other for servicing with some production capability in it to provide improved efficiency and flexibility. This concept is termed the New Shafts Complex. (Refer to Fig. 4)

The New Shafts Complex was selected despite a +5% capital cost differential over the "Supershaft" because of the following operating advantages:

- (1) It promotes early off-shaft development and production start-up
- (2) It provides for maintaining production rates at depth
- (3) It provides for future production expansion
- (4) It provides for future deepening
- (5) It provides flexibility of hoisting facilities

MINING METHODS

To optimise the economics the aims of a mining method are:

- (1) To maximise metal recovery
- (2) To minimise dilution
- (3) To maximise productivity

These will result in lowering the cost per unit of metal mined from insitu.

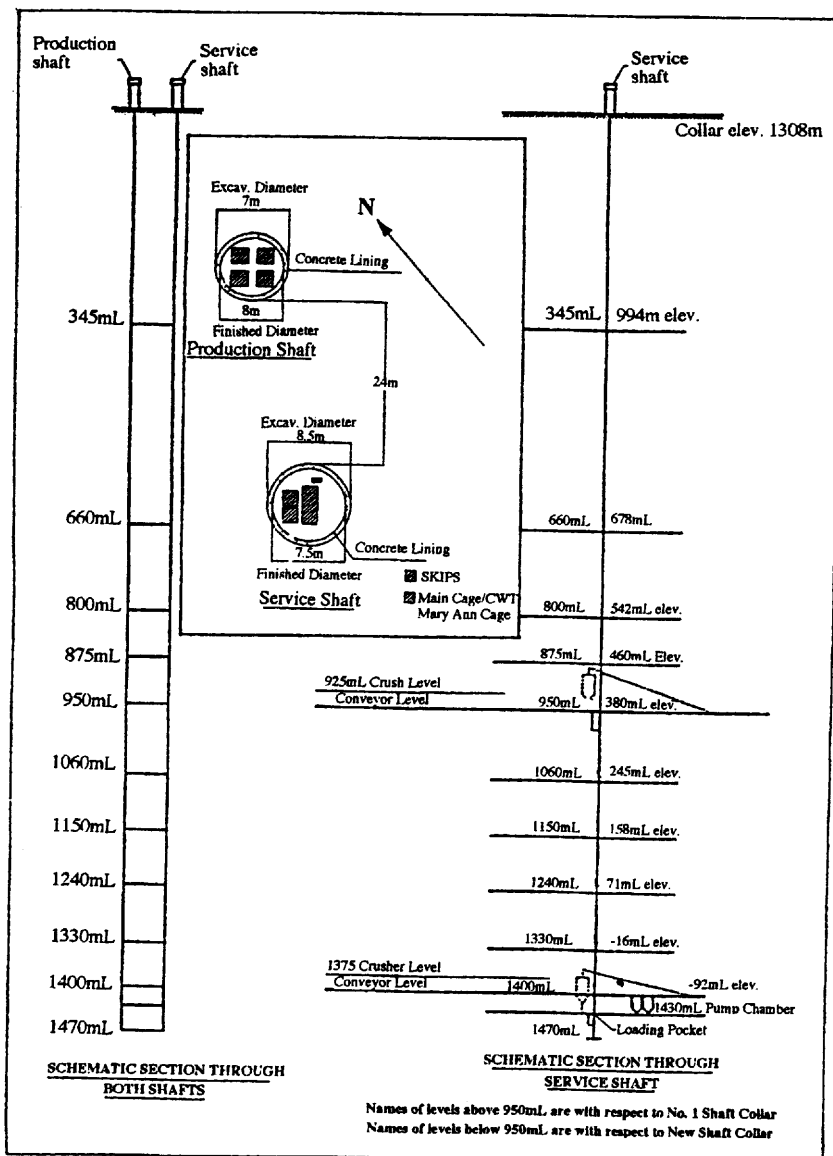


Figure 4. New shaft complex station

The currently practised open stoping methods allow limited amount of mining control resulting in high dilution and low recovery. Mining technology and equipment are far behind the contemporary ones resulting in low productivity. Therefore economics are adversely affected.

The Konkola Deep Mining Project has advocated the use of in-the-orebody cut and fill mining methods with contemporary technology and methodology involving high degree of mechanisation to improve upon the aforesaid drawbacks. Cut and Fill mining offers optimum potential for mining control thus improving recovery and minimising dilution.

It is intended to achieve standardisation of skills, materials, equipment, procedures and practices for efficient operation of the method.

The intended methods are variations of longitudinal cut and fill concept with variances to accommodate gravity or non-gravity dips and single or multi pass mining as dictated by the longitudinal orebody geometry with varying dips and horizontal width over its strike length.

Mining is planned in primary and secondary phases in order to achieve the maximum utilisation of the orebody.

As much flexibility as possible is incorporated in every aspect of design and planning to deal with adverse mining conditions without sacrificing reserves or mining rates.

Concurrent multiple working places and phasing and sequencing of mining activities will promote maximum utilisation of ore reserves without long term tie-up and increase the productivity.

MINE ROCK DEWATERING

Mine Rock Dewatering at Konkola has been and continues to be the key element of mine planning, development and subsequent production. Considerable expertise has been developed and exercised over the years in successfully handling dewatering. Arising out of this experience has come specific reason, ways and means of dewatering the various aquifers or aquifer influenced areas.

The deep mining plan is a significant variation from the traditional mining practice. The substantial increase in production rates in a relatively short time over the current rates will mean a corresponding increase in area dewatering demands, but not increased inflow rates.

The plan of up-dip mining compared with traditional down-dip advance will demand earlier area dewatering of aquifers for stoping.

The infrastructure development to send the mine broken rock to one common lower level which is located in the footwall, will necessitate earlier area dewatering of the footwall aquifer.

For the earliest and the most effective dewatering of the Hangingwall and Footwall Aquifers, the dewatering development will be based on the following policies:

- (1) Development will be as close to these aquifers as possible for shortest intersection by mining or drilling.
- (2) Development will not be in the Aquifer as that will retard the progress of mining, and inflow control.

- (3) Development will be in a reasonably competent ground to reduce ground support problems.

Mine Rock Dewatering programmes will follow time honoured procedures at this time, because they have been successful to date. Included are such procedures as:

- (1) Using cementation grout cover to cross the Lower Porous Conglomerate and Footwall Quartzite Aquifer for expediting development.
- (2) Using a system of drain drives and periodic crosscuts along the drives to intersect the footwall aquifers and provide diamond drilling access to Footwall and Hangingwall aquifers.
- (3) Using a series of 91 mm diamond drill holes, 300 to 400 metres in length, to dewater the hangingwall aquifers.
- (4) The use of diamond-drill holes to drain down aquifers in a controlled manner and regulate pressures and/or inflows.

The following revisions and/or improvements to current practices to accelerate draw down rates of mine rock dewatering will be implemented. As a general statement they will include:

- (1) Providing additional draindown crosscuts along strike.
- (2) Allowing provision for delineation and dewatering long hole drilling along the strike of the Footwall.
- (3) Allowing provision for dewatering of the Lower Porous Conglomerate in the vicinity of shaft access crosscut areas.
- (4) Allowing provision for the dewatering of known fissures from appropriate development.

The efficiency of Mine Rock Dewatering programmes will be improved by:

- (1) The mechanisation of access and drainage drifting to provide high speed drifting results, regardless whether track, trackless, or tunnel boring applications are employed.
- (2) The application of high speed longhole drilling equipment to reduce programme schedule timing in coring and non-coring situation.
- (3) The mechanisation of cementation cover grouting techniques to provide effective and efficient results.
- (4) The introduction of a range of grouting materials necessary to deal with various conditions in an expeditious manner.

Priority will be given to the dewatering development and drilling programmes. Dewatering programmes will be concentrating on only three levels, 590, 950 and 1400 metre levels. On each level, a multi-heading development programme will be followed which will consist of two parallel drives and regular dewatering crosscuts. This will provide equipment utilisation flexibility while any dewatering challenges are being dealt with in a particular heading.

The intended filling methods may substantially impact the requirement of dewatering the hangingwall aquifer. This will reduce the constraint on production and lower the cost. To be on the conservative side, this aspect has not been included in the current planning.

ROCK HANDLING

Current mining operations use the traditional rock handling arrangement consisting of multi-level track bound tramming and a central ore pass waste pass - crusher system near the min hoisting shaft.

Extensive strike length of the orebody and resulting long distances from the orebody to the shafts indicated the benefits to be gained by adopting a system consisting of multiple rock passes and crushers located at regular intervals along the strike loading the crushed rock on to a common conveyer system for transporting to the shaft loading system on one level serving for a reasonably large vertical portion of the mine.

This reduces the distances over which uncrushed rock is to be transported, provides flexibility due to the multiple passes and crushers and provides the cheapest mode of bulk transport over long distance viz conveying. It minimises the use of fossil fuel and promotes the use of electric power. The system is expandable to follow the advance of mining.

Adequate surge capacities will be provided at every stage along the route Adequate fragmentation will be ensured by following the correct mining procedures and practices and by secondary breaking of any big rocks within the stope to achieve efficient operation and effective utilisation of the rock handling system.

Two identical systems will eventually exist on 950 metre level and 1400 metre level to serve the two blocks. Programmable logic system will control the activities throughout.

EQUIPMENT

Mine planning is based on the use of contemporary large scale mechanised equipment so that the equipment efficiency is exploited to attain the production targets effectively with a minimum of resources.

The selection of underground mobile mining equipment is based on providing manoeuvrability, mobility and flexibility which are inherent requirements of the mining system.

Equipment sizing will be complementary to fleet and production requirements. The equipment will constitute a complementary homogeneous system with high productivity.

The following policy has been used in selecting the equipment:

- (1) Electrification where possible
- (2) Standardisation where possible
- (3) Equipment must have a proven record of working satisfactorily in similar situation

The following equipment is selected for direct mining activities:

- (1) Electric Hydraulic Development jumbos
- (2) Electric Hydraulic Rockbolt jumbos

- (3) Electric Hydraulic Longhole jumbos
- (4) 4.3 m³ Load-Haul-Dump units
- (5) 30 tonne Underground Dump Trucks
- (6) Secondary rock breaking machines

The following equipment is selected for indirect or supporting activities:

- (1) Scissorlift vehicles
- (2) Wheelloaders and Graders
- (3) Boom Trucks
- (4) Forklifts
- (5) Service Tractors 2.3 m³ LHDs

Apart from these there will be major mining equipment such as:

- (1) Raise borers
- (2) Raise climbers
- (3) Diamond Drilling machines
- (4) In-The-Hole Drilling machines

and minor mining equipment for various underground tasks related to mining, material handling, construction/installation, water control, ventilation, ground support, safety, sanitary etc.

BACKFILL

The selected backfill system will lead to optimum in-situ metal recovery because of the inherent selectivity of attacking and mining the orebody as well as dilution control. Additionally, the backfill can provide some geotechnical and hydrology related benefits.

The following items are considered to be important criteria for selection:

- (1) The property should be self-sufficient for backfill material
- (2) The system must be based on proven technology
- (3) The system should be amenable to pumpable distribution in view of the orebody geometry
- (4) The introduction of run-off water into the underground environment should be minimised
- (5) The requirement for stope fill preparation work should be minimised

Konkola ore needs a fine grind (85% to 90% passing 75 microns) to achieve optimum flotation response and mill throughput. When the tailings are deslimed to achieve optimum percolation rate for use as classified hydraulic tailings fill the sand recovered does not satisfy the quantity requirement of the mining operation. Therefore several other proven backfill concepts such as Cemented Rockfill, Cemented High Density tailings backfill. Cemented Paste Fill using tailings and crushed rock and their combinations were evaluated.

Materials for the above systems were sourced both locally and at some distance from the site. The logistics of supplying material to the site were considered.

Consideration was also given to the characteristics and placement procedures of the various options relative to the geometry, geology, geotechnical and hydrology conditions at Konkola. The risks associated with any system and/or its combination were considered.

The final choice was the High Density Backfill system modelled after the Preussag High Density Backfill concept. In this system, coarse crushed rock aggregate will be mined with dewatered, unclassified mill tailings to form a high density paste that is pumpable. The paste will be pumped by multi-cylinder positive displacement pumps and sized pipelines within close proximity of placement in the stopes to be filled, and a cement binder will be pneumatically injected prior to placement to obtain a high strength backfill. Most of the water in the mixture will be consumed in the cement hydration process or will remain as residual water in fill.

SCHEDULING

Important concepts in scheduling are:

Take advantage of the existing mine workings to promote the critical activities for early start and quick build-up of production. These include:

- (1) Extending dewatering development Northwards on 950 m level using No 1 Shaft
- (2) Mining of access development towards the New Shafts on 660, 800 and 950 m levels and infrastructure development near the shafts on 950 m level using No 1 Shaft
- (3) Mining towards the orebody and in the orebody in readiness for production from the first level viz. 660 m level soon after the 950 m level loading pocket is commissioned in the Service Shaft

Promote parallel activities to enhance the schedule. Examples are:

950 m level loading pocket in the Service Shaft is commissioned and development and production on upper level commenced while the shaft is being sunk below 950 m level

Service Shaft sinking is taking place while infrastructure development is being mined from existing workings to meet the Service Shaft

Mine or construct the facilities to suit the rate of production build up to promote economics by deferring capital expenditure. Examples are:

Production Shaft is constructed using the Service Shaft by raise boring and slyping which is cheaper than sinking after the Service Shaft has provided the initial production capability but before the production increases beyond the ability of Service Shaft

Concentrator and Backfill Plant is constructed in two stages to suit the production build-up from 2 million tpy to 6 million tpy over a period of two years

Schedule and evaluate Konkola Deep Mining as a stand alone Project irrespective of the on-going and planned activities of No 1 and No 3 shafts. After the Deep Mining Project comes on line, the decision to phase out the operations of No 1 and No 3 Shafts will be based on the economics.

MODERNISATION

In order to survive and prosper any industry must change with time. By adopting contemporary methodology and technology it endeavours to remain competitive.

Zambian Copperbelt by and large has remained indifferent to the process of modernisation which took place in the mining industry worldwide in the last three decades. There are various socio-economic and political reasons responsible for this. This paper is not meant to go into the details of these reasons.

It should suffice to say that the process of modernisation is long overdue on the Copperbelt in general and at Konkola in particular. Mining and processing technology of mid fifties can no longer sustain viability of operations well into the 21st century.

The benefits of mechanisation, use of electro-hydraulic drilling, cut and fill mining, conveying, semi-autogenous grinding translated finally into lowering of the cost per pound of copper have not been fully realised.

If a mine is to be built to exploit a deposit of exceptional quality and quantity such as the one at Konkola Division to run well into the twenty first century, there is no option but to go for the contemporary methodology and technology.

However, care has been taken throughout the planning not to go beyond the leading edge of technology. All practices procedures and equipment advocated are tried and true.

CONCLUSION

It is the hope of the authors that this paper would at least partially fulfil its purpose of making the reader aware of the conceptual planning behind the Konkola Deep Mining Project which has generated great interest in the mining and financial circles worldwide.

The authors wish to thank the ZCCM Management for their permission to publish the paper.