Determining A Suitable Method For Dewatering Of Ore Reserves Below 1040mL At Konkola Mine, Zambia.

By

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DECLARATION

I, Chella Makumba, do hereby declare that I am the author of this work. To the best of my knowledge and belief this dissertation contains no material previously published by any other person except where due acknowledgement has been made. This dissertation contains no material that has been previously submitted for any other degree or diploma at this University or any other institution for academic purposes.

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APPROVAL

This dissertation by Chella Makumba has been approved as fulfilling the requirements for the award of the degree of Masters of Mineral Science-Mining engineering by the University of Zambia.

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ABSTRACT

Konkola Mine is one of the wettest mines in the world. It pumps about 350,000m^3 of water per day. From the inception of mining operations in 1956, water has always posed a challenge to mining despite the use of dewatering strategy involving dewatering crosscuts and dewatering boreholes.

The groundwater at the mine was mathematically modeled using MODFLOW software in 1989 and predictions of dewatering operations were made for periods of 1989 to 2020. However, in the last two decades, the implementation of the dewatering plan has lagged behind as a result of financial constraints faced by the Mine. The Footwall Aquifer at shaft No. 3 is behind by almost 3 years and the Hangingwall Aquifer at shaft No. 1 is behind by at least 13years. This has resulted in most of the reserves being underwater which poses a safety and mining challenge. The reserves below 1040mL cannot be mined until the Hangingwall Aquifer is dewatered below 1150mL Cave line. The dewatered reserves have almost 2 years to depletion hence this study was undertaken to determine a suitable dewatering method for reserves below 1040mL. The study also sought to establish whether the use of backfilling mining methods would reduce dewatering requirement for the mine.

Drawdown simulations were done using the MODFLOW-VKD software. The water table was generated using MicroStation and Geovia Surpac software. The seepage points identified are the Kafue River, Kakosa Stream, discharge canals, Lubengele Dam, and Lubengele Stream. The study has established that more than 194,170m^3/d of water could be excluded from seeping through the mine. An analysis of mines that use backfilling methods established that the use of backfill reduces hydraulic inflow paths into a Mine and also reduces the dewatering requirement.

The results of the study indicate that the existing conventional dewatering approach (using crosscut and dewatering boreholes) coupled with surface water exclusion methods are the most viable for the reserves below 1040mL as opposed to deep surface wells. It was also found that dewatering requirement could be reduced in the mine by the application of backfill. The approximate cost of Conventional dewatering method (19 crosscuts and 190 boreholes) implementation was less than the cost for the Surface deep holes.

Key Word: Mine operations, dewatering, Hangingwall Aquifer, backfilling

DEDICATION

I dedicate this report to my beloved wife Chansa, daughter Isabella, mother Getrude, friends and family.

ACKNOWLEDGEMENTS

First and foremost, I would like to express my sincere and profound gratitude to Dr. Victor Mutambo, my principal supervisor, for his guidance, continued encouragement and constructive criticism during my research.

Special thanks go to the management of Konkola Copper Mines, for their support and for granting me permission to carry out the study on the mine site. I would also like to thank Mr. Mumba Mwansa (Head Hydrogeology-Konkola Mine), Bronislaw Smolski (Manager Hydrology) and Mr. Kasanda Alam (Manager Mining Projects-Konkola Mine) for their invaluable help and suggestions on many technical aspects of the project.

Finally, I would like to profoundly thank all those dear to my heart; my wife Chansa, daughter Isabella, mother Getrude, rest of the family, friends, and well-wishers for their support, prayers and love that kept me strong and determined during the study.

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ACRONYMS

KCM	Konkola Copper Mines
KDMP	Konkola Deep Mining Project
HW	Hanging wall
FW	Foot wall
HWA	Hanging wall Aquifer
FWA	Foot wall Aquifer
LPC	Lower Porous Conglomerate
FWQ	Foot wall Quartzite
LPCA	Lower Porous Conglomerate Aquifer
FWQA	Foot wall Quartzite Aquifer
OSU	Ore Shale Unit
URD	Upper Roan Dolomite
GFW	Geological Foot wall
AGSST	Argillaceous Sandstone
PPCF	Post Pillar Cut and Fill
SLOS	Sublevel Open Stoping
No	Number
mL	Meter Level
ZCCM	Zambia Consolidated Copper Mines

CHAPTER 1: INTRODUCTION

1.1 Introduction

This chapter looks at the background and problem statement. It also covers the objectives of the study and research questions. It further discusses the significance and gives a summary of the methodology used.

1.2 Background

Production at Konkola Mine (then called Bancroft Mine) started at No 1 shaft in January 1957. No. 3 shaft started operating in 1963. While No. 4 shaft commenced operation in 2010.

Konkola Mine is one of the wettest underground mines in the world; it pumps about 350,000m^3/day of water to the surface. Since the mine started production in 1957, water has always posed a challenge to mining, as a result, the mine has always had a robust dewatering strategy to dewater the footwall and hanging wall aquifer. However, in the last two decades, the implementation of the dewatering plan has lagged behind by almost 15 years due to the financial constraints faced by the mine in the last two decades. This has led the mine to lag behind in dewatering development. As a result, most of the reserves are under the water table and poses a safety and mining challenge. No mining can take place below 1040ml before dewatering the Hangingwall Aquifer below the 1150mL Cave line.

This study was therefore, undertaken in order to determine a suitable method for dewatering of reserves below 1040mL at Konkola Mine so as to make it possible to mine the deep orebodies safely. The study has reviewed the use of conventional dewatering method currently used at Konkola Mine and also looked at other dewatering methods used on the Copperbelt and globally. The study has suggested ways of improving the current dewatering method at Konkola Mine by excluding the surface water from recharging the Hangingwall Aquifer.

1.3 Problem statement

Konkola Mine is currently mining the upper orebody which is nearing depletion with about 2 years remaining. This being the case, the mine has invested over US$400,000,000 into the Konkola Deep Mining Project (KDMP) to expose the deep orebody (below 1040mL) and increase the production of copper ore from 2 to 6mt/y at 3.2% TCu. The mine has about 299mt reserves. However, most of the reserves are below the water table. Extraction of reserves below the water table possess a great danger to mining operations in terms of safety and flooding. Only Konkola Flats and Konkola Extension are mining above the water table as shown in Figure 1.1.

In terms of the Footwall Aquifer at No.3 shaft, the dewatering schedule is behind by 3 years while the dewatering schedule for the Hangingwall Aquifer at No. 1 shaft is behind by at least 13 years. Mining bellow 1040mL cannot take place until the Hangingwall aquifer is dewatered below 1150mL cave line.

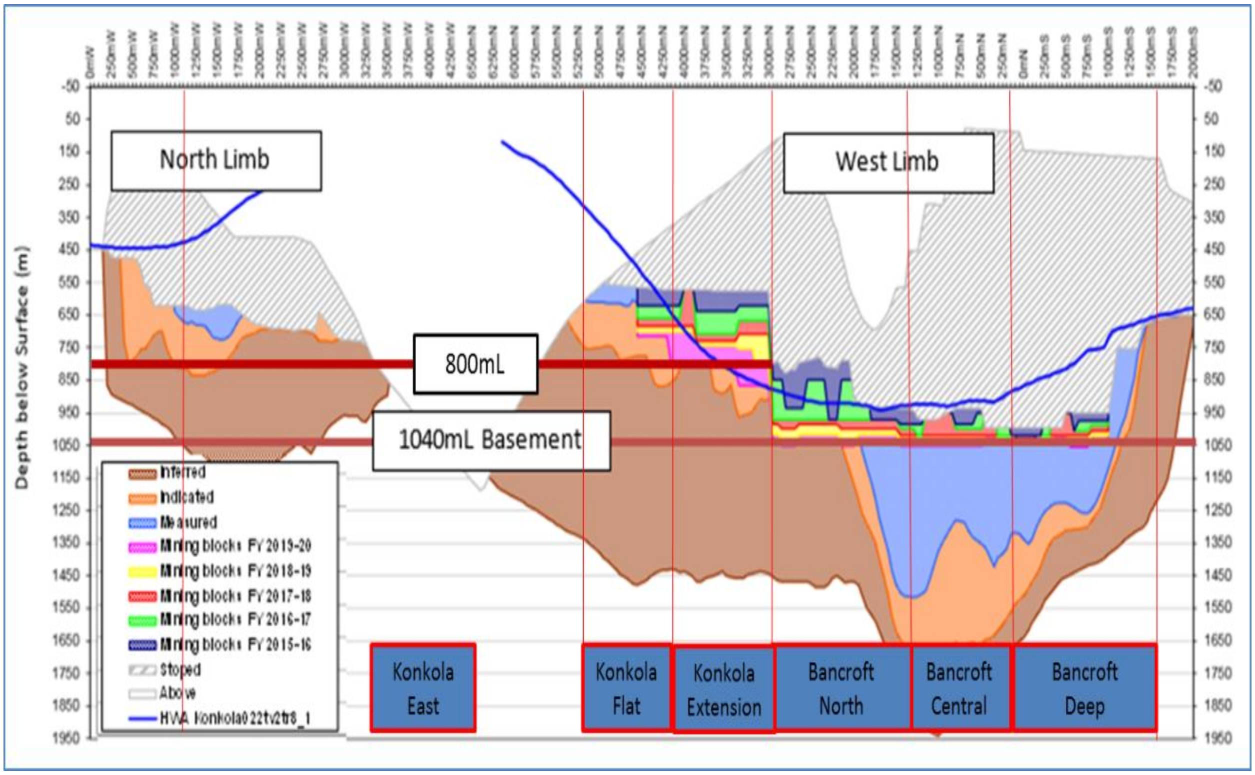

Figure 1.1: Konkola Mine Dewatered and Un-dewatered Reserves (Konkola Copper Mines, 2015)

Therefore, this study was undertaken in order to establish a suitable method for dewatering the ore reserves below 1040mL and to allow efficient mining operations to continue. The study has compared different dewatering approaches and selected a workable approach for Konkola mine based on cost and viability.

1.4 Objectives

Main Objective

- To determine a suitable method for dewatering of ore reserves below 1040mL at Konkola Mine, Zambia.

Sub-Objectives

- To establish a cost-effective technique of dewatering the aquifers
- To establish whether backfilling mining methods can be used to reduce the dewatering requirement for Konkola Mine

1.5 Research Questions

In order to achieve the mentioned objections, the following research questions needed to be answered.

a) What dewatering approach is best suitable for Konkola Mine?
b) Which dewatering method is cost-effective for Konkola Mine?
c) How fast can the water table be lowered by the cost-effective dewatering approach?
d) What impacts can backfilling methods have on future dewatering plans?
e) Can the sinking of surface boreholes solve the dewatering problems for Konkola Mine?
f) How can Konkola Mine avoid getting the dewatering process wrong again?

1.6 Methodology

In order to achieve the research objectives, the following research methods were used:

- ❑ **Main Objective 1**: To determine a suitable method for dewatering of ore reserves below 1040mL at Konkola Mine, Zambia

Methodology: The following were done:

- ✓ Sources of water into the mine were established
- ✓ Water profile were generated using Microstation and Surpac Geovia Software
- ✓ Drawdown simulation were run in MODFLOW-VKD software

- **Sub-objective 1**: To establish a cost effective technique of dewatering the aquifers

Methodology: Different potential dewatering methods (surface boreholes and Breakthrough methods) were reviewed and simulated using the Modflow VKD and the level of drawdown was then observed and each method evaluated in terms of cost

- **Sub-0bjective 2**: To establish whether backfilling mining methods can be used to reduce the dewatering requirement for Konkola Mine

Methodology: An analysis of mines that use backfilling methods was conducted to establish if the use of backfill would reduce hydraulic inflow paths into a Mine and the effects it would have on dewatering requirement

1.7 Research Significance

The outcomes of the research are very vital to Konkola Mine and are as follows:

1. The fastest way of dewatering the aquifers will be suggested;
2. the study will close data gaps and help in mine planning and design, geotechnical support design, quicken mine development, production, tramming and hoisting activities;
3. The ore will be mined safely;
4. It will reduce the amount of support installed as the unsupported spun will be increased;
5. The wear of tires will be reduced as most drives will become dry;
6. The life of installed support and other iron works will be increased;
7. A dry and pleasant working environment will be provided;
8. More efficient working conditions will be achieved: better trafficking and mining ability, reduced downtime due to mine flooding;
9. The haulage cost will be lowered: Dry ore and waste rock weigh less than wet material, so dewatering of rock provides a haulage cost saving;
10. Improved slope stability and safety: lowering of groundwater levels and reduction in pore water pressures can allow steeper slope angles to be used while maintaining or increasing factors of safety; and,
11. It will enable the mining of deep ores safely.

1.8 Organisation of Dissertation

- Chapter 1 looks at the introduction of the study. It discusses the background, problem statement and the aims and objectives of the study. Besides this, it also looks at the summary of the methodology and significance of the study.
- Chapter 2 looks at the study area location, characteristic and reviews past and present dewatering plans for Konkola Mine. This chapter also reviews regional and global dewatering approaches. It also reviews backfilling mining methods and their impacts on dewatering. Underground pumping systems and disposal of water are also discussed in this Chapter.
- Chapter 3 discusses the various research methodologies which were undertaken in order to meet the objectives of the study.
- Chapter 4 looks at data collected and analysis. The main objective of this research is to determine a suitable method of dewatering the ore reserves below 1040mL. Surface Exclusion methods, Surface Deep Wells and Conventional Dewatering Methods were considered as potential dewatering methods. Backfilling mining methods were studied and their impact on dewatering are outlined. This chapter included the determination of the sources of water to the mine.
- Chapter 5 looks at the finding of this study. It discusses the sources of water and the current dewatering status. It also compares the drawdown and cost of the Deep Surface Borehole and Conventional Dewatering Methods.
- Chapter 6 gives the conclusion and recommendation of the study. It discusses what dewatering method is viable and cost effective for Konkola Mine.
- Reference list the journals, books, conference papers and technical reports that were reviewed and cited in this study.
- Appendices contains the rainfall records from 1953-2016 and the different pumping rates by Copperbelt Mines. It also looks at the WatchDog weather monitoring specification and the prediction of Konkola Mine dewatering rates for the 20 years spun from 1988.

CHAPTER 2: LITERATURE REVIEW

2.1 Introduction

This chapter looks at the study area location, characteristic and reviews past and present dewatering plans for Konkola Mine. This chapter also reviews regional and global dewatering approaches. Backfilling mining methods and their impacts on dewatering and it also reviews underground pumping systems and disposal of water.

2.2 Study Area Location

Konkola Mine is located in the northern part of the Zambian Copperbelt in the district of Chililabombwe and lies 25 Km north of Chingola as shown in Figure 2.1.

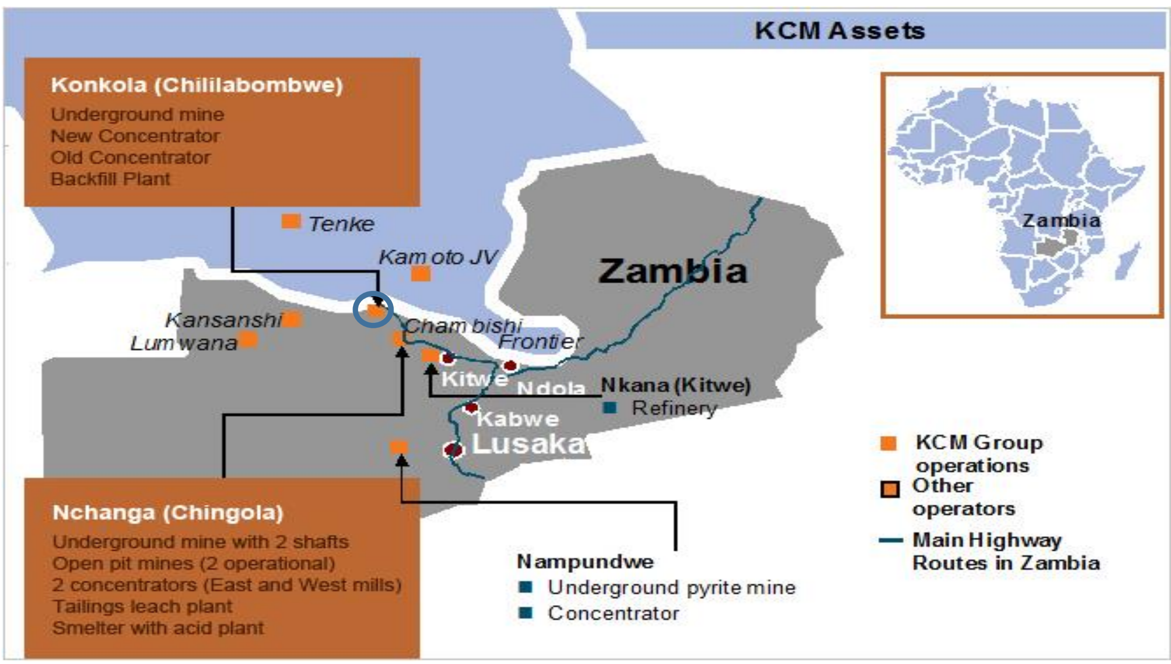

Figure 2.1: Location of Konkola Mine (kcm.co.zm)

2.3 Climate of Chililabombwe

The climate of Konkola Mine is influenced by its elevation of about 1,300m above sea level. There are three seasons experienced in Chililabombwe, These are: the cool and dry season from May to August with temperatures ranging from 6˚C to 27˚C, hot and dry season from October to November with temperatures ranging from 27˚C to 35˚C, and warm and wet season from December to April with temperatures ranging from 25˚C to 35˚C (en.clamate-data.org).

Most of the Precipitation at Konkola Mine occurs during the wet season with about 1300 mm/ year. The wettest months are December and January. Studies conducted by different Consultants between 1960 and 1988 to evaluate an overall water balance between precipitation, evapotranspiration, runoff and infiltration, are as shown in Table 2.1. However, recent studies show a 20% infiltration, 70% ran offs and 10% Evaporation (Water Management Consultants, 2000).

Table 2. 1: Infiltration as a proportional of precipitation in the Konkola mine Catchment Area (Zambia Consolidated Copper Mines (ZCCM) Limited, 1995)

Author	Rainfall mm/ Year	Evaporation %	Runoff %	Infiltration %
Howkins (1960)	1372	50	10	40
Raiston (1961)	1372	50	12	38
Starman and Shalash (1971)	1397	76.4	12.2	11.4
Leeds el at (1972)	1313	80	13	7
Hydro Geo (1988)	1334	75	13	12

2.4 Konkola Surface Hydrogeology

Konkola Mine area is wedged between two major faults; the Lubengele in the north and Luansobe in the south as shown in Figure 2.2. The other fault zones in the mining area are the Kirilabombwe Anticline Fault (which runs subparallel to the Lubengele fault) and the Cross-Anticline Axis Fault (this links up with the Lubengele and Luansobe faults) which are in between the Lubengele and Luansobe Faults. These fault areas are major recharge zones (Mulenga, 1993).

The Kafue River and its tributaries control the surface drainage system of the mining area and the surface hydrology pattern is controlled by the structural geology of the area (Mulenga, 1993).

The two important tributaries which recharge the aquifers are the Kakosa and Lubengele Streams. These streams flow over the mine Hangingwall Aquifers and in some areas they flow along and across the fault systems which are in hydraulic connectivity with the mine aquifers. Both streams discharge into the Kafue River at an approximate distance of 3 km from Shaft No. 1. The Lubengele Dam was built in 1964 in the Lubengele valley between the Lubengele and the Mingomba Stream areas, in the upstream of Shaft No.3. The Dam has been used as a tailing disposal from the time it was built. The Lubengele and Mingomba Streams seep through the dam’s embankment. The dam has a capacity to hold 73,873 million litres of water which would flood approximately 855 hectares.

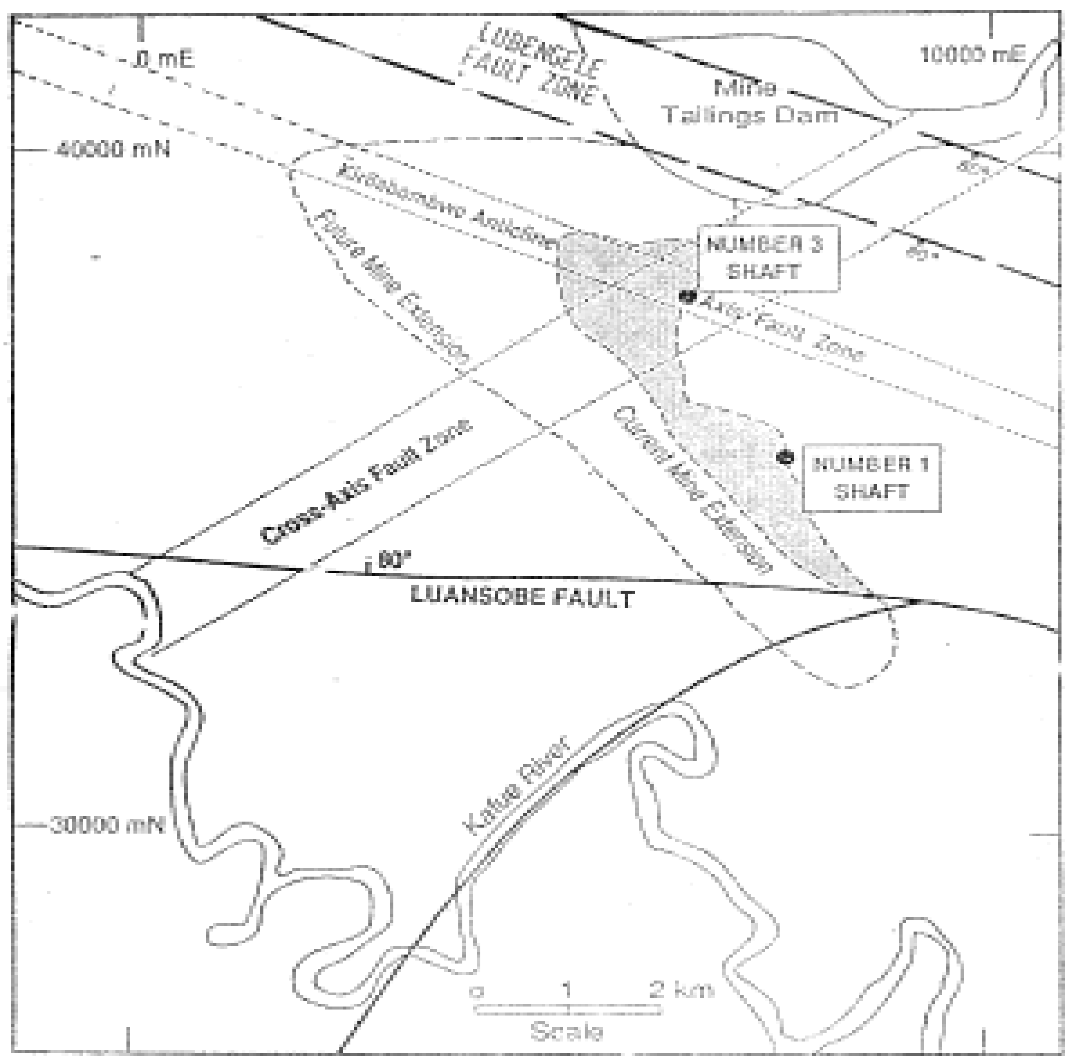

Figure 2.2: Surface Geology of Konkola Mine (Mulenga,1993)

2.5 Geology of Konkola Mine

The Bancroft mining area, which contains the Konkola orebodies, is the most north-westerly of the economically mineralized Kirila Bombwe Dome or anticline. The orebodies are subdivided into Kirilabombwe South (No 1 shaft), Kirilabombwe North (No. 3 shaft) and Konkola (No. 2 shaft). No. 4 shaft falls in the Kirilabombwe South as it is approximately 150m from No 1 shaft. The geological setting of Konkola Mine is shown in the stratigraphic column in Figure 2.3. Starting from the older rocks to the younger rocks that are from the Lower porous conglomerate (LPC) to the Hanging wall quartzite (HWQ).

Lower Porous Conglomerate (LPC) is a coarse, poorly sorted, porous and permeable rock. This rock type is of a very poor ground condition and it's usually not advised to place capital infrastructure in this rock. This is overlain by the Footwall Quartzite (FWQ) which is competent and where most ramps and haulages are placed in. Overlying the Footwall Quartzite is the Argillaceous Sandstone (AGSST) and above lies the Porous Conglomerate (PC) which is leached with fair to poor ground condition, porous and permeable. Above the PC is the Footwall Sandstone (FWSST) which is fair ground condition and where most extraction drives are placed and above the FWSST is the Footwall Conglomerate (FWC). These last three constitute the Footwall Aquifer, the second major aquifer at Konkola (Zambia Consolidated Copper Mines (ZCCM) Limited, 1995).

The Ore Shale formation lies above the Footwall Conglomerate (FWC) or the Footwall Sandstone. The contact between the Ore Shale Unity (OSU) and Footwall conglomerate (FWC) is termed as Geological Footwall (GFW). (Zambia Consolidated Copper Mines (ZCCM) Limited, 1995)

The OSU limit where mineralization becomes less than 1% is referred to as Assay Hangingwall (AHW). Above the Ore Shale the formations lie as follows: Hangingwall Quartzite formation (HWQ); Hangingwall Aquifer (HWA); Shale with Grit; Upper Roan Dolomite (URD); Mwashia Shale; Kakontwe Limestone; Kundulungu Shale. (Zambia Consolidated Copper Mines (ZCCM) Limited, 1995)

LPC, AGSST, PC, FWSST, FWC, and the OSU are sedimentary rocks whereas FWQ and HWQ are metamorphic rocks.

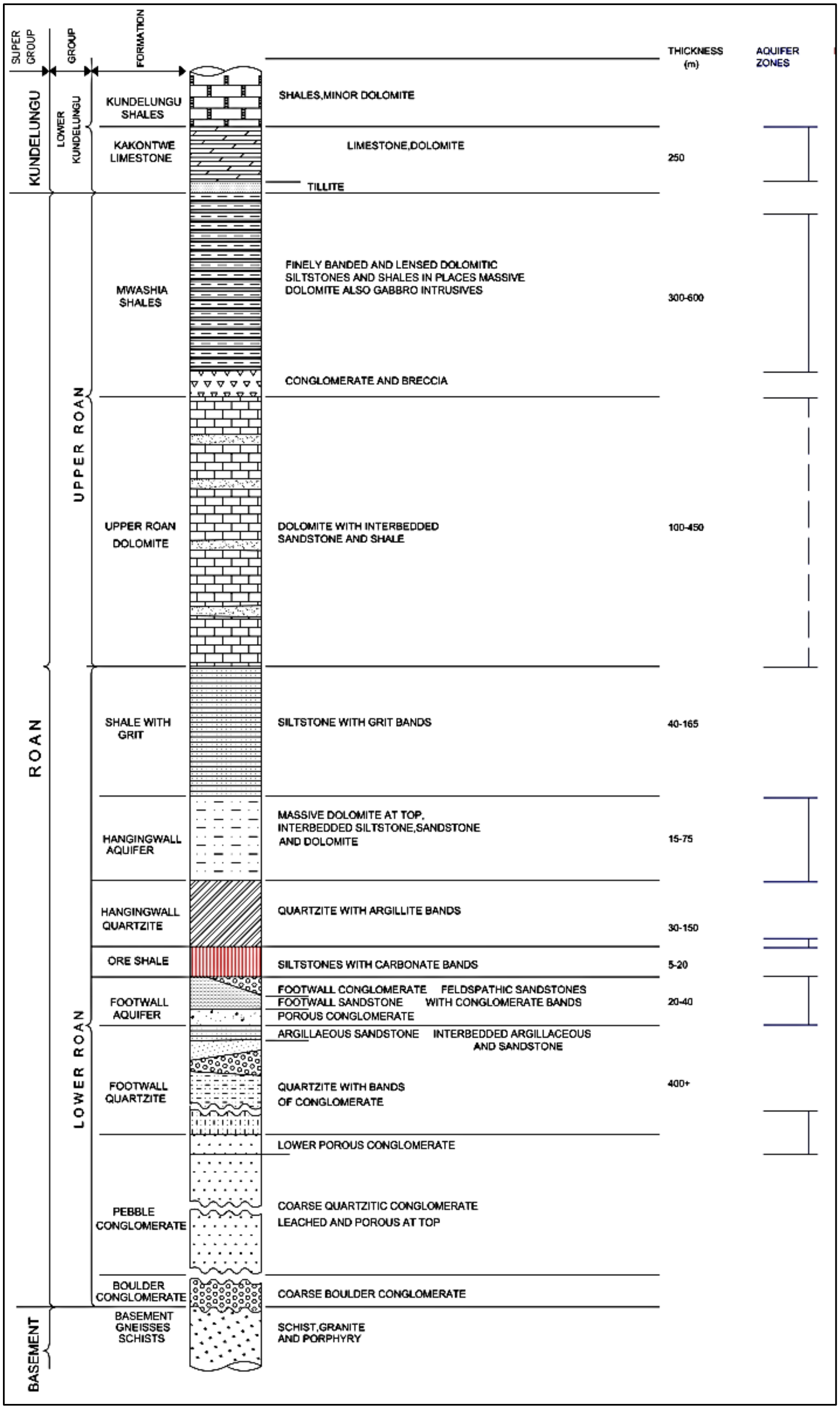

Figure 2.3: Stratigraphy of Konkola mine (Konkola Copper Mines, 2015)

2.6 Mining Methods

There are two main mining method used at Konkola Mine due to the changes in the dip and thickness of the rock. Figure 2.4 shows the mining methods employed at Konkola Mine.

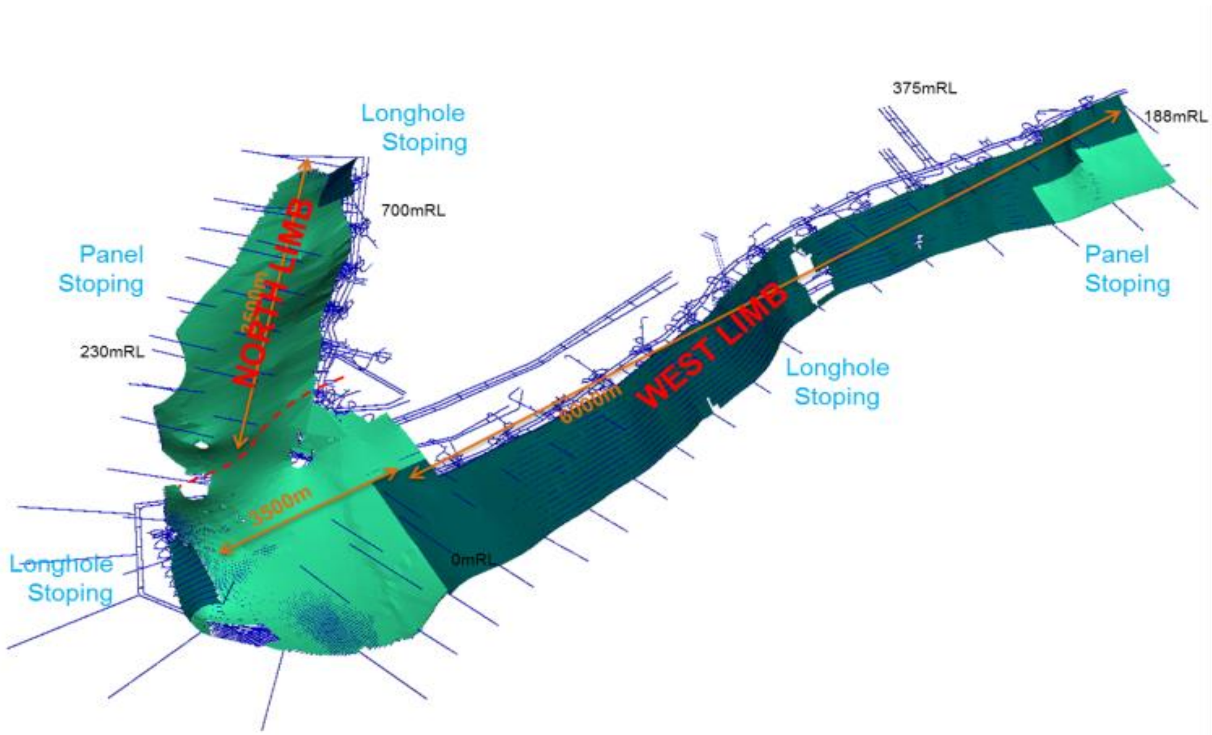

Figure 2.4: Konkola mining method (Konkola Copper Mine, 2015)

2.6.1 Sub Level Open Stoping (SLOS)

This method is a commonly used at both 1 shaft and 3 shaft, it involves subdividing the orebody into sublevels up dip with extraction haulages. The ring drilling pattern is used in the blast hole designs and holes are drilled using long hole drilling machine (Solo and Simba). The method is applied to both steeply and flat dipping ore bodies.

2.6.2 Post Pillar Cut and Fill (PPCF)

Generally, this method is suited for ore bodies with dips between 10º and 25º. This is an in-stope mining method and as such development and stoping are synonymous.

Post Pillar Cut and Fill (PPCF) mining method is a variant of the Room and Pillar mining method which advances up dip in a series of lifts. Waste rock or tailings are used as Backfill. The fill provides the working floor for the subsequent lift. Vertical Post Pillars are created from footwall to hangingwall to stabilize the hangingwall in the immediate mining area. This method is used at 3 shaft.

2.7 An Overview of Konkola Hydrogeology

The mine has five main aquifers but only three aquifers are dewatered. These are: the Hangingwall Aquifer, Footwall Aquifer and the Footwall Quartzite and Lower Porous Conglomerate Aquifer as shown in Figure 2.5. The upper roan dolomite aquifer is one of the largest aquifers at Konkola Mine and is dewatered for the sake of depressurisation and only when the pressures are too high.

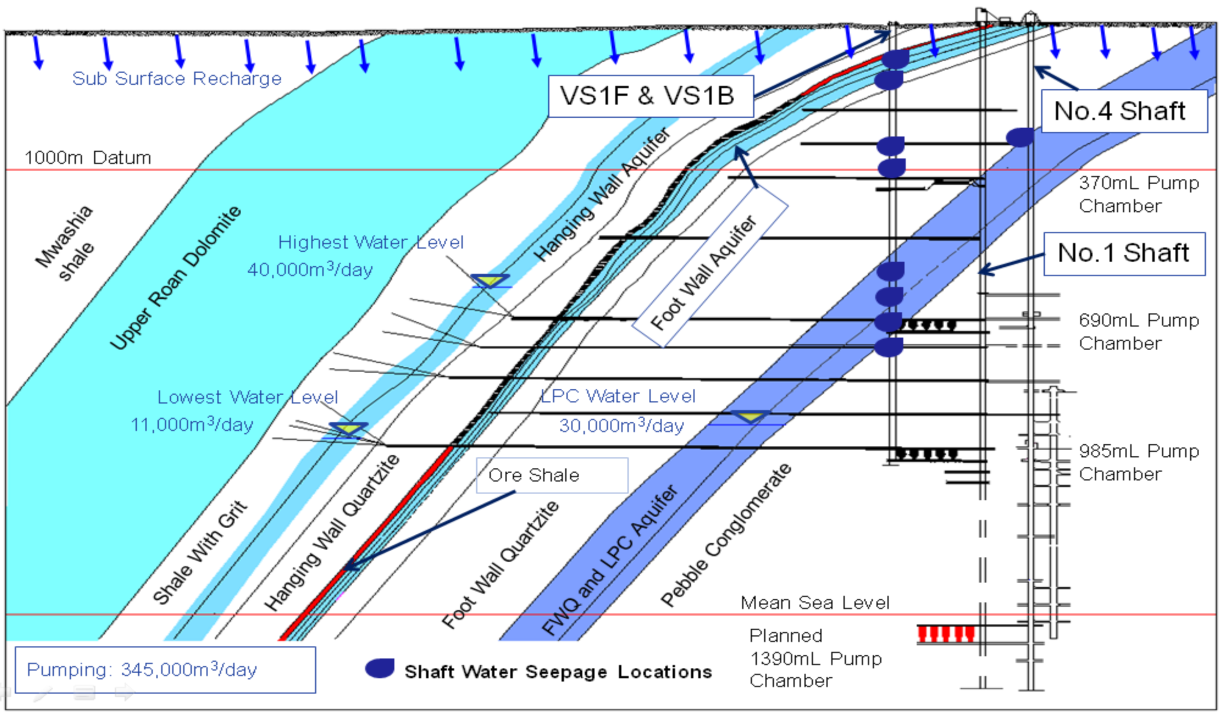

Figure 2.5: Konkola Aquifers (Konkola Copper Mine, 2015)

The Hangingwall Aquifer is made up of the dolomite, sandstone and siltstone formations. This aquifer lies above the orebody and is about 700m thick. Chemical and tritium dating (isotope) have revealed that the hangingwall water is young (post-1952) water compared to the footwall water. The hangingwall water is cold, slightly alkaline, high in total dissolved solids and has the same age as the water in the Kafue River (Mulenga, 1993). The aquifer is drained way ahead of stoping by mining of dewatering crosscuts in the hangingwall quartzite followed by the drilling of boreholes which intercept the Hangingwall Aquifer as shown in Figure 2.6. From the end of the dewatering crosscut, five long boreholes are drilled up into the aquifer to act as drains. The angle and length (usually 200m) of the boreholes are determined by the cave angle of stoping which is approximately 65°. The boreholes are drilled through a 3m long, 114mm diameter flanged standpipe which is grouted into the collar of the borehole. Once completed the borehole is equipped with high pressure valves and manometers to facilitate the measuring of groundwater pressure and to shut flow when required. A single drain hole may produce up to 9,000m^3/day and a dewatering cross cut can produce up to 80,000m^3/day. It takes about two years to lower the pressure in the HWQ after drilling the drain holes with an average drawdown of 10-15m a year.

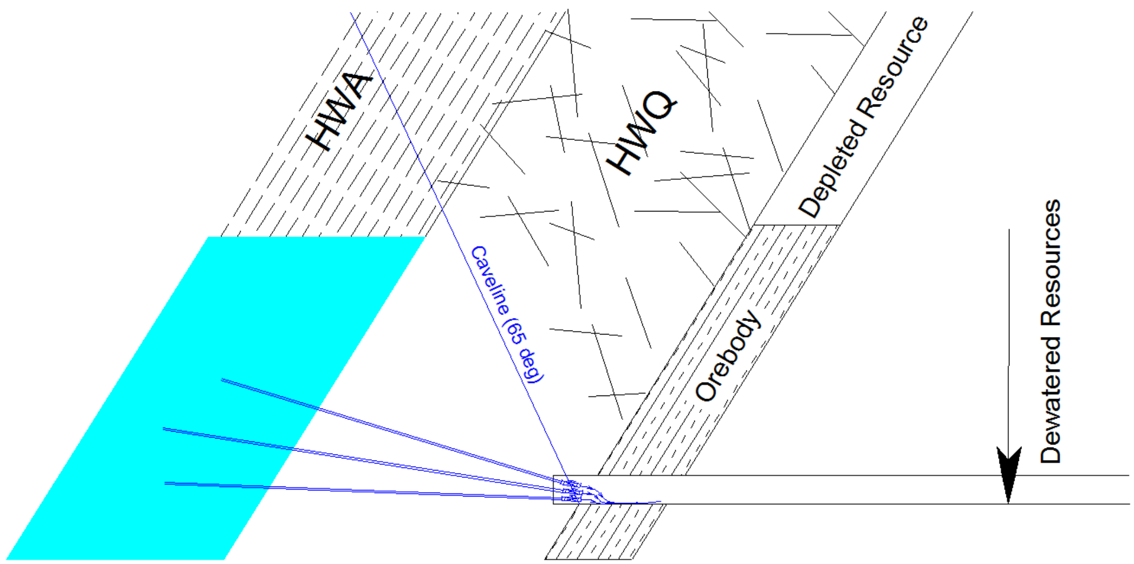

Figure 2.6: Dewatering of the hanging wall at Konkola (Konkola Copper Mine, 2015)

The Footwall Aquifer underlies the orebody and has a thickness of approximately 20m to 40m. It is made up of the Porous Conglomerate, Footwall Sandstone and Footwall Conglomerate. The footwall water is warm revealing that the water is old (Pre 1952) (Mulenga, 1993). This water is slightly acidic and low in total dissolved solids. The aquifer is drained by simply mining through it, this allows the aquifer to drain dry.

The Footwall Quartzite and Lower Porous Conglomerate Aquifers are the lower most aquifers. They consist of a fractured lower part of the Footwall Quartzite and a Lower Porous Conglomerate mainly in vugs. It's about 150m thick and is located 250m below the orebody. The water is warmer and slightly acidic with less total dissolved solids (Mulenga, 1993).

The other aquifers that exist at Konkola Mine are the Upper Dolomite Aquifer (one of the largest aquifers), the Kakontwe Limestone Aquifer (the largest) and other small isolated aquifers. These aquifers are only dewatered as a means of reducing the hydrostatic load.

2.8 General Dewatering Background

Dewatering at Konkola Mine started in July, 1955, pumping about 15 000m^3/d to a peak of 420, 500m^3/d in June 1978. Since then the pumping rates had gradually declined to 300, 000m^3/d in April 1995 and then increased to about 350 000m^3/d in 2014 as shown in Figure 2.7.

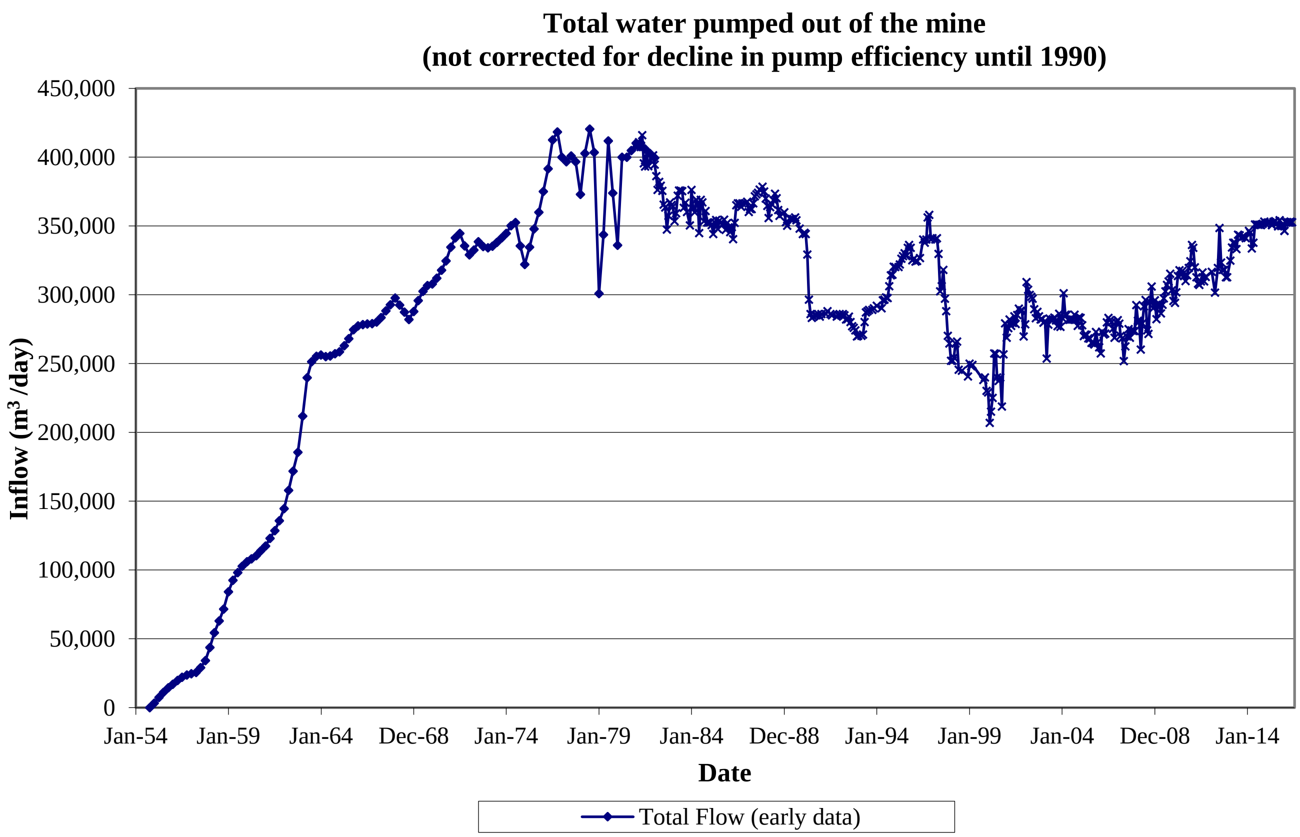

Figure 2.7: Average Volume of water pumped per day at Konkola Volume (m^3/day)

It should be mentioned that the method applied for measuring the flows from the inception of mining to 1990 was not accurate enough as it measured pumping hours and design flow rate of the pumps while neglecting the loss of efficiency of some pumps. In most cases, the actual pumping rates were overstated as indicated by Water Meyer in 1972 by approximately 25% (Zambia Consolidated Copper Mines (ZCCM) Limited, 1995). The following reductions were made to the reported discharges reported in the Zambia Consolidated Copper Mines (ZCCM) Limited, 1995 report.

1954- 1958- 100% of recorded

1959-1962- 90% of recorded

1963- 1971- 80% 0f recorded.

Since 1990, more accurate methods have been used, which measure flow rates this resulted in 17% reduction of the reported discharge from the inception of mining to May, 1990 (Zambia Consolidated Copper Mines (ZCCM) Limited, 1995). Figure 2.8 shows corrected pumping rates. The water pumped out has increased from inception to 2014 due to mine expansion and the effect of subsidence on surface and underground water bodies.

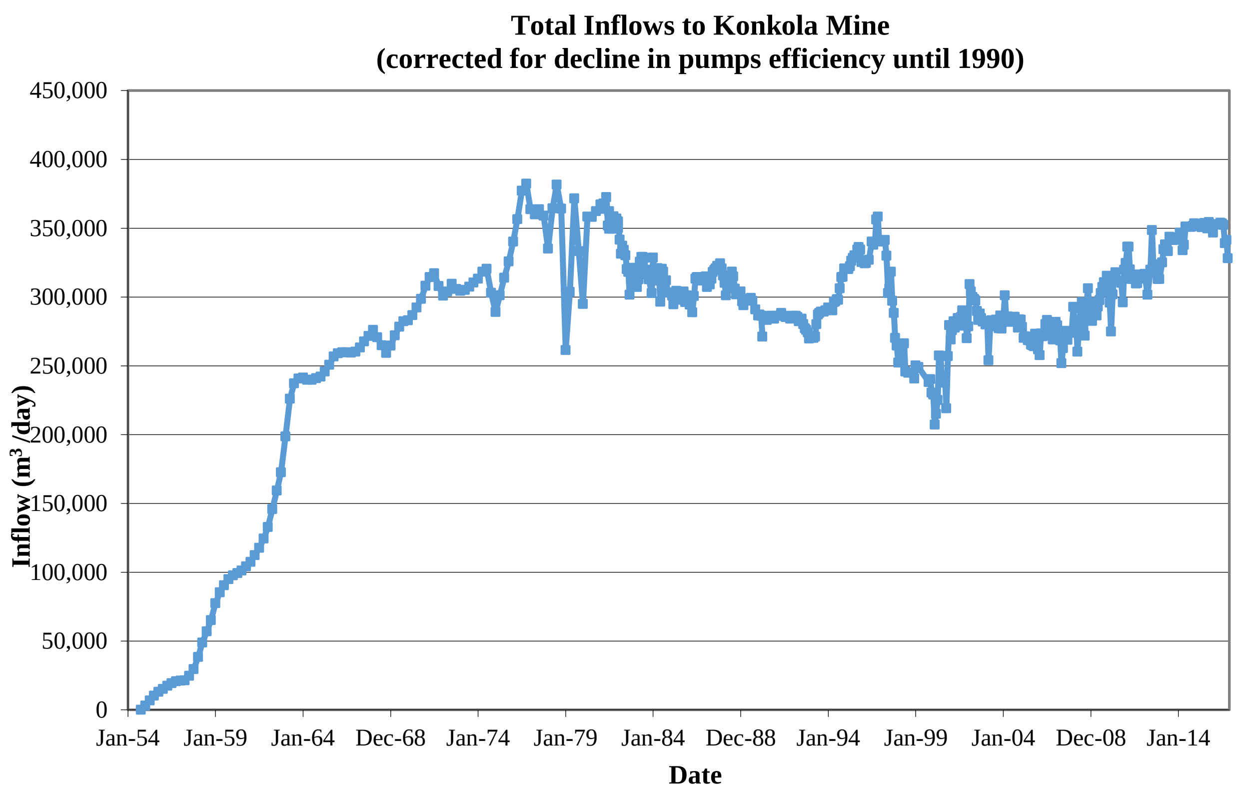

Figure 2.8: Corrected Average Volume of water pumped per day at Konkola Volume (m^3/day)

2.9 Ground Settlement

Ground settlement is the lowering of ground surface, known as subsidence or settling. Ground settlement is measured by field procedure either by survey, theodolite and transit rod, or as a manometer survey. From inception of mining, Konkola Mine has used mostly caving and open stoping mining methods which has resulted in the area experiencing subsidence. The highest settlement is 0.2m and the lowest is 0.01m per year as shown in Figure 2.9. This has increased the hydraulic conductive in areas that fall within the subsidence controlled region and it has also damaged the lining of discharge canals that pass through the caved region.

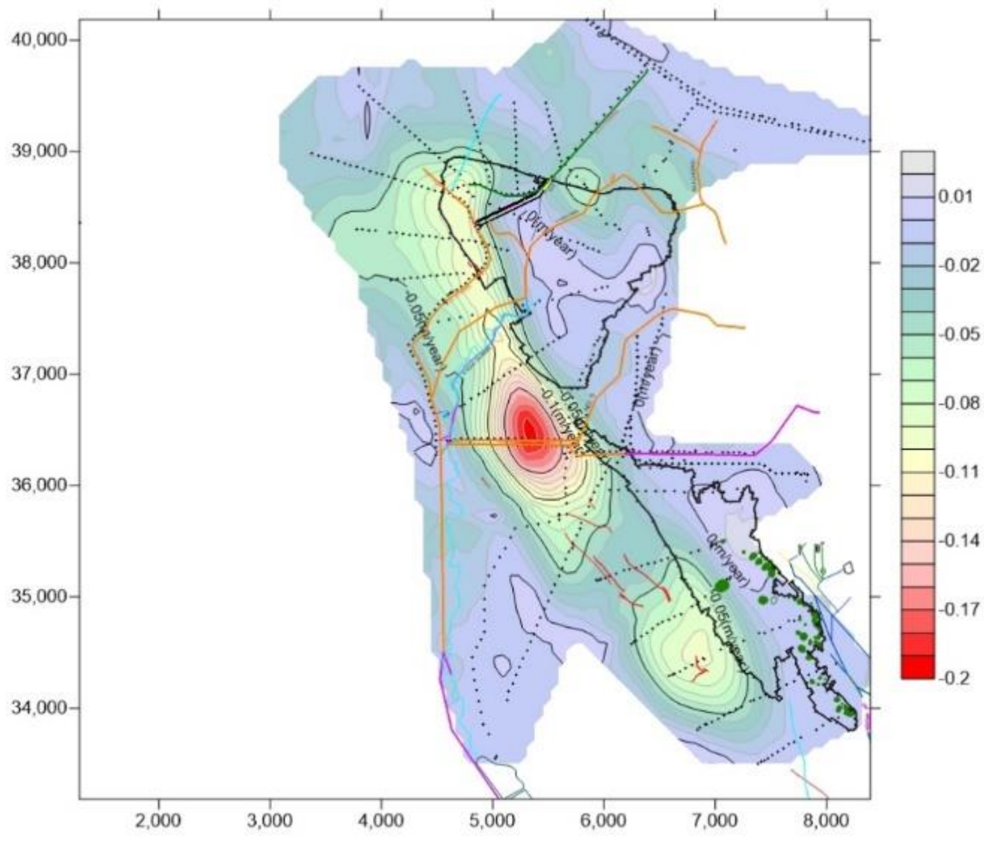

Figure 2.9: Rate of Settlement (m/year) (Konkola Copper Mines, 2015)

2.10 Copperbelt Regional dewatering approach

Water has always posed a challenge to mining on the Copperbelt, due to the presence of aquifers and surface water bodies on the mining sites. This led to mines during and before the Zambia Consolidated Copper Mines (ZCCM) era to come up with dewatering systems and strategies which enabled the exploitation of the orebodies safely. The ratio of water pumped to ore mined in 1992/93 was 11.4t of water to 1t of ore mined (Naish, 1993). The dewatering cost has always made up a significant part of the mining cost on the Copperbelt. The mines have utilized different dewatering method such as Surface Exclusion, Surface Boreholes, Interception of water, Simple Drainage, Breakthrough and Grouting. An in depth look into the dewatering methods used on Copperbelt is as outlined below.

2.10.1 Surface Control of water

The method excludes or reduces the amount of water that reaches the catchment area of the mine. This prevents recharge/inflow and reduces the water that reaches the mostly fractured parts of the mining catchment area (as a result of subsidence or due to natural phenomena such as the presence of faults that provides hydraulic conductivity to the mine). There are different methods used to control surface water, these include: dams (like the Lubengele Dam), Herringbone ditches to speed up runoffs, pipes to carry water over hydrological hazard zone (Like Lubengele Dam discharge channels), stream and river diversion, river lining, dambo filling and stream gauging. Different surface exclusions have been carried out on the Copperbelt, the Lubengele Stream at Konkola Mine was dammed and their other streams that were diverted in Mufurila and Luanshya Mine. Mulenga estimated that about of 180,000m^3/d of water could have been excluded by surface control at Konkola Mine, this represented 59% reduction in the water pumped out per day by the mine at that time (Mulenga, 1993).

2.10.2 Interception Method

This method intercepts water from flowing into an active mining area. It involves the driving of dewatering crosscuts at the lowest points and at the extremities of the orebody where drilling then intercepts the aquifer (Naish, 1993). This method lowers the dewatering cone from the active and top-levels thus preventing inflow into the mining areas. This method has been used at Mufulira Mine in controlling the flow of the dolomite water. When the dewatering crosscuts are initially mined and the boreholes drilled, they cause a steep-deep sided cone of dewatering, but as time progresses it becomes wide and flat.

2.10.3 Simple Drainage Method

This method is used by mines which have substantial pumping capacity that can cope with uncontrollable water. This is a widely used method on the Copperbelt and worldwide because of its simplicity in implementation. In this method, the drives (with a pilot hole for safety) are mined into the aquifer zone with a reduced hydraulic head (Naish, 1993). In order to cope with uncontrolled water, the mine is equipped with a substantial pumping capacity and a watertight door in case of power failure to protect the Shaft and other installations. This method was used at Nkana Mine, were the crosscuts to the orebody were mined at a position (lower conglomerate, footwall sandstone or footwall conglomerate) to the footwall side of the Footwall Aquifer. Holes were drilled which delineated the footwall aquifer prior to delineating the orebody. Once the hydrostatic load reduced, the heading was advanced into the footwall aquifer as an extraction drive.

However, at Mokambo Mine, they took the method lightly and literary as being simple and the suffered the unforgiving effects of that decision. The Romanian company Geomin got this method wrong in 1976. The pumping capacity at Mokambo Mine was not substantial to cope with the amount of water to be encountered and the watertight door was not suitable for the operations. As if this was not enough, the backup generator was of limited capacity. During the breakthrough of the footwall aquifer, the mine experienced a power failure which resulted in flooding of the mine and closure till present day though efforts are being made to pump out the water and restart operations before 2019 with a monthly production of 3000t by the new owners Changfa Mineral Resources Limited of China (Nkweto, 2016).

2.10.4 Breakthrough Method

This is similar to Simple Drainage method as it involves the mining of drives into the aquifer but in a more controlled manner. This method has been practiced at Konkola, Mufulira and Nkana Mines. It involves the mining of drives into the aquifer that have a suitable watertight door or suitable puddle pipe installed (Naish, 1993).

2.10.5 Dewatering Drilling

This approach is widely used on the Copperbelt in both surface and underground operations. In surface mines it has wide application in dewatering (Naish, 1993). In both underground and surface mine, holes are drilled to intercept the aquifer, the water flows from the boreholes either through gravity or pumping. At Nchanga underground mine, a twin haulage was mined under the open pit at 457m level to the east section for dewatering. At 10 East and 15 East some inclines were mined in which the dewatering holes which connect the Nchanga Open pit and the Nchanga underground were located. The holes were drilled on the base of the open pit thus ensuring the phreatic surface is below the pit, though this is not always the case due to the strata water being trapped in folds, requiring in-pit drilling of drain holes and in-pit pumping.

Underground dewatering drilling is widely used on the Copperbelt. In different mines, holes are drilled into the aquifer to lower the hydrostatic load. Most of these holes are equipped with gate valves which can be closed in case of a power failure e.g. at Konkola Mine and Nkana Mine. At Nkana Mine in certain areas where the water temperature reaches 43°C, the holes are equipped with valves made of stainless steel which are usually reclaimed and reused once the holes are dry.

Some mines like Nkana and Chibuluma took advantage of the already drilled exploration hole by driving dewatering crosscuts into exploration holes as a way of reducing the drilling cost. This gave tremendous results of great drawdowns and cost reduction. But this came with the risk of flooding by an unplanned intersection of exploration holes which are connected to a surface water source, this was experienced at both mines. Luckily both mines did not flood, because the drives were equipped with watertight doors and the mines had substantial pumping capacity. This risk was resolved by cementing all surface exploration hole, 50m above the orebody.

2.10.6 Grouting

This is the exclusion of water to an area by reducing the permeability of the surrounding strata through pumping of cement or resin which result in a closing of fissures and fractures hence reducing the permeability. This approach has been used in surface and underground mines. Cementitious and resin grout has been widely used on the Copperbelt especially in Underground mines. Holes are usually drilled around the excavation and resin or cement is pumped into the holes. At Konkola Mine, Shaft No. 4, 800mL level haulage drive, this method was used and is still being using. 5 cover holes are drilled at 36m length in a development heading as shown in Figure

2.10. The holes are then used for grouting and once grouting is done the hole is re-drilled if water comes out the grouting process is repeated. Once water does not come out, the end is mined up to 30m leaving a 6m collar for protection then the process is repeated. This is also used in tramming haulages that need to be kept dry because of the presence of the trolley line. Holes are drilled on the back of the excavation and resin is pumped into the hole. When water starts seeping into the tramming haulage the grout is redone.

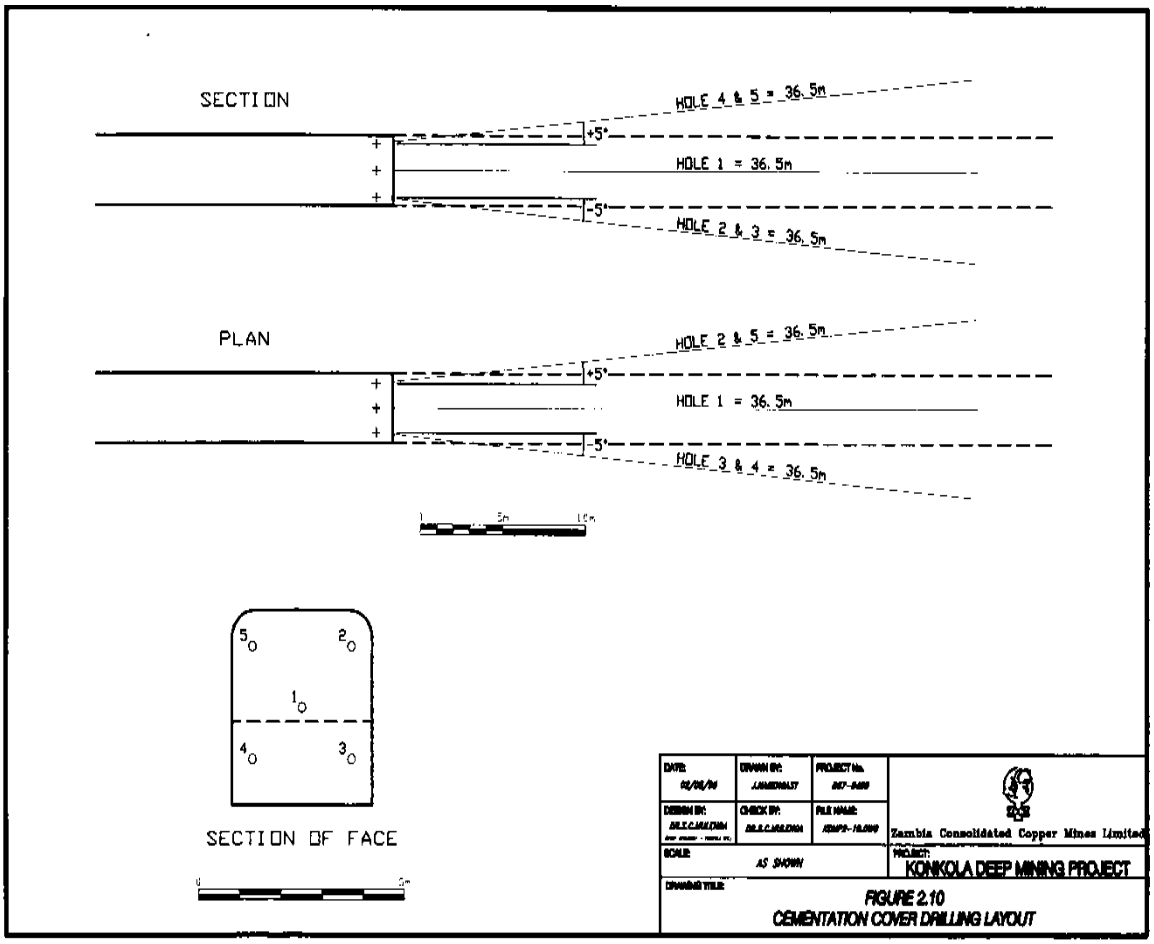

Figure 2.10: Cementation drilling layout at Konkola mine (Zambia Consolidated Copper Mines (ZCCM) Limited, 1995)

2.11 Global Dewatering Approach

Globally dewatering is categorised by three approaches, these are: Active Approach, Passive Approach and Hybrid Approach. The Active Approach is also known as Advanced Dewatering, this approach dewaters way ahead of mining and is done using boreholes and dewatering crosscuts or drainage adits while the Passive Approach which is also known as a Reactive Approach dewaters during mining by natural pore pressure dissipation through seepage. The third approach is the combination of the Active and Reactive Approaches known as the Hybrid Approach.

The Active Approach utilizes the dewatering galleries, boreholes and water diversions to lower the water table. The rivers or streams are diverted away from the pit or mine. The other technique involves the sinking of boreholes or wells to dewater through the principle of pumping interface either through gravity or mechanical means. As a borehole is pumped a cone of depression develops on the water table as shown in Figure 2.11. A combination of depression cones produces an interference effect which lowers the water table more effective and is achieved by placing pumping bores close to each other. The use of many pumping boreholes increases the effectiveness of this method.

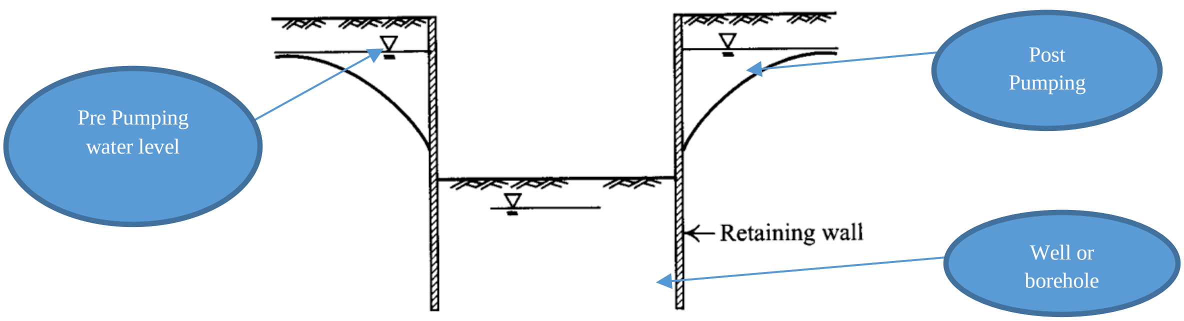

Figure 2.11: Borehole (Chang-Yu Ou, 2006)

To avoid the pumps from being clogged, the walls of the well are always encased or fitted with a screen and filters. The filters are made from materials that are fine such that they prevent soil particles from flowing through it. The filters have good permeability and flow through the casing or the screens. The filters must meet the following standards;

$$\frac{\mathrm{D15\ (filter)}}{D85\ (ground)} < 5\text{: insitu soil particle don't flow into the filter} \qquad \text{Eq (1)}$$

$$\frac{D\ 15\ (\,filter)}{D15\ (ground)} > 5\text{: material keeps good permeability} \qquad \text{Eq (2)}$$

$$\frac{D85\ (filter)}{D\ (screen\ diameter)} > 2\text{: Filter material does not flow in the screen} \qquad \text{Eq (3)}$$

The deep wells can be used in conjunction with the well points, especially in construction sites. The deep well have been used successfully in Nevada Goldstrike Mine with a drawdown of 520m (Johny, 2012).

The other method used in Active dewatering is dewatering galleries, these are drives mined to intercept and dewater the aquifer way ahead of mining. In open pits they are tunnels which encircle the pit and block water from reaching the pit while in underground mines these are drives which intercept the aquifer, they are usually mined at the lowest level so that the can dewater the aquifer above them before mining.

The Reactive Approach uses sumping, drains and in-pit boreholes. The in-pit bores are drilled to depressurise and drain the saturated slopes. The sumps and drains are widely used for low permeability sediments. Water seeps through the sediments gravitate to lowest points where it collects (sump), then it is pumped out to the surface as shown in Figure 2.12.

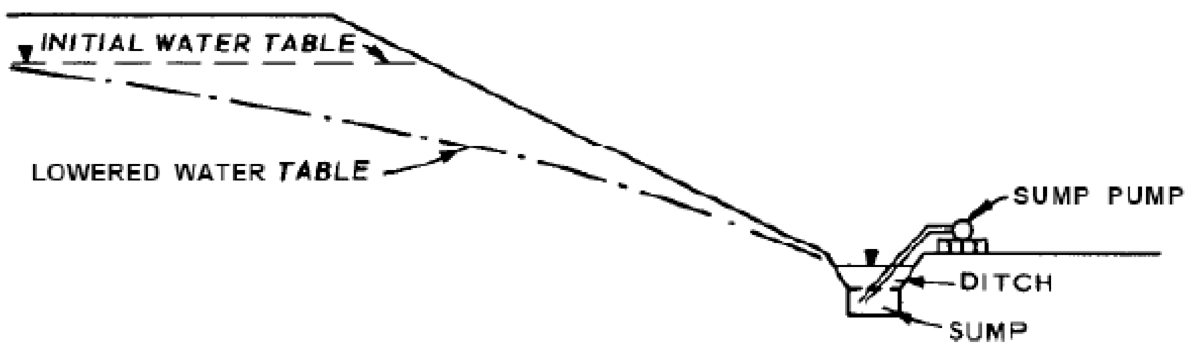

Figure 2.12: Dewatering through Sumps pumping (Department of the Army, Navy and Air Force, 2004)

2.12 Analysis of Groundwater Flow

Groundwater flow can be determined by a reliable hydrogeological study. The type of flow has to be determined whether artesian or gravity flow. The complete picture of the groundwater and subsurface must be understood and clearly outlined in the analysis of any dewatering system, seepage flow characteristics must be determined for the site and surrounding.

The drawdown, groundwater flow, transitivity, the quantity of discharge and other hydraulic parameters can either be determined by the analytical or numerical methods or a combination of both approaches.

2.12.1 Analytical Approach

The analytical approach applies both simple and advanced formulae to determine the flow of groundwater. There are several formulas for flow calculations, but in this report, only a few are reviewed.

2.12.1.1 Darcy's Law

Darcy in 1856 described flow rate (Q) for laminar flow as being directly proportional to the cross-section area (A) in which the fluid is flowing and also directly proportional to the hydraulic gradient. This is represented by equation 4, which can be used to estimate flow of water into a shaft or open pit. The law was formulated by Henry Darcy in1956 based on results of experiments on the flow of water through beds of sand, this greatly contributed to the forming of a branch of science called hydrogeology.

$$Q = k \text{ x } A \text{ x } i \qquad \text{Eq (4)}$$

Where: k- hydraulic conductivity (m/s), i- hydraulic gradient, A- Area (m^2), Q- Flow rate (m^3/s)

Where velocity of fluid flow is,

$$v=\frac{Q}{A}=-k*\frac{dh}{dl} \qquad \text{Eq (5)}$$

Where: v- Velocity (m/s), $\frac{dh}{dl}$ -hydraulic conductivity

Darcy clearly started that groundwater flows from a higher elevation to lower elevation or more precise from high potential energy (hydraulic head) to a lower potential energy (hydraulic head).

2.12.1.2 Steady-state Flow Equation

This is used to determine the total energy flow. This assumes that the total head is constant with the flow, which means a change in flow with time does not affect the head. This can also be understood as the input into a system is equal to the outputs.

$Q_1=Q_2$ Eq (6)

Therefore; $\Delta Q=0$ Eq (7)

Then;

$$\Delta Q = [K_x\frac{\partial^2 h}{\partial x^2}+K_y\frac{\partial^2 h}{\partial y^2}+K_z\frac{\partial^2 h}{\partial z^2}]\ \delta x.\delta y.\delta z = 0, \quad \text{Eq (8)}$$

Where: $\partial x.\partial y.\partial z$=volume of ground

$K_x=K_y=K_z=K$ is hydraulic Conductivity Eq (9)

Then,

$$\Delta Q=K[\frac{\partial^2 h}{\partial x^2}+\frac{\partial^2 h}{\partial y^2}+\frac{\partial^2 h}{\partial z^2}]\partial x.\partial y.\partial z=0 \quad \text{Eq (10)}$$

2.12.1.3 Unsteady State Flow Equation

This is frequently called transient flow and is defined as the fluid flowing condition at which the rate of change of pressure with respect to time at any position in the reservoir is not equal to zero or constant, the head is a function of both time and position. In this state, the recharge is not equal to the discharge and this condition can be presented by the equation below:

$$\Delta Q = [\frac{\partial\ Storage}{\partial\ time}]\ \delta x.\delta y.\delta z = [K_x\frac{\partial^2 h}{\partial x^2}+K_y\frac{\partial^2 h}{\partial y^2}+K_z\frac{\partial^2 h}{\partial z^2}]\ \delta x.\delta y.\delta z \neq 0 \quad \text{Eq (11)}$$

The amount of water released per unity is the volume of an aquifer per total head is S_s

S_s= [storage]. $\partial x.\partial y.\partial z$ Eq (12)

Thus the total volume capable of being released in a saturated aquifer is given by equation 13;

$S=S_sD$, D=thickness of aquifer (m), S=volume of water released (m^3)

Eq (13)

2.12.1.4 Well Equations (Well Theory)

Well theory looks at the relationship between discharge and drawdown. This relationship is affected by: number of wells, structures, geological conditions and pumping times.

The relationship between drawdown and discharge can be solved by either using mathematical well formulas or numerical ground modeling. The former is inferred by assuming ideal conditions. Numerical ground modeling is widely preferred and used because it's applicable to any geological or pumping condition.

There are two types of aquifers, these are confined and unconfined aquifers. A confined aquifer is one were the top and the bottom surrounding are impermeable layers, as shown in Figure 2.13.

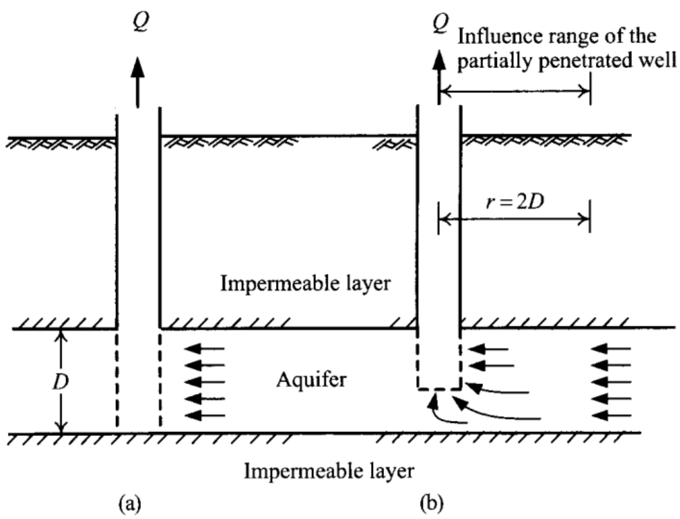

Figure 2.13: Partial and full penetration borehole in a confined aquifer (Chang-Yu Ou, 2006)

2.12.1.4.1 Confined aquifer

The wells can either fully or partially penetrate the aquifer. When calculating drawdown and discharge using the mathematical method whether an aquifer is fully penetrated or not is of significant importance as it affects the assumptions made in the equations. This is fully looked into below when it comes to the mathematical method.

2.12.1.4.1.1 Full Penetration wells

This is a well which fully penetrates an aquifer. The flow into the well is horizontal. Theis in 1935 looked at this type of well and came up with a drawdown equation which is based on the following assumptions**:**

- The aquifer has uniform thickness and is a homogeneous and isotropic confined aquifer
- The well fully penetrates the aquifer
- The well is 100% efficient

- Meets Dupuit-Theim assumption: hydraulic gradient any point on the draw curve is a slope at the point
- No recharge water is within the influence of the range
- Radius of well is too small that the amount of retained water can be ignored

Theis drawdown equation for non-equilibrium is given in Figure 2.14.

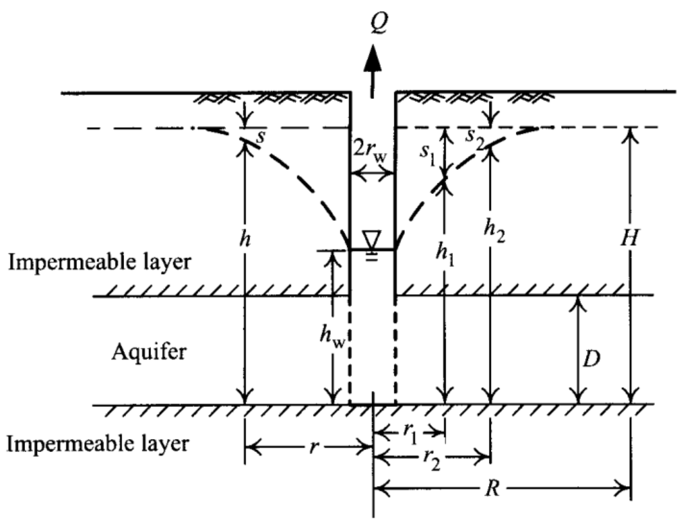

Figure 2.14: Full penetration in a confined aquifer (Chang-Yu Ou, 2006)

$$s=\frac{Q}{4\pi kD}W(u) \qquad \text{Eq (14)}$$

$$=\frac{Q}{4\pi T}W(u) \qquad \text{Eq (15)}$$

$$W(u)=\int_u^\infty \frac{e^{-u}}{u}du=-0.5772-\text{In } u-\sum_{n=1}^{\infty}(-1)\frac{u^n}{n.n!} \qquad \text{Eq (16)}$$

$$U=\frac{s\,r^2}{4Tt} \qquad \text{Eq (17)}$$

Where:

s=draw down (m), Q=discharge quantity (m^3/s), D= thickness of aquifer (m), T=kD= coefficient of transmissivity (m^2/s)

W(u)=well function, u= parameter of well function, r= distance from Centre of well

S=coefficient of storage or storativity t=time since pumping started (s), S,kD,k=hydraulic parameter

Coefficient of storativity is the drained volume of water due to the lowering of a unit head (hydraulic head) per unit surface area. This is dimensionless.

Jacob's non-equilibrium equation calculates s (Jacob, 1940).

$$s= \frac{0.183Q}{T}\log \frac{2.25Tt}{r^2S} \qquad \text{Eq (18)}$$

Jacob's equation only applies when u ≤ 0.05.

Assuming the range of pumping as distance where drawdown declines to zero

Then the influence range (R) will be;

$$R=\sqrt{\frac{2.25Tt}{S}} \qquad \text{Eq (19)}$$

Many people have suggested many different ranges for J. Kozeny in 1953 suggested that R (m) is;

$$R=1.5\sqrt{\frac{Hkt}{n}} \qquad \text{Eq (20)}$$

Where: n is porosity, H is ground water level before pumping (m).

Assuming s=0, that is the when drawdown does not expand with regard to pumping time. Thiem equation of equilibrium which can be derived from the differential equation of drawdown curves is given below;

$$Q=\frac{2\pi kT(s1-S2)}{in(\frac{r2}{r1})} \qquad \text{Eq (21)}$$

r1 and r2 are distances from 1st observation well and 2nd observation to the center.s1 and s2 are drawdowns in wells 1 and 2.

Letting R replace r gives s at any distance (m). The Thiem equilibrium equation is rewritten to::

$$s=\frac{QIn(\frac{R}{r})}{2\pi kD} \qquad \text{Eq (22)}$$

Letting drawdown in a well be (H-h_w) and the radius of the well be r_w.. we then have Thiem equilibrium equation rewritten as:

$$Q=\frac{2\pi kD(H-hw)}{In(\frac{R}{rw})} \qquad \text{Eq (23)}$$

2.12.1.4.1.2 Partial Penetration well

This is more widely used than full penetration well. When r≤2D, then the flow lines in the vicinity of the well are not necessarily horizontal, there is vertical flow line available. r>2D then the effects of partial penetration can be ignored and drawdown can be calculated by equations of full penetration. When r≤2D, the amount of drawdown is larger than a full penetration well. Kozeney in 1953 suggested an equation**;**

$$Q=\frac{2\pi k(H-hw)}{in\ (\frac{R}{rw})}\ \mu \qquad \text{Eq (24)}$$

$$\mu=\frac{D1}{D}(1=7\sqrt{\frac{rw}{2D1}}\mathrm{Cos}(\pi D/2D) \qquad \text{Eq (25)}$$

μ is coefficient of modification, D1 penetration of well (m), D is aquifer thickness (m).

2.12.1.4.2 Free Aquifer

These are also known as unconfined aquifers. These are exposed to the atmosphere and are underlain by an impermeable layer. There are also two types of penetration for free aquifers wells, its either they are fully penetrated or partially.

2.12.1.4.2.1 Full penetration wells

The mathematical equation in free aquifers for calculating drawdown is complicated because the coefficient of transmissivity is not constant but varies with time and distance.

When s<<D, D can be assumed constant during dewatering. T can also be assumed constant.

$$T= kD \qquad \text{Eq (26)}$$

Storativity (S) ≤ porosity

Depuity and Thiem assumption gives the equilibrium equation of drawdown of a full penetration (as shown in Figure 2.15) of a free aquifer as;

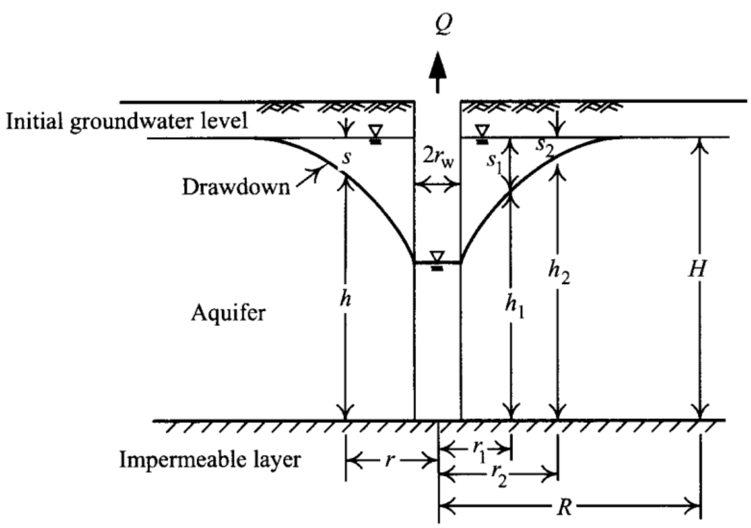

Figure 2.15: Full penetration well in a free aquifer (Chang-Yu Ou, 2006)

$$Q= \frac{\pi k(h2^2-h1^2)}{In\ (\frac{r2}{r1})} \quad \text{Eq (27)}$$

$$= \frac{\pi k(h2^2-h1^2)}{2.3\log(\frac{r2}{r1})} \quad \text{Eq (28)}$$

When R replaces r and r_w which is well radius (m) replaces r1 and r2. Then,

$$Q= \frac{\pi k(H^2-r_w^2)}{in(\frac{R}{r_w})} \quad \text{Eq (29)}$$

Draw down at any distance r in meters will be:

$$H^2 - h^2 = \frac{QIn(\frac{R}{r})}{\pi k} \quad \text{Eq (30)}$$

2.12.1.4.2.2 Partial Penetration Well

The equations for this sections are complicated. Marino and Luthin have written a lot of literature on them in 1982.in this section we are looking at Hausman proposed equation (Hausman, 1990) and referring to Figure 2.16.

$$Q = \frac{\pi k\left(H^2 - h_w^2\right)\alpha}{In(\frac{R}{r_w})} \qquad \text{Eq (31)}$$

$$\alpha = \sqrt{\frac{Hi - D_1}{H}}\ \sqrt[4]{\frac{H + D_1}{H}} \qquad \text{Eq (32)}$$

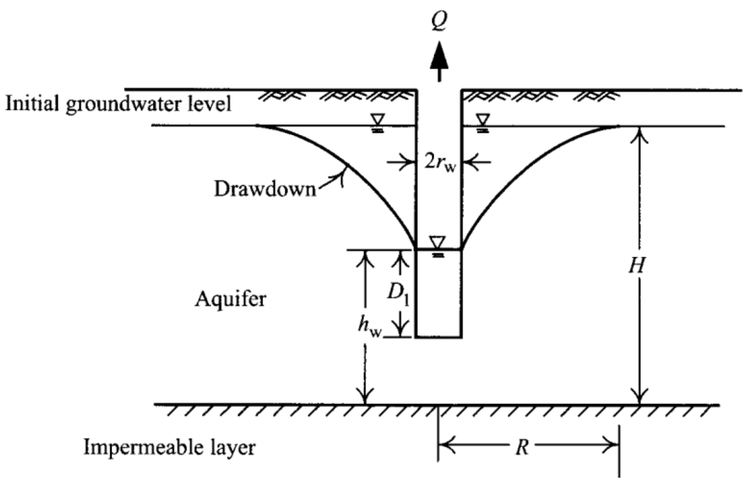

Figure 2.16: Partial Penetration in a free aquifer (Chang-Yu Ou, 2006)

Ollos proposed that the Value of C=0.5, thus the height of free discharge surface h_s in meters

$$h_s = \frac{C(H - \text{Ho})}{H} \qquad \text{Eq (33)}$$

- Sichart in 1928 suggested that the zone of influence of a borehole can be calculated by L

$$L = C' \, (H - h_w) \sqrt{k} \qquad \text{Eq (34)}$$

Where C=3000 for wells or 1500 to 2000 for single line well point

H, h_w in meters and k in m/s

2.12.1.4.3 Group Wells

Most of the times dewatering using wells is done by multiple wells. Figure 2.17 shows multiple well arrangement in a free aquifer.

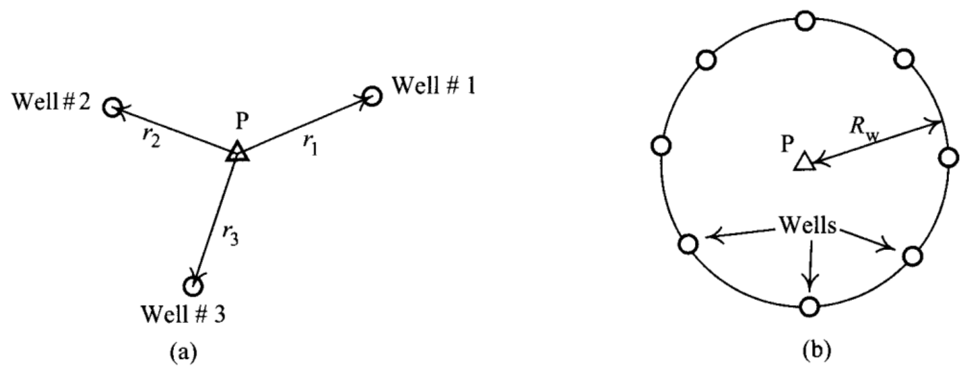

Figure 2.17: Group Wells (Chang-Yu Ou, 2006)

Assuming well 1 in (a) is used for pumping in an equilibrium state, then the quantity of discharge can be calculated by:

$$Q = \frac{\pi k(H^2 - h_1^2)}{In(\frac{R}{r_2})} \qquad \text{Eq (35)}$$

Where;

R is the Influence Range (m)

r_1 is the distance (m) from well # 1 to point P

H is the groundwater level before pumping (m)

h_1 is the groundwater level (m) at a distance of r_1 from the well.

Well # 2 and well #3 can be calculated in the same way.

Assuming there are n# of wells, pumping simultaneously in a free aquifer assuming equilibrium, forchheimer in 1930 suggested the equation below for calculating total quantity discharge.

$$Q_{tot} = \frac{\pi k(H^2 - h^2)}{In\, R - \left(\frac{1}{n}\right) In\, r_1 r_2 \ldots\ldots\ldots r_n} \qquad \text{Eq (36)}$$

Where;

Q_{tot} is the total quantity of water to be pumped out

n the number of wells

h is the groundwater level at P

$r_1 r_2 \ldots\ldots r_n$ are distances from well #1, #2…to point P.

If the wells are arranged in a circle the equation (36) changes to a simplified form shown below,

$$Q_{tot} = \frac{\pi k(H^2 - h^2)}{In(\frac{R}{R_W})}. \qquad \text{Eq (37)}$$

It should be mentioned that there is no reliable equation for calculating Q_{tot} for a confined aquifer or non-equilibrium free aquifer. Thus numerical simulation are used to find Q_{tot}.

2.12.2 Numerical Models for estimating mine flow

The main use of numerical (mathematical) models is to simulate the physical processes occurring in a hydrogeological system. This is usually done by a computer. Thus sometimes referred to as computer modeling.

A computer model can be set up to simulate inflows to the excavation over time as the quarry or mine enlarges. It can also be used to model the effect of say a ring of dewatering wells. More importantly, it can be used to determine the number and depth of dewatering boreholes or well points or a combination of both needed to draw water levels down below the planned base of the excavation. The main advantage of the numerical method over analytical methods is the ability to simulate three-dimensional flow covering great ground in a heterogeneous aquifer system. This gives a more representative detail of the groundwater system in a mine site and leads to more accurate prediction (Lloyd and Edward, 2000).

Numerical methods have yielded more success in the mining environment than analytical methods. Analytical methods have had little success in the mining application due to the complexity of the aquifer systems in the mining environments. Below are some of the widely used computer codes by mines:

- Modflow: - this is a U.S Geological Survey modular finite-difference flow model, which is a computer code that solves the groundwater flow equation. Used to simulate the flow of groundwater in aquifers. The family of modflow related programs are capable of simulating groundwater/surface-water systems, solute transport, variable-density flow, aquifer system compaction and land subsidence, parameter estimation and groundwater management.
- FEFLOW: - this a computer program for simulating groundwater flow, mass transfer and heat transfer in porous media and fractured media.
- UNSAT2: - this is a two-dimensional finite element model for horizontal, vertical, or axisymmetric simulation of transient flow in a variably saturated, non-uniform anisotropic porous medium

Modeling is usually done in three stages. At stage one the initial model is set up and used to plan the initial dewatering of the excavation. A trial dewatering system is set up and monitored as the initial cuts are made. At stage two the model is calibrated with real data and its accuracy enhanced. Stage three is the interactive use of the model throughout the life of the quarry or mine. (Naish, 1993)

2.13 Mathematical modeling of the Konkola Mine aquifers

The Konkola aquifer systems is one of the most complex in the world. Konkola Mine tried six calibration procedures from 1964 to 1988. The calibration procedure was used to simulate the three-dimensional character of flows at the mine site. The computer code which was used was MODFLOW. MODFLOW was selected because it gives a three dimension and strong transient, character of subsurface water flows and corresponding water pressures in the vicinity. It also depicts the presence of surface features such as rivers, lakes, streams, swampy areas etc. which serve or have the potential to serve as recharge sources. It also takes into consideration the complex nature of the local stratigraphy, fault structures, the position of the orebody, the mining method being used and the alternative mining methods to be investigated. As if this is not enough, it also takes consideration of the substantial spatial variability of material properties within the water-bearing formations and immediate surrounding formation. Lastly, because the domain is in the public view, it offers easy close scrutiny by review boards and peers.

The aim for this modeling was to simulate the response of complex water-bearing formations, or aquifers at the mine site to drainage and dewatering stresses. Mathematical modeling had three objectives:

- To adopt MODFLOW or any other software to specific three-dimensional hydrologic conditions in and around the mine site.
- To calibrate the model hydrologic conditions representing pre-mining to current dewatering practices. It is very important to accurately validate the pre-mining condition. Calibration is very important as it gives a reliable prediction of the known. This is based upon pre-mining or historic measurements and hydrological behavior.
- When the calibration is successful, then the calibrated model is applied to predict the future changes in water levels and dewatering stresses. The potentiometric heads in and around the mine site are caused by the projected dewatering rates. This was done to enable the design of a robust dewatering system where boreholes and pumps are located in areas which yield optimal drawdowns to enable safe mining.

2.13.1 Technical Approach

The building of the mathematical model at Konkola Mine involved eight rigorous steps. These were:

- ➢ Step one: this involved the gathering of historic data of the mine site. Site-specific data was assembled, this included geologic horizons as a function of vertical position, stratigraphic description of the water-bearing formations, parameter values and ranges for hydrogeologic variables, measurements of potentiometric heads at different mine locations and elevations and as a function of time, pre-mining hydrological conditions in as much spatial detail as possible, historic mine dewatering operations and mine water inflow rates and future projections in mine dewatering plans. The lack of early dewatering operation of the mine had limited the success within which the model could be calibrated to early periods at Konkola Mine.
- ➢ Step two: this involved the selection of a suitable framework to represent the site condition. The model domain was selected together with a framework of five successive layers. The layers contained the geological component from each formation, the surface recharge zones

known, the fracture zones, high permeability zones in the anticlines and water-bearing formation composite zones. The known conditions as completely as possible were represented by the selected vertical extents and boundaries of the model domain orientation.

- Step three: the numerical grid cell system was selected was suitable for predicting flow rates and dewatering scenarios on the site. The extent of the fine layers accommodated known mining sequences from 1955 to 1988. The numerical grids cell systems were subdivided into variably spaced rectangular cells of width 200m to 800m.
- Step four: MODFLOW data package was employed at this stage for simulation of dewatering operations during both the calibration and study stages. It was at this stage were pumping rates were inserted into the cells which reflected the observed responses of the aquifers to drainage boreholes.
- Step Five: this was the review stage and the selection of the time spun for model simulation. The review stage was based on the assembled historical data. The time spun looked at the changes in drainage locations, rates and duration which were known to have occurred. A five year time spun was taken and provided the starting of the next five-year run. But for stress durations, a period of one year was selected for each run. The stress period was traversed by twelve uniformly sized time steps representing month durations.
- Step six: data for model calibration was selected. Hydrologic property values lying within a suitable range were selected for each material along with dewatering data for each layer in a three-dimensional framework. The data which did not fall within the range was not selected for the model calibration. Carefully selection of the measured values of a potentiometric head was a basis for evaluation of a successful model.
- Step seven: a model calibrations was undertaken for a span of time representing historic dewatering operations undertaken at the mine. Five separate processes of calibration failed because of the complexity of the dewatering system. The five processes managed to calibrate some spans but not all. Only the sixth procedure worked as it calibrated a spun off five years from 1955 to 1988.
- Step eight: Application of the model in predicting the future. At this stage, the future dewatering was predicted. A scenario was selected to represent a dewatering location, rates and schedules to provide desired dewatering to reduce water level to desired levels which

were presumed suitable for mining. Six spans were run each approximately 5 years which gave a simulation of dewatering from the year 1989 to 2020.

2.13.2 Modeling Assumption taken at Konkola Mine

Assumption was made to overcome the deficiencies in detailed knowledge about spatial and temporal variations in the regional hydrogeology at the site. It was assumed that geologic units especially their stratigraphy representation was hydrologically connected to each other. It was also assumed that the water pressure was continuous in these units. Pathways were represented by numerical grid system which accommodated several different material types. The fractured zone within the specific geologic units was represented using the porous media approach. Boundary condition on the model boundaries for four layers which lay consecutively beneath each other were considered as identical in the model and carried the same properties of the 1st layer. The pattern of recharge and discharge through the boundaries of the model domain were un-altered with the vertical depth. Vertical recharge was applied as fluxes to each cell of the layer 1. The fluxes are constant in time but spatially variable.

2.13.3 Model calibration at Konkola mine

The selected procedure made use of the multi-layer approach but it restricted the number of layers during simulation only to those within which simulation dewatering operations are current. During the first time span which ended in1974, only layer 1 was simulated. The second time span ended in 1981 both layers 1 and 2 where simulated. The third time span ended in 1989, were the third layer was simulated. The model constituted five layers covering different depth of the mine as shown below;

- Layer 1: it was calibrated in 1974. It represented 0m to 660m (0ft to 2200ft)
- Layer 2: Calibrated in 1981 and represents data from 660m to 876m (2200ft to 2900ft)
- Layer 3: Calibrated in 1988 and represents data from875m to 950m (2900 ft to 3150ft)
- Layer 4: This represents the depth from 950m to 1180m (3150ft to 3900ft)
- Layer 5: This represents the depth of 1180m to 1405m (3900ft to 4650ft

Layer 1 was considered to be unconfined aquifer while the rest of the layers were considered confined until dewatering. In each layer, a large number of material properties were defined and modeled. The potentiometric head measured well with any point in the past thus giving success in calibration. The calibration success was quantitatively assessed for point measurement when the difference between calculated and measured was in the range of ±20m. If within range the model was successful. The range may seem large but it was dictated by the shape of the dewatered zone and the steep gradient of the potentiometric heads at its outer edges. The smallest cell was 200m by 200m thus it was impossible to achieve a closer match without adapting the implausible distribution of hydrologic properties.

2.13.4 Application of a Calibrated Model at Konkola Mine

When the sixth calibration was successful, the projected dewatering operation was made for the period of 1964 to 1988 and 1989 to 2020 as shown in appendix 4. This was dived into two simulation time frames. The first span involves layer 1, 2, and 3 began in 1989 and ended in 1998. The second span involved lays 2, 4 and 5, this continued up to 2020.

2.13.5 Model Update

The model has undergone several updates, the first updates was in 1995 to include the Konkola Deep Mining Project (KDMP) production options by ZCCM and in subsequent years the model was updated (2001, 2006, 2008 and 2013) to include latest data from the latest operations and modifications to the mining and schedule of dewatering. In 2013, the code was updated to new vision called MODFLOW-VKD. VKD stands for variable hydraulic conductivity with depth. The modifications beyond those documented for the VKD version of the code include the capability to change the hydraulic properties of the formations over time (Environmental Agency, 2003). This capability was required to represent potential changes to the hydraulic connection between different water-bearing formations due to the drilling of drain boreholes.

2.14 Selection of a Dewatering Method

In the selection of a dewatering method, there are key factors that are considered, these are;

- Mine Plan- this looks at how deep the mine is, how fast you want the aquifer dewatered and the shape of the aquifer
- Hydro-geotechnical Conditions- this looks at regional properties and relates them to the stability of the mine.
- Hydrogeological Conditions- this looks at Permeability (k), Storativity (S, Sy), recharge and through flow.

2.14.1 Mine Plans

This considers the constraints of mine planning which is mining. Mining looks at how to achieve what is possible, practical and safe. The dewatering method has to make mining practical, possible and safe. The depth of the mine is looked into and the rate at which the water table must be lowered too is predetermined. Besides this, the volume of water to be removed and whether pumping equipment can be installed in operational areas are considered in this phase. The availability of drilling and dewatering equipment is considered and also the contractor experience is looked into as a key factor.

This mining condition can (and do) change regularly and often at short notice. The mine planners normally have direct input to (mining) Mine Plan

2.14.2 Hydro-geotechnical Conditions

The selection method also looks at the impact of mining on hydro-geotechnical conditions such as:

- Blasting: induces “damage” in the sub-blast zone (increased permeability (k))
- Lithostatic unloading: this as a result of overburden removal which leads to a decrease in total stress leading to dilation/expansion (and increased k) and also an initial drop in pore pressure as voids open up.
- Hydrostatic unloading: this is as a result of dewatering of overburden

2.14.3 Hydrogeology Conditions

Here are the main hydrogeological conditions are considered:

- Permeability (k): this looks at pumping rates and borehole/drain separation. It also considers when to start dewatering (or drainage) because different rocks have different permeability so it’s important to quantify the permeability and start dewatering as per mine plan so that the dewatering process is not behind the mine plan.
- Storativity (S, Sy): this is the volume that needs to be pumped or the volume of water that a permeable unit will adsorb or expel from the storage per unit surface area unit change in head.it looks at residual moisture (how will drained material behave) too

- Recharge and throughflow: this looks at the inflow of water into the aquifer and the hydraulic conductivity of the strata

These can't be changed (mostly) – but the mine has to live with them

The characteristics of the water-bearing formation that must be determined before designing a dewatering system are:

- Whether the aquifer is confined or unconfined
- Transmissivity and storage coefficient of the aquifer
- Static water level
- Seasonality of potential inflows
- Depth and thickness of the aquifer
- Sources of recharge to the aquifer and location of these sources

The design of a dewatering system will be specific to the mine under scrutiny. To achieve the best method of dewatering a multi-disciplinary approach is recommended making the best use of management and mine engineers. This requires the collection of important information on the mine or quarry including:

- Dimensions of the area to be dewatered
- Depth to which the water levels must be lowered
- Plans for disposal of the water removed
- Whether the installation will be permanent or temporary
- Quality of the water that has to be removed

Once all the criteria have been assembled a scenario for dewatering can be designed. The effectiveness of the method chosen can be predicted through the use of a computer model.

2.15 Aquifer Characterisation

Aquifer characterisation is the prediction or assignment of aquifer parameters/properties, such as hydraulic conductivity (k), porosity and storativity. Many researchers have shown that these properties show a variation in results when it comes to aquifers with different stratigraphic units or aquifers of different rock formations (Ritzi et al., 2000).

This heterogeneity is controlled largely by the distribution of sedimentary materials in the aquifers (Ritzi et al., 2000). When determining the properties of such aquifers, the variograms correlation lengths should be independently evaluated for each stratigraphic unit instead of taking a wrong approach of generalization of the results of one unit on the whole aquifer. The results of the stratigraphic unit can be generalised within the same unit.

The mean and variance of the aquifer parameters, such as hydraulic conductivity vary by stratigraphic unit and thus not globally stationary. All geostatistical methods used should be applied to a stratigraphic unit in order to get effective estimates of aquifer parameters.

High-resolution borehole geophysical technology is used to measure continuous physical properties that can be used to delineate lithology and define stratigraphic correlations. Which can be used to include the heterogeneity in conceptual models (pore pressure distribution, effective and total porosity).

2.16 Subsidence

Subsidence can be defined as the gradual sinking, or abrupt collapse, of the rock and soil layers into an underground mine opening. Subsidence is caused by the caving and open stoping mining methods. Structures and surface features above the subsidence area can be affected. The following are some of the effects of subsidence:

- Open cracks and fractures on the ground surface;
- Restricted land use in areas of severe subsidence;
- Potential impacts on surface water sources as fractures propagate leading to surface water bodies seeping into the mine;
- Potential drainage of shallow and deep aquifers into the mine, impacts on vegetation and wildlife;
- Impact on water quality as a result of interconnectivity of aquifers;
- Interconnection of several aquifers;
- Increased mine inflow, a large zone of influence;
- Impacts on water wells in large areas;
- Increase in ore dilution;
- Potential air blast;

- Increased possibility of pillar run; and,
- Impacts on mine safety due to increased water inflow and weakened regional support.

There are two types of subsidence, these are continuous and discontinuous subsidence. Continuous subsidence is also known as trough subsidence. This involves the formation of a smooth surface subsidence profile that is free from step changes. It is the downwards deflection of the overlying and adjacent rocks towards the mined opening. This occurs when the relationship of the mining depth and caning-induced stress to the strength of the rock overlying the orebody is such that the fractures and discontinuous movements immediately move into the vicinity of the orebody. Continuous subsidence usually occurs in coal mining using Longwall mining method and this is usually caused by the mining of thin horizontal or flat dipping Orebodies overlain by weak, non-brittle sedimentary strata. The limit of the trough subsidence is defined by the angle of draw or limit angle. The angle of draw is defined as the angle of inclination between the vertical at the edge of the workings and the point of zero vertical displacements at the edge of the trough.

Discontinuous subsidence involves a wide range of mechanism which develops suddenly or progressively, this can occur on a large scales characterised by large surface displacements over limited surface areas and the formation of steps or discontinuities in the surface Profile. The angle of cave or the angle of draw limits the extent of the subsidence, for most Copperbelt mines it is between 60° to 75°. For Konkola mine the angle of draw is 65°. (Straskraba and Abel, 1994). There are different forms of discontinuous subsidence as outlined below

Crown hole: this arises from the collapse of the roof of the opening also known as the crown. These are regarded as special form of Chimney subsidence to be discussed latter.

Pillar Collapse: this usually occurs as a result of a pillar collapse due to increased load or due to deterioration of the pillar strength with time. When this occurs at a large scale pillar collapse causes catastrophic effects to the mine.

Chimney caving, **Piping or funneling**; this involves the progressive migration of an unsupported mining cavity through the overlying material to the surface. Chimney caving was responsible for the Mufulira mine disaster on 25th September, 1970. In which 89minners lost their lives. 450, 000$m^3$of tailing flooded part of the mine. A Chimney cave propagated upwards about 500m which connected to sublevel caving mining area.

Plug subsidence: this is a chimney variation which the overlying rocks slide in the opening as a result of a plane of weakness controlled by a dyke or fault. This usually produces an air blast.

2.17 Backfilling Mining Method

Most of the mines in the world are moving from caving and open stoping mining methods to backfilling mining methods because of the great benefits offered by backfilling methods. Backfilling methods offers increased ore recovery as the rib pillars can be mined out after backfilling the stopes. Besides this, the backfilling methods offer benefits of reduced environmental impact and a decrease in dewatering as most of the inflow paths are blocked by the backfill.

Backfilling methods reduce the potential of subsidence for the overlying and adjacent rock to a mine opening or stope as well as the subsidence of the surface. This reduces the ground movement which results in a reduction in the number of fractures developing at a mine thus reducing hydraulic inflow paths into the mine. Backfill helps in reducing subsidence but it must be noted that backfilling does not eliminate subsidence. The amount of subsidence is a function of time (de la Vergn. J.N., 2003).

The reasons for the application of backfill ranges from providing regional support to disposal of waste rocks. A backfill is a form of artificial support to a mine structure intended to control both local, stope wall behavior and mine near field displacement as shown in figure 2.18. It imposes a kinematic constraint on the displacement of key pieces in the stope boundary, this prevents spatially progressive disintegration of near-field rockmass in low-stress settings. Both the pseudo-continuous and rigid body displacements of the stope wall rock induced by adjacent mining are resisted by the fill. If the fill is properly confined, it may act as a global support element to the mine structure by the fill-rock interface inducing deformation in the body of fill mass thus reducing the state of stress throughout the mine near field domain.(Bradly and Brown , 2005)

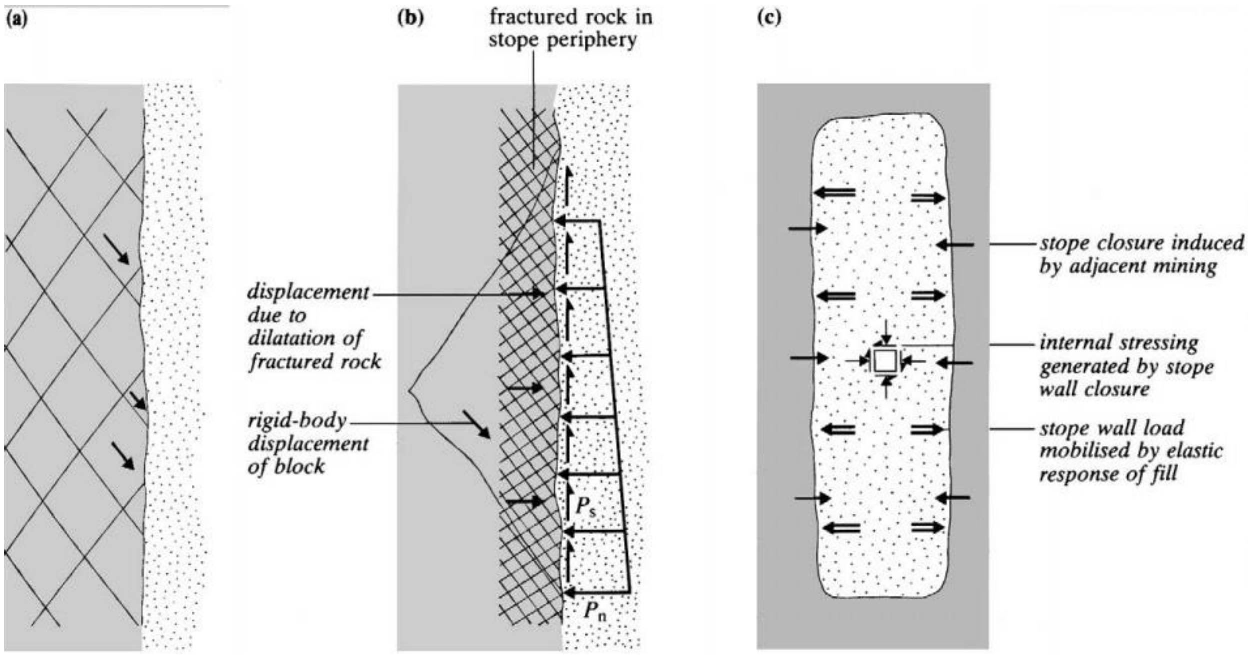

Figure 2.18: Backfilled stope (Bradly and Brown, 2005)

The only drawback to backfilling is the substantial increase in cost, depending on the type of fill to be used. The cost increases depending on the type of fill, the cost is slightly low in waste rock backfill but with paste and cemented fill the cost increases as infrastructure has to be built for the backfilling process.

The summary of the impacts of backfilling on a mine catchment area are as follows (Straskraba and Abel, 1994):

- Reduced hoisting of the waste ground as a result of the waste rock being used as part of backfill;
- Increased local and regional support;
- Improved mine `ventilation and refrigeration as a result of reduced air short-circuiting;
- Reduced groundwater inflow and reduced impacts on groundwater resource;
- Reduced damage to surface structures as a result of reduced ground movements;
- Low potential for impacts on surface water sources;
- Limited impact on the Hangingwall aquifer;
- Smaller or fewer tailing disposal facilities; and,
- Better land use of undermined areas.

2.17.1 Type of Backfill

There are seven types of backfill being used in the mine, these are:

- Rock Fill;
- Cemented Rock Fill (CRF);
- Concrete Fill;
- Paste Fill;
- Hydraulic fill/Sandfill;
- Cemented Hydraulic Fill; and,
- Ice Fill.

Rock Fill: - this involves the application of waste rock from development ends as backfill. The waste rock is backfilled into an open stope. LHDS and Dump trucks are used to transport and dump the material into stope. This is one of the cheapest backfilling methods as the material is readily available underground. This does not form a very rigid support wall thus some binders and cement may be added to form a cemented waste rock fill. This is not widely practiced but at Mindola sub-vertical shaft at Mopani copper mines, waste backfill is used to fill the VCR open stopes.

Cemented Rock Fill: - this originally consisted of spraying cemented slurry in a stope filled with waste rock but nowadays the cemented slurry is added before or as the waste rock fill is being placed in the open stope. Though more expensive than rock fill, it offers more advantages which include high strength due to cement content and provides a stiff fill that contributes to regional support. This was used at Cosmos nickel mine in Western Australia.

Concrete Fill: - this is a pre-mix and ready to use package. This is made of ready-mix cement concrete with a 10%-20% cement content.

Paste Fill: - this is a composition of complete solids content of dewatered mill tailings and cement. This is a high solid content ranging from 75%-85% and has high fine content with at least 15% by weight passing a 20µm. Water control plays a huge and very important role in ensuring that the fine exist as a colloidal suspension which acts as a transporting medium for coarser fractions of the fill. Cement usually ranges from 1-5% depending on the use of the fill. There is a 40% to 70% of saving in cement use than other alternative options of fill according to Landriault (2001).

Cement helps in preventing liquefaction which may result from the high fine content. The fill has physical properties of a semi-solid, thus requiring higher pressure gradients to induce flow.

Hydraulic Fill (HF): - HF was initially made from mill tailing after passing through the cyclone to remove fines. The fill was pumped underground in pipes as a slurry and applied in the stope using a hydraulic suspension, thus the name hydraulic Fill. The slurry contained 55% solids by weight and in instances when the tailing has a high fine content alluvial sand is used instead. The HF were sand is used is called Sandfill. The sand-water slurry has a composition of 70% solids by weight.

Cemented Hydraulic Fill (CHF): - the lack of true cohesion restricts the scope for mining application of Sandfill. This is overcome by the addition of a cementing agent, the obvious choice being Portland cement. A portion of the cement is replaced by other cheaper binders like fly ash (Pozzolan), ground slag, lime, or anhydrites. The cement ratios are varied depending on the use of the fill. The solid content of the CHF is 70% by weight.

2.17.2. Properties of Backfill

The Table 2.2 below shows typical properties of structural backfill.

Table 2. 2: Structural backfill properties (de la Vergn. J.N, 2003)

Backfill	Density	Binder Content Cement + Ash (%)	Water Content (%)	Water Bleed (%)	UCS (MPa)	Max Open Height (m)
Concrete	2400	11	5	0	20	-
CRF	1900	5-6	4 -4.5	0	2-4	90
Paste	2000	3-6	8-21	0-10	0.7-3	60
H Density	1900	4-8	25-35	0-15	0.3-0.7	45
CHF1	917	3-5	35-45	0-25	0.2-0.4	20

The Table 2.3 below shows typical properties of hydraulic and cemented backfill.

Table 2. 3: Properties of Hydraulic and cemented backfill (de la Vergn. J.N, 2003)

Cement/Solids Ratio	Strength (28 days) MPa	Strength (28 days) psi	Application
1:05	3.8	550	Chute Backing
1:06	3.1	450	Flooring
1:07	2.4	350	Flooring
1:08	1.7	250	Flooring
1:10	1.4	200	Flooring
1:20	0.6	85	Bulk Fill
1:30	0.4	55	Bulk Fill
1:40	0.3	40	Bulk Fill
1:50	0.14	20	None

2.17.3 Impact of backfill on groundwater inflow

Backfill reduces ground movement and subsidence thus greatly reducing groundwater inflow into the mines. Straskraba and Abel (1994) researched and found that there was no metal mines using exclusively mining methods with backfill, which experienced significant groundwater inflows, and/or with any major problems related to groundwater inflow. Groundwater inflow in backfilling methods is significantly less than methods without fill. Backfill fills the voids thus reducing the fracture formation and greatly reducing hydraulic inflow paths. Besides this, the backfilled stopes help in preventing the hangingwall from caving and thus preventing the formation of fractures that would puncture into the Hangingwall aquifer or any other aquifer that is within the angle of draw.

With backfill, the zone of increased fracture permeability is considered reduced. Backfill can reduce the groundwater inflow directly proportional to the compressibility of the backfill in relation to the extracted volumes for mine whose groundwater inflow is dependent on the volume of disturbed ground. Straskraba and Abel (1994) found that the groundwater inflow can be reduced by 90% using the 10% compressible uncemented backfill.

The reduction in groundwater inflow in mines using backfill could significantly bring down the cost of mining and dewatering by reducing; the number of boreholes drilled, drainage drifts mined and water pumped. This offers great opportunities for Konkola mines to reduce its pumping costs.

2.18 Disposal of Mine water

The disposal of water at Konkola Mine is as follows: the water underground flows from the roofs backs, side walls and dewatering boreholes into a drainage which leds to the drain drives. Water flow under gravity to the drain drives. The water flows from the drain drives to the settlers under gravity, then from the settlers, the water flows to the sumps and then the water is pumped to the surface (350,000m^3/d) and some of it back into the mine as industrial water (100,000m^3/d). Once the water reaches the surface some of it flow into the discharge canals which channel it to the Kafue River. Some of the water is channeled to Mulonga water treatment plants through pipes, and this is the water which is used for domestic consumption in Chililabombwe.

The dewatering borehole location is suggested by the dewatering plan and the simulated model. The number of boreholes depend on the amount of water to be pumped out and is also controlled by the model and dewatering strategy. The dewatering development which includes dewatering crosscuts, boreholes and drain drives is controlled by the dewatering model and the amount to be pumped out as per the method' s calculations. The size of drain drives is controlled by the flow of water that is to pass through and the ventilation requirement. Drain drives/ water drives are used as return routes for ventilation for most mines.

The traditional way of disposing of water by most mines is done by pumping but most metalliferous mining operations contain a range of impurities and suspended sediments or solids which may reduce the life of the pumping equipment. As a consequence, it is necessary to separate sediments from the mine water before pumping. There are three types of mine water usually found in mines as runoffs, these are:

- Acid water: - this is usually formed when water comes in contact with sulphide minerals. Acidic water is corrosive and harmful to humans and flora and fauna. Regulations don't permit any mine to dispose of acidic water thus the water in such mines is neutralised by the use of lime and soda mixture of 75%:25%. The mixture eliminates encrustation (formation of $CaSO_4$) and neutralises the water to an acceptable pH. The Copperbelt has unique situations where the sulphides are surrounded by the dolomite which act as neutralising agents for the water exposed to the sulphides, thus maintaining a pH of 7 in the runoffs;
- Aerated water: - this is water containing dissolved gases. This happens because of inadequate ventilation at the settlers and pump house. The common gases that dissolve in the water are carbon dioxide, oxygen, nitrous or nitri gases, hydrogen sulphide and other

gases that can be released in form of bubbles. This is mainly caused by the inadequate suction pipe or suction pressure; and,

- Muddy water: - water containing small particles. Clay particles don't erode the pumps but girts particles do. The particles are settled because of both protection of the dewatering infrastructure and legal requirement.

Impurities are removed from the water by separation, this can be done by making the sediments to settle by gravity or by adding a flocculant which will bind the sediments and cause it to settle below the water. The flocculants are used in instances where the sediments are unable (low specific gravity) to sink to the bottom on their own. Lime can be used as both flocculants and a neutralizer.

The suspended solids in the water are lowered below the total suspended solids (TSS) limit is reached to protect the pumping facilities and to abide by the environmental regulation. The Statutory Instrument No. 73 of 1993 of the water pollution control regulation guides on this matter in Zambia. For Konkola mine the TSS limit is 100mg/l.

In order to efficiently keep TSS below the limits the settler design plays a major role. There are two types of settler designs for underground mines, these are vertical and horizontal settlers. A close to 0mg/l TSS can be only achieved by the Ideal settler. An ideal settler is a sump or container in which the water containing solids in suspension is at rest or is stagnant. The water behaves like a homogeneous liquid with high specific gravity. As the solids settle the specific gravity of the liquid reduces. There is a classification in the settling, the bigger particles settle first followed by the smaller less dense particles.

$$V = A*v*t \qquad \text{Eq (37)}$$

Where;

V is the maximum volume of produced water (m^3)

t is the time (s)

A is the superficial area of the Sump (m^2)

V is the settling velocity relative to water of the smallest particles to be precipitated (m/s)

Therefore flow is:

$$Q=A*v=V/t \quad (m^3/s) \qquad \text{Eq (38)}$$

This gives a maximum rate at which water of required clarity can be produced under ideal stagnant conditions. It should be noted that depth and retention time have no impact on the amount of clear water produced in a unit time.

2.18.1 Horizontal Flow settler

The dirty water is introduced in one end of a long sump and clean water discharges at the other end over a battery of overflow lip launders as shown in Figure 2.19. The overflow moves into the clear water sump. As water flows from one end of the settler to another, the coarse particles sink to the depth of equal specific density rapidly building a sludge at the inflowing end. The fine particle sludge forms under the overflow launders where almost wholly vertical flow exists.

When the sludge build up grows to the extent of covering all the ends of the settler and increases in height, an efficiency of materials settling reduces, thus causing a buildup of material in the clear water sump. In order to prevent wear and damage to pumps, the settler is cleaned. When cleaning the settler water is bypassed from settler into the clear water sump. The water in the settler is lowered, the sludge is agitated into a slurry by either high-pressure air or water jet. The slurry is either moved to the stope bellow the settler by gravity or pumps to a higher level by a plunger pump. When the settler is clean, the water is re-introduced and the clear water sump is cleaned.

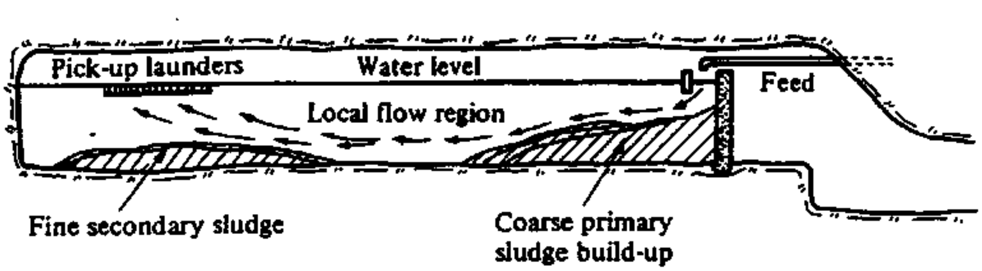

Figure 2.19: Horizontal Settler (Vutukuri and Singh, 1993)

The dirty water is led in the bottom of the settler at such a rate of velocity were the velocity of the rising water is less than the velocity of the sinking particles. The denser particles sink first to the bottom followed by the fine particle which settles on top of the coarse particle according to their specific densities. The sludge is very sensitive to the flowing current of the water.

2.18.2 Vertical Flow settler

There are two types of vertical flow settler one with one wall and the other with two walls. The two types of vertical settlers are shown Figure 2.20.

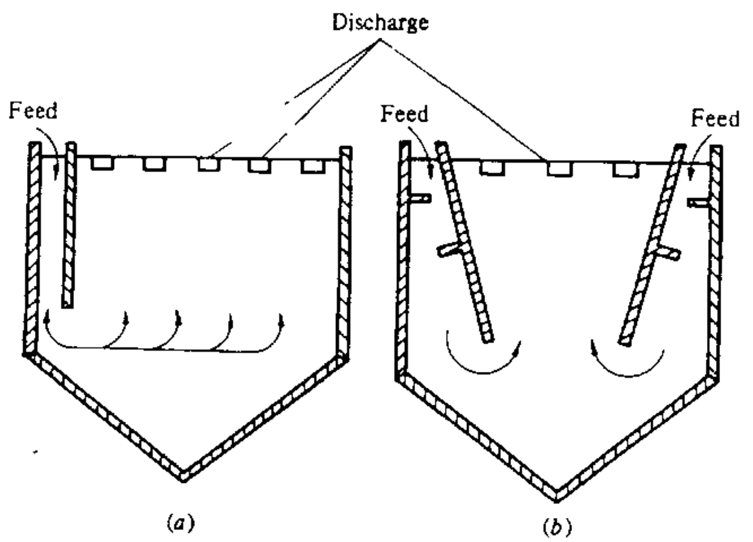

Figure 2.20: Vertical Settler (Vutukuri and Singh, 1993)

A more stable condition is attained with design (b) but may be uneconomic to build thus a compromise may be used (a) (Vutukuri and Singh, 1993)

The sludge buildups in the settler are cleaned as the slurry flows by gravity to the stope below.

2.18.3 Flocc Bed Settler

This can be a single or double wall vertical settler which has been modified to facilitate the use of flocculating agents. This use of flocculating agents enables the mine to avoid large settling areas underground thus reducing the development costs. Flocculants are long-chain polymers which bridge between solids in suspension resulting in agglomerates of entangles particles. The floccs or agglomerates is usually of 1.5mm and settles at a velocity of 500mm/minutes. The flocc form rapidly and reduces in the concentration of the fluid rapidly as well. The remaining floccs can be trapped by passing the fluid upwards through a filter bed of previous flocculated particles.

2.19 Pumping Systems

A pumping system consists of the following subsystems: drainage to sumps; drainage to settlers; section and delivery ranges; pumps; and, its monitoring and control units. These maximize the efficiency and economy of the entire pumping systems.

The pumping system for the mine plays a crucial role in the dewatering plan of a mine and special care is taken in the design. When designing a pumping system, every mine desires a high operational efficiency and low maintenance system. Besides this, the other desire is for the overall cost to be low and have the capacity to deal with seasonal inflows fluctuations, mechanical and electrical breakdowns including power cuts/failures and sudden increase in water.

When designing a pumping system, the mine pump must be related to the proposed duty. This requires a number of factors and must be considered in the design. The key factors are water inflow, water quality and mine layout and development.

Konkola mine has three pump chambers (370mL, 690mL and 985mL) at No 1 shaft housing 60 HPH high-lift centrifugal pumps (Sulzer). These pumps have a capacity of pumping about 586, 800m^3/d but most of them have low efficiency and some have broken down. Table 2.4 shows the types of pumps used at Konkola Mine.

Table 2.4: Pump specification deployed at Konkola Mine

Pump Chambers	Pump Type	Number of stages	Impeller Diameter (mm)	Capacity		Head (m)	Number of Pumps	Purpose of Pump
				m^3/hr	m^3/d			
370mL	Sulzer HPH 54/25	4	540	815	19560	>400	15	Lift to surface
690mL	Sulzer HPH 54/25	4	540	815	19560	>401	7	Lift to 370mL
	Sulzer HPH 54/25	8	540	800	19200	>750	8	Lift to surface
985mL	Sulzer HPH 54/25	4	540	815	19560	>400	8	Lift to 690mL
	Sulzer HPH 54/25	7	540	725	17400	>700	8	Lift to 370mL
	Sulzer HPH 54/25	9	580	839	20136	>1000	14	Lift to surface

CHAPTER 3: METHODOLOGY

3.1: Introduction

This chapter discusses the various research methodologies which were undertaken in order to meet the objectives of the study. The research methodologies used to collect the data are: desktop studies, site visits both underground and surface, interviews with Mine Officials, and review of the mine dewatering records/ technical reports. The software used to analyse the data are Microstation, Surpac Geovia and Mudflow VKD.

3.2 Desktop Studies / Literature Review

A deep and thorough literature study was conducted to fully understand the dewatering methods. Several articles on dewatering of Konkola and other Mines were studied. The Copperbelt and other global dewatering approaches were also reviewed. Old and latest Consultant's findings were also reviewed at the Amano center located at Konkola Mining Training School in Chililabombwe. Besides this, books and journals on dewatering and subsidence were reviewed.

Backfill literature was reviewed and effects of backfill on hydraulic inflow paths was studied. Mines that use backfill were studied in order to determine the effects of backfill on the dewatering requirement.

3.3 Site Visits

Several site visits were undertaken at Konkola Mine. Visits to the following areas were done:

- Dewatering crosscuts- Dewatering crosscuts where visited at 800mL (4500mN, 4300mN, 4050mN, 3700Mn) and 950mL (3800mN, 3350mN, 2700mN, 2450mN, 2120mN, 800 mS, 1000mS and 1200mS) in order to measure pressures and flow rates from the bore holes. The method of drilling of the boreholes is as outlined in section 2.7. The flow rate is used to calculate discharge while the pressure is used to measure the depth of the water table. The pressure and flow rates were measured as follows: first the valves on the boreholes were closed for 15 minutes, then the pressure gauge was fitted onto the valve, Figure 3.1 shows the author closing the valve. The water was opened and the pressure was recorded. The flow rate for the boreholes was measured by recording the time taken to fill a container of a known volume. Then equation 39 was used to calculate the flow rate per second. Then the calculated discharge is converted into m^3/d.

Discharge (Q) = Volume (m^3)/Time (s) Eq (39)

Figure 3.1: Closing of a dewatering Borehole at 4300mN

- Drain drives, settlers, sumps and pump chambers- these were visited in order to familiarise myself with the way water moves from the crosscuts to the surface discharge points. The main aim was to familiarise myself with the underground dewatering circuits /channels and water quality controls.
- Discharge canals: these were visited to determine how water moves from discharge points on surface to the Kafue River and see any potential seepage points.
- The Kakosa stream, Lubengele stream, Lubengele dam and the Kafue River were visited to establish the potential seepage points as they cross the faults zones, caved areas and limestones formations. This was also to familiarise myself with the surface hydrogeology of the catchment area.

3.4 Weather Data

The rainfall data was collected using the two WatchDog 2000 series weather monitoring station installed at one shaft and three shaft. The WatchDog 2000 series provides full weather sensing capabilities, including sensors for air temperature, wind speed, wind direction, relative humidity, dew point, rainfall and solar radiation. Appendix 3 shows the weather monitoring station specification.

The rainfall records from 1956 to 2017 were collected from the hydrogeology department and are as shown in appendix 1. Figure 3.2 shows the weather monitoring station located at number 1 Shaft.

Figure 3.2: Weather station at No 1 shaft

3.5 Consultation with Mine Officials

Consultations with the geology, survey, and geotechnical departments was done. Several meetings were arranged during the data collection and analysis stages of the study with the long range mine planning and hydrogeology teams for guidance and review of the collected data.

3.6 Water profile

The water profile was made using the measured pressures from the dewatering bore holes. A smart line was generated using MicroStation as shown in Figure 3.3. Microstation is a CAD Software platform for two and three dimensional design and drafting. It generates 2D/3D vector graphics from objects and elements. The elements include lines, circles, arcs, French curves, point, text and more.

The depth for water table was calculated by converting the pressure into depth (m), this was done by multiplying the pressure (bars) by 10.

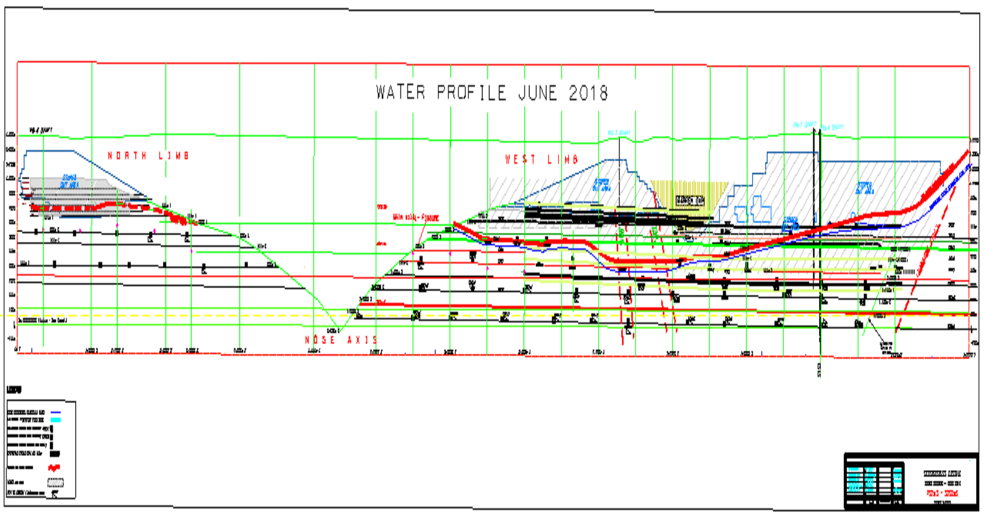

Figure 3.3: Water profile Smartline

Then the profile was transposed into a plan for the catchment area in MicroStation as shown in Figure 3.4. This is done in order to fit the generated smartline to the plan of the orebody. The smart line was then imported to Surpac GEOVIA and generated into a solid as shown in Figure 3.5.

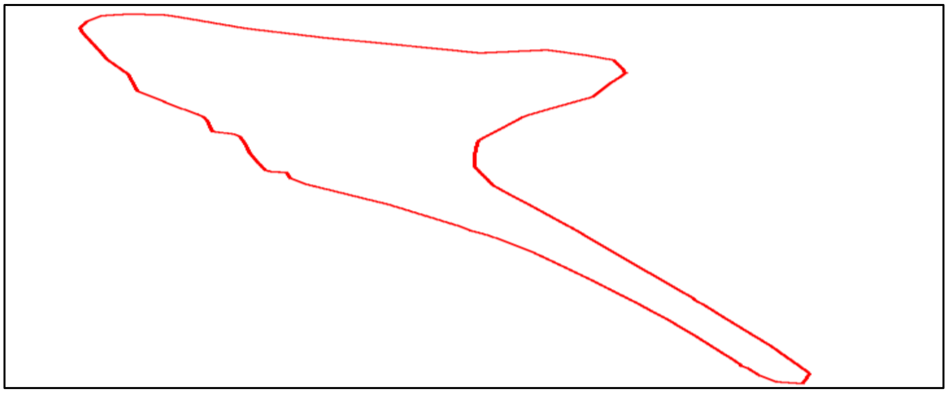

Figure 3.4: Plan

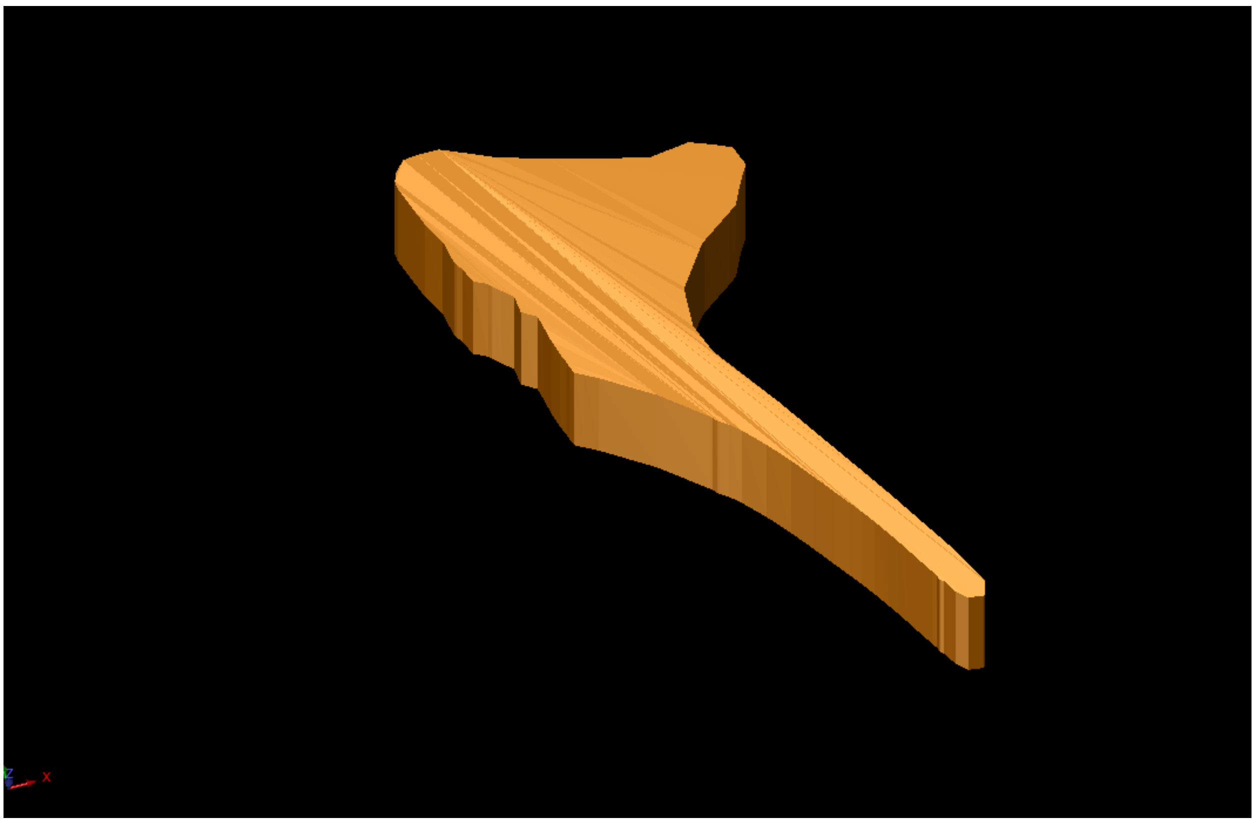

Figure 3.5: Surpac Solid

From the solid a three dimension water profile was generated. The dewatered and un-watered reserves are shown by inserting the ore reserves on the solid. The water table will demarcate the reserves showing which ones are dewatered and those which are not. From the solid, the water profile can be generated showing mined out areas, un-dewatered reserves and the water table level. The water profile is very imported as it shows the current position or level of the water table, which allows us to calculate the desired drawdown before simulating any scenarios.

3.7 Drawdown Simulation

Drawdown simulations where done using the MODFLOW-VKD software. MODFLOW is a Modular Three-Dimension Finite-Difference Ground-Water Flow Model. VKD stands for variable hydraulic conductivity with depth. The modifications beyond those documented for the VKD version of the code include the capability to change the hydraulic properties of the formations over time (Environmental Agency, 2003). This capability was required to represent potential changes to the hydraulic connection between different water-bearing formations due to the drilling of drain boreholes.

The drawdown for these possible dewatering methods (Conventional dewatering and Surface boreholes) were run. The targeted area for dewatering was the 1150mL cave line.

The simulation of various dewatering strategies which had the objective of meeting corresponding dewatering targets for the 5 Year Mine Plan were run. In all of the predictive scenarios which were run, the existing dewatering infrastructure remained operational and supplemented the new dewatering infrastructure for the dewatering alternatives. Each of the predictive scenarios also included the proposed development of tunnels on the 800mL, 875mL, 950mL, 1150mL and 1350mL during the five year plan, which have all been included in the model as drain cells. Several scenarios for both the surface boreholes and the conventional dewatering approaches were simulated with different borehole depths, location and borehole numbers.

An accelerated dewatering strategy was established by comparing the simulated drawdown for the conventional method and surface borehole method. The target area to be dewatered was 1150mL.

CHAPTER 4: DATA COLLECTION AND ANALYSIS

4.1 Introduction

This chapter looks at data collected and analysis. The main objective of this research was to determine a suitable method of dewatering the ore reserves below 1040mL. Surface Exclusion methods, Surface Deep Wells and Conventional Dewatering Methods were considered as potential dewatering methods. Backfilling mining methods were studied and their impact on dewatering are outlined. This chapter included the determination of the sources of water to the mine.

4.2 Sources of water to the mine

The study established three sources of water to the mine from the updated dewatering model, as; Rainfall infiltration, seepages from water bodies on the surface and water stored in aquifers.

4.2.1 Rainfall Infiltration

The infiltration from the rain into the mine catchment area is estimated to be about 20% of average rainfall. Rainwater infiltrates the catchment area directly across the region and via dambo areas created as a result of subsidence. The high infiltration of water into the mine is confirmed by the presence of an extensive amount of Saprolite (chemically weathered but return fabric of the decomposed parent rock) in the stratigraphy of Konkola which indicate the high possibility of rapid infiltration from the rainfall and the storage of large amounts of water in the Saprolites layers. The low thermal gradient of $1^{o}C/ 438m$ at Konkola mines against $1^{o}C/30m$ which is normal for other Copperbelt Mines shows that cool water flowing into the mine lowers the rock temperature to below the regional geothermal gradient.

The average rainfall for Konkola catchment area is 1274.3mm as shown in Table 4.1 which is a summary of Appendix 1. The average rainwater infiltration was found to be 20%, while 70% of the rainwater ended up in runoff and 10% evaporated (Water Management Consultants, 2000).

Average infiltration from rainfall= 20%* Average Rainfall

= 20% *1274.3mm

=**254.9mm/year**

The average recharge to the mine is 254.9mm for year on the groundwater catchment area for the mine which is estimated to be 240km^2 (Schlumberger Water Services, 2017).

The infiltration/d= (average infiltration*Area of catchment)/ 365days

$$= (0.2549\text{m/year}*240000000\text{m}^2)/365\text{d/year}$$

$$= \mathbf{167{,}605.5m^3/d}$$

The presence of dambos in the Konkola stratigraphy has contributed to the high infiltration of rainwater into the mine. The dambos located at 2900mw near Lubengele dam have made the area between 660m to 875m Levels un-minable, the drives which were mined in this area collapsed due to poor ground condition caused as a result of water. The dambos act as small lakes during rainy season as the get filled up with rainy water which seeps into the catchment area. The mine has a lot of dambos contributing to high infiltration of rainfall into the mine. Figure 4.1 shows a picture of a dambo and Figure 4.3 shows the location of dambos in Bancroft mining area.

Figure 4.1: Picture of a Dambo or Sinkhole

Table 4. 1: Rainfall Pattern Summary for Konkola Mine catchment from 1953 to 2016/17 rainy seasons

Month	September	October	November	December	January	February	March	April	May	June	July	August	Total Annual
Average Monthly Rainfall (mm)	3.9	27.9	141.7	277.8	291.9	255.4	215.3	52.9	7.4	0	0	0	1274.3
Average Evaporation (mm/month)	195.2	215.2	167.2	130.4	122.8	119.4	131.6	134.2	136	118.6	134.4	157.8	

4.2.2 Water Contained in aquifers

The aquifers release water into the mine as a result of decompression of the rocks and dewatering. This is caused by lithostatic unloading as a result of under burden remover due to the extent of mining. The lithostatic unloading leads to a decrease in total stress that results into dilation and expansion of the overburden rocks thus causing an increase in the permeability K. Water also flows from other distant catchment areas into the mine catchment area as a result of hydrogeological connectivity and fractures, the Lubambe water table has been lowered to about 400mbgl or 973mamsl due to the pumping at Konkola Mine. Fractures play a major role in water inflow into the mine. The fractures act as water conduits or drainages, once they intercept a water source. The fault zones are areas of high hydraulic conductivity, Konkola Mining area is sandwiched by three major faults as shown in Figure 4.2. These faults are connected to aquifers of the mine. The storage removal is estimated to be between 40,000-50,000m^3/d, this is controlled by the extent of mining and new developments but this, however, can increase to about 200,000 m^3/d depending on the extent of development. The storage removal is measured by chemical analysis of the water from crosscuts.

The Hangingwall aquifer and footwall aquifer contribute 80 to 90% of the water dewatered while the10-20% comes from the Lower Porus conglomerate and footwall quartzite aquifer.

4.2.3 Concentrated recharge from surface water bodies

Konkola Mine has several recharge zones because of its unique surface hydrogeology and the effects of subsidence on the surface water bodies. The mining area is sandwiched between three major fault zones (Lubengele, Luansobe and axial Fault zones) which are recharge zones in areas where they intersect a water source as these are connected to the mine aquifers.

The study identified the following possible seepage points into the mine catchment area, these are: Kakosa Stream, Kafue River, unlined canals, canal with damaged lining, Lubengele stream and Lubengele Dam which are shown in Figure 4.2.

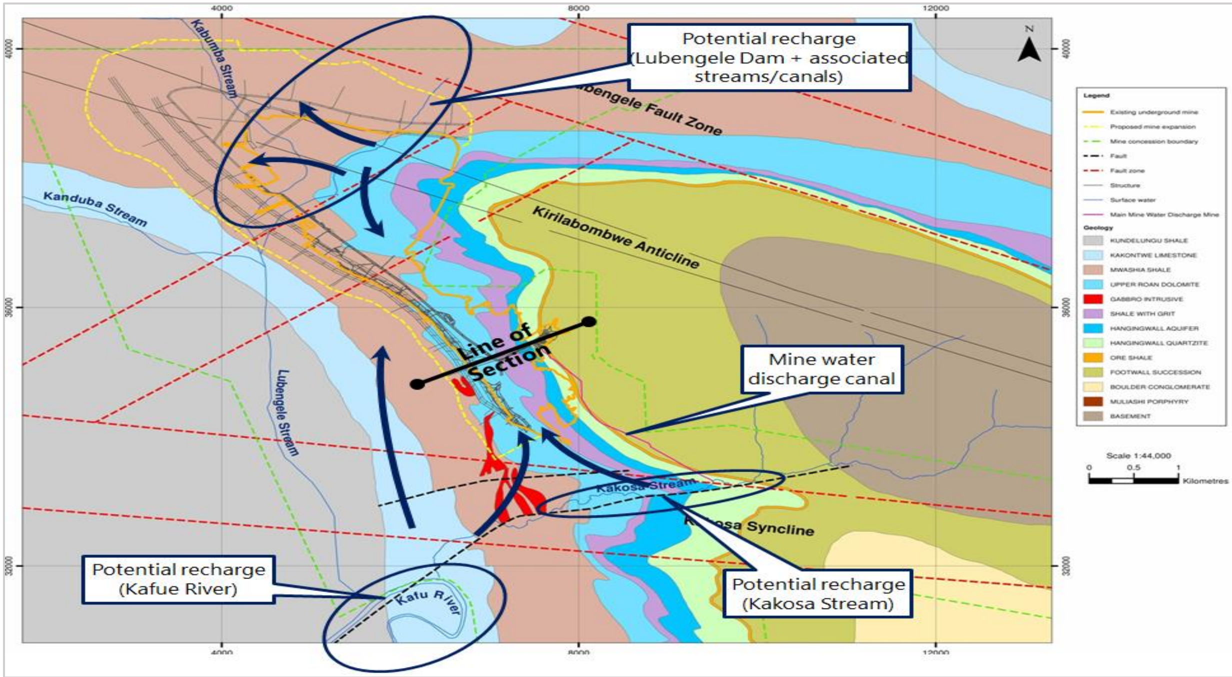

Figure 4.2: Seepage Points in the Konkola Catchment area

The recently developed groundwater model by Schlumberger water services indicates seepage of about 27,000m^3/day (possibly much more) from Kakosa Stream. Figure 4.3 shows concentrated seepage at the Kakosa stream.

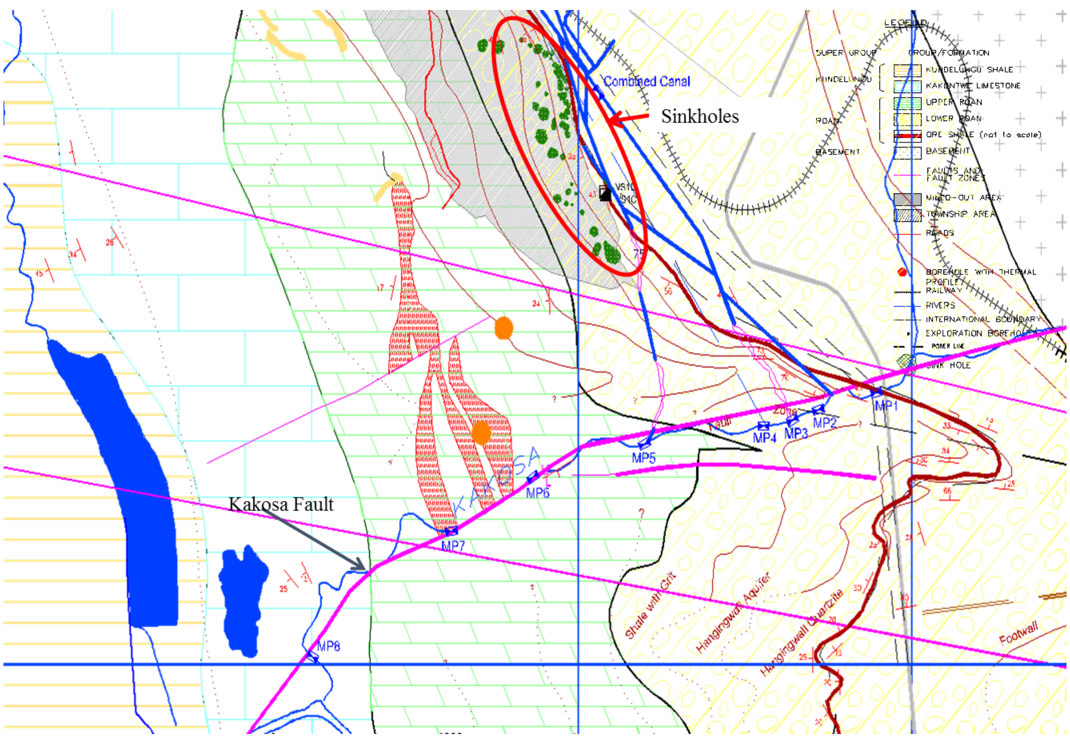

Figure 4.3: Possible Seepage at Kakosa Stream

The seepage on the Kafue River was quantified into two seepage limbs (eastern and western). The Kafue River Western Limb intersects the Kakosa fault which is connected to the mine aquifers and acts as a recharge conduit, about 58,840m^3/d of water seeps on this limb while about 93,230m^3/d seeps through the eastern limb which intersects the Kakontwe limestone (Karstic). A river discharge measurement approach was used to determine seepage. Teledyne Acoustic Doppler Profiler was used to measure the discharge between 16 sites on the western and eastern limbs. Measurements were made in a run or where the streamflow was constant and straight. The gains or losses in discharge between sites gave the estimate net volume of water exchanged between the river and the groundwater (Banda, Phiri and Nyambe, 2017). Figure 4.4 shows the western and eastern limbs seepage points.

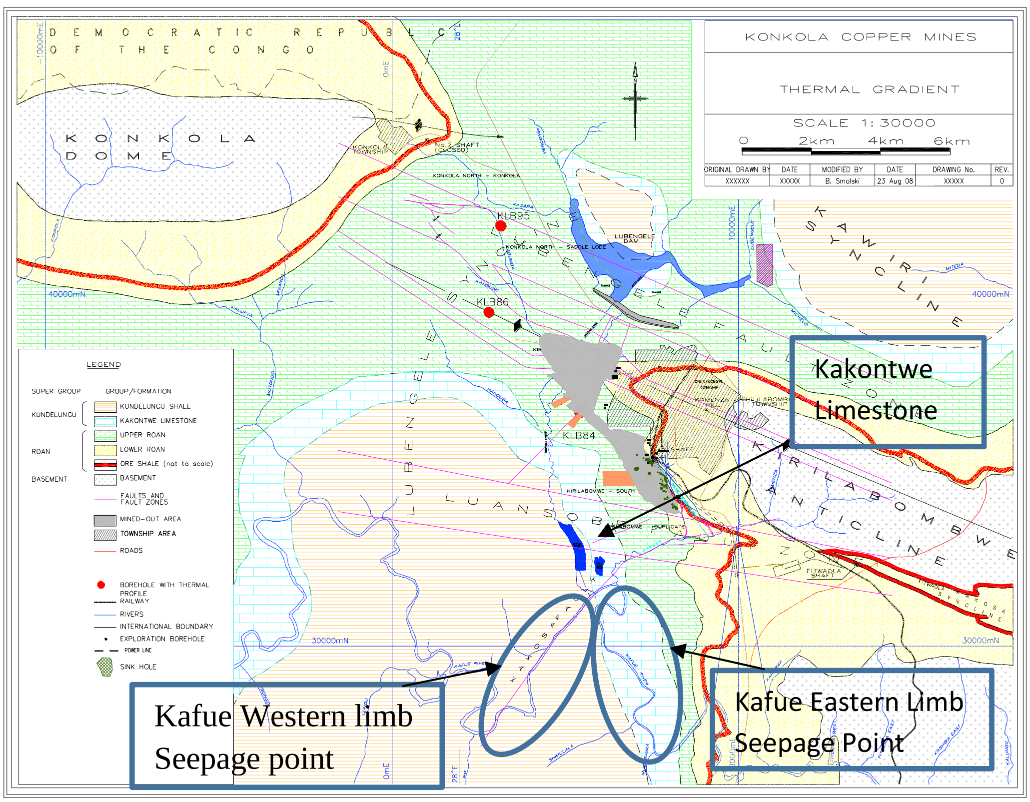

Figure 4.4: A map showing seepage points on the Kafue River

Besides these, the other sources of surface water seepages into the mine catchment area are the cracked and unlined mine discharge canals. The canals are suspected to be leaking water into the mine as they pass through the caving areas on potions which are cracked and not lined as shown in Figure 4.5 and Figure 4.6.

Figure 4.5: Channel Five Damage

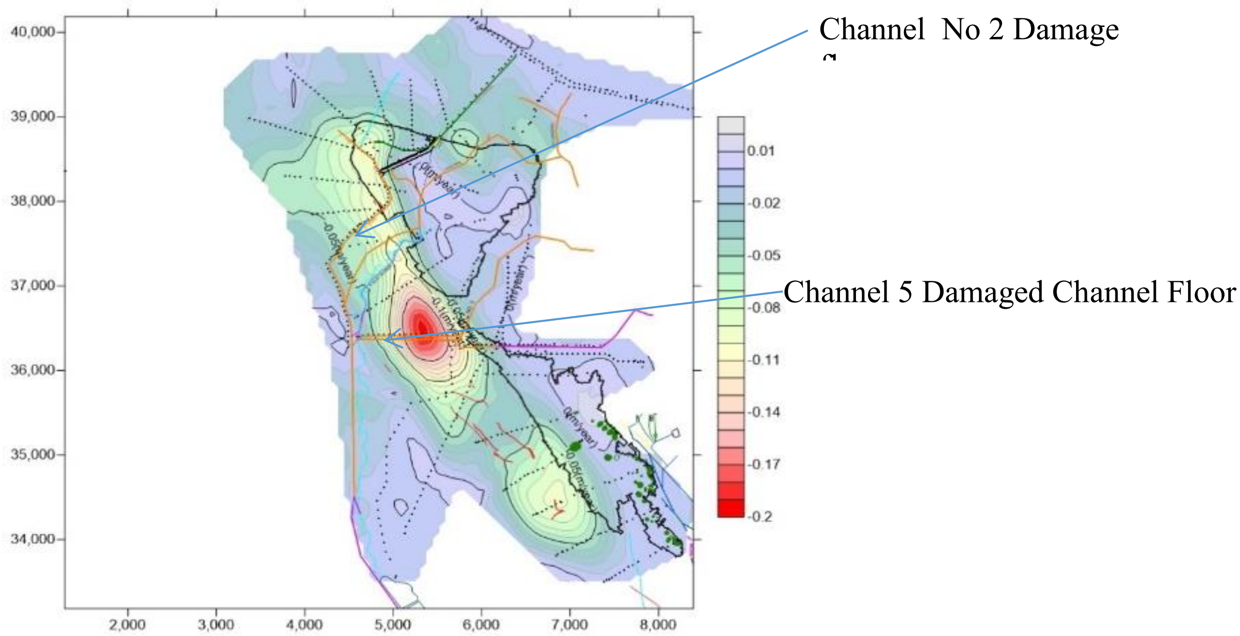

Figure 4.6: Damaged discharge Canals due to Settlements in the Catchment area

Water infiltrates into the catchment area through cracked portions of the discharge canal. The mine has got 22km of lined canals and 11km of unlined canals. The unlined canals have high hydraulic conductivity with the catchment area, especially when passing through the caving areas.

Subsidence has greatly affected the lined canals and most of them cracked as a result of tension cracks as shown in Figure 4.6 and Figure 4.7. Ground settlement is the lowering of ground surface, known as subsidence or settling, at Konkola Mine ground settlement of is high as 0.2m/year and as low as 0.04m affects lined and unlined discharge canals. The high ground settlement areas have high hydraulic conductivity.

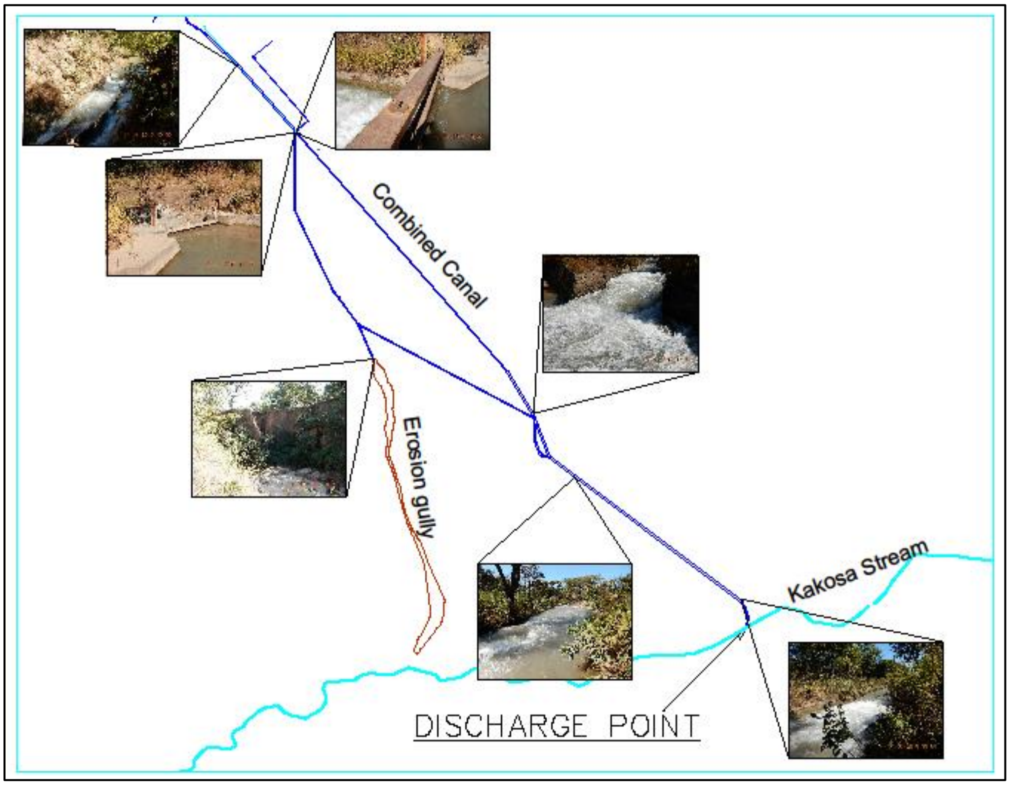

Figure 4.7: Combined Canal

4.3 Current Pumping Rates

The mine currently pumps about 350,000m^3/d of water, a trend of water pumped per day from 2002 to 2016 is as shown in Figure 4.8. All the water is pumped out of the mine through one shaft as shown in Figure 4.9, the blue arrows show water direction for areas that are on the same elevation with the pump chambers or above it, were mostly water flow under gravity while the green arrows show water direction for areas that are below the pump chambers in terms of elevation, for the green arrow water is pumped upward into the drain drives that lead to the pump Chamber at 985mL. The water flows from No 3 Shaft to one shaft through drain drives as shown in Figure 4.9. About 450,000m^3/day of water is received at the pump chambers and only

350,000m^3/d is pumped to the surface. The remaining 100,000m^3/d of water is pumped back to the mining sections to be used as industrial water. During a power failure only 200,000m^3/d is controlled by closing the boreholes while the emergency power from the backup generator is only capable of pumping about 200,000m^3/d and the remaining uncontrollable water 50,000m^3/d starts to flood the mine. Whenever there is a power cut, the Bancroft deeps floods.

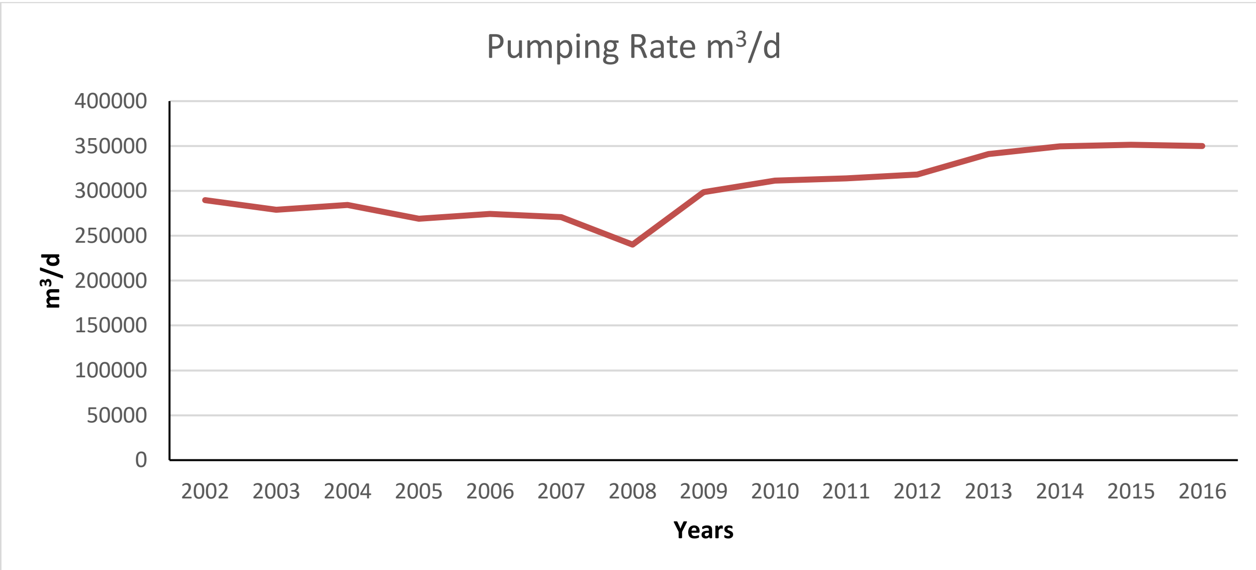

Figure 4.8: Pumping rates

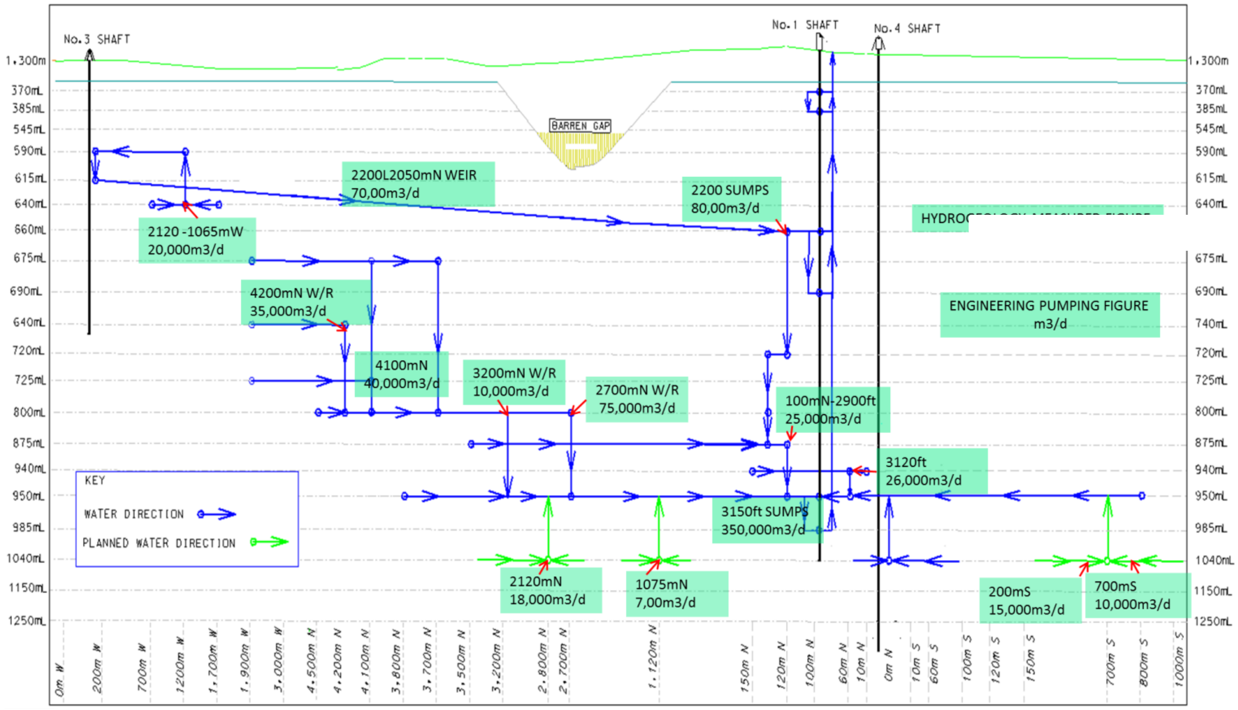

Figure 4.9: Water Drainage for Konkola Mine

The water to ore ratio is extremely high for Konkola Mine with an average of about 92t of water to 1t of ore delivered at the concentrator compared to the Copperbelt regional average 4t of water per 1t of Ore. The Figure 4.10 shows the trend pumped out water to ore delivered to the concentrator. December 2016 to February 2017, the production declined due to low equipment availability of trucks and Loaders.

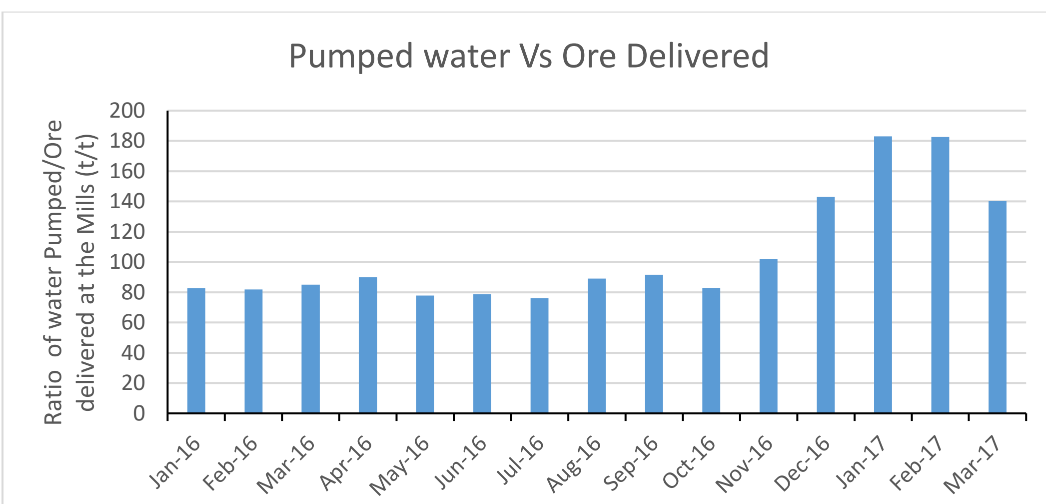

Figure 4.10: Daily ratio of pumped out water to ore delivered to the Concentrator

4.4 Methods of Dewatering

The Bancroft mining area dewatered reserves have about two year's life to depletion. Mining below 1040mL cannot take place before dewatering the aquifers that fall within the 1150mL cave line, this has led to the mine seeking a suitable dewatering method that will lower the Hangingwall aquifer water below the 1150mL cave line in order to avoid an ore gap. Three methods have been identified, these are: Surface Exclusion (also known as Surface water control), Surface deep wells and Conventional underground dewatering methods also known as Breakthrough method with underground boreholes.

4.4.1 Surface Exclusion Method

Surface Exclusion methods as the name suggests involves the exclusion of surface water from reaching or passing through the catchment area. This method does not lower the water table but it prevents recharge. The water from streams, rivers and other surface sources is prevented from passing through the catchment area (fractured) thus avoiding the recharge of the mine aquifers. Konkola Mine catchment area has a significant number of surface water bodies that recharge the aquifers. Table 4.2 shows the amount of recharge into the mine from surface water sources.

Table 4. 2: Recharge Quantities at Konkola

Source	**Recharge m^3/d**
Kakosa	27,000
Kafue Western Limb	58,840
Kafue Eastern Limb	93,230
Lubengele Dam	12,800*
Lubengele Stream	2,300*
Total	**194,170**

*Note: * No current recharge has been determined since the works done during the KDMP volume three report. The numbers used are as determined by the KDMP Vol. 3 report (Zambia Consolidated Copper Mines, 1995). Though these numbers may not represent the current seepages as the area affected by mining induced subsidence has increased as compared to the time the study was done but can still be used as a minimum.*

Table 4.2, shows that by surface exclusion alone a minimum of 194,170m^3/d of water can be excluded which accounts to about 56% of the daily pumped water. This number does not include the seepage from canals, hence more water can be excluded from the Mine by this method. It is important to note that surface exclusion method cannot lower the water table but can prevent recharge of the aquifers which can reduce the amount of water being pumped per day, hence this method cannot be used as an exclusive dewatering method but needs to be coupled with another dewatering method.

There are various ways the surface water bodies can be prevented from recharging the aquifers at Konkola Mine, these are:

- Stream or river diversion from fault zones and caving areas; Kafue river and Kakosa stream;
- Stream, river and discharge canal lining using flexible waterproof material like Geomembrane or concrete lining: Kafue River and Kakosa Stream;
- Lubengele Dam Sealing;
- Lined canal refurbishment; and,
- Unlined canals sealing or lining.

4.4.2 Surface Deep Wells

The surface wells target to dewater the hangingwall aquifer that is above the 1150mL cave line in the Bancroft mining area. The simulation of drawdown for this method gave a positive result in lowering the water table below the 1050mL elevation and also in avoiding the ore gap in the Bancroft mining area. This method involves sinking ten holes targeting the 1150mL cave line area.

This method requires comprehensive site investigations as a pre-requisite and are planned to take 6 months. The site investigations are critical for this method in order to acquire the geological, geophysical and hydrogeological data for the site. These studies will assist the drillers to accurately assess the drilling conditions of the area and associated risks. Besides this, the survey will also show a more detailed characterization of the lithology and groundwater flow condition. This will measure also, the fractures and bedding planes of the area where wells will be located and also, determine the approximate thickness of the bedrocks and location of faults, buried river channels, fissures and solution cavities. The studies to be done are listed below:

- Surface geophysics surveys for characterisation of ground conditions and structural discontinuities at target drill sites;
- Diamond core drilling of pilot investigation boreholes;
- Downhole packer permeability (Lugeon) testing;
- Downhole wireline logging;
- Installation of multilevel vibrating wire piezometers;
- Installation of time domain reflectometer equipment; and,
- Laboratory testing of geomechanical properties of main lithological units.

Three holes (13-inch diameter) are to be drilled during the site investigation at a depth of 1650m, one well is to be equipped with a submersible pump while the other two wells are to be equipped with piezometers. These wells will show the expected performance of the ten wells and any changes needed to be implemented before the ten dewatering wells are drilled.

Ten wells are to be drilled in the Bancroft area targeting the HWA. These wells are to be accurately drilled and positioned. The location of the holes plays a crucial role as they have to be located outside the caving area and in a position where they intercept the targeted portions of the HWA. The proposed location of the wells are shown in Figure 4.11.

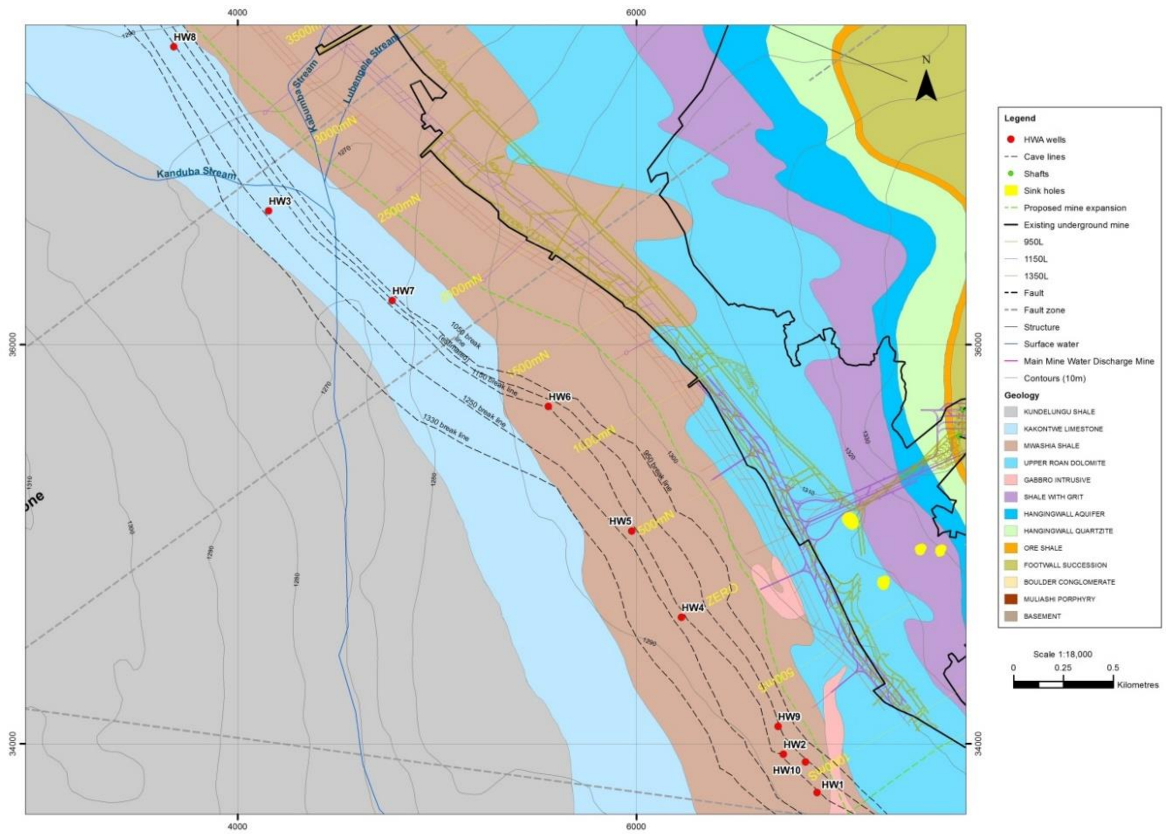

Figure 4.11: Map showing the Proposed Surface well location

The ten holes are planned to be drilled in 1 year six months by two oil drilling machines, drilling of one hole is planned to take about 3 months. The diameter of the wells is about 33cm and the expected flow is 50L/s. The depth of the holes are shown in Table 4.3.

Table 4. 3: Proposed Well depths

Proposed Dewatering Well	HW1	HW2	HW3	HW4	HW5	HW6	HW7	HW8	HW9	HW10
Depth to base of HWA according to SWS model (m)	1121	1191	1337	1413	1603	1584	1386	1124	1043	1049

All the holes with an exception of HW9 and HW10, are located in the 1150mL cave line, HW9 and HW 10 are located in the 1050mL.

The expected flow rates simulated using updated groundwater model for the wells are shown in Table 4.4

Table 4. 4: Proposed Well Flow rates

Well	Available drawdown assuming 30m water head in the well	Initial simulated pumping rate (L/sec)	Simulated pumping rate 3 months after commencement of pumping (L/sec)	Simulated pumping rate 6 months after commencement of pumping (L/sec)	Simulated pumping rate 12 months after commencement of pumping (L/sec)
HW1	153.1	50	50.0	50.0	33.7
HW2	89.7	50	41.6	37.9	31.6
HW3	112.7	50	38.2	32.3	26.9
HW4	83.7	46.7	33.2	28.8	24.0
HW5	266.0	50	45.4	39.1	28.8
HW6	626.9	50	37.6	29.2	21.4
HW7	107.4	50	35.1	27.6	19.2
HW8	52.8	50	38.7	37.0	35.7
HW9	126.3	50	50.0	50.0	8.6
HW10	152.4	50	50.0	50.0	16.5

Table 4.4 shows significant drawdown after just a year of pumping. The highest available drawdown is in HW6 (626.9m) and the lowest available drawdown was found in HW8 (52.8m). The simulated water profile after about two years of pumping, is shown below in Figure 4.12.

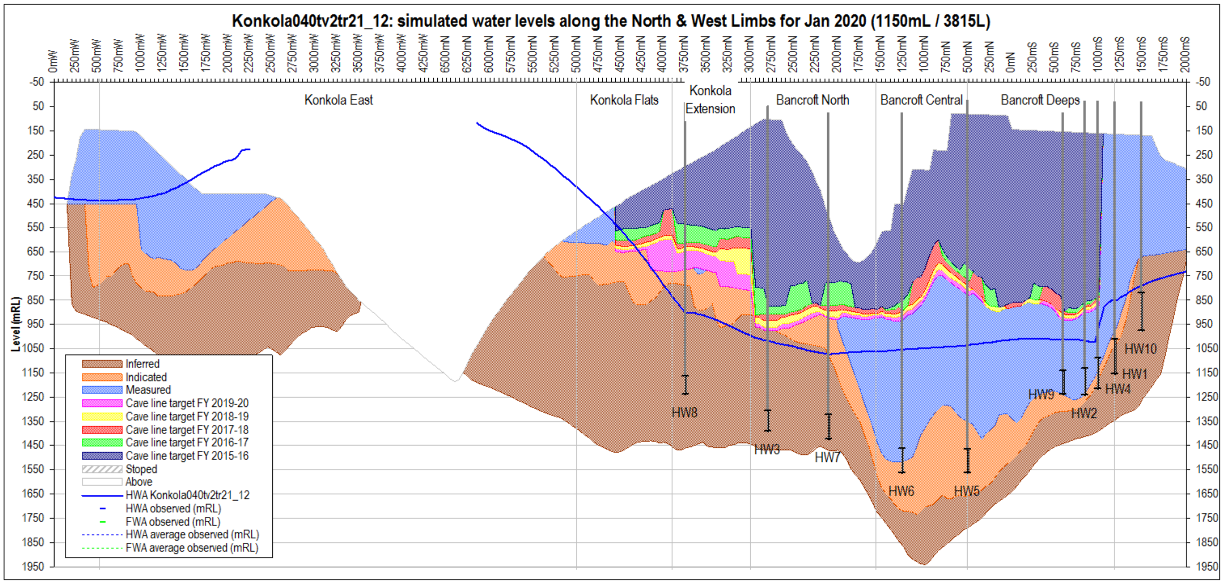

Figure 4.12: Water profiles after two years of pumping

The dewatering target for the elevation of 1050mL are achieved along most of the strike of the west limb and particularly Bancroft North, Bancroft Central and Bancroft Deeps sections. Once implemented on time, the method helps the mine avoid the ore gap in the Bancroft Mining areas. The drilling, completion, testing and commissioning of each hole will take 3 months, thus in those areas (Bancroft central and Deeps) where the holes be commissioned early and the water table is below the cave line, mining will start even before completing the whole project.

4.4.2.1 Costs

The cost are of this method are illustrated in Table 4.5. The price quoted in Table 4.5 are the average price from the quotations received from interested companies. The site investigation management and management studies are an average of what the bidders for the site investigation gave. The cost of all the test (Surface geophysics surets, Insitu packer permeability testing, downhole geophysical logging, multilevel vibration wire piezometer installation, time domain reflectometer installation and laboratory testing of geomechanical properties) required in site investigation was $720,112 and the cost for three test hole of length 1650m, diameter 33cm is $2,970,150. Besides this, the site studies come with management cost totaling to about $450,261 taking the overall cost of site investigation and management cost to $4,140,523.

Table 4.5: Cost of Surface Dewatering holes

Details	Costs $
Site Investigation and Management cost (market average)	4,140,523
Drilling Cost for the ten wells ($3.5mn/ Hole market average as per bids)	35,000,000
Cost of Pumps (10) (market average is 550,000 for a pump)	5,500,000
Total Cost	**44,640,523**

4.4.2.2 Disadvantages

Though the surface wells have shown great success in lowering the water table below the cave line, this method has some disadvantages. The following are the identified disadvantages associated with its implementation:

- Drillers experience in drilling in the Konkola stratigraphy which is highly faulted and fractured. Drilling in this stratigraphy beyond a certain depth becomes problematic as experienced by Lubambe Copper Mines PLC when they attempted the surface holes. During the drilling a surface deep borehole at depths of 500m to 600m, the drilling become problematic, the driller experienced fluid losses and at 850m the drill bit became stack and this led to the hole being abandoned. The entire project was abandoned as result of failure to complete the first hole and due to financial challenges experienced by the mining company during the attempt of the second hole.
- Missing the targeted points on the aquifers. This is a possibility as the holes are long and susceptible to deviation.
- Time sensitive: any delay in the implementation of the project would result in a gap in ore production in the Bancroft mining area.
- Long-term security of the associated surface infrastructure. New infrastructure would need additional security.
- Some surface infrastructure will be located outside current mine concession area and will thus require negotiation for access and acquisition.
- The drying up of some holes after one year of commissioning.

4.4.2.3 Positives

The entire project needs about two years to implement and significant drawdowns are achieved within a year of operations. The method may be implemented way in advance of development of mine haulages on 1,150mL and 1,350mL. The method has shown successful results in mines where it has been used, at Barrick Gold Nevada Goldstrike Mine, the 520m drawdown was achieved as at 2014. (Johny, 2014). The method can be used to dewater 1150mL whilst the mine is developing conventional dewatering infrastructure at 1350mL.

4.4.3 Conventional Dewatering

Conventional dewatering methods also known as breakthrough method has been used at Konkola Mine since inception of mining in 1957. The targeted area for dewatering is the area that falls within the 1150mL cave line. The method requires the setting up of the new main level, 1150mL, requiring 30,734m to be mined, which include a main level, drain drive, 19 dewatering centers from 200mW to 1200mS and associated dewatering infrastructure.

Before the LPCA is intercepted, the phase one of the 1390mL Pump chamber needs to be constructed, this will involve the mining of 2269m development which includes the establishment of sumps at 1350mL. Figure 4.13 shows planned 1150mL development. It should be mentioned that regardless of the dewatering method used (Conventional or surface boreholes), the main levels, sumps and pump chamber have to be constructed.

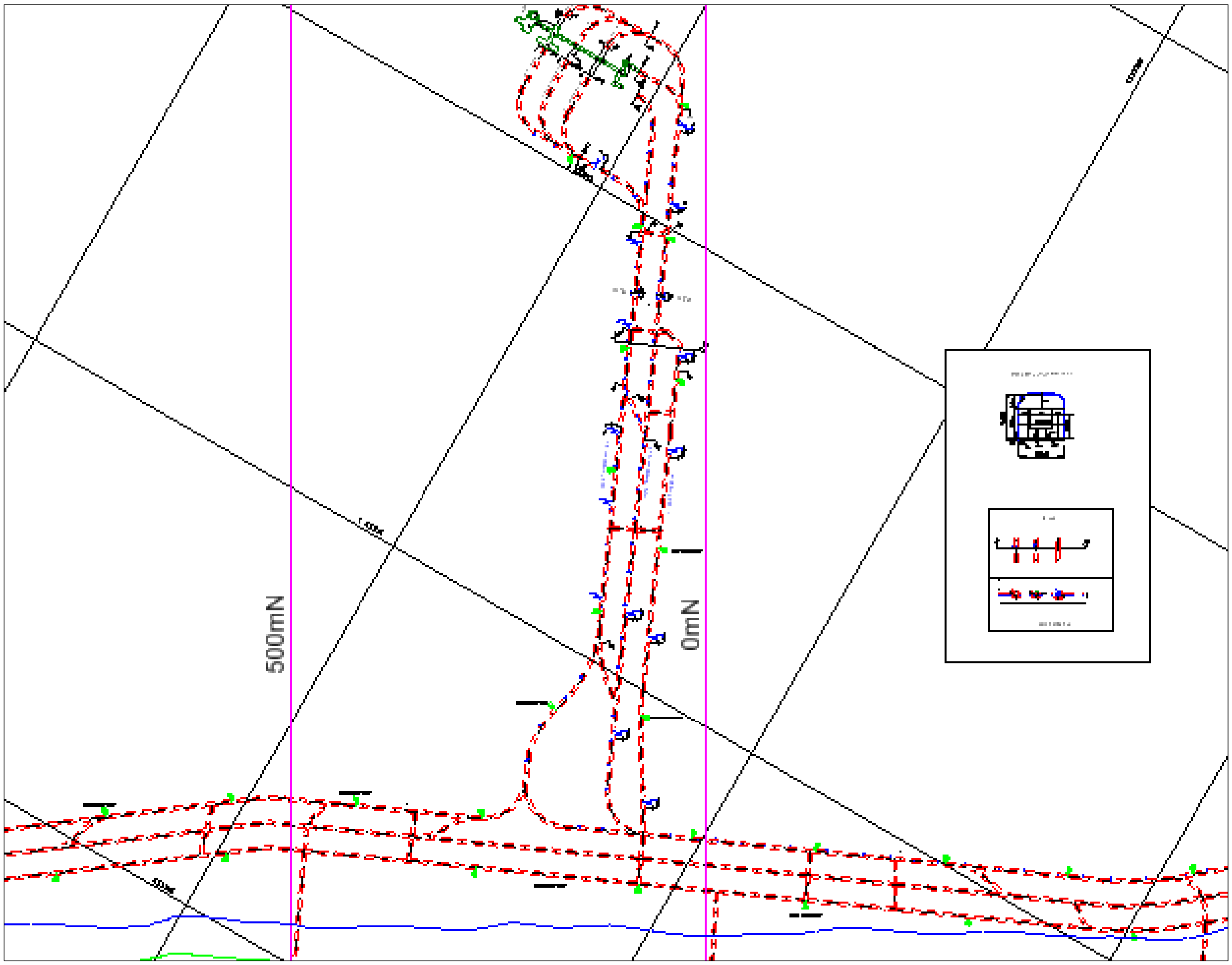

Figure 4.13: 1150mL development Plan

The 1150mL development will take 36months to mine and 18 to 24 months to dewater. The entire project needs approximately 60 months to mine and dewater that is 5 years. Historically 18 months of dewatering has been enough for dewatering of aquifers at Konkola. The development rate required to keep the project on track is 34m/day. The watertight door installation and associated developments will take 6months and this has been taken into consideration in the five years. The Gant chart shows the schedule of this project in Figure 4.14.

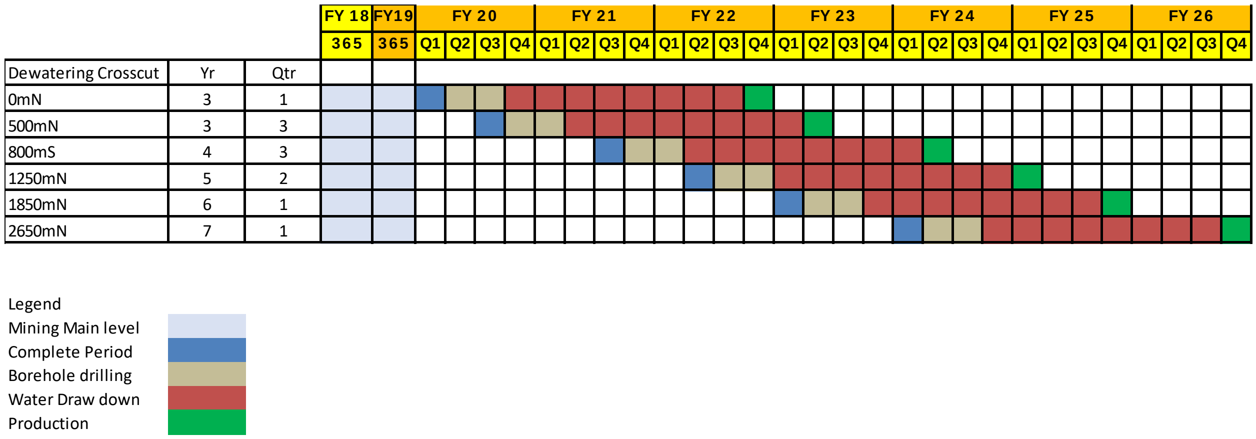

Figure 4.14: Schedule of development

In the schedule one dewatering borehole is planned to be drilled and equipped within 1 month, the drilling rate is approximately 300m/month and the average length of hole is 200m. A dewatering crosscut constitutes 10 holes which are planned for 6 months using two drilling machines, the period for drilling has taken into account that it's not all holes that are drilled which reach the planned target and those that miss need to be re-drilled. This method offers a chance of re-drilling once the target is missed, hence offering flexibility.

The mining of 1150mL will start simultaneously with the mining of the 1350mL main Level and 1390mL pump chamber. Figure 4.15 shows the planned 1150mL and 1350mL

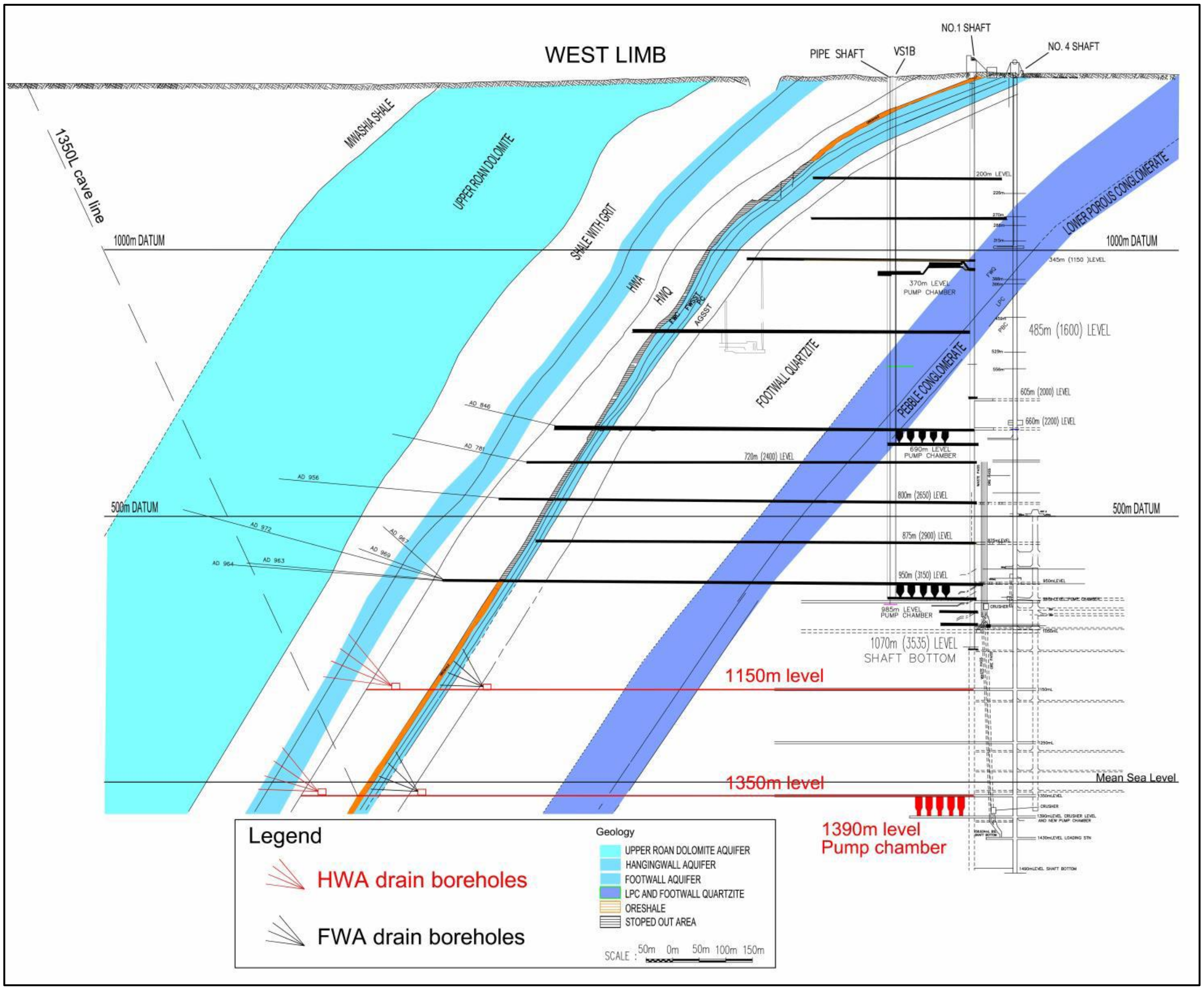

Figure 4.15: 1150mL and 1350mL development

4.4.3.1 Cost

The cost considered for the project is the cost of dewatering crosscuts and not the cost of the main level and pump chamber, because these costs irrespective of any method used have to be incurred. The cost per cubic meter and cost per meter for HQ size are rates the mine payed to Contractors during data collection stage (2017).

The average linear meters for the dewatering crosscuts: 70m

The width and height are: 4.2m * 4.2m

The volume of the excavation: 1235.8m^3

Cost per Cubic Meter: $156

Cost of the crosscut= 1235.8*$156 =**$192,629**

The average length of holes (HQ=96mm diameter): 200m

No of holes in a cross cut is 10, thus total meter drilled in one crosscut: 2000m

Cost of the 10 holes: 2000m* $107.67/m= **$215,340**

Total cost for a Crosscut: **$215,340+ $192,629= $407,969**

Approximate cost for 19 crosscuts: <u>$ 7,751,411</u>

4.4.3.2 Disadvantages

The biggest risk to this method is mining the planned meters in the stipulated timeframe. The timeframes are highly sensitive as missing the targets means the ore mining gap becomes a reality. The Figure 4.16 shows the decline in production associated with this method.

4.4.3.3 Positives

There are several positives to implementing this method, these are:

- Experience in method execution, this method has been used from inception of mining at Konkola Mine;
- Averts a gap in ore mining once implemented on time;
- The method offers flexibility: when a hole misses the targeted area in the aquifers, another hole is easily drilled at less cost; and,
- Holes are short hence less susceptible to deviation.

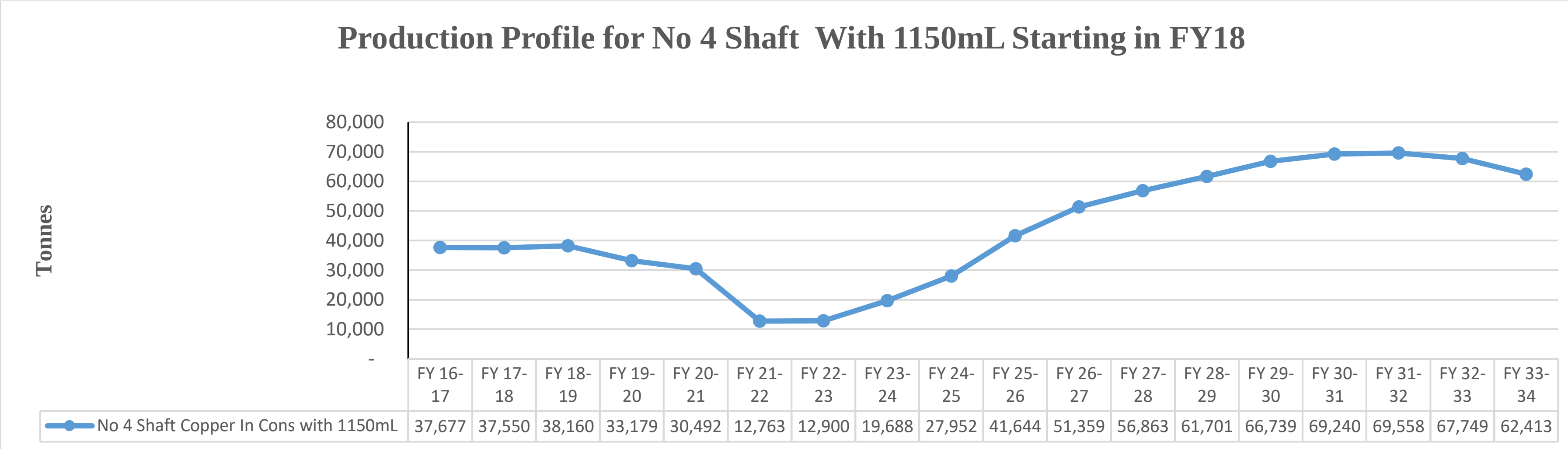

Figure 4.16: Production Profile for No. 4 shaft

4.5 Impact of Backfilling Mining Method on the Dewatering Requirement

The main reason for dewatering is to lower the water table below the cave line (because the area above the cave line is affected by subsidence, the effects of subsidence are discussed in section 2.17) and to reduce hydrostatic pressure. Fractured zones are one of the main groundwater flow channels to the Mine (Mulenga, 1993).

Backfilling methods reduces the rate of subsidence by 80 to 90% in comparison to caving methods (Straskraba and Abel, 1994). The backfill fills the stope void forming artificial support to the mine structure controlling both local stope wall behaviour and mine near field displacement, hence reducing ground movement. The cave line displacement from A level to B level is **X** for a backfilled area (backfill shown in red) as shown in Figure 4.17, which is less than an area that is not backfilled: A level to C level is **Y**. This simply means that for a backfilled stope with back length of **Y** and backfilled from C level to B level as shown in Figure 4.17 in red, the cave line displacement will be **X** compared to a movement of **Y** downwards if the stope was not backfilled. This translates to a lower dewatering requirement for mine that use backfill compared to those that don’t.

The dewatering requirement is significantly reduced in mines that use backfill as shown in Figure 4.17. Typically, with backfill mining methods it is necessary to dewater nearby Footwall and Hangingwall aquifers which are adjacent to the ore zone. However, Hangingwall aquifers approximately 40m or more above the mined ore would not probably drain into the mine (Straskraba, 1991). The need to drain these is site-specific and is determined by various factors: the presence of faults, anticlines and fractures; pore pressure; and, caved areas. Any substantial decrease in the dewatering is be based on geotechnical and hydrological studies and should be supported by rock mechanics and hydrogeologic monitoring. A reduction in dewatering requirements would result in a significant reduction in costs, these are the cost of: mining water drives; mining dewatering cross cuts; drilling deep dewatering bores; and, pumping. This can help in offsetting the cost of backfill. Straskraba and Abel in 1994 found that the introduction of backfill mining method reduced the requirements for mine dewatering.

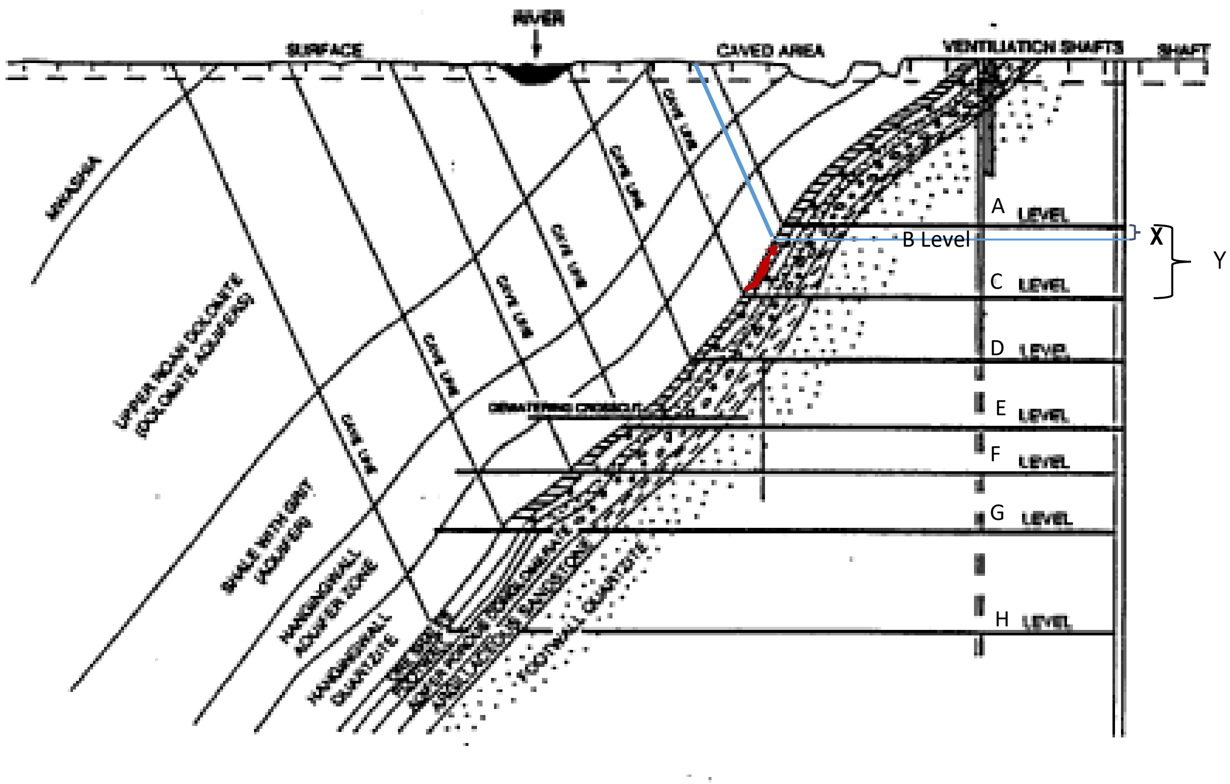

Figure 4.17: Cave line in mines with and without backfill

Backfill infrastructure already exists at Konkola Mine, thus reducing the costs of migration from stoping methods to backfill as no new plant would be set up, though the only costs to be encountered are the costs of plant repairs and pipe rehabilitation/ relining. The Mine has three backfill plants, these are KCM 1 BF, KCM 2 BF and KCM 3 BF plants. The KCM BF1 and 2 are located at No. 1 shaft while the new plant KCM 3 BF is located at No. 3 Shaft. The oldest backfill plant is the KCM 1 BF followed by the KCM 2 BF built in 2002 and the KCM 3 BF built in 2012. The KCM 1 BF is designed to produce Hydraulic fill while the KCM 2 BF is for cemented fill and the KCM 3 BF which is also known as the Waste Rock Crushing and Milling Plant is designed to produce coarse material. The hydraulic fill is to be mixed with the coarse material to form a high strength material with good permeability.

The mine once used Hydraulic Fill in the areas where Post Pillar Cut and Fill method (PPCF) was being used but due to the plant being un-available (broken-down), the mine is currently using waste rock backfill. PPCF is applied in areas with dips between 10° and 25°. The Fill provides a floor for subsequent lift as shown below in Figure 4.18.

The suitable backfill blend for Konkola is a mixture of Hydraulic Fill and waste rock crushed at 2.83mm aggregate, this blend must be in the ratio of 3:7 by weight of hydraulic fill and the aggregate, respectively, with the addition of 4% Portland cement. The mixture requires not less than 28 days to attain a required uniaxial compressive strength of 1MPa (Mutawa. A, 2011).

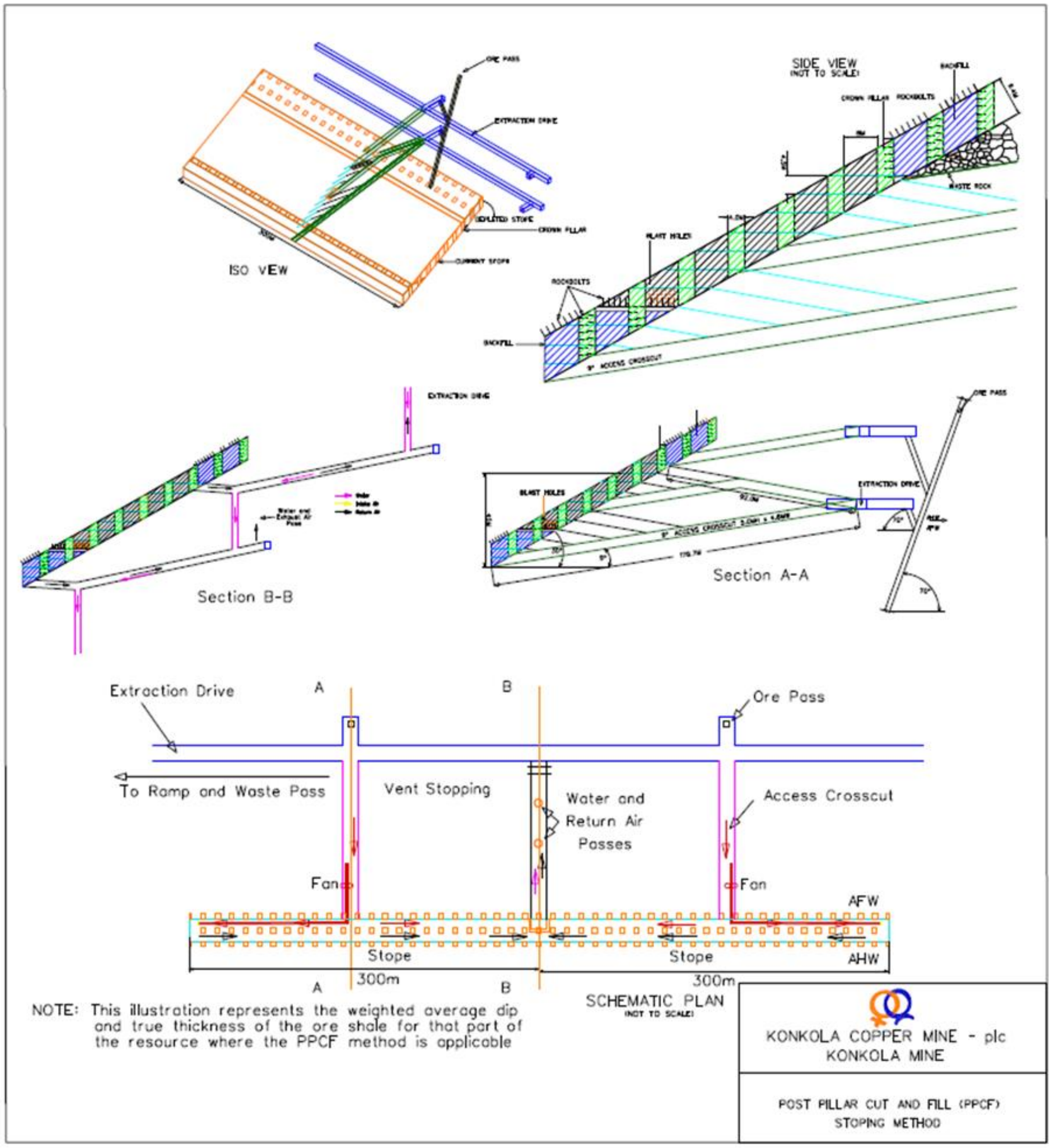

Figure 4.18: Post pillar cut and fill

CHAPTER 5: DISCUSSION

The Konkola Mine catchment area which is about 240km^2 has three sources of water, these are rainfall infiltration, seepage from surface water bodies and water stored in aquifers. The rainfall infiltration is 20% of the average rainfall of 1274.3mm/year, which translates to 254.9mm/year and 167,605.5m^3/d over the entire catchment area. The rainfall infiltration rate is high because of the presence of an extensive amount of saprolites in the Konkola stratigraphy. A small amount of water from the rainfall can be prevented from infiltrating into the mine by the burying the dambos. The total seepage of the surface water sources is more than 194,170m^3/d which is about 55.5% of the water pumped per day by the mine. The storage aquifer release is between 40,000and 50,000m^3/d.

The mine pumps about 350,000m^3/d of water to the surface and between January, 2016 and March, 2017 had a water to ore ratio of 92t:1t. The mine pumped 92t of water to 1t ore mined. Konkola Mine pumps more water than any other mine on the Copperbelt as shown in appendix 2.

There are three dewatering methods which were considered, these are: surface water exclusion method, surface deep wells method, and conventional dewatering method. The surface water exclusion method offers more than 194,170 m^3/d reductions in the water recharging of aquifers, which will significantly reduce the amount of water pumped out per day. This can reduce the daily pumping rate by more than 55.5%. The surface exclusion cannot be used alone but must be used with other dewatering methods because it cannot lower the water table.

The surface deep wells showed great success in lowering the water table, though the method had many grey areas. The method's success will depend on the results from the site investigation. The cost of implementing this method is about \$44,640,523. The simulation showed some wells going dry after one year of being commissioned casting a shadow on the \$3,500,000 cost per well.

The other method reviewed was the conventional dewatering method, this method is less disadvantages compared to its advantages and has been used since the inception of mining. The mine management and technical department has built enough experience in the implementation of this method. This method showed a decline in production from the year 2019 to 2026 accounting for a loss in copper in cons of 130,146t in 7 years. The lowest Copper in Concentrate will be experienced in the financial year (FY) 2021-22 (12,763t). During this time the dewatered reserves will be at their lowest but however, due to the completion of dewatering complex at 0mN in the year 2022, an upswing in production will commence. From the year 2022 onwards, the production trend increases until it reaches 69,240t in the year 2030 as shown in figure 4.16. The approximate cost for this method is $7,751,411 which is about 17.4% of the cost of the surface boreholes.

The surface deep wells when compared to the Conventional methods offers an opportunity for the Mine to lower the water table faster and mine without any slump in production. The method has several disadvantages which need to be sorted out before commencement of this method and it has a high cost when compared to the conventional method, while the greatest advantage of the Conventional methods is the mine has been using the method from inception of mining in 1957 and low cost bearing the fact that the Mine is currently facing financial difficulties. The conventional methods having been executed at Konkola Mine successfully from the inception of mining gives confidence to the mine management, unlike the surface deep wells which have never been used at the Mine site. The surface deep wells require site investigation (cost $4,140,523) while the conventional method does not need any costly site investigation. In terms of practicality and cost the Conventional dewatering methods is better than the surface deep wells.

Mines that use backfill have a less dewatering requirement compared to mines that use caving and open stoping methods (Straskraba and Abel, 1994). Straskraba and Abel in 1994 did extensive mining case studies and they indicated that the experience with adverse conditions associated with cut-and-fill mining and ground water is limited. They also, did not find any metal mine using exclusive backfill mining method, with significant groundwater inflow, and/ or with major problems related to ground water inflow. This is as a result of subsidence, which is more pronounced in mines that use caving or open stoping methods compared to those that use backfill. Backfill reduces the area affected by subsidence hence reducing the dewatering requirement. The suitable Backfill for Konkola Mine is a mixture of Hydraulic Fill and waste rock crushed at 2.83mm aggregate, the blend being in the ratio of 3:7 by weight of hydraulic fill and the aggregate, respectively, with an addition of 4% Portland cement (Mutawa, 2011).

CHAPTER 6: CONCLUSION AND RECOMMENDATIONS

6.1 Conclusion

The study has established that the best and viable method of dewatering Konkola Mine is the conventional dewatering method as it is easy to implement and cost-effective. The approximate cost for using the conventional method was determined to be $7,751,411 while the dewatering method using surface deep wells was determined to be $44,640,523. This presents a potential serving of about $36,889,112. Surface exclusion method can exclude more than 194,170m^3/d of water (which is more than 55.5% of the water pumped out of the mine) from recharging aquifers. Besides this, about 176 605m^3/d of rainfall infiltrates into the mining catchment area. The use of backfilling method compared to caving and open stoping methods has the potential to significantly reduce the dewatering requirement.

6.2 Recommendation

The study recommends that the Mine:

I. strongly adheres to the dewatering plan and schedule of development of dewatering infrastructure

II. implements surface exclusion methods of water

III. moves from open stoping to method that employ backfill

IV. conducts more seepage test

V. consistently updates the dewatering model

VI. conducts site investigation to get more information on the Konkola stratigraphy

VII. conducts hydraulic tests in the selected existing and future drain holes to obtain true hydraulic parameters of the rocks.

VIII. rehabilitates all canals in the Konkola area

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APPENDICES

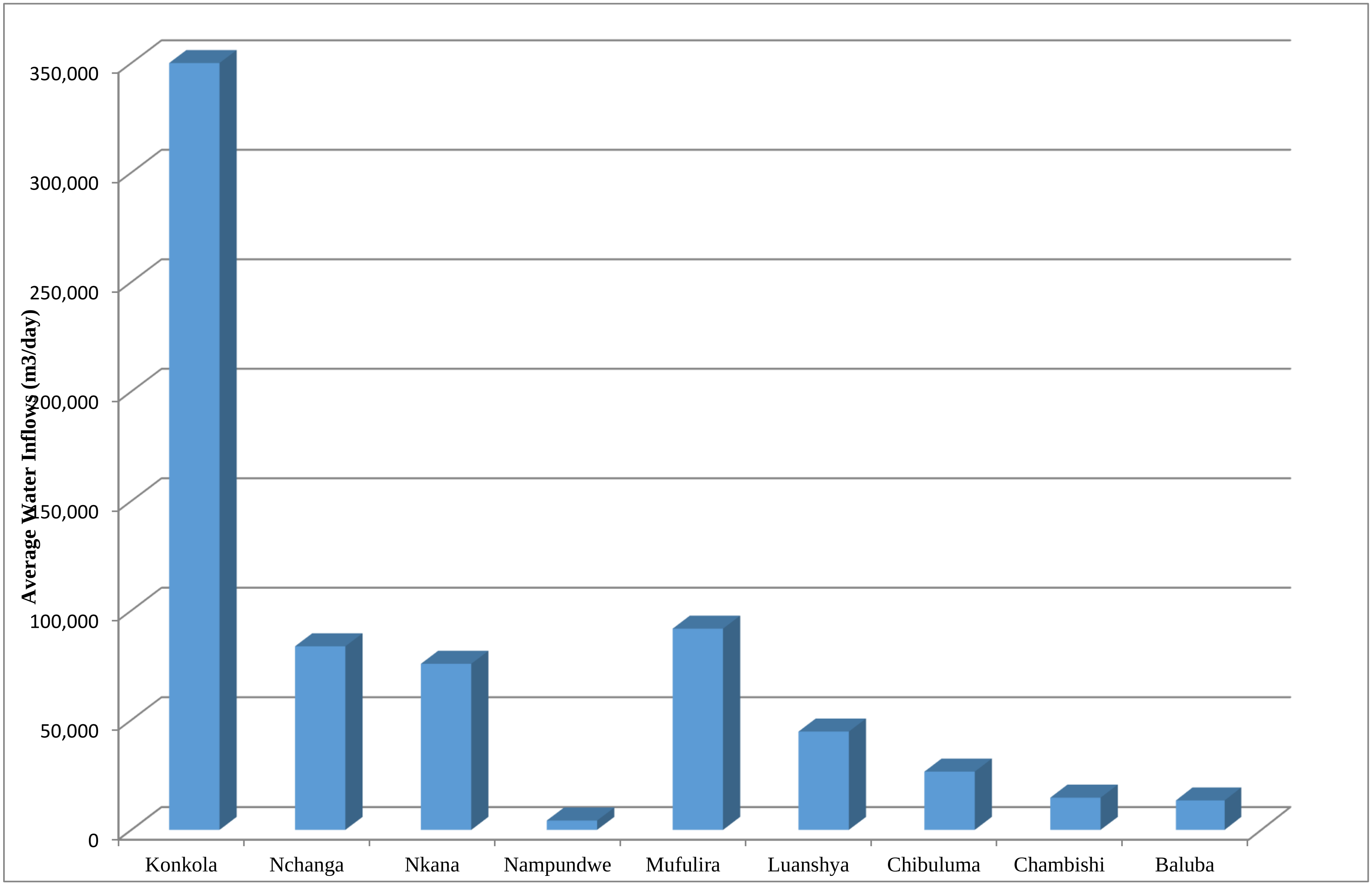
Appendix 1: Rainfall Data from 1953-2016

SEASON	SEASON	SEPT	OCT	NOV	DEC	JAN	FEB	MAR	APR	MAY	Annual
1953	-1954	0	19	162	343	301	273	203	6	0	1307
1954	-1955	0	0	160	172	351	424	165	87	0	1359
1955	-1956	0	65	158	252	399	331	287	237	0	1729
1956	-1957	0	20	166	266	426	378	398	4	10	1668
1957	-1958	0.0	25.7	46.7	310.4	291.1	144.3	154.2	13.5	0.0	985.8
1958	-1959	3.3	19.3	113.0	332.0	109.5	257.3	174.8	49.5	0.0	1058.7
1959	-1960	0.0	18.288	82.042	351.724	347.472	262.128	237.744	33.274	0	1332.7
1960	-1961	0.0	8.382	133.35	316.23	263.398	325.628	290.83	45.212	0	1383.0
1961	-1962	0.0	39.37	276.86	295.402	368.554	405.638	234.696	180.34	0	1800.9
1962	-1963	0.0	1	200.66	344.424	319.786	227.076	293.878	27.2	0	1414.0
1963	-1964	0.0	28.194	163.576	193.3	312.166	200.7	78.74	0	0	976.7
1964	-1965	0.0	11.43	163.83	248.92	294.386	322.326	210.82	10.4	0	1262.1
1965	-1966	14.0	10.414	74.676	205	271.018	159.8	75.946	42.9	49.276	903.0
1966	-1967	0.0	0.5	57.912	216.408	241.808	374.396	320.04	12.446	4.6	1228.1
1967	-1968	3.0	143.51	171.196	91.44	267.462	129.032	172.974	49.784	4.6	1033.0
1968	-1969	1.5	7.9	146.558	487.426	387.096	302	175.514	174.244	0	1682.2
1969	-1970	0.0	50.5	120.65	305.308	273.812	306.6	27.94	18.8	0	1103.6
1970	-1971	30.48	33	234.696	233.934	295.656	247.142	187.198	47.8	1.3	1311.2
1971	-1972	4.1	13.2	151.88	335.28	280.47	193.29	228.34	29.97	20.84	1257.3
1972	-1973	3.8	57.66	118.86	170.4	164.58	186.43	171.7	24.9	0	898.3
1973	-1974	0.0	11.68	177.3	241.55	290.06	143.59	155.43	42.42	117.08	1179.1
1974	-1975	0.0	0.76	113.28	445.98	320.55	144.79	455.5	77.47	0	1558.3
1975	-1976	0.0	0	58.41	401.32	380.24	210.55	372.6	191	7.11	1621.2

SEASON	SEASON	SEPT	OCT	NOV	DEC	JAN	FEB	MAR	APR	MAY	Annual
1976	-1977	33.8	35.57	229.36	205.99	329.7	304.27	269.49	28.44	0	1436.6
1977	-1978	15.2	31.5	180.75	432.2	315.96	199.13	383.3	105.16	0	1663.2
1978	-1979	0.0	74.17	207.21	376.27	210.85	198.63	264.01	20.83	0	1352.0
1979	-1980	0.0	62.74	183.12	346.65	204.46	214.64	220.73	173.74	0	1406.1
1980	-1981	9.14	66.3	101.59	263.38	233.68	322.58	244.85	34.8	0	1276.3
1981	-1982	0.0	22.6	150.1	156.22	266.69	304.79	89.17	81.03	22.35	1093.0
1982	-1983	0.0	69.6	276.86	323.61	237.48	240.82	123.44	139.96	0.51	1412.3
1983	-1984	0.0	36.82	130.05	476.52	299.22	336.66	147.62	16	0	1442.9
1984	-1985	1.0	11.43	153.33	383.03	254.74	216.2	161.45	95.6	17	1293.8
1985	-1986	0.0	8.89	272.11	356.1	277.1	270.6	316.2	99.4	0	1600.4
1986	-1987	0.0	91.4	162.2	158.3	193.1	253.2	147.6	7.3	0	1013.1
1987	-1988	2.0	0	94.7	193.9	240.9	342.9	335.2	7	0	1216.6
1988	-1989	0.0	14.3	134.9	198.5	279.7	257.3	290.7	21.1	0	1196.5
1989	-1990	0.00	5.10	138.60	284.90	147.20	151.90	85.60	69.40	54.80	937.5
1990	-1991	0.00	4.00	66.10	204.00	343.40	145.30	211.70	60.10	9.20	1043.8
1991	-1992	0.00	39.10	107.80	230.90	342.20	115.00	207.30	10.50	0.00	1052.8
1992	-1993	0.00	14.20	103.60	393.90	337.60	493.60	336.20	96.80	0.00	1775.9
1993	-1994	0.00	0.00	61.20	286.40	268.00	396.40	98.60	7.60	0.00	1118.2
1994	-1995	0.00	69.80	61.50	176.50	279.50	330.20	152.30	5.50	0.00	1075.3
1995	-1996	0.00	5.60	117.30	220.50	340.80	299.70	277.30	41.60	32.50	1335.3
1996	-1997	0.00	2.00	48.60	294.30	269.80	146.50	90.50	34.70	0.00	886.4
1997	-1998	29.40	15.40	200.60	399.30	345.00	217.00	244.70	2.40	0.00	1453.8
1998	-1999	0.00	0.00	109.10	133.50	290.50	140.80	317.70	30.90	2.40	1024.9
1999	-2000	0.00	0.00	173.90	159.80	256.70	296.10	256.50	8.00	12.00	1163.0
2000	-2001	0.00	4.00	191.40	405.50	392.60	260.40	337.60	53.80	0.00	1645.3
2001	-2002	30.50	39.21	69.48	183.32	309.35	216.42	119.37	8.88	0.00	976.5
2002	-2003	1.27	52.34	123.09	204.84	397.76	204.48	237.17	17.91	0.00	1238.9
2003	-2004	0.00	30.47	136.65	259.31	250.44	396.61	113.54	106.93	0.00	1294.0
2004	-2005	0.00	89.12	151.62	248.30	285.53	115.07	54.87	0.00	0.00	944.5

SEASON	SEASON	SEPT	OCT	NOV	DEC	JAN	FEB	MAR	APR	MAY	Annual
2005	-2006	18.54	0.00	123.58	208.28	316.48	199.13	205.69	13.97	17.78	1103.5
2006	-2007	0.00	8.12	153.14	227.57	346.70	204.22	133.85	0.00	0.00	1073.6
2007	-2008	3.05	20.83	63.52	688.85	299.44	381.64	131.58	4.32	0.00	1593.2
2008	-2009	0.00	10.67	337.11	232.75	358.85	196.13	310.77	46.48	39.88	1532.6
2009	-2010	0.00	1.02	259.14	314.90	236.22	357.31	252.70	14.73		1436.0
2010	-2011	0.00	3.81	202.87	295.41	203.19	182.91	257.32	22.61		1168.1
2011	-2012	4.06	26.67	147.63	228.86	304.25	304.33	400.31	2.29	0.00	1418.4
2012	-2013	9.14	0.51	80.87	416.26	351.22	145.28	154.92	0.00	0.00	1158.2
2013	-2014	0.00	13.21	103.89	325.85	211.59	360.69	147.84	111.76	0.00	1274.8
2014	-2015	9.80	3.05	67.79	298.20	294.94	158.25	211.07	219.21	0.00	1262.3
2015	-2016	0.00	3.05	113.31	168.65	404.12	315.98	257.547	37.588	0	1300.2
2016	-2017	0	0	147.9	221.9	316.70	380.1	216.10	120.80	0.00	1403.5

Appendix 2: Copperbelt Pumping Rates



Appendix 3: WatchDog Weather Monitoring Specifications

Sensor	Available on	Measurement	Accuracy
Wind Speed	All but 2800	0, 2-150 mph 0, 3-241 km/h	±2 mph (±3 km/h), ±5%
Wind Direction	All but 2800	1° increments	±4°
Air Tempera-ture	All but 2800	-32° to 100°C -25° to 212°F	±0.6°C ±1°F
Relative Humidity	2550, 2700, 2900ET	10% to 100% @5° to 50°C	±3%
Dew Point	2550, 2700, 2900ET	-73° to 60°C -99° to 140°F	±2°C ±4°F
Rainfall	2600, 2700, 2900ET	0.01" (0.25mm) resolution	±2% at < 2 in (5 cm) /
Solar Radia-	2900ET	1-1500 W/m^2	±5%

Appendix 4: Simulated Mine Dewatering Rates in 1988 (Sharma and Straskraba, 1991)