GROUNDWATER FLOW THROUGH KONKOLA (BANCROFT) COPPER MINE - ZAMBIA

A thesis submitted to the University of London

by

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ABSTRACT

This thesis describes the results of field based research on the Konkola (Bancroft) underground copper mine groundwater flow problem, in Zambia. Konkola is one of the wettest mines in the world, pumping an average of 360,000 m^3/d of water.

To understand how groundwater flows into and through the mine and subsequently formulate a long-term cost-effective groundwater management solution, an integrated study of historical and current dewatering and mining records, structural geology, surface hydrology, rock chemistry and groundwater chemistry, was carried out. The results enabled the development of a numerical model, to be used to make groundwater discharge and water level drawdown predictions for future groundwater management decisions.

The results obtained clearly reveal that; mine water can be divided into that which originates from surface recharge close to the mine and that which originates from regional aquifers at depth. This has enabled the origin of water in the mine to be essentially established. A considerable percentage of the water entering the mine comes via the Hangingwall Aquifer with its connection to the surface sources of recharge. Flow to the mine is dominated by flow through fractures and fissures which form zones of high conductivity. The flow in these zones can be modelled by finite difference networks. Significant reduction of water inflow into the mine can be achieved by implementing a water-exclusion groundwater management solution.

DEDICATION

This work is dedicated to my family. My wife Rebecca, my son Chileshe and my daughter Chitalu for their unwavering support and understanding. To my mother Eufemia Mbewe Mulenga and late father Gabriel Lyantinta Masakamike Mulenga, for teaching me to walk the talk.

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STRUCTURE OF THESIS

The thesis is made up of eight chapters divided into four parts.

Part I is composed of Chapters 1 and 2. Chapter 1 deals with the research problem studied and the strategy used for its solution. Chapter 2 contains the general literature review on groundwater flow problems in mines all over the world and methods for solving these problems.

Part II contains Chapters 3 to 5. In this part, the geological and hydrogeological setting of the mine is analysed (Chapters 3 and 4) and results of field and laboratory investigations in mine water chemistry and rock chemistry are analysed (Chapter 5). This part ends with a description of the conceptual model of groundwater flow at the mine.

Part III, composed of Chapters 6 and 7, addresses the subjects of numerical modelling of groundwater flow (Chapter 6) and the control of groundwater recharge into the mine (Chapter 7).

Part IV consists of Chapter 8 which deals with the conclusions for the research, References and Appendices.

PART I

RESEARCH PROBLEM, LITERATURE REVIEW, AND STRATEGY FOR SOLVING THE PROBLEM

Konkola underground copper mine has the largest known ore reserves of all the eight operating mines owned by Zambia Consolidated Copper Mines Limited but, because of the presence of large volumes of groundwater encountered in mine workings, its full ore-production potential cannot presently be realised. Pumping the water constitutes a major operational cost component. Mining has now reached a depth where the economic viability of the operation is in the balance and management decisions concerning future production require not only as accurate an assessment as possible of groundwater inflows, but above all how to solve the problem.

The correct approach to tackling groundwater flow problems in mines is to find the cause(s) of the ingress in the first place. Most of the case histories around the world reveal the problem of severe mine water inflow to be associated with the presence of hydrogeological boundaries adjacent to sources of recharge.

Chapter 1 deals with how the scientific problem was identified and defined, the reasons for the need to do so, and how and why the listed strategy to solve the problem was formulated.

Chapter 2 addresses the problems of dealing with groundwater inflow in mines in the mining world in general, with a view to discern the main causes and solutions to the problem, and ascertain how this knowledge would help in tackling the Konkola Mine groundwater flow problem.

CHAPTER 1

THE PROBLEM AND RESEARCH STRATEGY

1.0 INTRODUCTION

Konkola Underground Copper Mine, formerly known as Bancroft, is located in Chililabombwe District (Figure 1.0), at the northern end of the Zambian Copperbelt Province, near the Zambia - Zaire border. The mine is owned by Zambia Consolidated Copper Mines Limited (Z.C.C.M. Ltd.). It lies about 450 kilometres north-west of Lusaka, the capital city of Zambia, altitude, about 1300 metres above mean sea level.

Konkola is the fifth largest copper mine in Zambia. There are three main orebodies; the KirilaBomwe South, Konkola and KirilaBomwe North, served by the Numbers 1, 2 and 3 Shafts, respectively. Only Number 1 Shaft is currently in operation. Number 2 Shaft has remained closed since the 1958 world copper price depression. Number 3 Shaft was placed on care and maintenance basis in July 1987, mainly due to high and ever-increasing dewatering costs. Production resumed in 1990.

Prior to the closure of Number 3 Shaft, production stood at a monthly average of 150,000 tonnes of ore at a grade of 3.5% Total Copper. It has now dropped to an average of 95,000 tonnes per month. Although not the largest copper producer in Zambia, Konkola has the largest known ore reserves in the country, enough to last well into the next century. Currently known copper resources stand at over 200 million tonnes at an average grade of 3% total copper.

Mining is the mainstay of Zambia's economy. Metal exports account for over 90% of the national foreign exchange earnings. Therefore, Konkola having the largest known copper ore resource in the country, has the greatest potential to

LOCATION MAP OF KONKOLA UNDERGROUND COPPER MINE

Figure 1.0

play the pivotal role in generating foreign exchange, not only in the immediate future but more so in the long term as the shortfall from other mines begin to take effect.

1.1 BRIEF HISTORY OF THE MINE

According to Nichols (1956), Mackay and Johnson (1957), Bancroft and Guernsey (1961), copper was first discovered in the Chililabombwe area in 1924 by two Anglo American Corporation prospectors, James Williams and Babb Williams. Prospecting work was suspended during the world economic depression of the nineteen thirties and the Second World War, and did not resume until 1949.

Bancroft Mines Ltd. was formed in May 1953, named after Dr. Austen J. Bancroft in recognition of his great role during the exploration period. Sinking of Number 1 Shaft started in 1953 and in January 1957 production commenced at both Numbers 1 and 2 Shafts. However, in 1958 the mine was closed due to the fall in world copper prices. Production resumed in 1959 but only at Number 1 Shaft. Number 2 Shaft has remained closed ever since. Number 3 Shaft came into production in 1962.

1.2 THE PROBLEM

Ever since mining begun at Konkola in 1955, the major operation problem has always been the control of large inflows of groundwater encountered and expected during mining (Nichols 1956, Whyte and Lyall 1969, Coleman 1971, and Mulenga 1986).

The stratified copper deposit is sandwiched between two major aquifers as illustrated in Figure 1.1. The Hangingwall aquifer rocks are mainly calcareous; dolomites and limestones interbedded with siltstones. The Footwall Aquifer rocks are mainly siliceous; feldspathic sandstones, conglomerates and quartzites.

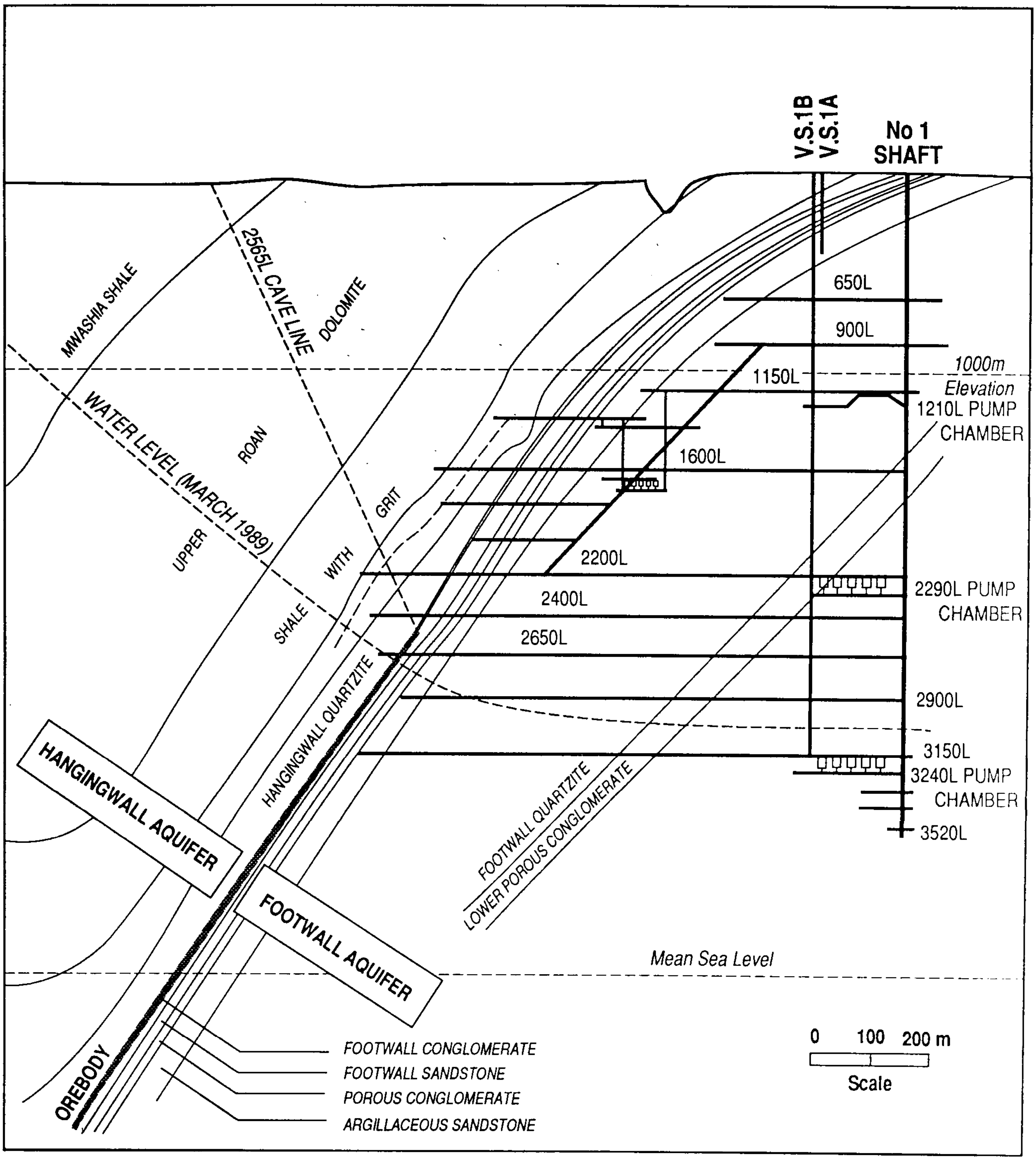


Figure 1.1 : Konkola Mine: Generalized geological section at Number 1 Shaft showing orebody, aquifers and water level.

Immediately south and west of the mine there is a large river, the Kafue, flowing over the Hangingwall Aquifer rocks (Figure 1.2). North of the mine lies the mine Tailings Dam.

The unsupported open stoping mining method used at the mine requires dewatering of the aquifers during mine development so that collapse subsequent to ore extraction occurs in dewatered strata.

Number 1 Shaft is the deepest of all the shafts at Konkola, with a depth of just over 1 kilometre. Pumping costs are so high that the long-term economic viability of the mine is seriously threatened.

Konkola is the wettest mine in the world as can be clearly seen in Figure 1.3. Currently an average of 360,000 m^3/d is pumped from the mine to surface. Pumping has increased from 15,000 m^3/d in July 1955 to a peak of 425,000 m^3/d in June 1978 and to an average of 321,650 m^3/d in June 1990 as shown in Figure 1.4. It appears that groundwater pumped from the mine and deposited into the Kafue river down stream of the mine, is finding its way back into the mine through the fault system that crosses the river. In addition it has been thought that the river directly recharges the mine.

Figure 1.5 shows that the water pumped to ore hoisted ratio increased from 36:1 in 1959 to 113:1 in 1989 (Konkola Mine Annual Reports 1955 - 1989). The average ratio of volume of water pumped to the ore hoisted at all other Zambian Copperbelt mines is a mere 4:1. Plate 1.1 shows a typical dewatering site in the mine.

Sufficient pumping capacity has to be installed to lower the water level in the various aquifers in time to meet production targets and watertight doors constructed on each main level to safeguard personnel, equipment and the shaft in the event of a water burst. The total installed pumping capacity to surface as in June 1990 was about 790,000 m^3/d .

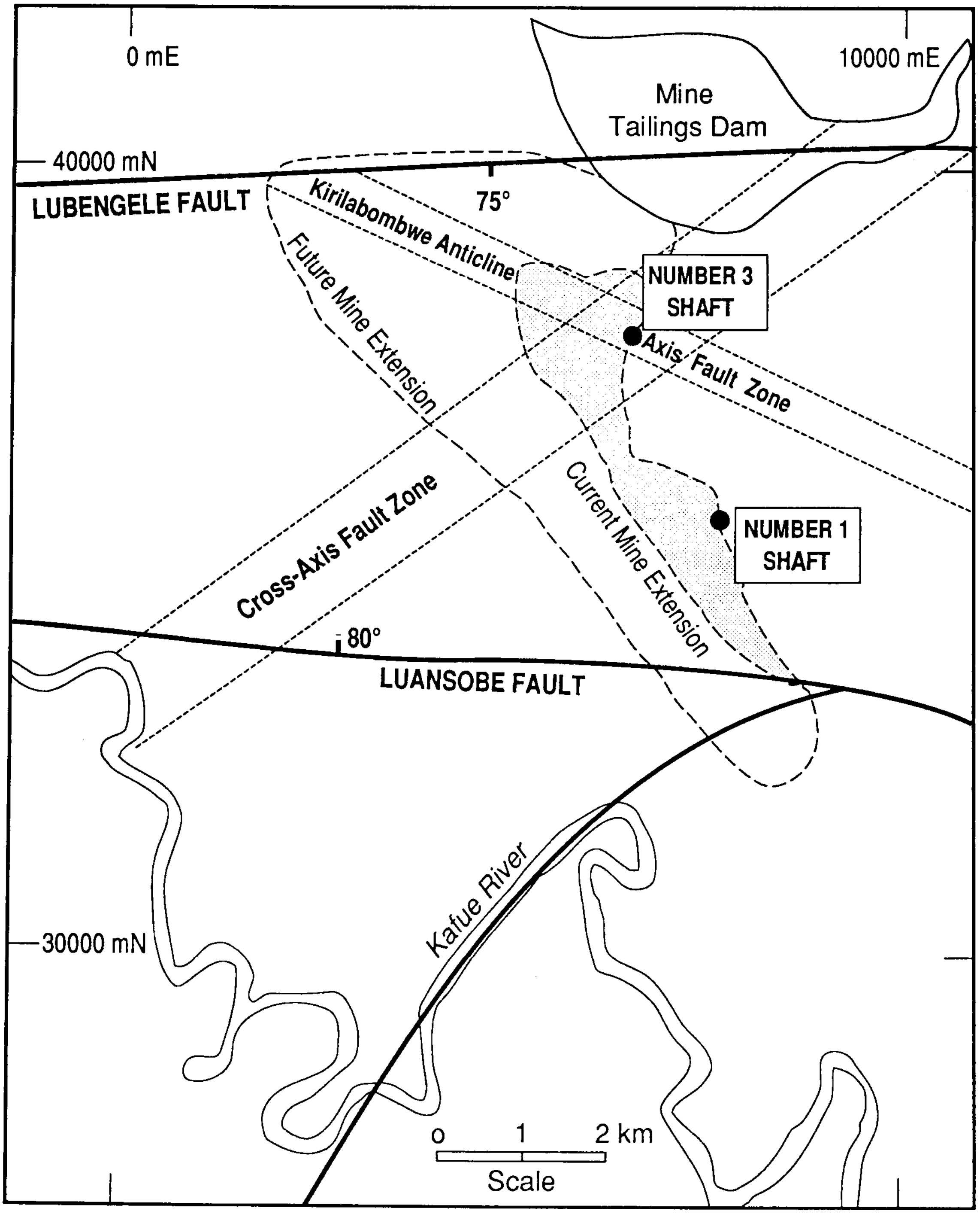


Fig. 1.2 : Map of Konkola Mine showing the faults, river, dam, and approximate limits of mining

Figure 1.3 : PLOT SHOWING THE RATIO OF MINE DISCHARGE TO ORE EXTRACTED IN MAJOR MINING AREAS OF THE WORLD (After Armstrong 1988)

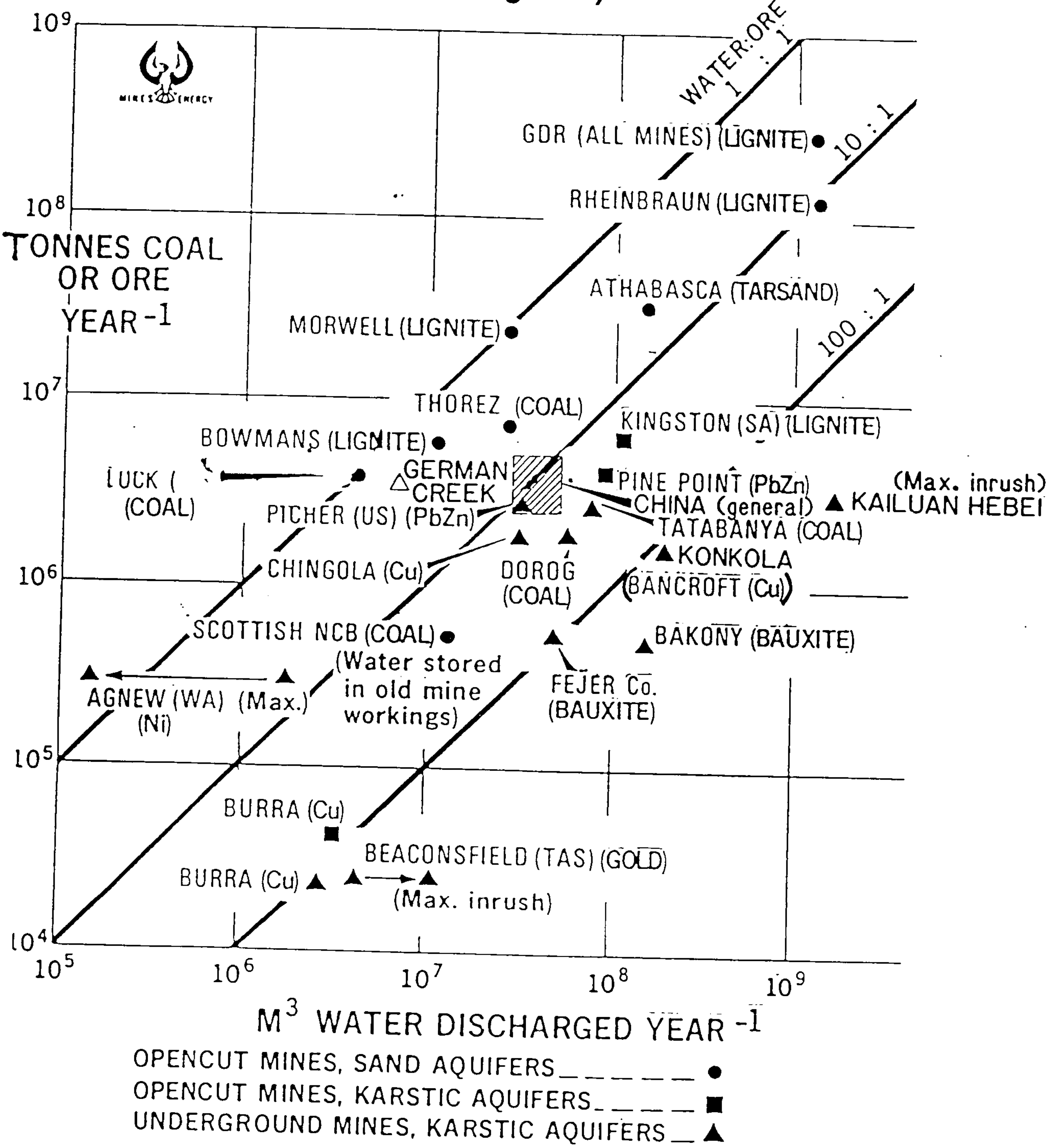


Figure 1.4 : KONKOLA MINE : AVERAGE VOLUME OF WATER PUMPED PER DAY (1955 - 1990)

Figure 1.5 : KONKOLA MINE : RATIO OF VOLUME OF WATER PUMPED FROM THE MINE TO TONNES OF ORE HOISTED (1959 - 1990)

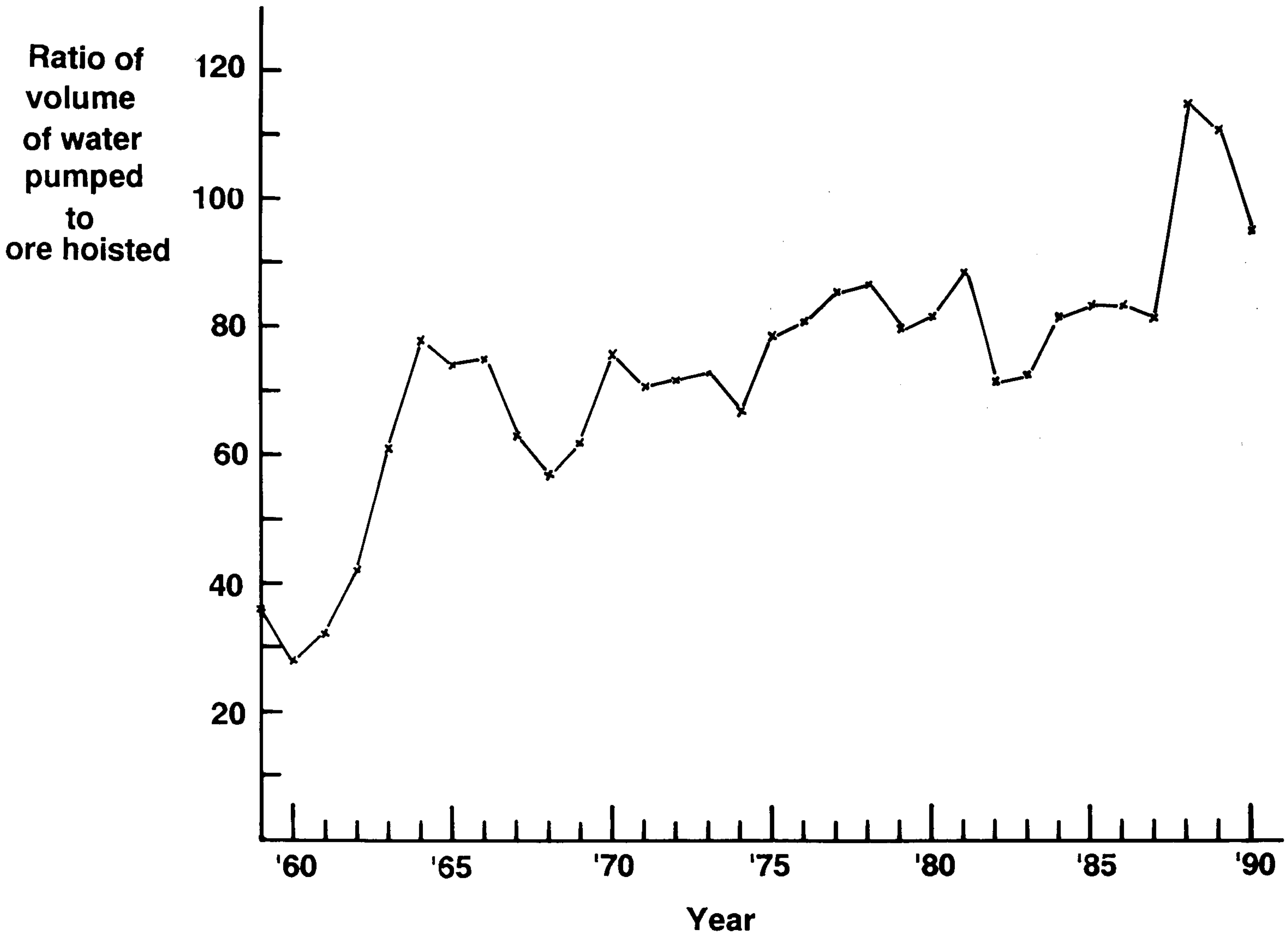
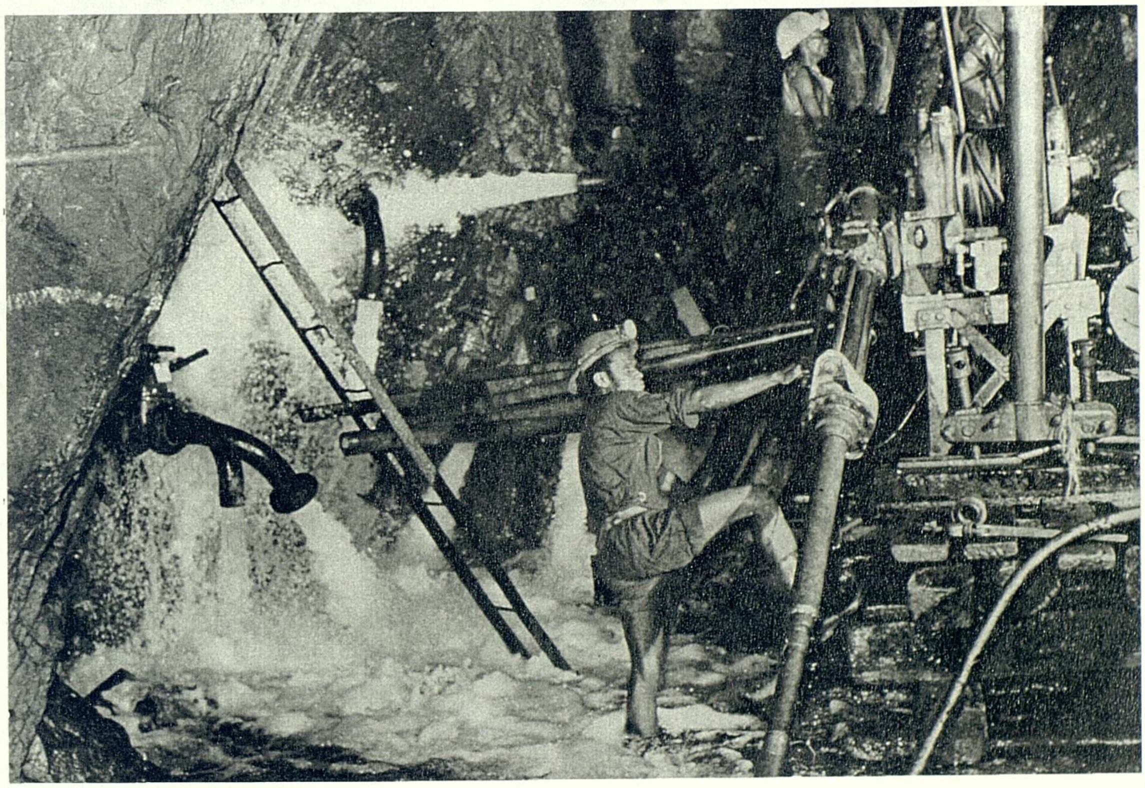


Plate 1.1 : KONKOLA MINE : TYPICAL UNDERGROUND DEWATERING - DRILLING SITE



Konkola's history abounds with flooding incidences. This is partly due to the fact that at the time shaft sinking begun, mine management had very little relevant hydrogeological knowledge and data to make a proper assessment of what lay ahead in terms of groundwater conditions. The location and distribution pattern of major fissures, volumes of water and hydrostatic pressure heads to be intersected were not adequately known.

According to Nichols 1956, when the sinking of Number 1 Shaft and Number 2 Shaft begun in 1953, it had been anticipated that Number 1 Shaft would encounter water; mine management did not expect the same problem at Number 2 Shaft as it was located on high ground of the Konkola Dome. However, within the first few hundred feet both shafts were making sufficient water to require cementation and installation of additional pumps.

On 15 February 1955, the pilot holes in a haulage development in Number 1 Shaft area intersected water under heavy pressure which flooded the haulage. It took six months to clear the problem because apart from cementation and installation of additional pumps, watertight doors had to be constructed to protect men and equipment from any future water burst and prevent flooding of the shaft.

At Number 2 Shaft, in the tunnel towards the orebody no water was anticipated and the pilot holes revealed nothing to indicate trouble. They were, however, to have a worse time here than at Number 1 Shaft. Towards the end of May 1955, the face blasted into a bed of wet running sand which filled the tunnel completely for a distance of over 20 metres.

The last major incidence took place in 1983 at Number 1 Shaft when 3150 foot level (960 metres) was flooded for almost a week, after a pilot borehole discharge became uncontrollable during a watertight door test exercise.

As can be clearly seen, the associated costs of mine drainage are very substantial. the cost of pumping, mining pump chambers, power sub-stations, underground water reservoirs and equipment represent over 50% of the total mine cost. Pumping alone, excluding the cost of mining and setting up pumping complexes accounts for about 10% to 15% of the total mine cost (Konkola Mine 1984 - 1990 Annual Reports). Unless a cost effective groundwater control solution is found, dewatering costs will continue to rise as the mine gets deeper and expands.

In the report on "The Future of Konkola" (Fluor 1988), the 20 year mine plan (1987 - 2007), schedules production to more than double the current output. The plan for Number 1 Shaft aims at continuous production rate of 1,100,000 tonnes per year and an expansion of the mine from the current production level of 3150 foot level (960 metres) to 3650 foot level (1112 metres). The production from Number 3 Shaft should gradually increase to 1,140,000 tonnes, and deepened from the 1850 foot level (565 metres) to 3150 foot level (960 metres).

To achieve this target substantial lowering of groundwater levels will be required. In the Hangingwall Aquifer drawdowns of at least 20 metres per year at Number 1 Shaft and 15 metres per year at Number 3 Shaft will be necessary. Pumping costs will increase from their current value of 10% to 15% and could approach and even exceed 20% of the annual total mine cost. These requirements clearly demonstrate that the cost of water handling will need to be substantially reduced in order for Konkola Mine to maintain long-term economic viability.

Despite considerable and expensive attempts to resolve the problem in the past, no satisfactory solution emerged other than continued and increased pumping. By July 1987, the situation became critical. Number 3 Shaft was closed and placed on care and maintenance, mainly due to high dewatering costs.

The source(s) of groundwater recharge to the mine, the pattern of groundwater flow and the effective dewatering catchment were not known with any confidence. Neither were the aquifer characteristics such as hydraulic conductivity, transmissivity, storage coefficient known nor were the boundary conditions, which are essential to a proper understanding of the hydrogeological regime, clearly defined.

Water inflows and drawdown predictions were, and still are, based on practical experience. The present average yield for a specific section is taken as a base for estimation of future inflows. The monitoring of discharge and pressure in underground boreholes and water levels in surface boreholes is used for prediction of drawdowns.

Much as this approach may have worked fairly well when the mine was still at shallow depth, it has proved highly inadequate as the mine went deeper. The accuracy of predictions have increasingly become unreliable as exemplified by the over-capacity pump chambers that have been mined. In the seventies, in order to cope with the planned mine expansion, two huge pump chambers were mined, one at Number 1 Shaft (3240 ft.) and the other at Number 3 Shaft (1940 ft.), to cope with the predicted increase in groundwater inflow. However, on completion of the project, there was only enough water to be handled by one pump chamber. the 1940 ft. Pump Chamber at Number 3 Shaft had to be abandoned. Clearly, it is evident that there is an urgent need for a new approach.

1.3 RESEARCH OBJECTIVES

As stated earlier, it was imperative that the hydrogeological regime of the Konkola Mine area needed to be well understood if a permanent solution was to be formulated. The long term solution to the inflow into the mine and thereby remove the necessity for costly pumping. With this in mind, the main objectives of the research were set as follows:

(i) To study how groundwater is moving into and through Konkola Mine in order to locate the source(s) of groundwater recharge to the mine.

(ii) To develop a conceptual hydrogeological model of Konkola Mine. This would make possible the development of an appropriate numerical hydrogeological simulation model, to facilitate accurate prediction of mine water discharge and water level drawdown.

These research objectives provide the scientific basis for formulating a hydrogeological programme which primarily aims at finding a permanent solution for reducing water inflow into the mine. Such a programme would investigate various groundwater control measures such as:

(i) grouting the major groundwater recharge pathways and other corrective measures to either minimise or stop groundwater recharge into the mine.

(ii) the formulation of a cost-effective dewatering system, and

(iii) a programme of long term groundwater monitoring to create a hydrogeological data bank for updating the mine model.

1.4 RESEARCH APPROACH

From the time mining started at Konkola, 35 years ago, it had always been well known that the mine had a groundwater inflow problem. Mine management understood it as being an engineering problem of water control and the creating of safe working conditions. The solution adopted was that of pumping the water. Enormous pump chamber complexes were thus established. Over the years as the mine went deeper, bigger pumping complexes were installed to cope with the increasing groundwater inflows. The mine pumping costs have now reached a level where the whole economic viability of the mine is in balance. An alternative solution to the problem is urgently required.

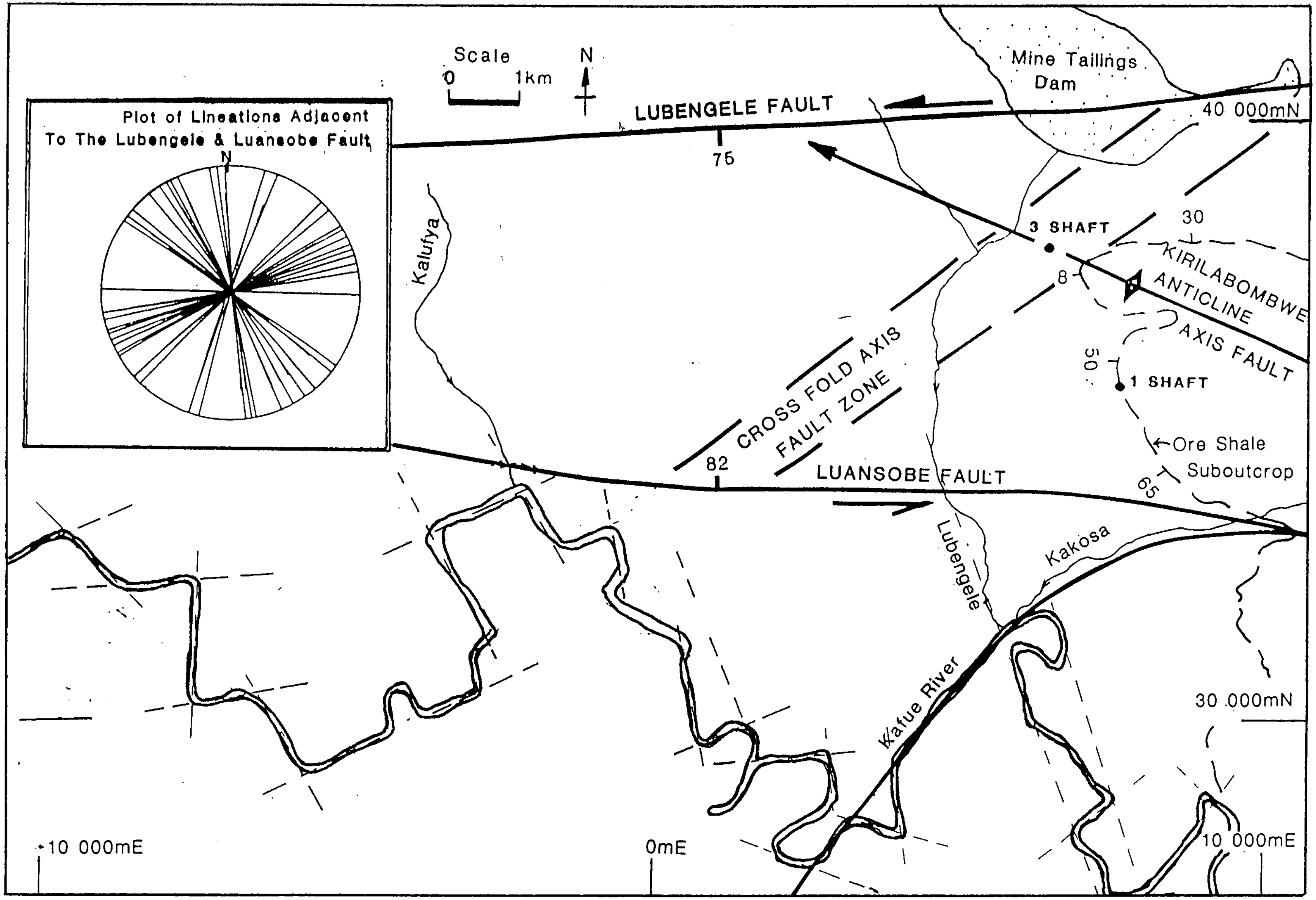
The approach of this research was first and foremost to define the scientific problem. Once this was achieved, it was necessary to determine whether or not it is amenable to a scientific solution and, finally, formulate a cost-effective permanent solution. This was done by reviewing and evaluating all mine historical and current data on geology, hydrogeology, hydrology, mining and all relevant aspects.

Mine records contained abundant information on water levels measured in open uncased boreholes over the mine area; they also record the mine discharge of any period. These data were combined to produce maps of water levels and water level change for given periods so that volumes of ground dewatered could be compared with volumes of water discharged. Manipulation of the great quantity of data available was made feasible by the use of computer programs for contouring.

From these basic studies it became apparent that the water level surface contained anomalies in its elevation with highs and lows existing instead of a smooth drawdown surface. These maps were superimposed upon maps of basic geology of the mine, at similar scale and revealed that the basic geological controls existed for the movement of water in the mine area.

This conclusion agreed with experience of working in the mine and prompted a study of the drainage patterns in the area in an attempt to reveal any fundamental controls exerted by geological structures upon drainage (Figure 1.6). These three lines of evidence clearly pointed to the importance of appreciating the structural geology of the mine and its hydrological significance.

Figure 1.6 : KONKOLA MINE AREA SURFACE HYDROLOGY AND FAULT SYSTEM



Against this scientific background the strategy was evolved. The strategy is illustrated in Figure 1.7 and uses six basic components:

- Historical dewatering and mining records
- Current dewatering and mining records
- Structural geology
- Surface hydrology
- Groundwater chemistry and rock chemistry
- Hydrogeology

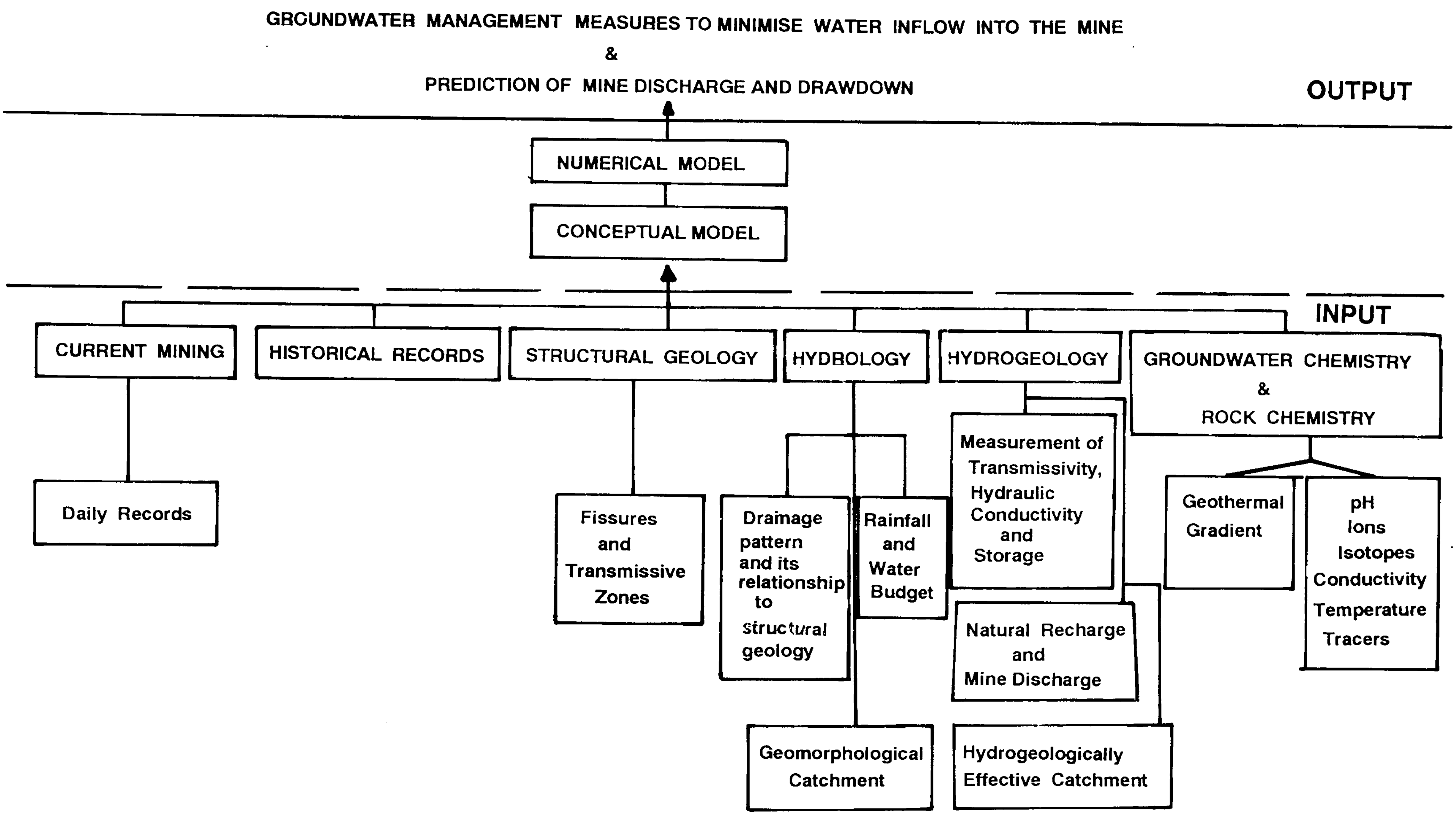
A review of current and historical dewatering and mining records revealed what was already known and what was not known and provided a framework from which to decide what further information was required.

For example, values are needed for aquifer characteristics such as hydraulic conductivity (K), storage coefficient (S), and transmissivity (T) in their in-situ undisturbed state as well as how these values can change as the effects of open - stoping are felt in the Hangingwall Aquifer and the effects of stress relief are felt in the Footwall Aquifer.

Of equal importance is data on aquifer types, mine discharge, groundwater level drawdown rates over the years and the extent of the cone of dewatering depression. In addition it is essential to have a proper understanding of the mine catchment characteristics, such as the hydrologically effective catchment area and rates of natural recharge over the mine area, and the water budget.

The new elements of the strategy never used before at Konkola were the incorporation of surface hydrology, structural geology, geophysics, the relationship between surface hydrology and structural geology, and groundwater chemistry into the programme of groundwater studies.

Figure 1.7 : STRATEGY FOR SOLVING THE KONKOLA MINE GROUNDWATER PROBLEM



Structural geology is of immediate relevance to the location of fissures and transmissive zones. Such fractures and zones are of fundamental importance to groundwater flow and they have to be mapped so as to enable a usable map of geological structure to be created from which boundary conditions that appear to be pertinent and aquifer characteristics that seem to be operating can be identified. The structural map facilitated prediction of permeable zones in ground yet to be developed and the order of magnitude of transmissivity and storage to be expected.

The review of surface hydrology provided an opportunity to study the drainage patterns in the area with a view to reveal any fundamental controls exerted by geological structures upon drainage and ascertain potential source(s) of recharge to the mine groundwater flow regime. In this context therefore it was essential to determine the extent to which the river flow pattern in the Konkola Mine area is controlled by the structural geology, and whether or not the Kafue River and surrounding streams flow along and/or cross the mine fissures and fracture zones. Areas where the surface river system intersect these transmissive zones would be zones of potential leakage and targeted for further investigations. The water budget provided an estimated water volume that is available to the mine.

Groundwater chemistry has the potential to reflect both the source of water and its flow path provided the characters revealing this can be reliably discriminated. A wide range of studies was initiated to sample the mine overall in a limited period of time (ten days) so as to provide a geochemical snapshot of the whole mining area. The studies were done in two consecutive years, at the same time of the year each time, July 1988 and July 1989. These studies were based on ionic concentration, pH, total dissolved solids, conductivity, temperatures of local river water, stream water, groundwater intersected in the mine, and rocks in the mine respectively, dissolved oxygen, bacteria and isotopes. A total of twenty three elements were analysed for each site. A total of eighty eight sites were sampled.

The studies described above were accompanied by a review of the range of types of numerical models that are available for prediction of flows and a comparison of their sophistication with the quality of data available on the mine and collected during the research period. The basic need was to know the boundary conditions that are pertinent, the aquifer characteristics that seemed to be operating and the local water budget. By combining model simulations with field data, these characters could be established with reasonable certainty. The model was calibrated using historical data.

With this strategy the results of all previous investigations and records could be incorporated for the first time into an overall picture for the mine hydrogeology, thus permitting proper understanding of how groundwater is moving into and through the mine, and the creation of an adequate data base from which a numerical model was to be made, and a programme of drying the mine formulated.

CHAPTER 2

GENERAL LITERATURE REVIEW

2.0 INTRODUCTION

The presence of large volumes of water in mines and the complications so caused, is a problem that has taxed all concerned since mining began. Mines are either developed above or below the water table. Evidently, those mines that are located below the water table generally have disproportionately high dewatering costs and are more susceptible to flooding (Zhongling 1988, Williams *et al* 1986, Sammarco 1986, Singh 1986, Fernandez-Rubio 1986, Brawner 1986, Straskraba 1984, Hofer 1979, Neate and Whittaker 1979, Davies and Baird 1977, and Venter 1969).

Flows of water have an important effect on the cost of and progress of many mines. The existence of water limits the methods which can be used in some and presents hazards in others.

The control of groundwater inflow to mine workings is often a long term and expensive proposition due to the cost of pumping, and the secondary effects of increased equipment wear particularly if the inflow is acidic. Above all, groundwater inflows into underground mines present health and safety problems. These problems range from the sudden catastrophic flooding of the mine and decreased wall stability, to lowering the morale of workers by the inconveniences of working in continuously wet environment. Excessively high dewatering costs or a catastrophic inflow of water into the mine ultimately threatens the economic viability of the operation.

2.1 PROBLEMS OF WATER INFLOW IN MINES

The problem varies in magnitude and importance at different mines according to the conditions which prevail, particularly that of depth. Some mines are so dry as to require little or no pumping, while others may be so heavily watered that the weight of water pumped exceeds many times the weight of ore hoisted (Figure 1.3).

Throughout mining history, inundation by water into mine workings has been responsible for a large number of major mine disasters. To mention but a few case histories, the following have been selected to illustrate the problem.

Atkinson (1988) lists the following:

Riding Underground Colliery, Falkirk, Stirlingshire, U.K.
On 25 September 1923, forty lives were lost in a mine flood. The inundation occurred due to failure of a dyke which had previously been considered impervious. Old workings on the other side of the dyke had been full of water since about 1873 and the abandonment plans did not accurately reflect the situation in the vicinity of the dyke.

Montagu Underground Colliery, Scotswood, Northumberland, U.K.
On 30 March 1925 the mine was flooded and thirty eight lives were lost. Many years previously a neighbouring colliery had encroached on Montagu. Montagu colliery workings were 86 metres from the boundary and they were proposing to leave a 37 metre barrier, when the inundation occurred.

West Driefontein Underground Gold Mine, South Africa
On 26 October 1968, the mine flooded following a water burst in one of the working faces. Although no lives were lost at this instance, almost four years before, in December 1962, much of the surface installation at West Driefontein disappeared into a huge sinkhole with the loss of 29 lives. Mine subsidence had been magnified and accelerated by collapse of aquifers following dewatering.

The inundation clearly demonstrated the terrible power of water. The rush of water down the shafts sucked the air out of some workings to create vacuums. Fierce air currents over 200 km/hr followed as the vacuums were filled. Water continued to flow at up to 450,000 m^3/d.

The inrush occurred after workings penetrated an impervious vertical dyke of 50 metres thickness. The water table in the "compartment" on the other side of the dyke was considerably higher than in the mined area which had been continuously pumped. The inrush was effectively stopped off by dams constructed on two levels.

Mufulira Underground Copper Mine, Zambia

On 25 September 1970, eighty nine lives were lost as a result of the mine flooding. In human terms, this is the worst disaster in the world resulting from water inrush.

Caving over stopped areas produced a sink hole of 320 metre diameter by 15 metre deep on the surface under a tailings dam which caused an inrush of material at a point 500 metres below the surface. This caused all parts of the mine below the 700 metre level to be flooded. The estimated inflow was 708,000 m^3 of water. Much of the mine was recovered and restored to production.

Lofthouse Underground Colliery, Yorkshire, U.K.

On 21 March 1973, previously unknown old workings were intersected resulting in violent inundation. Seven lives were lost. The violence of inrush was due to the pressure head caused by old shafts.

China Coal Mining Belt

In China's coal belt records of mines flooding abound. The mining belt is located in the vast karst environment of water bearing Carboniferous - Permian carbonate rock. The karstic limestone aquifers are a constant source of water hazard to the mines. Over one hundred incidences of flooding have been recorded. Zhongling (1988) sites the major mine flooding disasters as follows:

(i) **Kailuan Hebei Coal Mine, North China:** On 2 June 1984 the mine flooded following collapse of a karst column. An estimated inflow of 2,930,400 m^3/d entered the mine. This is the largest single mine water burst ever recorded. No figures for casualties are given.

(ii) **Jiangbei Sichuan Coal Mine, Southwest China:** On 27 August 1966 an estimated 2,160,000 m^3/d flooded the mine following karst caving. No casualty figures are given.

Hungary

In Hungary the bauxite and coal mining region is concentrated in the Transnubian Mountain karstic region. This area has had more than its fair share of mine flood incidences, starting as far back as the early 1900s. Alliquander (1982) and Bocker and Vizy (1982) quote the following examples to highlight the problem.

(i) **Dorog Coal Mine:** Between 1950 and 1970, eighteen rushes leading to flooding and production losses were recorded. As a result, by 1976 less than 40% of the originally planned mining capacity was available.

(ii) **Nyirad Bauxite Mine:** The mine was flooded following a water inrush of about 216,000 m^3/d , in February 1963.

Australia

Australia's mine water problems are mainly those of scarcity of water, but Hancock (1982) states that there have been some incidences of mine flooding. As early as the nineteenth century groundwater flow problems were experienced. At Beaconsfield Gold Mine in Tasmania water burst lasting up to three months were experienced, pumping approximately 13,000 m^3/d. At the Langi Logan Deep Lead Goldfields, 70,000 m^3/d were pumped from the mine.

India

In South India at the Neyveli Lignite Open Pit, 314,182 m^3/d must be continuously pumped each day of the year to avoid flooding the mine. The mine aquifer is extensive, its recharge area being some 130 kilometres away in the Western Ghats. Its recovery rate is such that in the event of pump failure, the mine would flood within 35 minutes.

It must be said though the loss of life, property and production from flooding is grim, scores of mines have worked safely for generations under groundwater reservoirs, lakes and even the sea. Nevertheless, most of the case histories reveal the problem of severe mine water inflow to be associated with the presence of hydrogeological boundaries adjacent to sources of recharge.

2.2 CONTROL OF GROUNDWATER INFLOW IN MINES

Historically, the presence of water in mines has been regarded as a water disposal problem. Mines all over the world have coped by developing in-mine drainage systems, deliberately over designing the dewatering and pumping systems, and maintaining an adequate supply of stabilisation and control equipment at all times (Atkinson 1988).

However, with recent developments and better understanding of mine hydrogeology, management of groundwater flow in mines is rapidly changing. Mines have got deeper and larger, and pumping costs have soared, thus necessitating a new outlook. The new trend is one of water exclusion rather than water pumping.

Loofbourow (1973), states that the need for better control of mine water is measured mainly by the direct and indirect costs of working wet ground.

Direct operating and capital costs of pumping are more evident than indirect items, such as provision of extra standby power and maintenance facilities. The total of all these items is the cost of pumping water from a mine. This is only a part of the extra cost of working wet as compared to dry ground. Less productive methods, less efficient equipment and more expensive explosives may have to be used in wet work. Wet rock plugs certain equipment. Maintenance is higher and labour less efficient. The cost of any means of keeping workings drier should be weighed against the high cost of working in a wet environment. The true difference in the cost of working a wet or dry mine, probably is known at only a few places.

Loofbourow (1973), cites Ambrosia Lake Mine in New Mexico, United States Of America. Between 1963 and 1967, a wet and dry mine were worked by the same company under conditions otherwise comparable. Average operating cost at the wet mine, pumping 10,368 m^3/d and 11,664 m^3/d was 16.06 US dollars per tonne of ore. The comparable figure for the dry mine was 7.5 US dollars. Principal differences included the fact that the wet mine was worked from the drainage levels below the ore with rail haulage, the dry mine in the ore with off-track equipment loading and hauling to the shaft. In the wet mine, ore packed in chutes, cars and skips and froze in surface bins, stockpiles and trucks. Wet-mine costs included drilling drain holes, building and maintaining sumps and pumping out water containing abrasive solids.

The extra costs and hazards of tunnels and shafts in wet, weak and wet, hot and wet ground can be several times those for corresponding work in tight, competent dry rock.

The current trend is to find ways of reducing inflows and thereby reduce the cost of pumping and in some cases eliminate the need of costly pumping altogether.

The measures include the following:

(i) Diverting or intercepting surface water from the mine (Loofbourow 1973 & 1979). For example at Steep Rock Lake Mine and Caland Mine. There are other ways of protecting workings from inflow. In some mines this is achieved by leaving enough solid ground between the mine and water as at Wabana in Newfoundland in Canada, and in the submarine coal mines in Durham, U.K., and Nova Scotia, Canada, and in the metal mines in Ontario and Quebec in Canada. In others it is achieved by leaving pillars on fissures or prevent or minimise movement, as in the South African Gold mines.

(ii) Grouting of highly permeable rock zones to reduce hydraulic conductivity. For example, in the Donetz coal basin and other coal mining areas of the USSR, grout covers and curtains eliminate the need to perform dewatering by maintaining depression cones (Kipko 1988). Other mines in the world that have employed similar approach are Port Radium, Deep Creek and Leadwood in the U.S.A. (Loofbourow 1979).

(iii) Installation of dewatering wells around the mine in order to lower the water table. For example at Loy Yang Coal Open Pit, Victoria, Australia, Homer Wauseca Iron Mine, Michigan, U.S.A.; and Orapa Diamond Open Pit in Botswana, Southern Africa (Wood *et al* 1988, Loofbourow 1979, Connely and Gibson 1985).

(iv) Selective shaft location, and mining from the bottom up. For example at Naica, Kimballton and San Antonio mines in the U.S.A., the shafts had to be relocated in less permeable areas. At Frood Mine near Sudbury, Ontario, Canada and West Driefontein Mine in South Africa, the mines were developed from bottom up. Lower levels are less permeable than the upper weathered levels. Therefore there is relatively low water ingress in the mine and there is more time to dewater the upper levels. More important, the advantage of this approach is that while pumping is at its maximum rate, the head could be reduced instead of increasing as is usually the case as a mine is deepened (Loofbourow 1979, and Tress 1974).

(v) Dewatering prior to mining, if economically viable. This was done at Bendigo Goldfield in South Australia (Forbes and Showers 1988).

The main sources of water inflows in mines are generally as follows:

(i) Rainfall
(ii) Local and/or regional aquifer system.
(iii) Leakage from surface water bodies such as rivers, lakes and surface reservoirs. This leakage may take place either through faults, fissures and high permeable rock formation zones respectively. For instance, a stream that is higher in elevation than the groundwater system may loose flow thereby recharging the groundwater system. The magnitude of stream gain or loss is dependent upon the hydraulic conductivity and related properties of the underlying hydrostratigraphic units. Groundwater moves constantly from recharge to discharge areas.

Although precipitation is the ultimate source of all surface water and essentially all groundwater, the total hydrologic cycle must be considered when evaluating potential groundwater problems in a mining environment. A portion of rainfall that falls in a topographic basin moves as sheet flow to run-off channels (Figure 2.1). Some of the precipitation infiltrates the unsaturated soil zone where it can either move laterally as interflow to the surface system to augment stream flow or move downward as recharge to the groundwater systems. The proportion of rainfall that ends up in the groundwater system varies not only with climate but also with rock condition.

Figure 2.1 : HYDROLOGIC CYCLE OF A PORTION OF GROUNDWATER FLOW SYSTEM (Modified After Williams et al 1986)

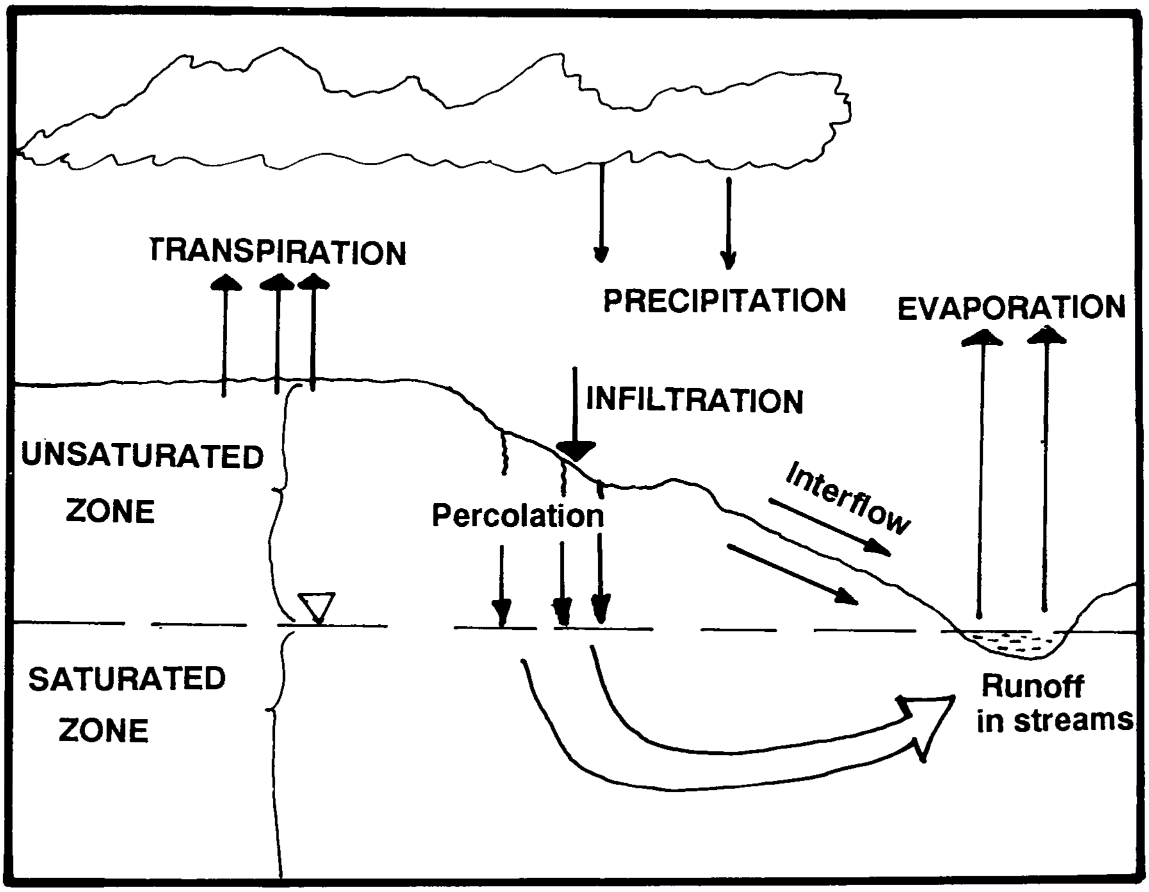
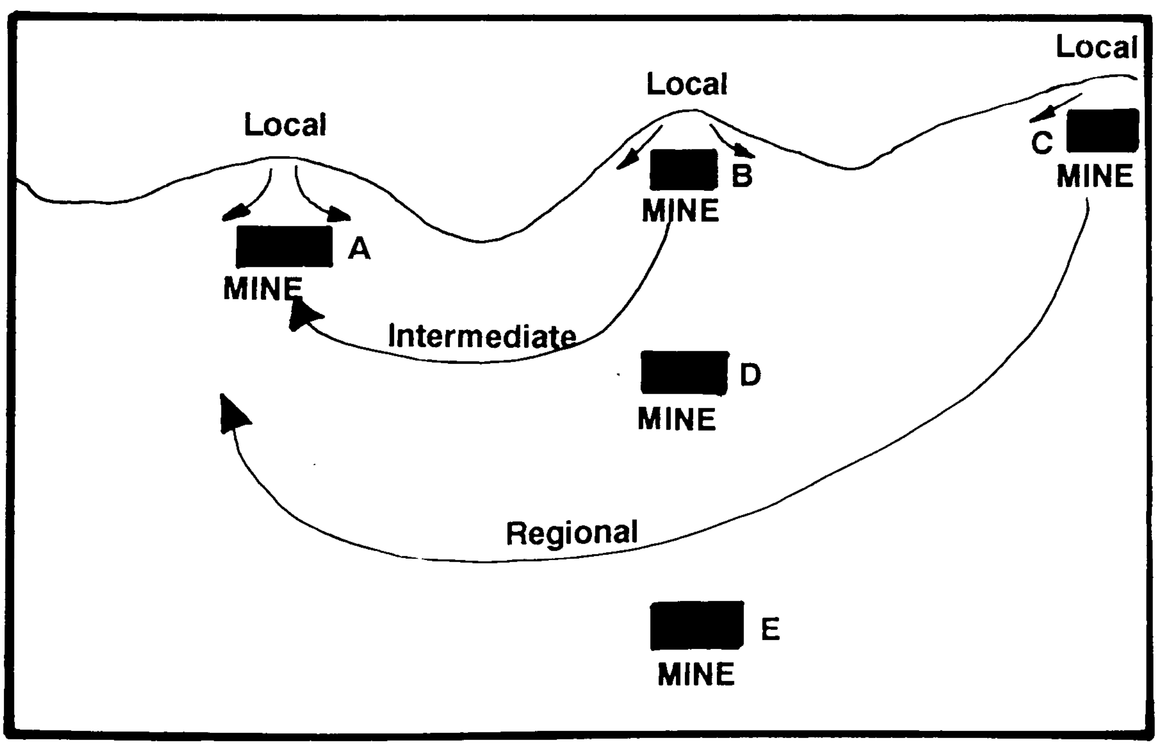


Figure 2.2 : POTENTIAL MINE LOCATIONS WITH RESPECT TO LOCAL, INTERMEDIATE, & REGIONAL GROUNDWATER FLOW SYSTEMS



2.3 GROUNDWATER FLOW SYSTEMS

Groundwater flows when there are pressure and elevation differences between one point and another. Water moves from higher potential to lower potential. The rate of groundwater movement is governed by the hydraulic conductivity of an aquifer and the hydraulic gradient. Groundwater flow in a porous medium is governed by Darcy's Law.

A groundwater flow system includes the entire set of pathways that water flows through the ground between hydraulic boundaries. It includes the recharge area, lateral flow area, and discharge area. The flow pattern is controlled by hydraulic boundaries, the quantity and distribution of recharge, topography, and the spatial distribution of hydraulic conductivity. Figure 2.2 illustrates one combination of these factors. Flow lines are shown on the diagram to illustrate the path a unit mass of water would follow from a specific recharge site.

Groundwater flow systems are designated local, intermediate or regional. These designations aid in categorizing field data and in assessing potential mining impacts (Williams *et al* 1986). Toth (1963) defined these systems as follows:

(i) **Local System** - This has recharge and discharge areas adjacent to each other at a topographic high and low respectively.

(ii) **Intermediate System** - It has one or more topographic highs and lows located between its recharge and discharge areas that do not occupy the highest and lowest elevations in a basin.

(iii) **Regional System** - This has its recharge and discharge areas occupying the highest and lowest elevations respectively in a basin.

The patterns of groundwater flow to a large extent control concentrations of dissolved constituents and temperature of the groundwater. The mineralisation of groundwater is dependent on the time of contact of the water with soluble minerals in a porous medium and thus on velocity of movement and length of flow path (Williams *et al* 1986). Higher concentrations of dissolved solids in groundwater normally occur along longer flow paths where velocities are low. Seasonal variations in air temperature are usually damped out at a formation depth of about 9 metres (30 feet) (Domenico 1972). Groundwater at that depth has a nearly constant temperature that approximates the mean annual air temperature of the region. Below that depth, the water temperature is controlled by the flow of heat from the interior of the earth. This is why groundwater temperature can be used to estimate the depth of groundwater circulation.

2.4 ANALYTICAL TOOLS

Most mine groundwater problems can be approximated using a direct and simple application of darcy's Law, for example as the flow net equation, the application of the state equations describing steady and non-steady flow and their utilisation in the form of the well flow equations for leaky, and non-leaky, conditions as based on the Theis formulation.

(i) **Darcy's Law:** This states that the flow rate (Q) is directly proportional to the cross-sectional area (A) through which flow is occurring, and directly proportional to the hydraulic gradient

$$Q = K \times A \times i$$

and apparent flow velocity,

$$\upsilon = \frac{Q}{A} = -(Ki)$$

where $-(Ki)$ indicates flow in the direction of reducing total head.
Darcy's Law is valid so long as the flow is laminar.
A simple application of Darcy's Law is seen in the use of Flownets.

Flow Net Equation: Here Darcy's Law states that

$$Q = K \times \frac{\Delta h}{L} \times A$$

expressing A in terms of $n\psi$ and L in terms of $n\phi$

$$Q = K \times \Delta h \times \frac{n\psi}{n\phi}$$

where: $n\phi$= number of equipotential drops, and $n\psi$= number of flow channels.

Flownets are a graphical representation of the Laplace equation for steady state flow in an isotropic medium, where

$\frac{\delta\psi}{\delta\phi}$=0 reflects the orthogonal intersection of flow paths with potential surfaces.

(ii) **Steady State Flow Equation:** The steady state flow describes a condition in which there is no change in head with time (ie $Q_{in}=Q_{out}$ and $\Delta Q=0$). This is a situation where discharge is equal to recharge. The potential flow which describes the change in flux in response to a change in potential is given by **Laplace Equation:**

$$\Delta Q = [K_x\frac{\delta^2 h}{\delta x^2} + K_y\frac{\delta^2 h}{\delta y^2} + K_z\frac{\delta^2 h}{\delta z^2}]\,\delta x.\delta y.\delta z = 0$$

where $\delta x.\delta y.\delta z$ = volume of ground. For homogeneous and isotropic aquifer reduces to:

$$\Delta Q = K[\frac{\delta^2 h}{\delta x^2} + \frac{\delta^2 h}{\delta y^2} + \frac{\delta^2 h}{\delta z^2}]\,\delta x.\delta y.\delta z = 0$$

(iii) **Unsteady (Transient) State Flow Equation:** This is a groundwater flow condition in which there is a change in water level and thus hydraulic gradient with time, hence Storage becomes involved as voids drain or fill. Discharge is in disequilibrium with recharge. The equation for such flow through a saturated anisotropic porous medium is:

$$\Delta Q = [\frac{\delta Storage}{\delta time}]\delta x.\delta y.\delta z = [K_x\frac{\delta^2 h}{\delta x^2} + K_y\frac{\delta^2 h}{\delta y^2} + K_z\frac{\delta^2 h}{\delta z^2}]\delta x\delta y\delta z \neq 0$$

given that the *[Storage]δxδyδz* $= S_s$, where S_s = water released per unit volume of aquifer per unit change in total head. Thus, within a column of aquifer of area (*δxδy*), the total volume of water capable of release in an aquifer of saturated thickness (b) is,

$$S_s b = S$$

As the amount released depends upon the units of thickness drained, ie upon change in head (h) with time (t)

$$\Delta Q = [\frac{\delta h}{\delta t}.S_s] = [\frac{\delta h}{\delta t}\frac{S}{b}]$$

where:

$$\Delta Q = \frac{\delta h}{\delta t}\frac{S}{b} = [K_x\frac{\delta^2 h}{\delta x^2} + K_y\frac{\delta^2 h}{\delta y^2} + K_z\frac{\delta^2 h}{\delta z^2}] \neq 0$$

The change in flow rate, expressed in two dimensions, is given by:

$$\Delta Q = K[\frac{\delta^2 h}{\delta x^2} + \frac{\delta^2 h}{\delta y^2}] = \frac{S}{b}\frac{\delta h}{\delta t}$$

and because Kb=T, we can write

$$\Delta Q = [\frac{\delta^2 h}{\delta x^2} + \frac{\delta^2 h}{\delta y^2}] = \frac{S}{T}\frac{\delta h}{\delta t}$$

(iv) **Theis Well Equation:** The application of the Laplace equations for flow to non-steady radial flow towards a central sink was solved by Theis (1935) such that the basic relationship between drawdown, transmissivity, storage and discharge could be written as follows:

$$s = \frac{Q}{4\pi Kb} \times W(u) = \frac{Q}{4\pi T} \times W(u)$$

$$u = \frac{r^2 \times S}{4Tt}$$

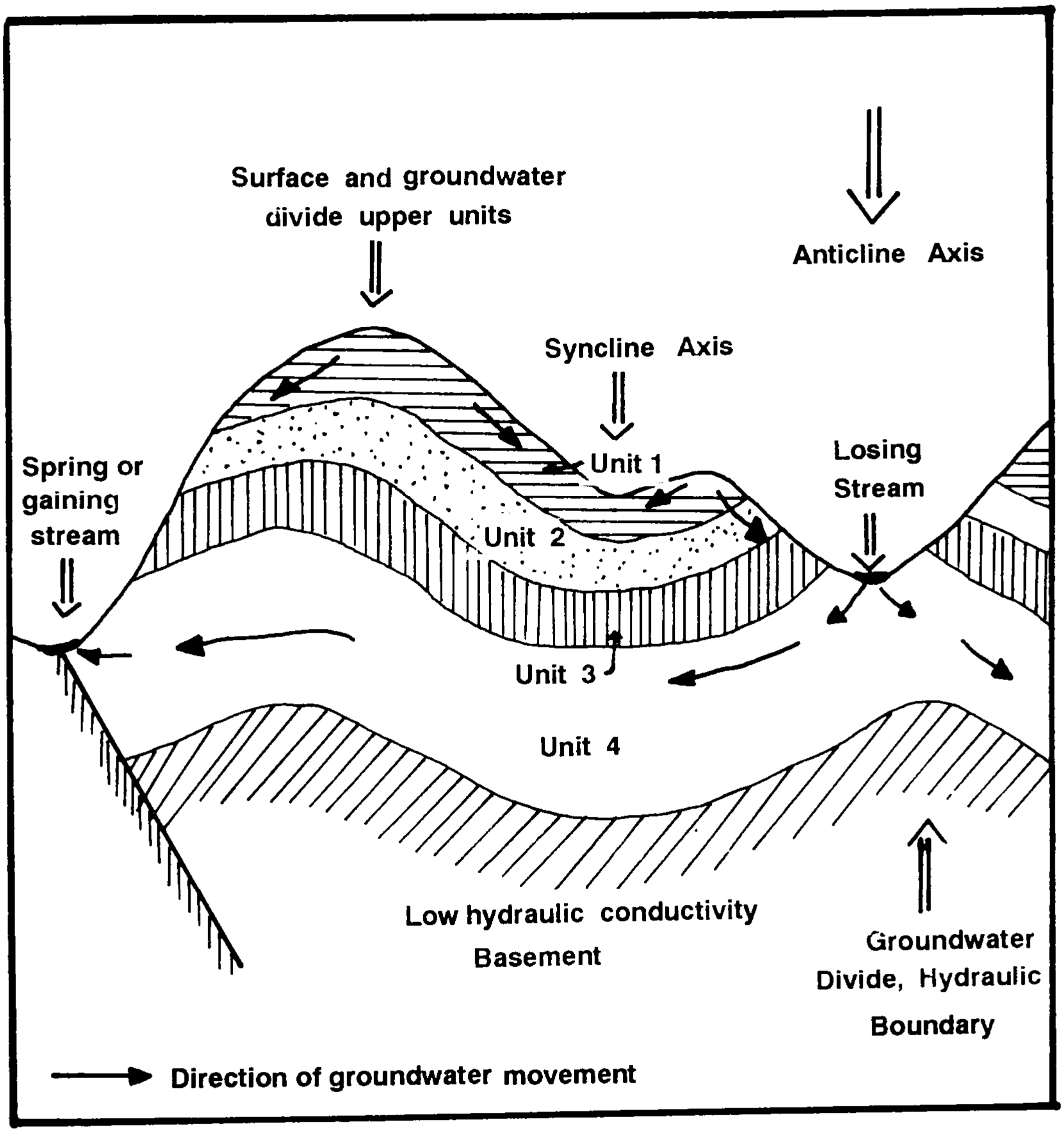
where: s=drawdown, Q=discharge, K=hydraulic conductivity, b=thickness, T=transmissivity, t=elapsed time, W(u)=well function where (W)= an exponential value and (u) its argument; where r=distance from pumping well to the observation hole where drawdown is measured or predicted, S=storage coefficient.

2.5 MINE HYDROGEOLOGICAL SETTINGS

A mine can be located in either a single or multiple aquifer setting. These aquifers may be unconfined, confined or both, and bounded by various types of boundary conditions. Boundary conditions can either be boundaries across which flow is permitted or boundaries across which no flow occurs. The former are designated variable-head, where the head changes with time, and constant-head where the head does not change with time.

Figure 2.3 illustrates the type of hydrogeological settings that can exist (Williams *et al* 1986). Hydrostratigraphic units 2 and 3 support only shallow groundwater flow near the axis of the anticline. These low hydraulic conductivity units support more flow at this area because of weathering of a fractured zone, since more fracturing can be expected near the axis of anticline or syncline. Unit 1 has a high hydraulic conductivity and supports only local flow systems due to the influence of topography. Unit 4 supports an intermediate flow system due to its high hydraulic conductivity and the influences of topography. The barrier boundary created by the fault is effective in blocking flow. The fault forces the groundwater inflow in unit 4 to the surface where it discharges as a spring. Conversely, the spring flowing along axis of the anticline looses water to the groundwater flow system.

Figure 2.3 : GEOLOGIC INFLUENCE ON GROUNDWATER FLOW SYSTEMS (After Williams et al 1986)

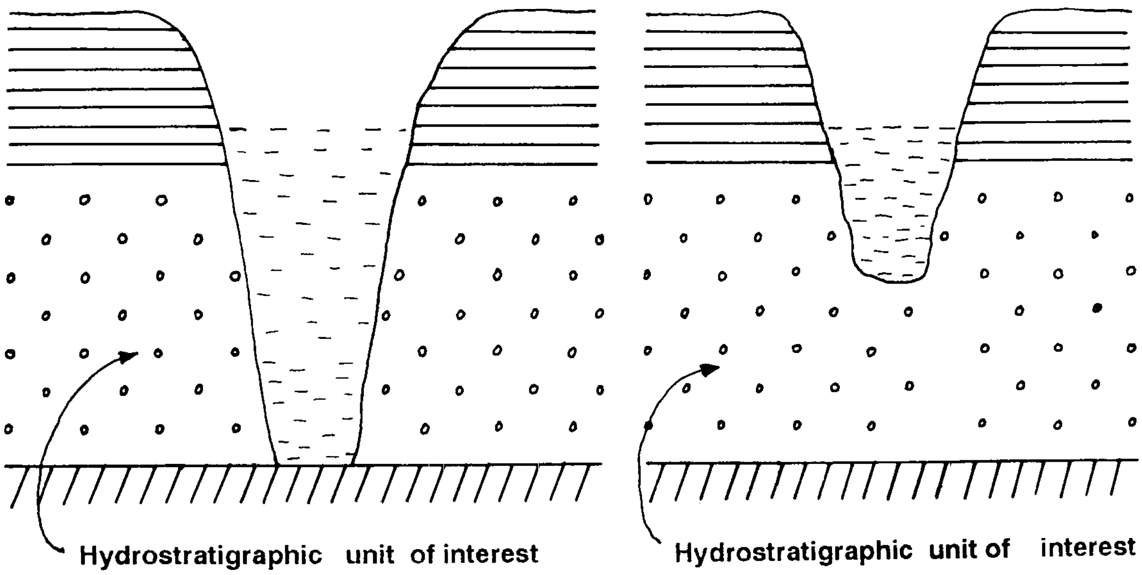


Geological boundaries in groundwater flow regimes exist where there is a lateral discontinuity in hydraulic conductivity. A significant decrease in the hydraulic conductivity creates a barrier boundary across which little movement of groundwater can occur. Recharge boundaries are created where the aquifer unit intercepts surface water systems or geologic units with significantly higher storage capacity and hydraulic conductivity than the aquifer in question.

Barrier boundaries can exist in various forms. The fault noted in Figure 2.3 is one type of barrier boundary that can block groundwater flow. Unit 4 has been displaced, it now lies opposite the low hydraulic conductivity basement material. According to Noorishad *et al* (1971), although faults in almost any rock can create such boundaries (Davies & deWeist 1966), faults can act as barriers blocking the flow of groundwater in rocks such as basalts. The result is a high potentiometric level on the up gradient side of the fault with a much lower potentiometric level on the down gradient side. Fault gouge can be a primary cause of low hydraulic conductivity in fault zones, but this is very much dependent on the type of material involved and its thickness Hydraulic boundaries account for most of the inrushes of water that have occurred throughout the history of mining.

Recharge boundaries can occur due to a units' contact with a river, stream, dam, lake or a zone of very high hydraulic conductivity. The water body can fully penetrate the unit of interest as shown in Figure 2.4a or only partially penetrate the unit as in Figure 2.4b. Recharge boundaries can consist of saturated zones of very high hydraulic conductivity. An example of this type of boundary is a highly brecciated fault zone as illustrated in Figure 2.5. Water stored in the extensive fault zone in Figure 2.5 is not confined so a considerable amount of water can be transmitted to Unit 2 with only a small water level decline in the fault zone. Fault zones may also place highly fractured rock units in hydraulic connection with minable rocks. Faults can behave as recharge boundaries as shown in Figure 2.6.

Figure 2.4 : Surface water bodies as recharge boundaries
(After Williams et al 1986)



(a) Water body fully penetrates unit

(b) Water body partially penetrates unit

Figure 2.5 : Potential recharge boundary as brecciated fault zone
(After Williams et al 1986)

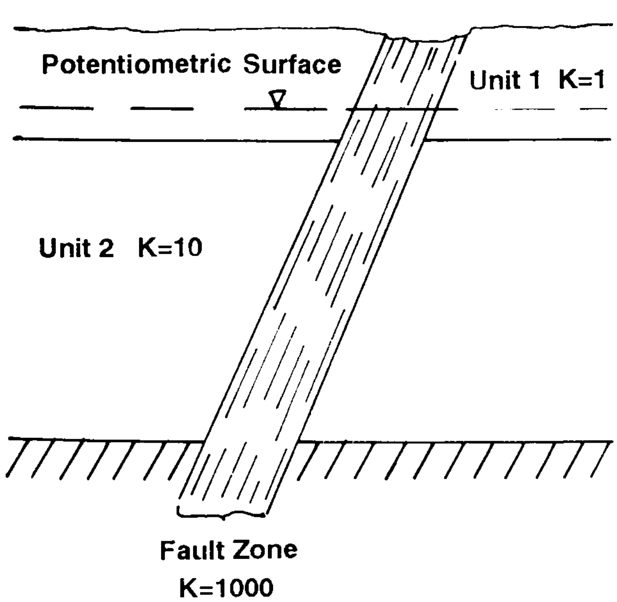
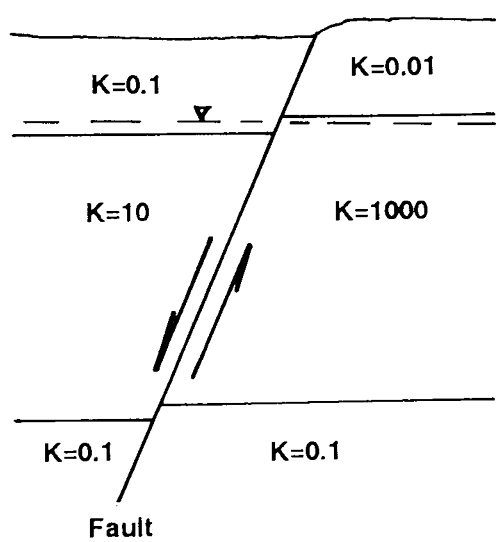


Figure 2.6 : Potential recharge boundary created by fault displacement
(After Williams et al 1986)



If the water comes from storage only, and the mine drainage is greater than the water released from storage, the mine flow rate will fall with time. On the other hand, if the aquifers in the mine are constantly recharged either by the regional aquifer system or surface water leakage such as rivers, lakes, and dams, the mine inflow will rise with depth. An excellent example is Carbones de Berga, S.A underground coal mine in Spain (Fernandez - Rubio (1986)), where coal seams are interbedded with limestones and sandy carbonaceous marl beds. The rising inflow comes from aquifer storage, leakage from the nearby Llogregat River, and from direct rainfall infiltration into the footwall aquifer, mainly via the faults.

Pumping Statistics for Carbones de Berga

Year	Production	Pumping	Ratio
1980	240,445	2,880	4.4
1981	372,385	4,320	4.2
1982	248,974	5,760	8.4
1983	278,889	8,640	11.3
1984	433,326	18,720	15.8

Similar examples above have clearly demonstrated the importance of a mine's hydrogeological setting in determining the kind of water inflow conditions that prevail at any particular mine. Mine inflow can be affected in a matter of hours by precipitation events where there is a direct hydraulic interconnection between the mine and the surface, through fractures of a mine opening. Examples include Barrio Longos underground gold mine in Paracale district, Camarines Norte Province in the Philippines (Caringal 1988), and Nampundwe underground pyrites mine in Zambia (Sweeney 1988). In these areas there are seasonal fluctuations in groundwater volumes to the mines. On the other hand, where sources are mainly controlled by the regional aquifer system, generally there are no significant variations with change in season, as observed at Nkana underground copper mine in Zambia (Sweeney, 1988).

2.6 HYDROGEOLOGICAL INVESTIGATIONS IN MINES

The dynamic nature of a mine environment coupled with the heterogeneous and anisotropic distribution of hydrogeological parameters, make groundwater studies complex and generally costly. The problem necessitates a well thought through programme of investigations. The programme should be tailored to suit the problem at hand.

First and foremost, investigations should start with a desk study. If the researcher is unfamiliar with the site, the desk study should be complemented with a visit to the mine. Desk study enables the determination of what is known and unknown about the problem and thereby facilitates the formulation of an appropriate strategy to solve the problem by carrying out investigations at the mine.

In order to develop a correct perspective of the hydrogeological regime under investigation, a proper understanding of the following parameters is essential:

(i) geological stratigraphy and structure,
(ii) topography and geomorphology,
(iii) surface hydrology and its relationship to the fracture and fissure zones, at both the local and regional scales,
(iv) mine pumping records,
(v) rainfall,
(vi) groundwater pressure profile,
(vii) aquifer(s) setting,
(viii) aquifer coefficient values and their distribution,
(ix) aquifer chemistry, and
(x) water chemistry.

It is evident that in order to evaluate a groundwater problem, conditions and parameters which have a significant bearing on the mine should be determined.

2.6.1 Geology

An understanding of both stratigraphy and structure can be achieved by conventional mapping methods. Geophysical methods can be used to delineate fracture systems where it is difficult to apply conventional geological mapping methods.

Structural geology mapping is essential to delineating fracture and fissure zones. This enables us to identify potential pathways for groundwater flow and hydrological boundaries, estimate the order of magnitude of hydraulic conductivity, transmissivity and storage coefficient to be expected, and facilitate the prediction of permeable areas in ground yet to be developed.

Aquifer Coefficients

The term refers to hydraulic conductivity, transmissivity, and storage coefficient. Hydraulic conductivity can be determined by a variety of techniques, including calculation from formulae, laboratory methods, tracer tests, auger hole tests, and borehole pumping tests.

(i) **Formulae:** Numerous investigators have studied the relationship of hydraulic conductivity to the properties of porous media. Several formulae have resulted based on analytic or experimental work. Most hydraulic conductivity formulae have the general form:

$$k = c \times d^2$$

where, c = a dimensionless coefficient, or

$$k = f_s \times f_\alpha \times d^2$$

where, f_s = grain (or pore) shape factor, f_α = porosity factor, and d = characteristic grain diameter (Todd, 1980).

Few formulae give reliable estimates because of the difficulty of including all possible variables in porous media and they were never designed for use with fissured systems. Because of these problems, other techniques for determining hydraulic conductivity are preferable.

(ii) **Laboratory Methods:** In the laboratory, hydraulic conductivity can be determined by a permeameter, in which flow is maintained through a small sample of material while measurements of flow rate and head loss are made. There are two types of permeameters; the Constant-Head and Falling-Head (Todd, 1980).

(a) **Constant-Head:** (Figure 2.7a). This can measure hydraulic conductivities of consolidated or unconsolidated formations under low heads. Water enters the medium cylinder from the bottom and is collected as overflow after passing upward through the material. From Darcy's law it follows that the hydraulic conductivity can be obtained from:

$$K = \frac{VL}{Ath}$$

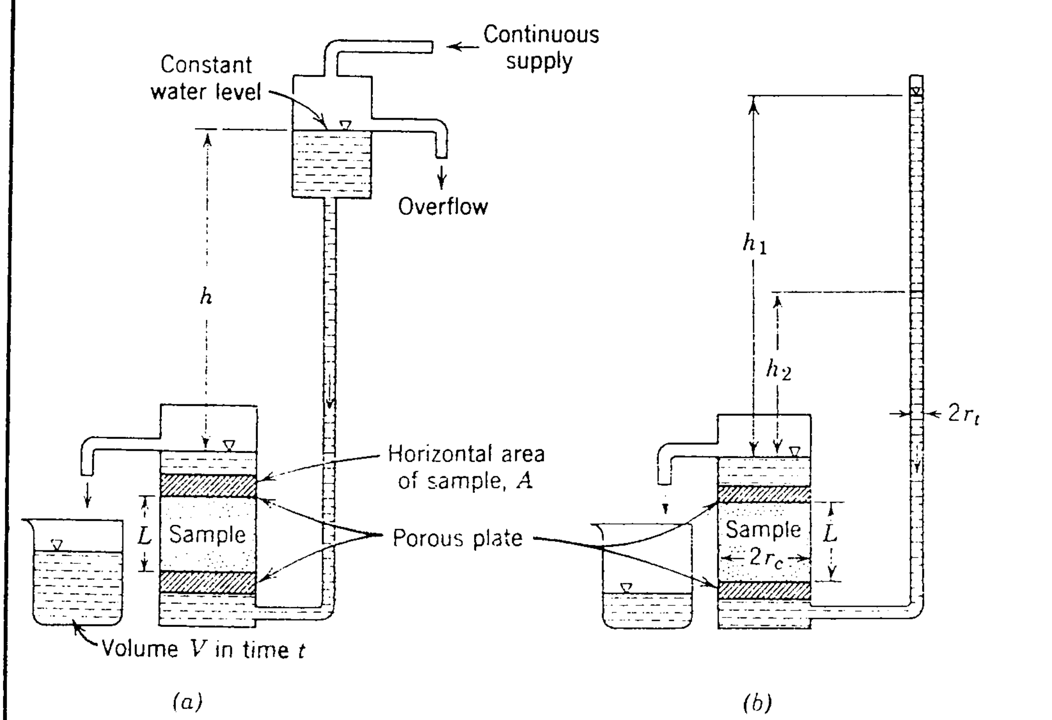
where, V = the flow volume in time t, A = Horizontal area of sample, L = length of sample, and h = head difference.

(b) **Falling-head:** (Figure 2.7b). In this method water is added to the tall tube; it flows upward through the cylindrical sample and is collected as overflow. The test consists of measuring the rate of fall of the water level in the tube. The hydraulic conductivity can be obtained by noting that the flow rate Q in the tube,

$$Q = \pi \times r_t^2 \times \frac{\delta h}{\delta t}$$

must equal that through the sample, which by Darcy's law is

Figure 2.7 : PERMEAMETERS FOR MEASURING HYDRAULIC CONDUCTIVITY OF GEOLOGIC SAMPLES



(a) Constant Head

(b) Falling Head

$$Q = \pi \times r_c^2 \times K \times \frac{h}{L}$$

After equating and integrating,

$$K = \frac{r_t^2 L}{r_c^2 t} \ln \frac{h_1}{h_2}$$

where, L, r_t, and r_c are shown in Figures 2.7a and 2.7b, and t is the time interval for the water level in the tube to fall from h_1 to h_2.

Permeameter results may bear little relation to actual field hydraulic conductivities when fissures are important conduits for flow. However, they are useful in as far as they give an indication of the lowest values to be expected.

Undisturbed samples of unconsolidated material are difficult to obtain, while disturbed samples experience changes in porosity, packing, and grain orientation, which modify hydraulic conductivities. Then, too, one or even several samples from an aquifer may not represent the overall hydraulic conductivity of an aquifer. Variations of several orders of magnitude frequently occur for different depths and locations in an aquifer. Furthermore, directional properties of hydraulic conductivity may not be recognised.

(iii) **Field methods:** Several methods of obtaining values of aquifer coefficients are available. These include tracer tests, packer tests and pumping tests. The most reliable method in a mine environment is pumping tests, because they give the most representative values as the test inevitably cover a large area.

(a) **Tracer Tests:** Field determination of hydraulic conductivity can be made by measuring the time interval for a water tracer to travel between two test holes. For a tracer a dye, such as sodium fluorescein, or a salt, such as calcium chloride, is convenient, inexpensive, easy to detect, and safe (Todd, 1980). Because the tracer flows through the aquifer with the average interstitial velocity (υ_a), then

$$\upsilon_a = \frac{K}{\alpha} \times \frac{h}{L}$$

where, K=hydraulic conductivity, α=porosity, h=head difference between the point at which the tracer is added and at which the tracer is sampled, and L=distance between the point at which the tracer is added and at which it is sampled.

But υ_a is also given by

$$\upsilon_a = L/t$$

where t is the travel time interval of the tracer between the holes. Equating these and solving for K yields

$$K = \frac{\alpha \times L^2}{ht}$$

Although this procedure is simple in principle, results are only approximations because of serious limitations in the field. The limitations are as follows:

The holes need to be close together, otherwise the travel time interval can be excessively long and unless the flow direction is accurately known, the tracer may miss the downstream hole entirely. Multiple sampling holes can help, but these add to the cost and complexity of conducting the test.

If the aquifer is stratified with layers having differing hydraulic conductivities, the first arrival of the tracer will result in a conductivity considerably larger than the average for the aquifer. Use of tracers also assumes that the retardation of the tracer, or its loss of concentration if concentration is the parameter being measured, are affected only by flow and not by chemical reactions occurring along the path of flow or absorption of the tracer by the ground through which it is flowing.

(b) **Pumping Tests:** There are several variations of pumping tests currently in use. They include single (bail and slug tests) and multiple well tests. Where there is only a single borehole, the Pressure - Recovery method is the most reliable method of obtaining representative values of transmissivity and hence hydraulic conductivity. The analysis is based on the Theis (1935) solution of transient pressure regime in a confined aquifer.

After pumping at a constant rate, the borehole is shut and the pressure build-up measured until it reaches the original pressure. This rise of the water level is measured as residual drawdown "s", (i.e. the difference between the original water level prior to pumping and the actual water level measured at a certain moment "t" since pumping stopped (Figure 2.8a).

Residual drawdown is given by:

$$s = \frac{2.303Q}{4\pi T}\log\left(\frac{t + \Delta t}{\Delta t}\right)$$

where, s = residual drawdown, (h_o - h_w), h_o is the original piezometric head, h_w = piezometric head at a certain moment during the pressure build-up, Q = flow rate, T = Transmissivity of the aquifer, and δt = time elapsed since the borehole was closed.

The form of the above equation is similar to Jacob's approximations to the Theis Well function (Jacob 1940, Cooper and Jacob 1946). In this

Figure 2.8a : TYPICAL PRESSURE RECORD FOR PRESSURE BUILD - UP AND RECOVERY TEST

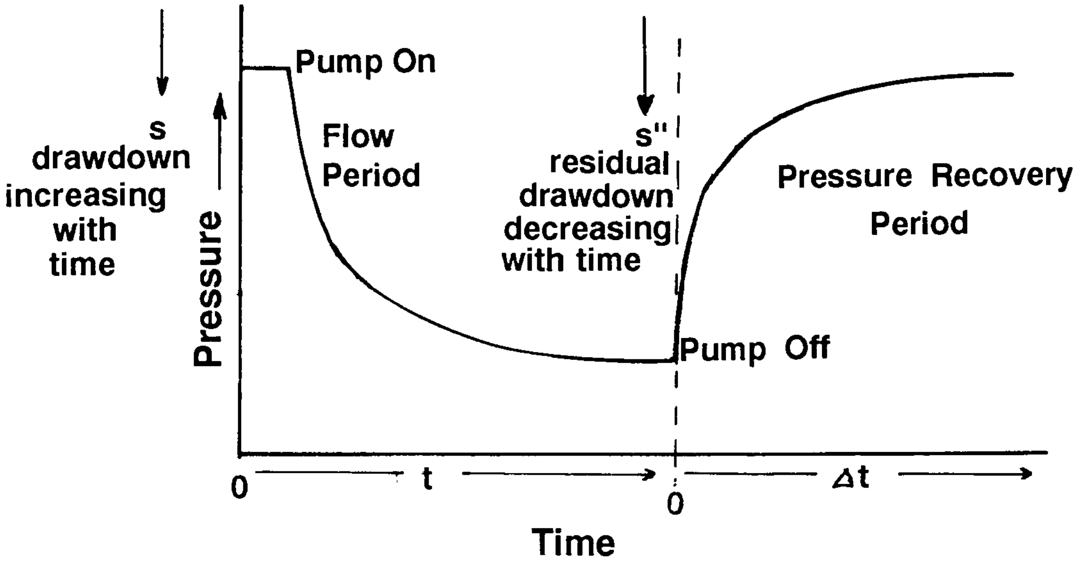
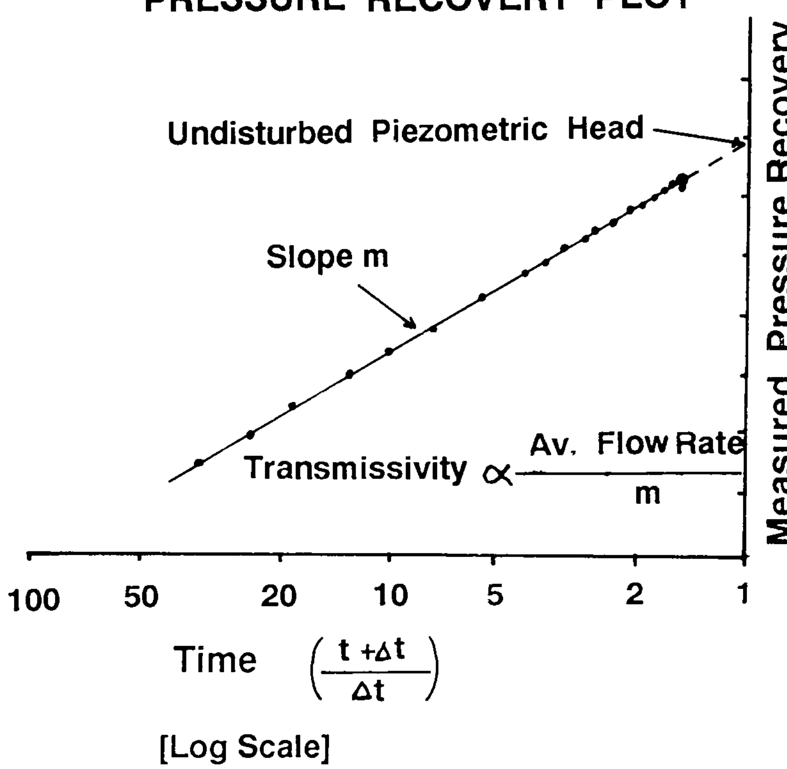


Figure 2.8b : GENERAL FORM OF HORNER PRESSURE RECOVERY PLOT



form, the piezometric pressure during the recovery period, h_w, is related linearly to the logarithm of the time function $(\frac{t + \delta t}{\delta t})$ and the corresponding semi-log plot is referred to as the Horner plot as shown in Figure 2.8b (Horner 1951).

The aquifer transmissivity T is determined from the slope m and the rate of withdrawal Q during the flow period.

$$T = \frac{2.303Q}{4\pi m}$$

where, m = piezometric head per log cycle.

In situations where multiple boreholes are present, drawdown data are plotted as per standard drawdown plot. Using The Theis equation for unsteady state flow in a confined aquifer, the plot is then superimposed on a prepared plot of W(u) vs. (1/u) values (Kruseman and De Ridder 1983). The type curve is matched to the data plot while keeping both axes parallel. Values of W(u), 1/u, t, and s are read from match point on the plots and substituted into:

$$T = \frac{Q}{4\pi s} W(u)$$

where, T = transmissivity (L^2/T), Q = constant borehole discharge (L^3/T), s = drawdown in the observation well, and W(u) = well function (dimensionless).

The storage coefficient can be obtained by substituting into:

$$S = 4T(t/r^2)u$$

Where, S = storage coefficient (dimensionless), t = time since pumping started, and r = distance from the pumping well to the observation hole.

2.6.2 Boundary Conditions

These include sources of recharge and barriers. Surface water basin divides usually can be determined from existing topographical and geological maps. Recharge boundaries can be estimated from the groundwater level profile, tracer results and controlled pumping tests. However, in a mining environment there is an additional complexity. Apart from the subsurface groundwater boundaries not necessarily matching with the surface water boundaries, the groundwater boundaries change with time as a result of mining subsidence.

2.6.3 Surface Hydrology

River and stream patterns can be obtained from topographic maps and aerial photographs. River flow measurements can be divided into two categories; viz periodic measurements and continuous measurements.

Periodic measurements can be made with a variety of portable flumes, examples of which include V-notch, trapezoidal and Parshall flumes (US Bureau of Reclamation, 1975). Flumes are best suited for low flows.

A semi-permanent to permanent installation can be made with a weir. Weirs can be used most effectively when there is a fall of 0.15m or more available in the discharge channel (US Bureau of Reclamation, 1975).

Flow meters, stage recorders and tracer velocity or dilution techniques, are most suited to the measurement of large flows. They are the most widely used methods for measuring river flow and the only suitable technique in turbulent flow.

Stream flow usually is calculated using the **mid-section method**. This method requires that the fluid velocity be measured at prescribed depths along a number of verticals on the cross section of the stream. The mean of the velocity is multiplied by the total water depth at the vertical and by the sectional width extending halfway to the verticals on each side of the vertical just measured. Flow is calculated as follows:

$$q = \frac{V_1+V_2}{2}\left[\frac{(L_2-L_1)+(L_3-L_2)}{2}\right] \times d_2$$

where: q = Flow through the section (L^3/T), $V_1 + V_2$ = Fluid velocities at vertical L_2 at prescribed depths of 0.2 and 0.8 or 0.6 of the depth of the water (L/T), L_1, L_2, L_3 = Distances from the initial point for any three consecutive verticals (L), and d_2 = The water depth at vertical L_2 (L) (US Bureau of Reclamation, 1975).

The total stream flow is the summation of the flow through all the sections.

River flow pattern as well as measurement of volume of flow, are important with respect to determining the influence of geological structure and estimating leakage. Areas where the river or any body of surface water crosses faults or highly permeable strata, are potential sources of surface water leakage into the groundwater flow system.

Most mines have rain gauge stations on site. For those that do not have, rainfall records can be obtained from appropriate government departments.

Existing data usually are analysed by one of three methods:

(i) **The Station Average Method:** All the depth of precipitation at each station are summed up and the sum divided by the number of stations as follows:

$$P_{avg} = \frac{\Sigma_{i=1}^{N} P_i}{N}$$

where, P_{avg} = Average precipitation at station i (L), and N = The total number of stations.

This method is applicable where there are a relatively large number of stations uniformly distributed across the area and the area has little topographic relief (Hjelnfelt and Cassidy, 1975).

(ii) **Thiessen Polygon Method:** This method weights the precipitation at each station in direct proportion to the area it represents without consideration of topography. The stations are plotted on a map and perpendicular bisectors are drawn on the lines connecting the stations. The polygons formed by these bisectors constitute the area that the precipitation for that stations represents. The overall precipitation can be calculated with the following equation:

$$P_{avg} = \frac{\Sigma_{i=1}^{N} P_i A_i}{\Sigma_{i=1}^{N} A_i}$$

where: A_i = The area assigned to station i (L^2), (Hjelnfelt and Cassidy, 1975). All other variables are as described in the Station Average method.

(iii) **Isohyetal Method:** This is the most accurate method for averaging precipitation over an area (Linsley *et al*, 1958). This method is the only technique that allows consideration of orographic or other topographic effects in the computation of an overall precipitation value. Contours of equal precipitation are drawn on the map using the data from each station. The area within each set of contours is measured and multiplied by the average precipitation for that area. Each area is summed and the total averaged by the following equation:

$$P_{avg} = \frac{\Sigma_{j=1}^{m}[(P_j+P_{j+1})(\frac{A_i}{2})]}{\Sigma_{j=1}^{m}A_j}$$

where: P_j = The precipitation at contour j (L), A_j = The area enclosed between contours j and j=1 (L^2), m = Total number of contours (Hjelnfelt and Cassidy, 1975).

2.6.4 Local Water Budget

The local water budget of an aquifer refers to the long term allocation of the available inflow water from precipitation, regional flow or recharge sources to components of natural or artificial discharge, for example, mine pumping. The goal of a water budget is generally to determine the perennial or seasonal replenishment that a mine drainage system has to handle after a requisite amount of local aquifer storage is depleted.

2.6.5 Origin and Flow Pattern of Mine Water Inflow

Dissolved constituents in the water provide clues on its geologic history, the soil and rock masses through which it has passed, and its mode of origin within the hydrologic cycle (Freeze and Cherry, 1979; Mazor, 1972 and 1991).

Therefore, the source of water and its flow path can be traced using the water chemistry, provided the characters revealing this can be reliably discriminated. The whole mine should be sampled in a limited period of time so as to provide a geochemical snapshot of the whole mining area. Measurements of temperature, pH, total dissolved solids, conductivity, dissolved ions, dissolved oxygen, simple tracers and careful use of isotopes, should provide enough data to carry out the analysis. Radioactive isotopes, Tritium (^3H and Carbon 14(^{14}C) are used to determine the age of the water. Non-radioactive isotopes; Oxygen 18(^{18}O) and Deuterium (^2H) serve mainly as indicators of groundwater source (Mazor, 1972).

The pattern of flow deduced by these methods can then be compared with that derived from a knowledge of hydraulic potential, as measured from piezometers and similar installations.

2.6.6 Groundwater Level Profile

Piezometers and boreholes fitted with gate valves for measuring pressure heads are the most widely used tools. The conventional methods of test drilling not only provide information for the definition of the hydrostratigraphic units, but also information on hydraulic gradient.

It should be pointed out that the primary purpose of a mine water inflow problem investigation is to determine the conditions and parameters which have the significant bearing on the problem. Since the accuracy of geologic and hydrologic data is usually low, it is inappropriate to use highly sophisticated and expensive methods to evaluate conditions and formulate a solution.

2.7 MATHEMATICAL MODELLING OF MINE WATER INFLOW

Following a reliable interpretation of site investigation data the hydrogeological conditions in a mine area can then be modelled.

There are broadly two mathematical approaches:

(i) Analytical methods
(ii) Numerical methods.

Analytical methods are based on simple groundwater flow formulae, commonly derived for the analysis of flow to a well. While these methods are relatively cheap and easy to use, they are generally only suitable for simple flow problems involving homogenous hydraulic conductivity conditions and geometrically regular boundaries. The typically complex conditions that characterise many practical mine inflow problems cannot be described (Lloyd and Edwards, 1988).

Numerical methods involve the mathematical solution of flow problems, usually by digital computer. The advantage of numerical modelling over analytical methods lies in the greater range of ground and boundary conditions that may be represented in a mine inflow problem. Provided that sufficient data are available, this allows a more detailed representation of the groundwater system in the mine environment and leads to more accurate predictions (Lloyd and Edwards, 1988).

Several mathematical modelling methods have been developed for the numerical solution of groundwater flow problems. These include finite-difference methods (Remson *et al*, 1971; Rushton and Redshaw, 1979), finite-element methods (Huyakorn and Pinder, 1983) and the boundary integral equation method (Liggett, 1977).

However, only a limited number of these packages have been used successfully to model mine water flow because of the difficulties involved. In simulating groundwater flow in a mine environment, the major difficulty is accommodating the ever changing pattern of hydraulic properties and boundaries as caused by the effect of mining. Some examples of modelling packages that are known to have been used to model mine water flow are as follows:

(i) **UNSAT2.** This package was used at a Uranium mine in New Mexico and Lead mine in Missouri, USA (Williams *et al*, 1986).
(ii) **WATGEN:** Used at a mine in Pennsylvania, USA.
(iii) **MODFLOW:** Used at the Dikuluwe-Mashamba copper-cobalt mines in the Kolwezi area, in Zaire, Central Africa.

Models are used primarily for making predictions of discharge and drawdown. The reliability of predictions from a groundwater model depends on how well the model approximates the field situation. Inevitably, simplifying assumptions must be made in order to construct a model because the field situation is too complex to be simulated exactly.

A detailed discussion of numerical modelling of mine water flow is given in the appropriate section of the thesis.

PART II

KONKOLA MINE GEOLOGICAL AND HYDROGEOLOGICAL SETTING

The effects of rock blasting and extraction and their consequent effects of stress relief on the rock mass in a mine makes the study of groundwater flow in a mine environment complicated.

Much as geology is the framework upon which the groundwater flow pattern and volumes of flow depend, the effects of mining activity forces the hydrogeological regime to change with time as well as with level of mining activity. For instance, hydrogeological boundaries that may have prevailed prior to the onset of mining may change after the commencement of mining.

Thus, in order to establish why the problem exists at all, that is the cause of continued large water inflows into Konkola Mine, it was essential that a clear understanding of the hydrogeological regime and setting be obtained, and conceptual model of groundwater flow developed.

Study of the geological history, stratigraphy, hydrostratigraphy, structure and its influence on the surface drainage pattern, would reveal potential major groundwater flow routes and likely sources of recharge, and facilitate the delineation of hydrogeological boundaries. Hydrogeological boundaries are essential to defining the effective groundwater catchment and estimation of the water budget. Water level maps drawn using mine records gave an insight into the behaviour of groundwater flow regime with time and mine level depth. Groundwater chemistry gave the means through which groundwater flow paths and patterns, and the overall nature of the flow regime could be defined and confirmed.

This part consists of three chapters; Chapters 3, 4 and 5. Chapters 3 and 4 deal with the geological and hydrogeological setting of the mine and estimation of the water budget. Chapter 5 analyses the results of field and laboratory investigations in mine water chemistry and rock chemistry and development of the conceptual model of the groundwater flow regime at the mine.

CHAPTER 3

KONKOLA MINE SETTING

3.0 INTRODUCTION

Although Konkola mine is not the largest producer of copper in Zambia, it has the largest known copper ore reserves in the country as well as one of the largest in the world. Proven reserves currently stand at over 100 million tonnes at an average grade of 3.5% total copper, enough to last at least the next 50 years.

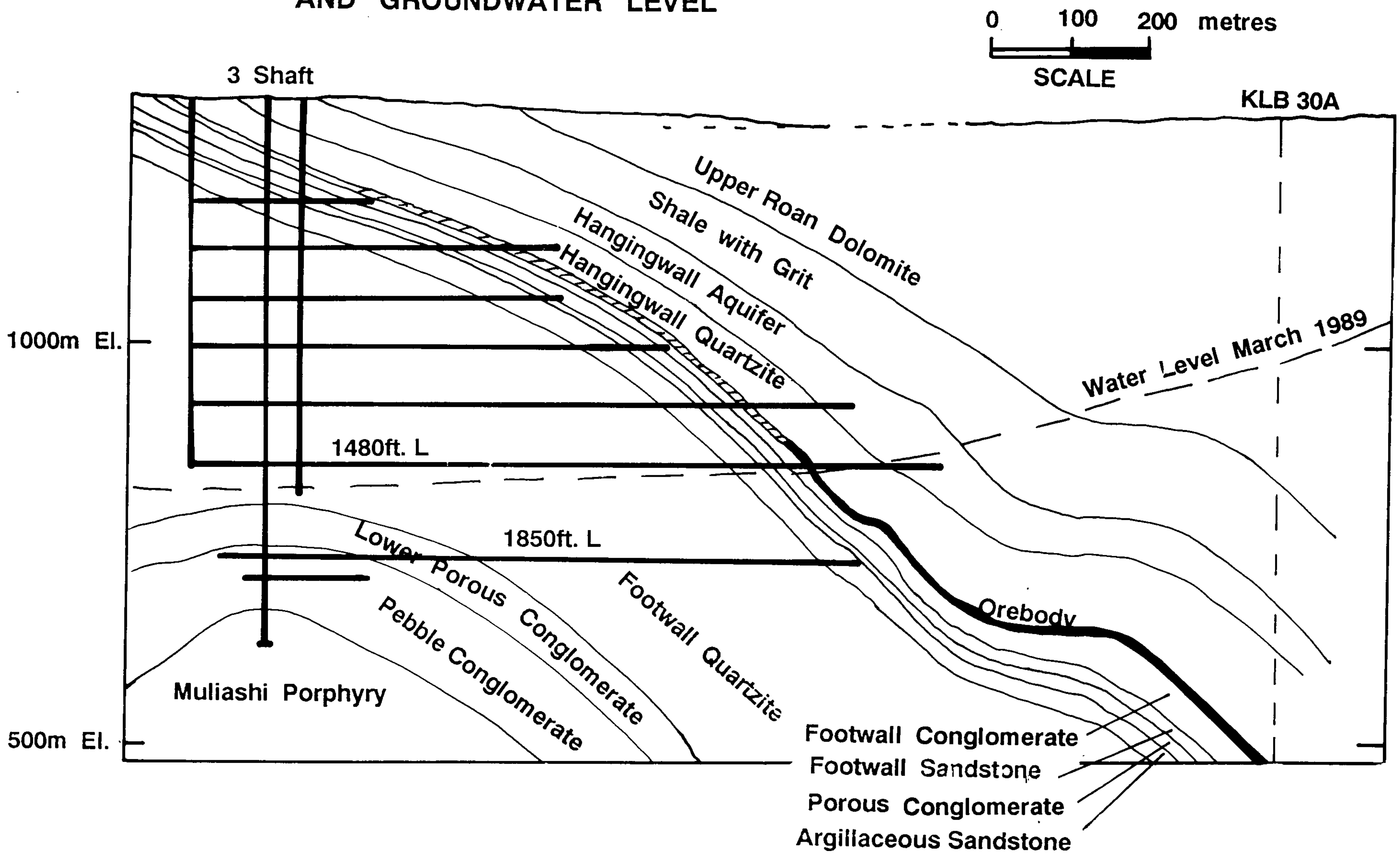
Monthly average production hovers around the 150,000 tonnes mark. However, production has recently been scaled down by about a third, mainly due to high dewatering costs.

There are three orebodies, the Kirilambombwe, Konkola and Fitwaola. Konkola orebody extracted through Number 2 Shaft was closed in 1958 and has not been opened since. Fitwaola orebody was never developed beyond the exploration/feasibility phase. The Kirilabombwe Orebody has a strike length of over 12km and has thickness ranging from about 5m to 30m. It is subdivided into the North and South orebodies. The North Orebody is mined through Number 3 Shaft and the South Orebody through Number 1 Shaft.

Number 1 Shaft is the deepest and currently stands at 1080m Level (3520ft.) below surface as shown in Figure 1.1 in Chapter 1. Number 3 Shaft is 915m Level (3020ft.) deep (Figure 3.1). Main production is at 780m Level (2650ft.) at Number 1 Shaft and 450m Level (1480ft.) at Number 3 Shaft.

Konkola mine is located in Chililabombwe district (formerly called Bancroft). It lies on a plateau 1300m above mean sea level as shown in Figure 3.2 in Appendix 1.

Figure 3.1 : KONKOLA MINE - GENERALIZED GEOLOGICAL SECTION AT NUMBER 3 SHAFT SHOWING OREBODY AQUIFERS AND GROUNDWATER LEVEL



3.1 GEOLOGY

3.1.1 Stratigraphy

The rocks present in the area range stratigraphically from the Archean Basement Complex to the late Precambrian Katanga System as shown in Figure 3.3 in Appendix 1 and in Figure 3.4 (Garlick 1961, Drysdall *et al* 1972). The Basement Complex rocks are mainly granites, gneisses and schists. The Katanga System is composed of sedimentary rocks, ranging from quartzites, conglomerates, sandstones, siltstones, to dolomites and limestones.

As illustrated in the stratigraphic column, Figure 3.5, the Katanga System lies unconformably over the Basement Complex. Ore mineralisation is in the ore Shale formation, which is part of the Lower Roan succession of the Katanga System. The formations lying below the Ore Shale are referred to as Footwall and those above as Hangingwall. Footwall formations are composed of siliceous rocks; mainly quartzites, feldspathic sandstones and conglomerates. Schwellnus (1961) and Fleischer *et al* (1976) note that the pebbles and boulders in the footwall formations are predominantly quartzite and grey granite. Hangingwall rocks consist mainly of dolomites, limestones, dolomitic siltstones and shales.

3.1.2 Hydrostratigraphy

(a) Aquifers

There are four main aquifers; two above and two below the orebody (Figure 3.5).

(i) **Footwall Quartzite Aquifer** This is the lower most aquifer. It consists of the Lower Porous Conglomerate and the lower part of the footwall Quartzite. It is about 150m thick. Large volumes of water are intersected in this aquifer.

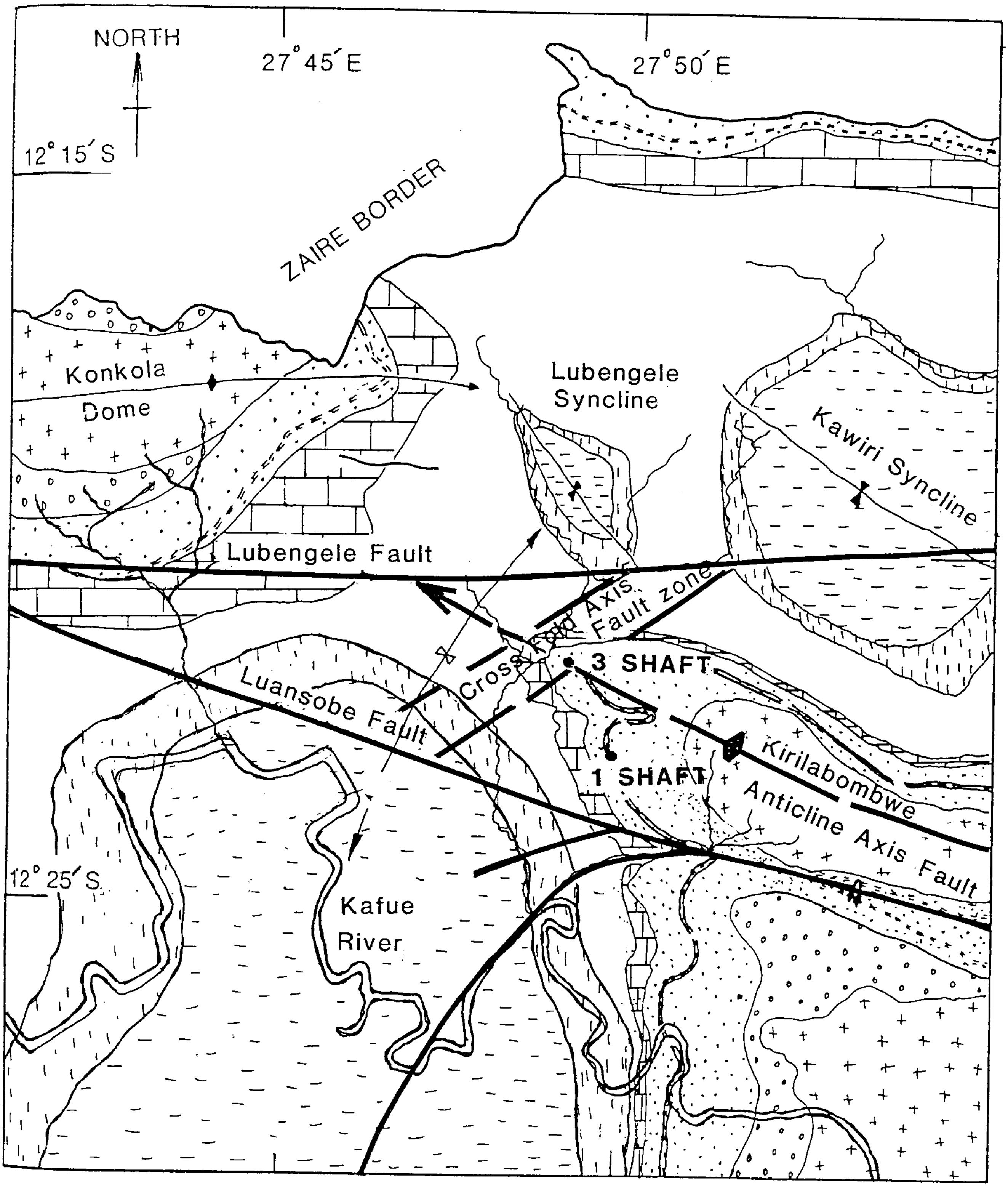


Figure 3.4 : GEOLOGICAL MAP OF KONKOLA MINE

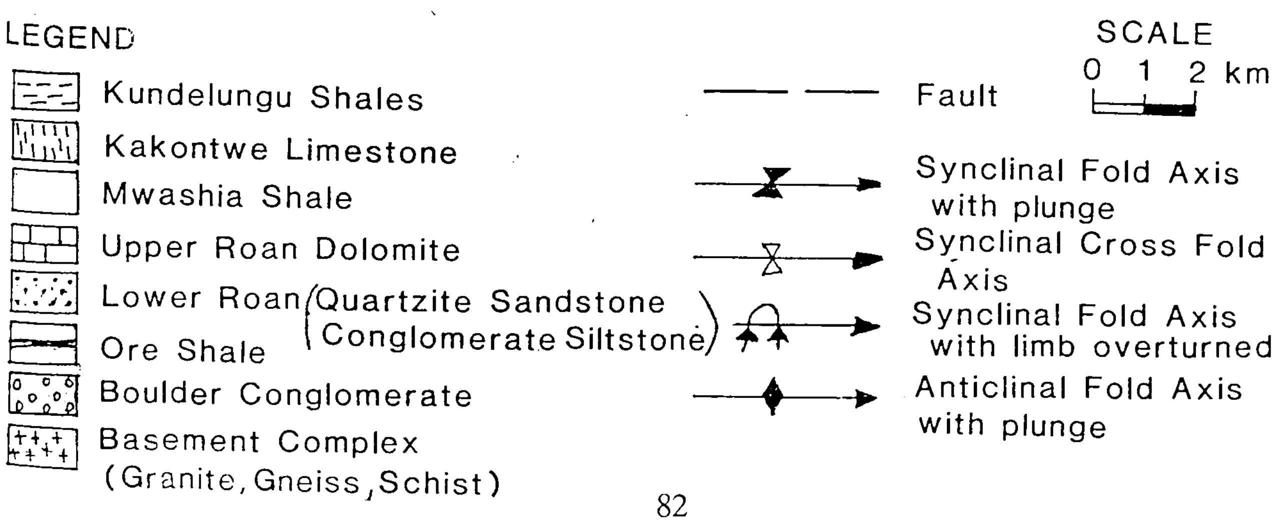
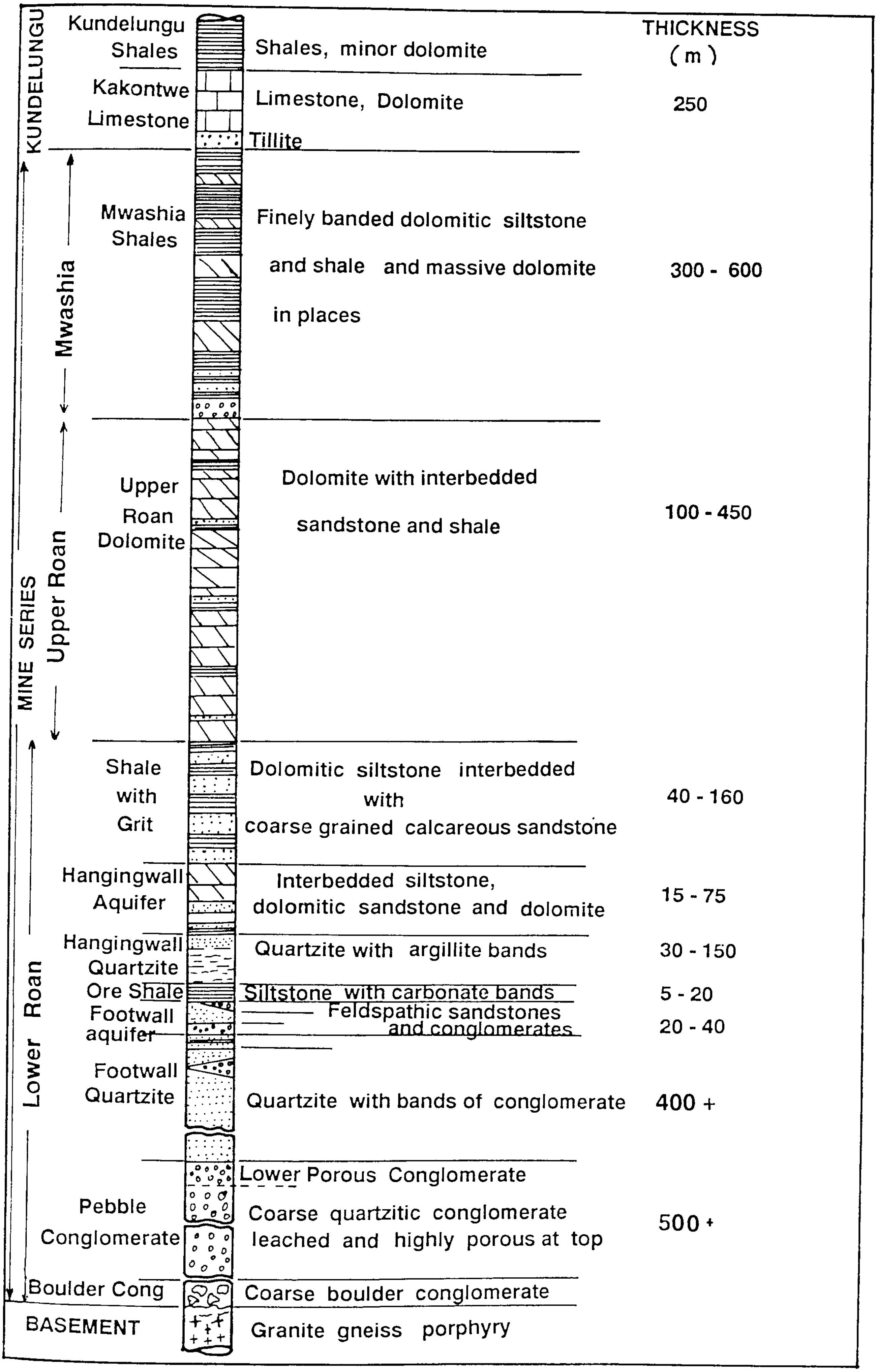


Figure 3.5 : KONKOLA MINE STRATIGRAPHIC COLUMN



The lower part of the Footwall Quartzite is heavily jointed and fissure controlled. In the Lower Porous conglomerate water is mainly confined to vugs and predominantly porous flow. The Lower Porous conglomerate because of its sponge-like water bearing characteristics, drains at a very slow rate as compared to the jointed and fissured Footwall Quartzite.

Two major breakthroughs into open fissures have occurred in the past. In 1976 at Number 3 Shaft on 1240ft. Level, 60,000 m^3/d at a pressure of 11bars (110kN/ m^2) was intersected in a main haulage development. The water continued to flow from the fissure at an average rate of 30,000 m^3/d and did not dry until seven years later in 1983. In 1982 at Number 1 Shaft on 2900 ft. Level, a 72mm diameter pilot borehole drilled from the Drain Crosscut West through the Lower Porous conglomerate, intersected 11,300 m^3/d at a pressure of 37bars (370kN/m^2). Mining was halted for several days before the hole could be grouted.

(ii) **Footwall Aquifer** The Footwall Aquifer is composed of three formations, the Porous Conglomerate, Footwall Sandstone and Footwall Conglomerate. This aquifer directly underlies the orebody. The aquifer has a thickness of about 20m to 40m.

The Porous and Footwall Conglomerates are lithologically similar. They are composed of fairly well sorted feldspar and quartz pebbles ranging in diameter from 2.5cm to 7.5cm. The matrix is siliceous. Footwall Sandstone is mainly medium to coarse grained and feldspathic with interbedded argillite and conglomerate bands.

Water flow in this aquifer is stratigraphically controlled and is generally easily drained. This is due to the high porosity and permeability of the matrix. The average yield for an advancing face, with dimensions 3m x 4 m, development in this aquifer hovers around the 5,000 m^3/d. However, in fissured zones up to 10,000 m^3/d can be intersected.

(iii) **Hangingwall Aquifer** This aquifer lies above the orebody. Because it is situated close to and above the orebody, this aquifer creates the greatest water inflow problems at the mine. It is about 70 m thick and outcrops in the immediate vicinity of the mine area.

The aquifer is composed of the Hangingwall aquifer, Shale-with-Grit, and Upper Roan Dolomite formations.

(iv) **Kakontwe Limestone Aquifer** The Kakontwe Limestone outcrops very near to the mine. It has an average thickness of 250m. Since the Kafue River flows over an extensive area of this aquifer, most of which lies in the Luansobe Fault zone, the bulk of any river leakage into the mine is most likely to take place here.

(b) Aquicludes

Figure 3.5 shows the three important aquicludes. These are the Argillaceous Sandstone and unfractured part of the Footwall Quartzite, Hangingwall Quartzite and Mwashia Shale. However, because of the present of open joints, fissures and fracturing caused by faulting and blasting, there is hydraulic interconnection between aquifers. Records of water level during exploration borehole drilling shown in Figures 3.6a to 3.6c, clearly demonstrates the hydraulic interconnection of aquifers. Hence these aquicludes are essentially leaky aquicludes.

Figure 3.6a : PLOT OF GROUNDWATER LEVEL BEHAVIOUR DURING DRILLING OF BOREHOLE KLB86 - CROSS ANTICLINE AXIS & LUBENGELE FAULT ZONES

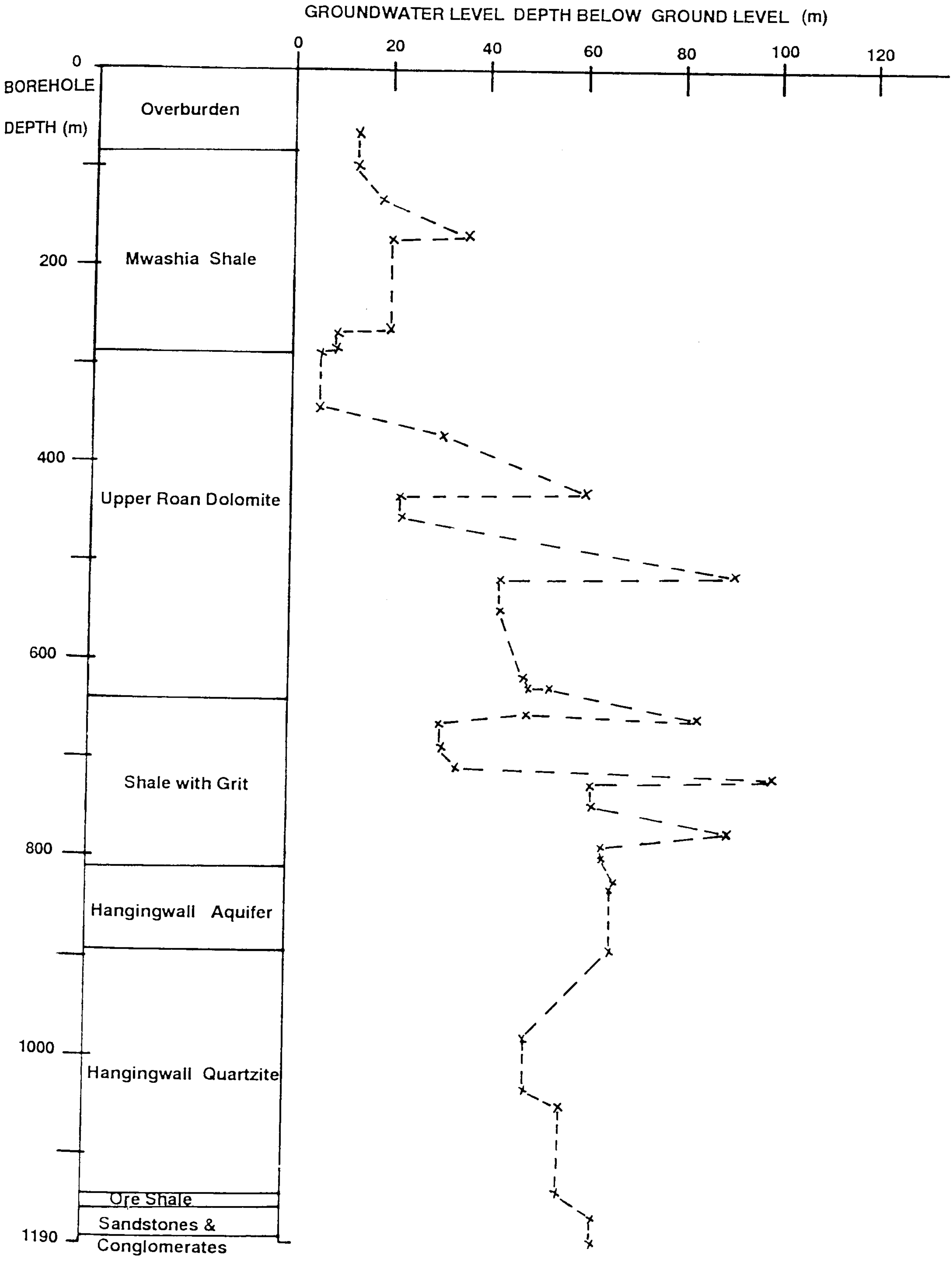


Figure 3.6b : PLOT OF GROUNDWATER LEVEL BEHAVIOUR DURING DRILLING OF BOREHOLE KLB84 - CROSS ANTICLINE AXIS FAULT ZONE

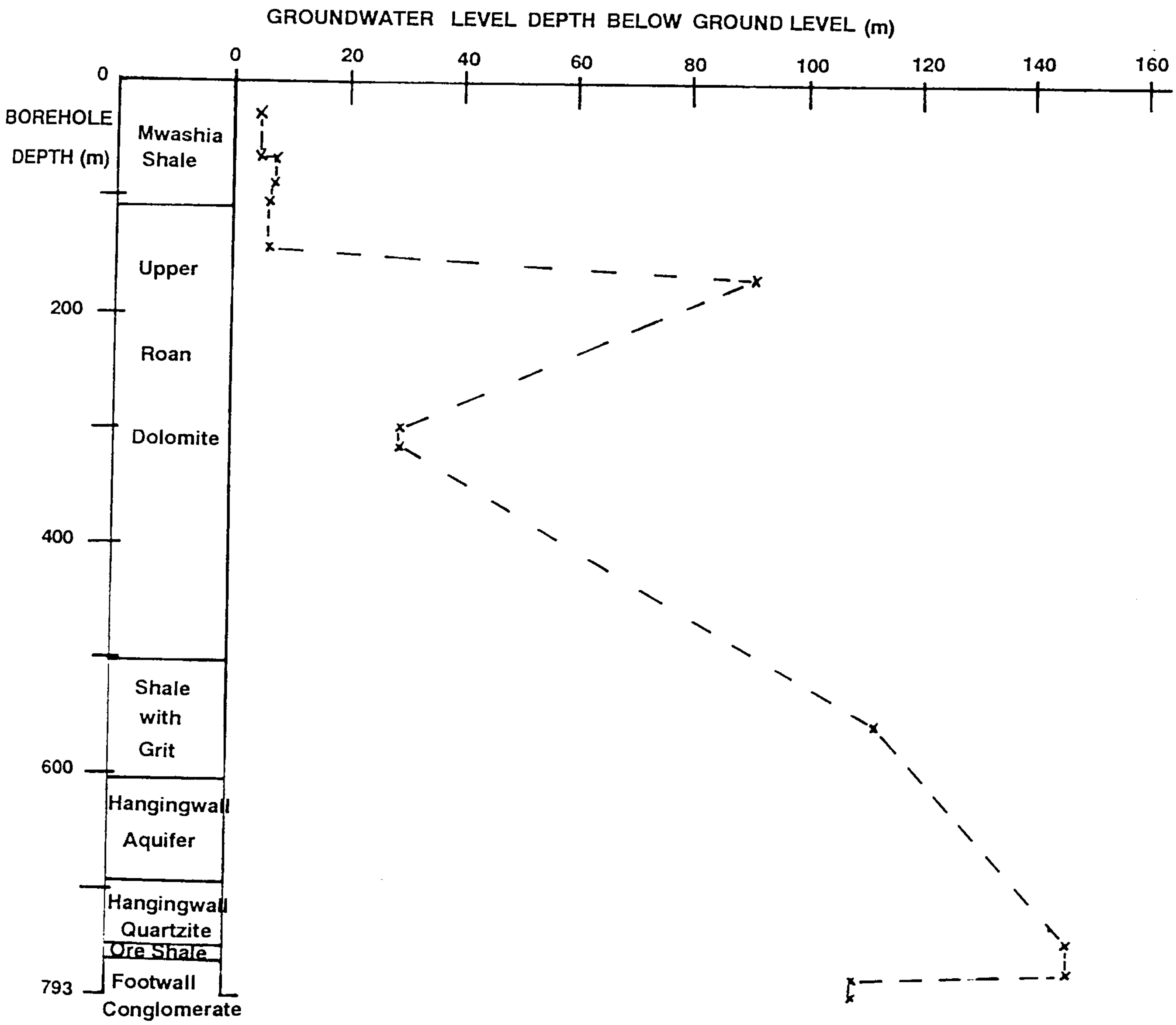
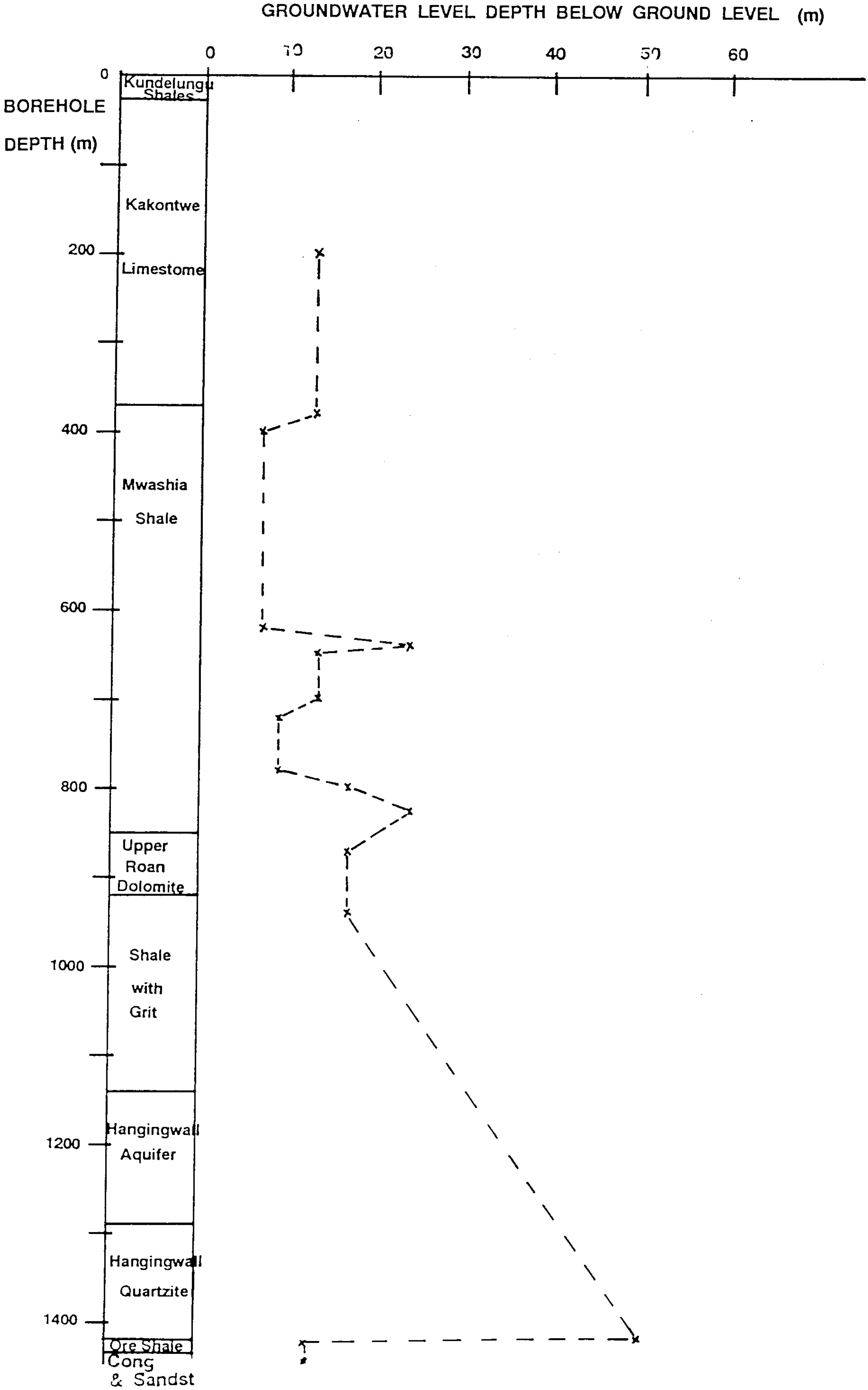


Figure 3.6c : PLOT OF GROUNDWATER LEVEL BEHAVIOUR DURING DRILLING OF BOREHOLE KLB94 - LUANSOBE FAULT FRINGE ZONE



The records of water level behaviour during drilling show that as we go down from ground level, we go through zones which allow water to be lost from the borehole and therefore fairly permeable. These zones are interspaced with otherswhere groundwater level rises in the borehole, indicating that they are separated from the draining zones by relatively impermeable thickness of strata and they themselves contain water under greater pressure head than the overlying aquifer.

There are also zones where no substantial changes of pressure heads occurs with depth, indicating either no flow of groundwater or equipotential surfaces which are essential vertical, as would occur with horizontal flow.

Outside the area of mining influence the strata is generally less disturbed and there is a distinct difference between the hydrostratigraphic and lithostratigraphic units respectively. However, in the mine area, where the rock has been further fractured by blasting, there is no difference between lithostratigraphic and hydrostratigraphic units. Fractures have created hydraulic connection between the strata.

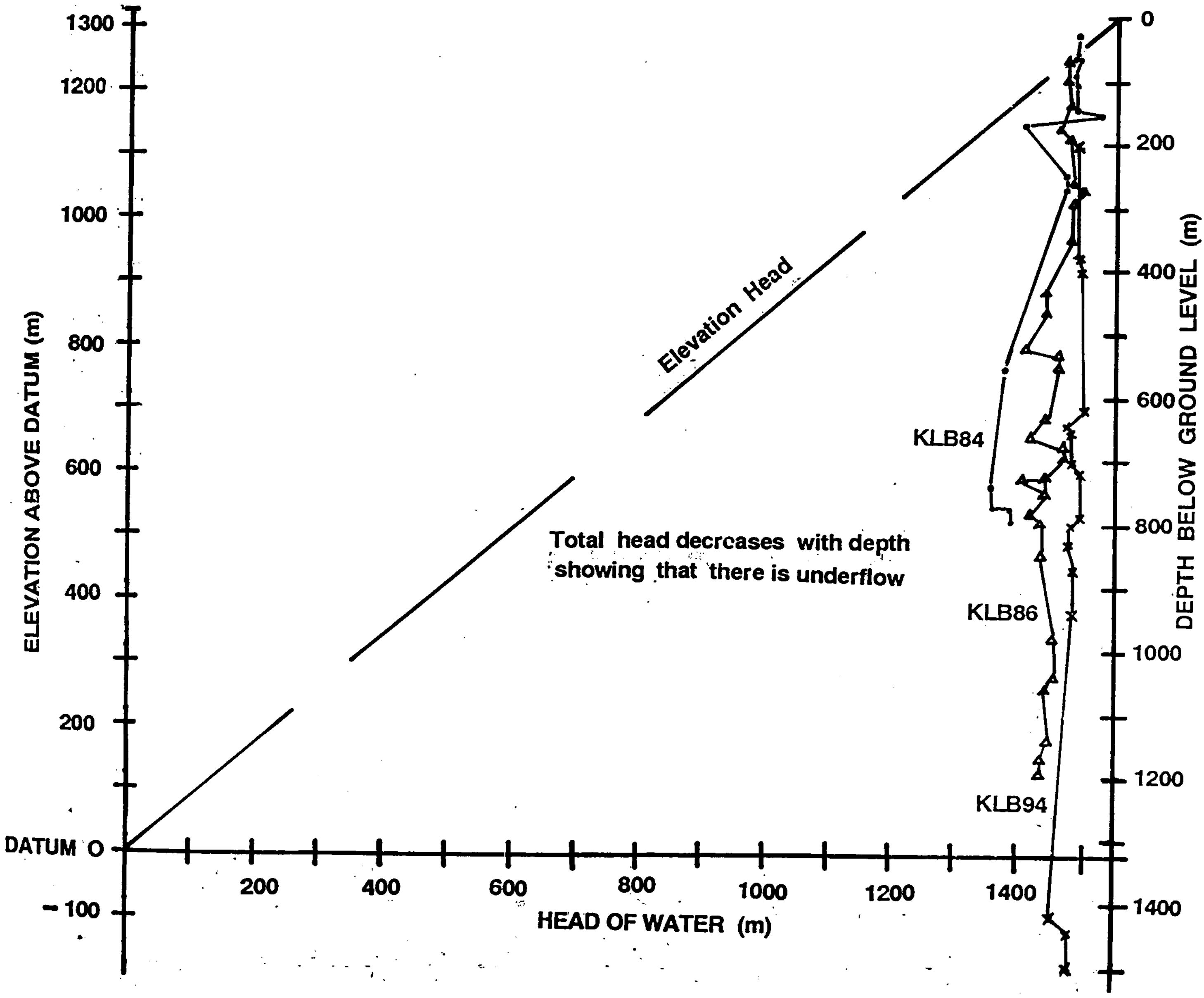
A plot of the behaviour of boreholes' groundwater total head during drilling (Figure 3.6d) show that there is underflow. This demonstrates that one is dealing with a water level situation controlled by drainage at depth and is not hydrostatic.

3.1.3 Structure

The geological structure of the Konkola mine area comprises a system of west-northwest plunging folds forming part of the anticlinorial complex known as the Kafue Anticline (Figure 3.3 and 3.4). The Kafue Anticline is the dominant regional structure on the Zambian Copperbelt (Daly 1986, Coward and Daly 1984, Daly *et al* 1984 and Garlick 1961).

Figure 3.6d : COMPOSITE DIAGRAM SHOWING BEHAVIOUR OF GROUNDWATER LEVEL DURING DRILLING OF BOREHOLES KLB86 KLB84 & KLB94

All borehole ground levels about the same elevation above sea level



The major structures at Konkola are the Kirilambombwe Anticline which forms the northern tip of the Kafue Anticline, the Lubengele Syncline and the Konkola Dome as shown on the geological map. The Kirilabombwe anticline has a plunge of about 60° in the northwest direction. Beds dip about 8° in the nose area to up to 75° on the limbs extremities.

The mine is wedged between two major faults, the Lubengele in the north and the Luansobe in the south as shown on the geological map. In between these two faults are the Kirilabombwe Anticline Axis Fault and the Cross Anticline Axis Fault. These faults have been intersected in the mine as shown on mine level plans in Figures 3.7a to 3.7d in Appendix 1.

The Lubengele and the Luansobe faults form a conjugate shear set with predominantly lateral movement direction. The Lubengele dips steeply to the south at about 75° and has created a heavily fractured zone on the north limb of the North Kirilabombwe Orebody (Number 3 Shaft). The Luansobe dips north with an average dip of 85°. These faults are part of the regional "phantom" fault that extends southeast to Mufulira mine and northwest into Zaire. The Luansobe fault has created a brecciated zone of more than 2000m strike length on the southern extremity of the South Kirilabombwe Orebody (Number 1 Shaft). This breccia zone is called the "Duplicate Orebody".

The Kirilabombwe Anticline Axis Fault has a subvertical dip. It has formed a graben-like structure with vertical displacements ranging from about 1.5m to about 6m. The Cross Anticline Axis Fault zone, is made up of a series of host and graben-like structures. It dips steeply in the southeasterly direction and has vertical displacement of up to 12m.

The major joints trend at N135°, approximately parallel to the Anticline axis. The minor joint direction of N045° parallels the Cross Anticline Axis Fault. These joints are mainly open and provide excellent channelways for water transfer between formations. Where closed they are infilled mainly with carbonates, vein quartz, clay, specularite and secondary copper miners. These joints represent typical tension joints.

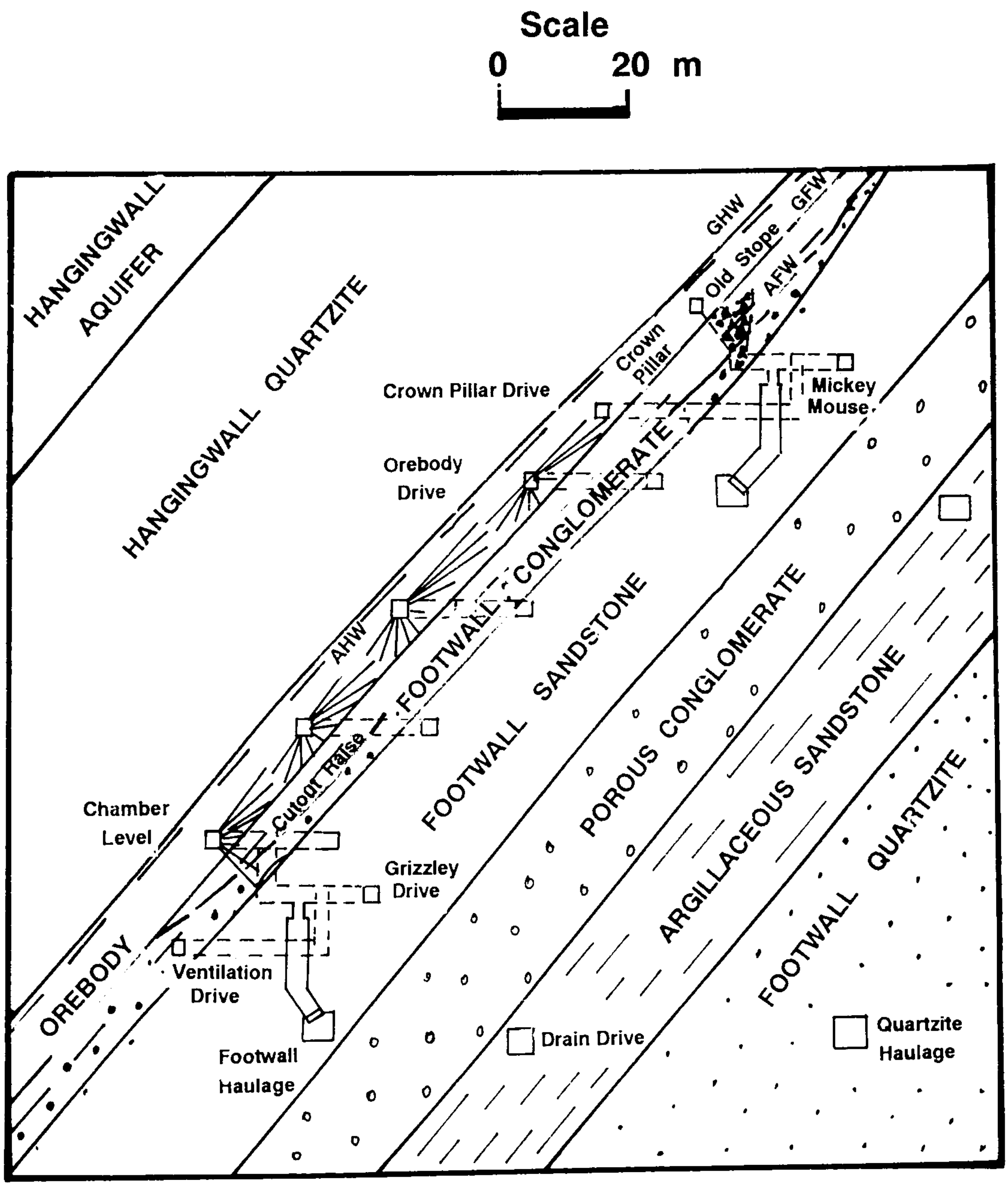
Preliminary analyses of structural geology suggests sinistral movement over a long period of time. Both ductile and brittle forms of deformation exist and can be seen clearly in the mine, all having orientation in harmony with the patterns of strain occurring in the ground between the Luansobe and Lubengele Faults during their period of sinistral movement. If the ductile deformation occurred at depth and the brittle deformation at higher levels in the crust it can be argued that the area has been progressively strained during a long period of uplift. A rational basis for explaining the detailed structure in the mine thus begins to emerge upon which an understanding of fractures and their water-bearing characters can be based.

The structures in the Konkola mine area were formed during the Lufilian orogeny (Garlick 1961, Coward and Daly 1984). The Lufilian Arc defines a large arcuate orogenic belt containing the copper-bearing sediments of the Katanga Supergroup (Cahen and Snelling 1984, and Raybould 1978). Coward and Daly (1984) note that during the Lufilian orogeny (600 - 500Ma) the Katanga cover and Basement were deformed together. The Katanga sediments were pushed over the Basement. Movement was predominantly in the ENE direction. This has been well preserved in the orientations of the discontinuities at the mine.

3.2 MINING METHOD

The method of ore extraction is sub-level open stoping with longhole drilling. At Number 1 Shaft the orebody dips steeply. The dip ranges from 40° to about 80°. Ore gravitates into grizzleys and subsequently into loading boxes in the haulage (Figure 3.8a).

Figure 3.8a : Konkola Mine Mining Layout for Gravity Stoping at Number 1 Shaft



At Number 3 Shaft the orebody has a shallow dip. The dip ranges from about 8° in the nose area of the anticline to about 35° on the limb extremities. Ore is extracted by scraping down dip into a drawpoint (Figure 3.8b).

After the ore has been extracted, the open stope is filled by either hangingwall collapse or waste gravitating from old stopes. This is why the presence of a large aquifer above the orebody creates such a great problem at Konkola. The Aquifer has to be dewatered before the orebody can be safely mined (Mulenga and Shamutete 1984, Rijken and Clutten 1970).

3.3 HYDROLOGY

The drainage system in the mine area is controlled by the Kafue River, which is the second largest river in Zambia. The two important tributaries are the Lubengele and Kakosa streams. The mine is located in the Lubengele Catchment.

The Kafue (Plate 3.1) and the Lubengele flow over the hangingwall formations. The Lubengele Mine Tailings Dam (Plate 3.2) is located immediate north of Number 3 Shaft, and is built over the Lubengele Fault (Figure 1.2).

The angularity and linearity of river pattern seem to suggest that the river pattern in this area is controlled primarily by the fault system. Coward (1984) and Garlick (1961) note that this area lies in the Pan African regional ENE and WSW lineaments and discontinuities formed during the Lufilian orogeny. The river flow pattern fits this setting.

The zones in which the river and streams intersect the faults and fissures are areas of potential surface water leakage into the hangingwall aquifer and subsequently the mine.

Figure 3.8b : Konkola Mine Mining Layout for Scraping Stopes at Number 3 Shaft

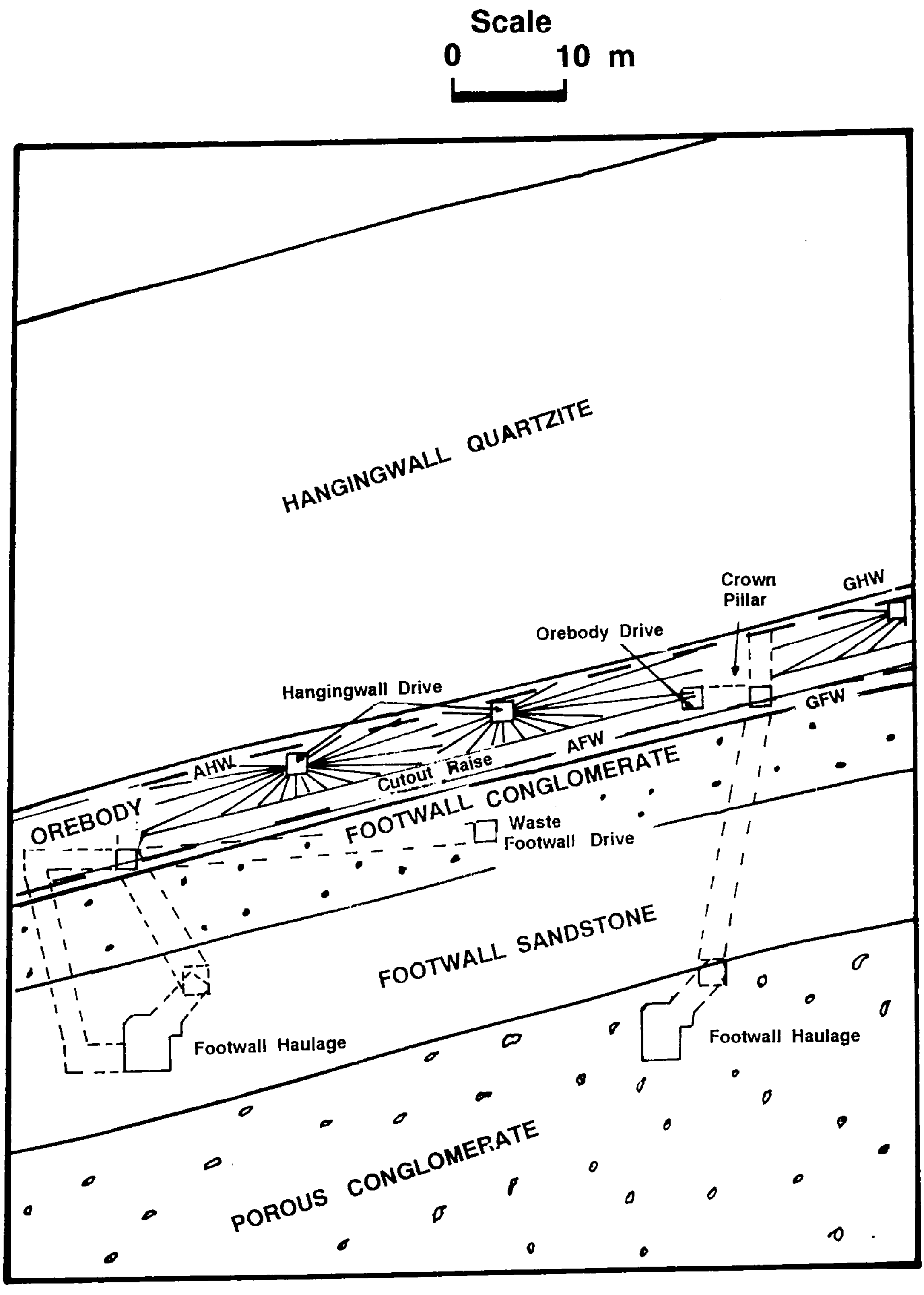


PLATE 3.1 : KAFUE RIVER AT KONKOLA MINE

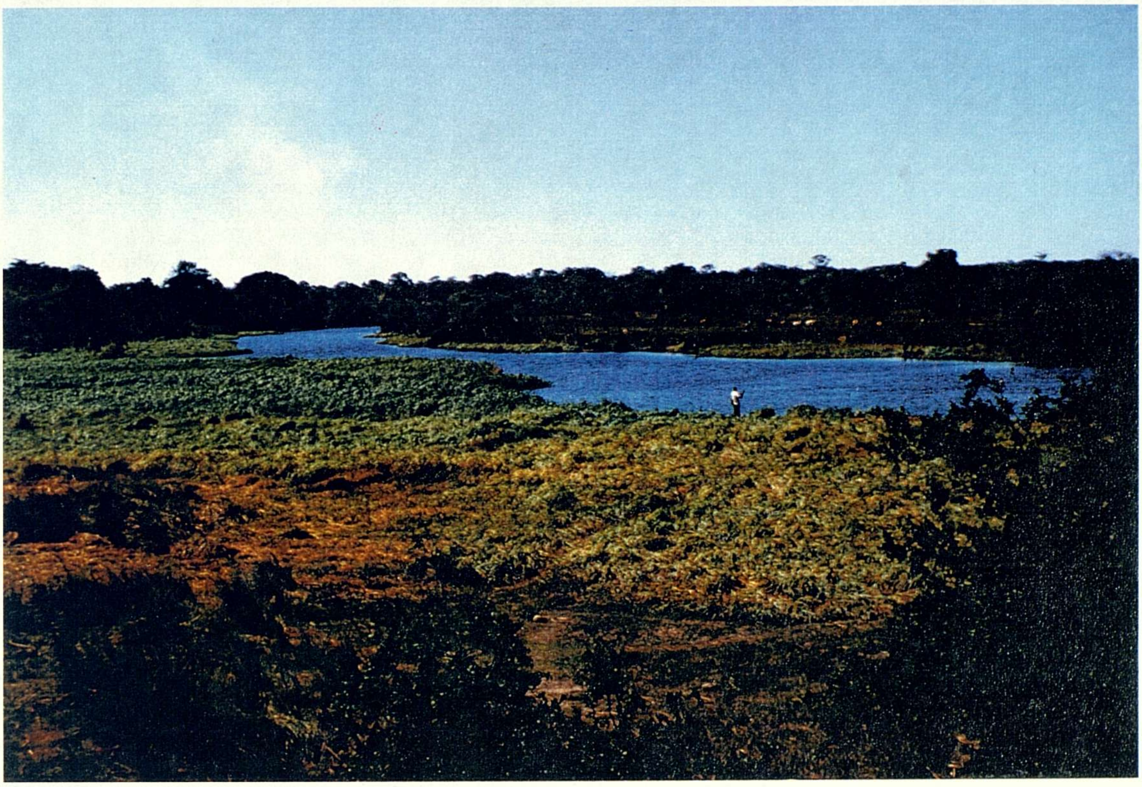


PLATE 3.2 : THE LUBENGELE MINE TAILINGS DAM IMMEDIATE NORTH OF KONKOLA MINE NUMBER 3 SHAFT

3.4 CONCLUSION

Combining the surface hydrology with the solid geology demonstrated, for the first time, that there is a direct relationship between the geological structure and surface hydrology. this strongly indicated that there was a potential leakage of surface water into the mine groundwater system through these channelways which had to be investigated. Equally important is the realisation that these faults form the major hydraulic boundaries of groundwater flow into the mine.

Although the distribution pattern of the aquifers and transmissive zones is now better understood, there is virtually no data on values of hydraulic conductivity, transmissivity and storage coefficients of the rock formations at the mine.

Verwey (1962) conducted groundwater flow velocity tests using sodium fluorescein as a tracer applied to a surface borehole. The tests were inconclusive as no trace of fluorescein could be found in any of the underground water samples. However, on the basis of experience gained working in similar environments, an estimated groundwater flow velocity of 1.2m/d towards the mine was proposed.

In order to accurately determine groundwater flow rates to the mine and predict drawdowns it is essential that representative values of hydraulic conductivity, transmissivity and storage coefficient of rock formations in the area of interest are obtained.

Tests had to be conducted at the mine during the research period in order to obtain representative values for the data required.

CHAPTER 4

KONKOLA MINE WATER BALANCE AND GROUNDWATER LEVELS

4.1 INTRODUCTION

This chapter focusses on determining the total volume of water inflow to the mine and the potential source(s) of the groundwater input. Groundwater level behaviour and mine pumping data are compared with rainfall in an attempt to ascertain the presence of any seasonal variations in quantities of water pumped to surface and water level drawdowns. The chapter concludes by stating what has been learnt from the review.

4.2 WATER BUDGET

The mine water budget refers to the long-term allocation of the available catchment inflow water from; precipitation, regional flow and other recharge sources to the mine. Primarily, the calculation of a water budget on a mine is aimed at determining the amount of groundwater inflow expected to enter the mine and dewatering that should be implemented to allow safe working conditions.

A mine catchment area (drainage basin) is the entire area providing run-off to, and sustaining part or all of the groundwater flow, to the mine water flow regime. In general, the catchment system can be visualized in terms of the water balance equation, which, according to Gregory and Walling (1976), may be stated as follows:

Run-off = Precipitation - Losses ± Change in Storage

where:

Run-off = output of water in surface channels

Precipitation = input

Losses = evaporation + transpiration

Similarly, this can be expressed as:

Inflow = Outflow + Change in Storage

where:

Change in Storage = Natural Drainage + Mine Pumping

The catchment area must be delineated as precisely as possible. However, difficulties do arise in accurately defining the effective catchment, as most catchments delineation are based on topographic maps without due consideration to the subsurface groundwater divide.

Therefore, in view of the existence of surface run-off and groundwater flow it is possible to have a drainage basin area on the surface that does not correspond to the boundaries of the basin below the surface. This is most likely to be the case in areas where geological rather than topographical structure has the greatest influence on groundwater flow. This appears to be the most likely setting at Konkola Mine as shown in the map relating surface hydrology and fault system, and in the geology and topographic maps respectively. The fault system crosses the adjacent topographic catchments and thus creates a groundwater catchment which is much larger than the local topographic catchment of the mine.

The parameters needed in the accurate determination of a mine water budget are as follows:

(i) Topographic (surface run-off) catchment area.
(ii) Groundwater catchment area, as determined by likely hydraulic boundaries and groundwater levels.
(iii) Rainfall.
(iv) Evapotranspiration.
(v) Infiltration.

(vi) River flow records, covering at least one cycle of all seasons.

(vii) Mine pumping figures and groundwater levels.

4.2.1 Topographic Catchment Area

Konkola Mine lies in the Lubengele-Kakosa streams watershed catchment (Figure 3.2 in Appendix 1). The catchment is bounded by the Kakosa stream on the east and by the Kafue River in the south. The total catchment area is about 187 km^2.

4.2.2 Rainfall

The climate of Zambia is divided into two major seasons. A hot and wet summer from about October to April, and a cool to warm dry winter from May to September. Summer temperatures are in the range of 25°C to 35°C and winter temperatures 10°C to 15°C.

Konkola Mine lies in the high rainfall belt of the country. Annual rainfall is in the range of 1000 mm to 1800 mm as shown in Table 4.1. The wettest months are generally December and January (Mine Rainfall Records 1953-1990).

Studies have been carried out in the past by different authors, in an attempt to calculate an overall water balance between precipitation, evapotranspiration, run-off and infiltration for the Konkola mine area. The results were as follows:

	Evapotranspiration	Run-off	Infiltration
Starmans and Shalash (1971)	76.4%	12.2%	11.4%
Leeds et al (1972)	80%	13%	7%

Recent review of these parameters by staff of the geology department at Konkola Mine indicate that infiltration is in the range of 7% to 12% of precipitation. Therefore, considering that the average annual rainfall at Konkola is 1310 mm, and average infiltration is 9.5% of precipitation, the estimated annual infiltration

Table 4.1 KONKOLA MINE RAINFALL RECORDS 1953-1990

(Measurements are in millimetres)

Year	Sept	Oct	Nov	Dec	Jan	Feb	Mar	Apr	May	Total per/year
1953-54	0	18.8	162.1	342.9	301.5	272.8	202.7	6.1	0	1306.9
1954-55	0	0	159.5	172.0	351.3	423.7	164.8	87.1	0.3	1358.7
1955-56	0	64.8	158.2	252.0	399.3	331.2	286.5	237.2	0	1729.2
1956-57	0	19.8	165.9	265.9	425.5	378.2	397.8	3.6	9.7	166.4
1957-58	0	25.7	46.7	310.4	291.1	144.3	154.2	13.5	0	985.9
158-59	3.3	19.3	113.0	332.0	109.5	257.3	174.8	49.5	0	1058.7
1959-60	0	18.3	82.0	332.2	347.5	262.1	237.7	33.3	0	1313.1
1960-61	1.5	21.3	136.7	403.1	270.3	294.9	275.8	90.9	8.1	1502.6
1961-62	0	39.4	276.9	295.4	368.6	405.6	234.7	180.3	0	1800.9
1962-63	0	1.0	200.4	344.2	319.8	227.1	293.9	27.2	0	1413.6
1963-64	0	23.9	163.6	193.3	312.2	200.7	78.7	0	0	972.4
1964-65	0	11.4	163.8	248.9	294.4	322.3	211.3	10.4	0	1262.5
1965.66	14.0	10.4	74.7	205.0	270.0	145.8	75.9	42.9	49.3	888.0
1966-67	0	0.5	57.9	216.4	241.8	374.4	320.0	12.4	4.6	1228.0
1967-68	3.0	143.5	171.2	91.4	267.5	129.0	172.9	49.8	4.6	1032.9
1968-69	1.5	7.9	146.6	487.4	387.1	302.0	175.5	174.2	0	1682.2
1969-70	0	50.5	120.7	305.3	276.1	306.6	27.9	18.8	0	1105.9
1970-71	30.5	33.0	234.7	233.9	295.7	247.1	187.2	47.8	1.3	1311.2
1971-72	4.1	13.2	151.9	335.3	280.5	193.3	228.3	29.9	20.8	1257.3
1972-73	3.8	57.7	118.8	170.4	164.6	186.4	171.7	24.9	0	898.3
1973-74	0	11.7	177.3	241.6	290.1	143.8	155.4	42.4	117.1	1179.4
1974-75	0	0.8	113.3	446.0	320.6	144.8	455.4	77.5	0	1558.4
1975-76	0	0	58.4	401.3	380.2	210.6	372.6	191.0	7.1	1621.2
1976-77	33.8	35.6	229.3	206.0	329.7	304.3	269.5	28.4	0	1436.6
1977-78	15.2	31.5	180.7	432.3	316.0	199.1	383.3	105.210	0	1669.3
1978-79	0	74.2	207.2	376.3	210.9	198.6	264.0	20.8	0	1352.0
1979-80	0	62.7	183.1	346.6	204.5	214.6	220.7	173.7	0	1406.0
1980-81	9.1	66.3	101.6	263.4	233.7	322.6	244.9	34.8	0	1276.3
1981-82	0	22.6	150.1	156.2	266.7	304.8	89.2	81.0	22.3	1092.9
1982-83	0	69.6	276.9	323.6	237.5	240.8	123.4	139.9	0.5	1412.3
1983-84	0	36.8	130.1	477.5	300.2	326.7	147.6	16.0	0	1442.9
1984-85	1.0	11.4	153.3	383.0	254.7	216.2	161.5	95.6	17.0	1293.8
1985-86	0	8.9	272.1	356.1	277.1	270.6	316.2	99.4	0	1600.4
1986-87	0	91.4	162.2	158.3	193.1	253.2	147.6	7.3	0	1013.1
1987-88	2.0	0	94.7	193.9	240.9	342.9	335.2	7.0	0	1216.6
1988-89	0	14.3	134.9	198.5	279.7	257.3	290.7	21.1	0	1196.5
1989-90	0	5.1	138.5	284.9	147.2	151.9	85.6	69.4	54.8	937.4

is about 125 mm. This gives an estimated total of about 23, 375,000 m^3 per ammum of water infiltrating into the mine, from the topographic catchment rainfall (Table 4.2).

4.2.3 Mine Dewatering

Mine dewatering is achieved by drilling boreholes in the aquifers and through gravity drainage resulting from normal mine development.

For each dewatering borehole, discharge is recorded and hydrostatic pressure head measured using a pressure gauge. These pressure heads are then used to plot groundwater profiles, for each aquifer.

Most of the dewatering drilling is concentrated in the Hangingwall Aquifer. The Footwall aquifers are mainly dewatered by gravity drainage consequent to normal mine development. Ten years ago in 1980, active dewatering drilling of the Footwall Quartzite Aquifer began.

In most cases it takes about five to ten years for these dewatering boreholes to run dry.

The total volume of water pumped from the mine as at end of June 1990 averaged 321,650 m^3/d. Of this, 35.1% (112,899 m^3/d) was intersected in Hangingwall Aquifer drilling, 29.1% (93,600 m^3/d) in Footwall Aquifer, and 35.8% (115,151 m^3/d) in Footwall Quartzite Aquifer.

4.2.4 Mine Water Balance

If we take the ten years from 1979-80 to 1989-90, as an example, the annual volume of water pumped from the mine during this period, has been in the range of 125,000,000 m^3 to 147,000,000 m^3. Of this topographic catchment rainfall accounted for 16,000,000 m^3 to 29,000,000 m^3.

Table 4.2: KONKOLA MINE WATER BUDGET 1979-1990

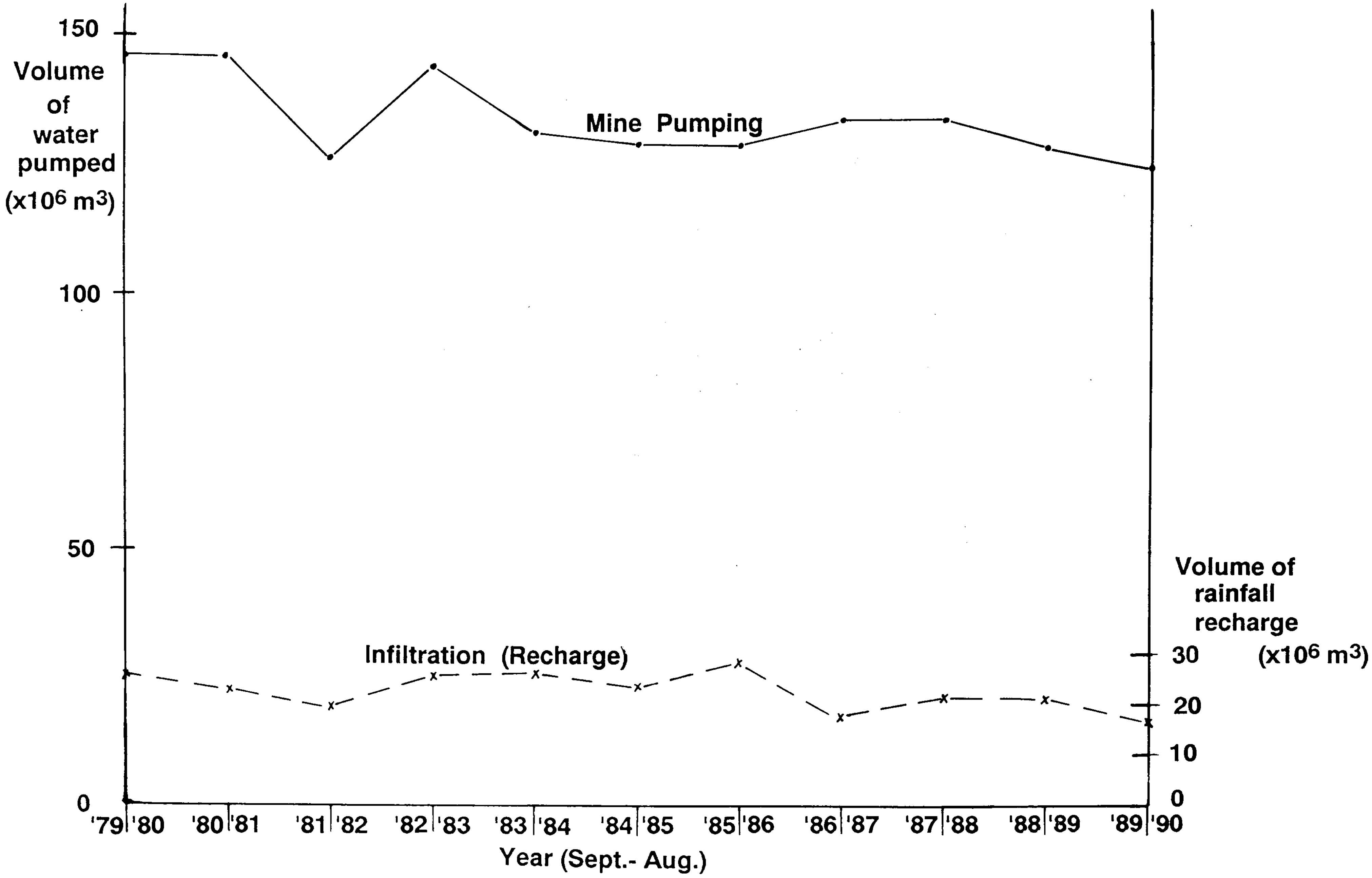
Year (Sept-Aug)	Total Volume Pumped (m^3)	Total Rainfall (m)	Infiltration (9.5% of Rainfall) (m)	Total volume of infiltration into mine topographic catchment area (187 km^2) (m^3)	Infiltration ÷ Mine Pumping (%)
1979/80	146,886,000	1.406	0.134	25,058,000	17.1
1980/81	146,115,000	1.276	0.121	22,627,000	15.5
1981/82	126,413,000	1.093	0.104	19,448,000	15.4
1982/83	144,118,000	1.412	0.134	25,058,000	17.4
1983/84	131,436,000	1.443	0.137	25,619,000	19.5
1984/85	129,585,000	1.294	0.123	23,001,000	17.8
1985/86	129,565,000	1.600	0.152	28,424,000	21.9
1986/87	134,578,000	1.013	0.096	17,952,000	13.4
1987/88	134,588,000	1.217	0.116	21,692,000	16.1
1988/89	129,718,000	1.197	0.114	21,318,000	16.4
1989/90	125,408,000	0.937	0.089	16,643,000	13.3

As can be clearly seen in Figure 4.1, there is more water pumped from the mine than can be accounted for by the local catchment rainfall. About 75% of the water pumped from the mine is from sources other than topographic catchment rainfall. Other workers, namely: Campbell (1973), Warren (1972) and Watermeyer et al (1972), reached similar conclusions that there was more water being pumped out from the mine than could be accounted for by the mine topographic catchment rainfall. Even if this is not completely accurate, the fact remains that water is clearly originating from sources other than topographic catchment rainfall.

The water is certainly coming from one other if not two sources, viz, recharge to the ground from surface water, especially the Kafue River and its tributaries which flow over the Hangingwall Aquifer and along the faults in the vicinity of the mine, and the Lubengele Mine Tailings Dam which is located on the Lubengele fault which is close to and dips towards Number 3 Shaft, and from the regional aquifers at depth.

The Luansobe Fault, has created a heavily brecciated zone in its immediate vicinity on the southern extremity of Number 1 Shaft and so has the Lubengele on the north limb of Number 3 Shaft. All these fractured zones provide major paths for groundwater inflow into the mine.

Figure 4.1 : KONKOLA MINE : VOLUME OF WATER PUMPED AND VOLUME OF RAINFALL RECHARGE



4.3 WATER LEVEL MAPS

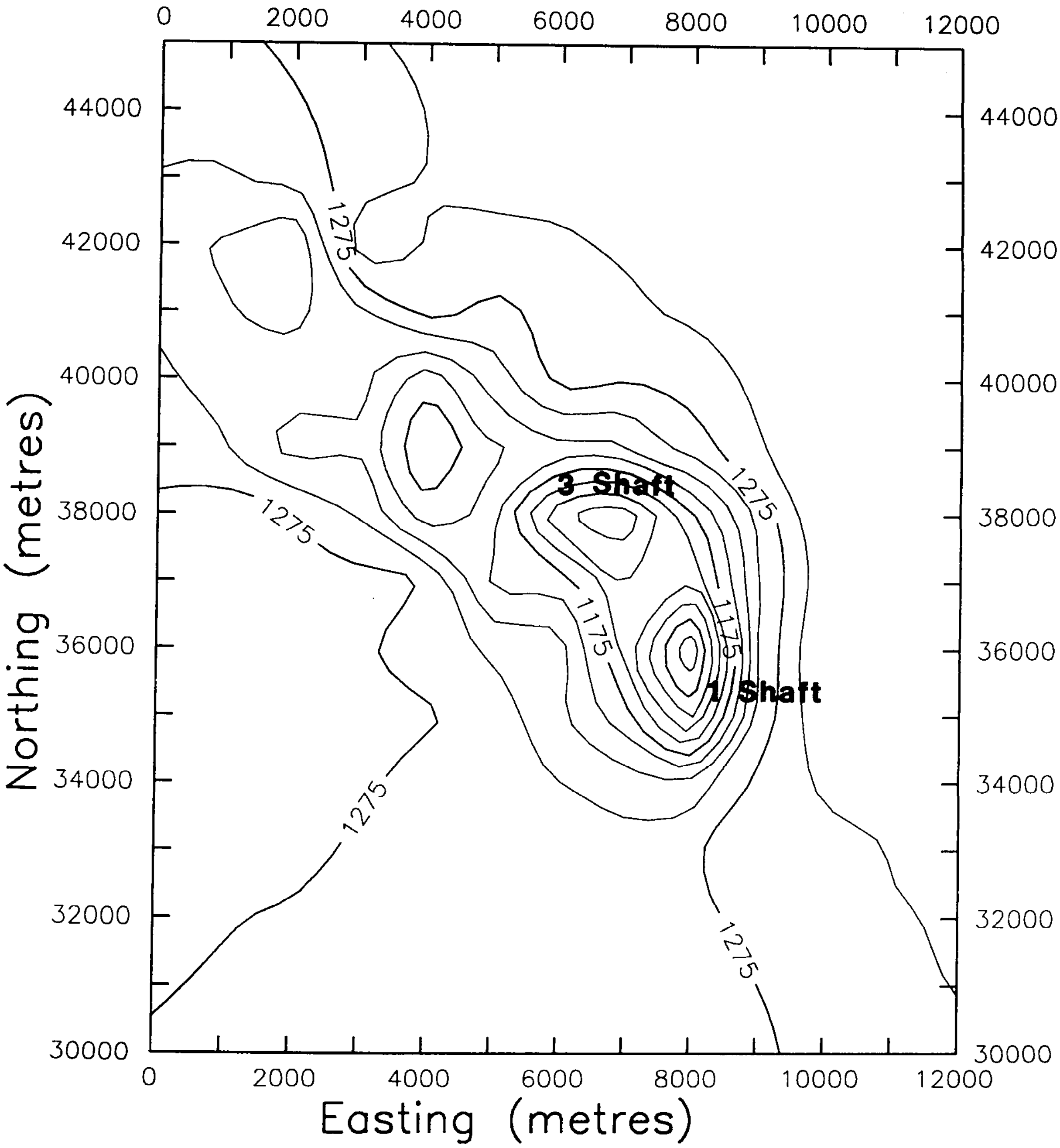
Mine records contain abundant information on water levels measured in open uncased boreholes over the area of mining, as well as in underground dewatering boreholes. These data were combined to produce maps and sections of water levels and water level change for given periods so that volumes of ground dewatered could be compared with volumes of water discharged. The data would then be used to obtain values of storage. Water level data from the uncased open boreholes had to be thoroughly scrutinized in order to weed out blocked boreholes and avoid obtaining erroneous results.

A cone of dewatering depression, elongated in the north west - south east direction, has developed over the mine as clearly demonstrated in the groundwater level, contour maps of 1970 to 1987 shown in Figures 4.2 to 4.6. The flow directions as indicated bycontour gradients, are mainly from the south of Number 1 Shaft and north of Number 3 Shaft. Number 1 Shaft forms the deepest part of the cone as shown in Figure 4.7, highest drawdowns have been achieved in this area of the mine. This is because Number 1 Shaft is the deeper of the two and dewatering drilling is at greater depths than at Number 3 Shaft.

As stated earlier, one of the main purposes of this exercise was to obtain values or a value of storage [volume pumped ÷ (change in water level x area affected by change in water level)] but, change in water level is dominated by near-steady flow, which prevents a clear value of storage from being obtained.

The water level surface does not form a smooth drawdown surface. Instead it has anomalies in its elevation with highs and lows existing rather than a smooth surface. These maps were superimposed upon maps of basic geology of the mine, at similar scale and revealed that the basic, geological and hydrogeological controls existed for the movement of water in the area.

Figure 4.2 KONKOLA GROUNDWATER LEVEL CONTOUR MAP JUNE 1970

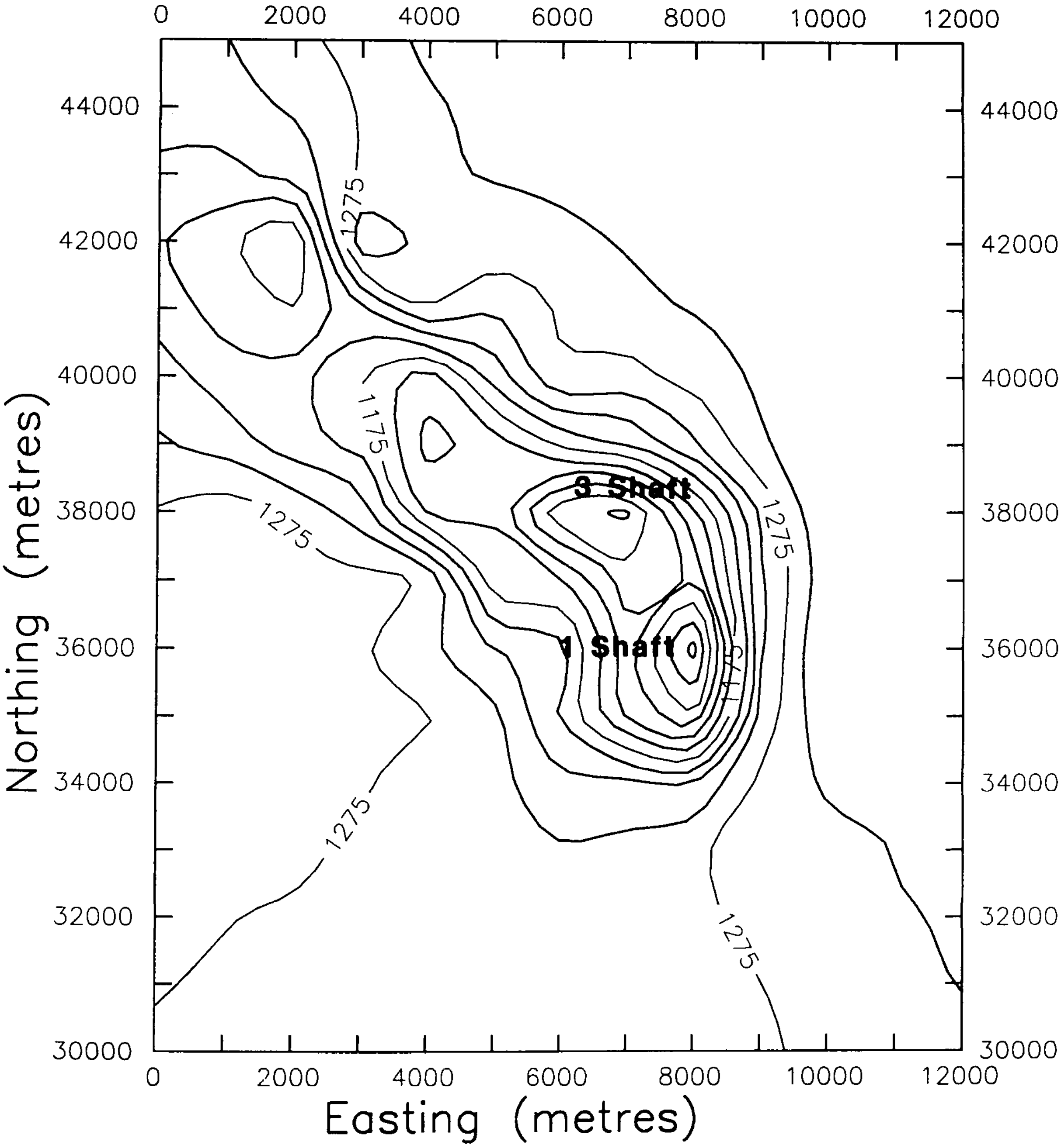


Scale

0 1000 metres

Figure 4.3 : KONKOLA GROUNDWATER LEVEL CONTOUR MAP

JUNE 1975

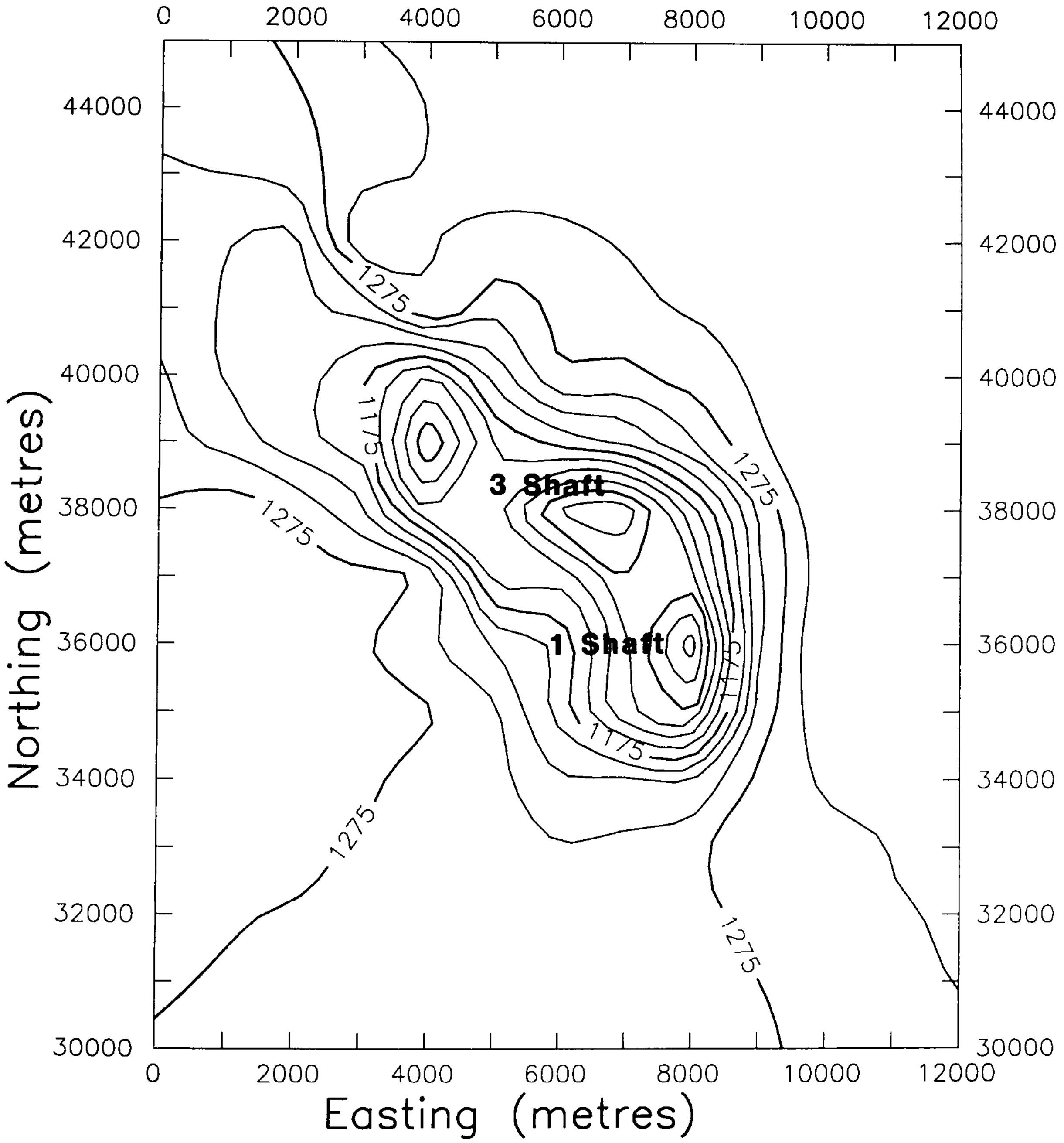


Scale

0 1000 metres

Figure 4.4 : KONKOLA GROUNDWATER LEVEL CONTOUR MAP

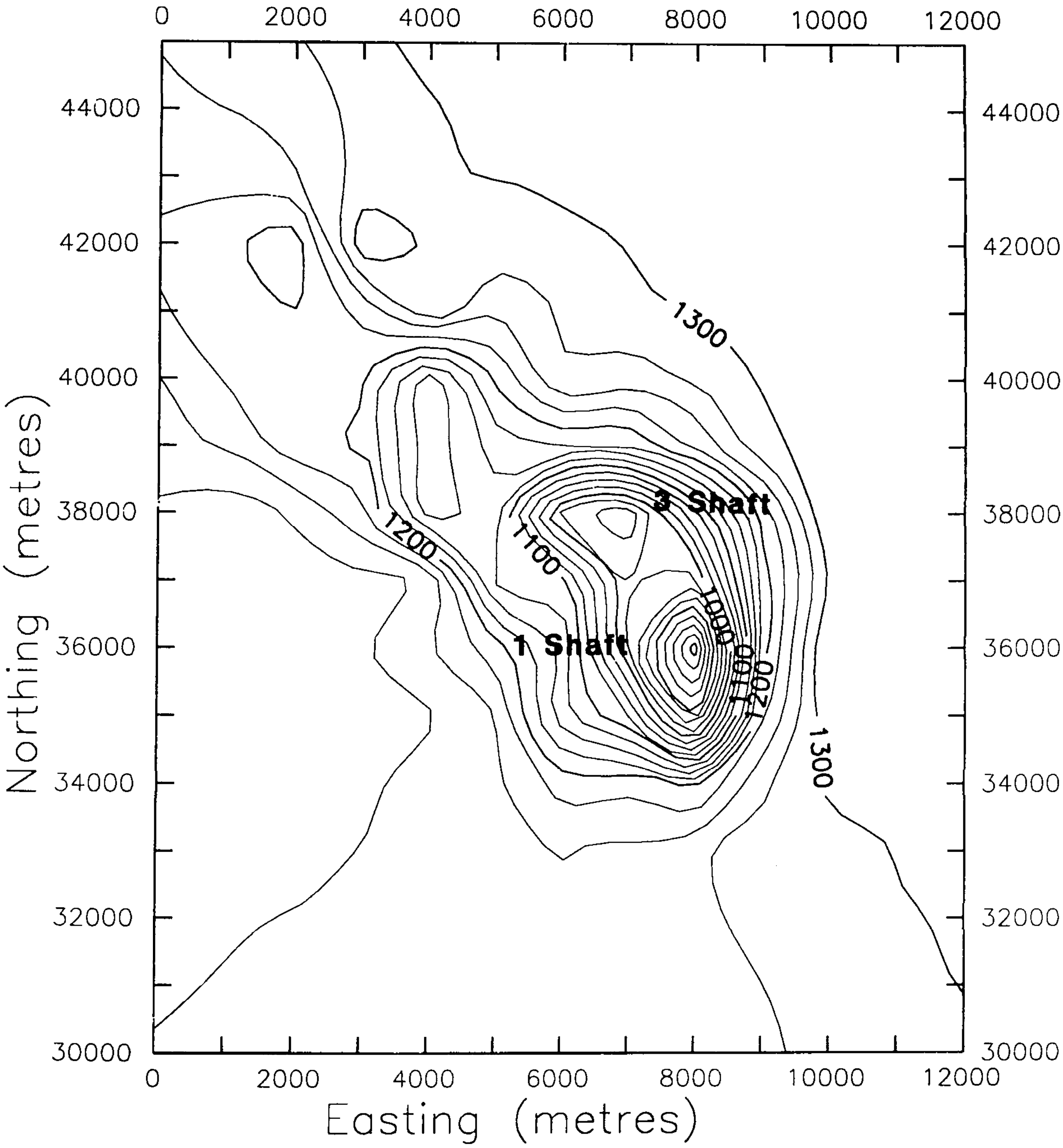
JUNE 1980



Scale

0 1000 metres

Figure 4.5 : KONKOLA GROUNDWATER LEVEL CONTOUR JUNE 1985



Scale

0 1000 metres

Figure 4.6 : KONKOLA GROUNDWATER LEVEL CONTOUR MAP

JUNE 1987

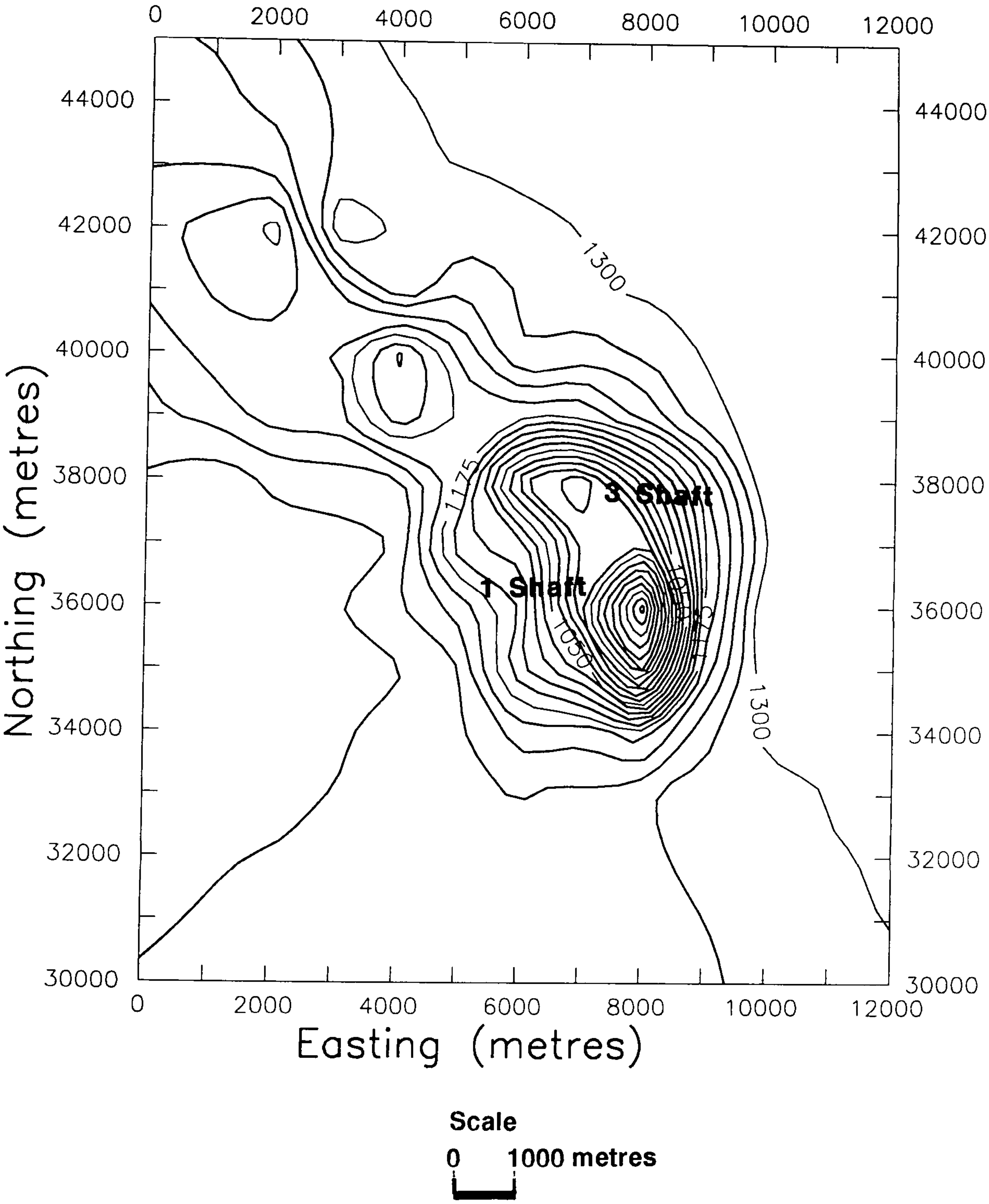
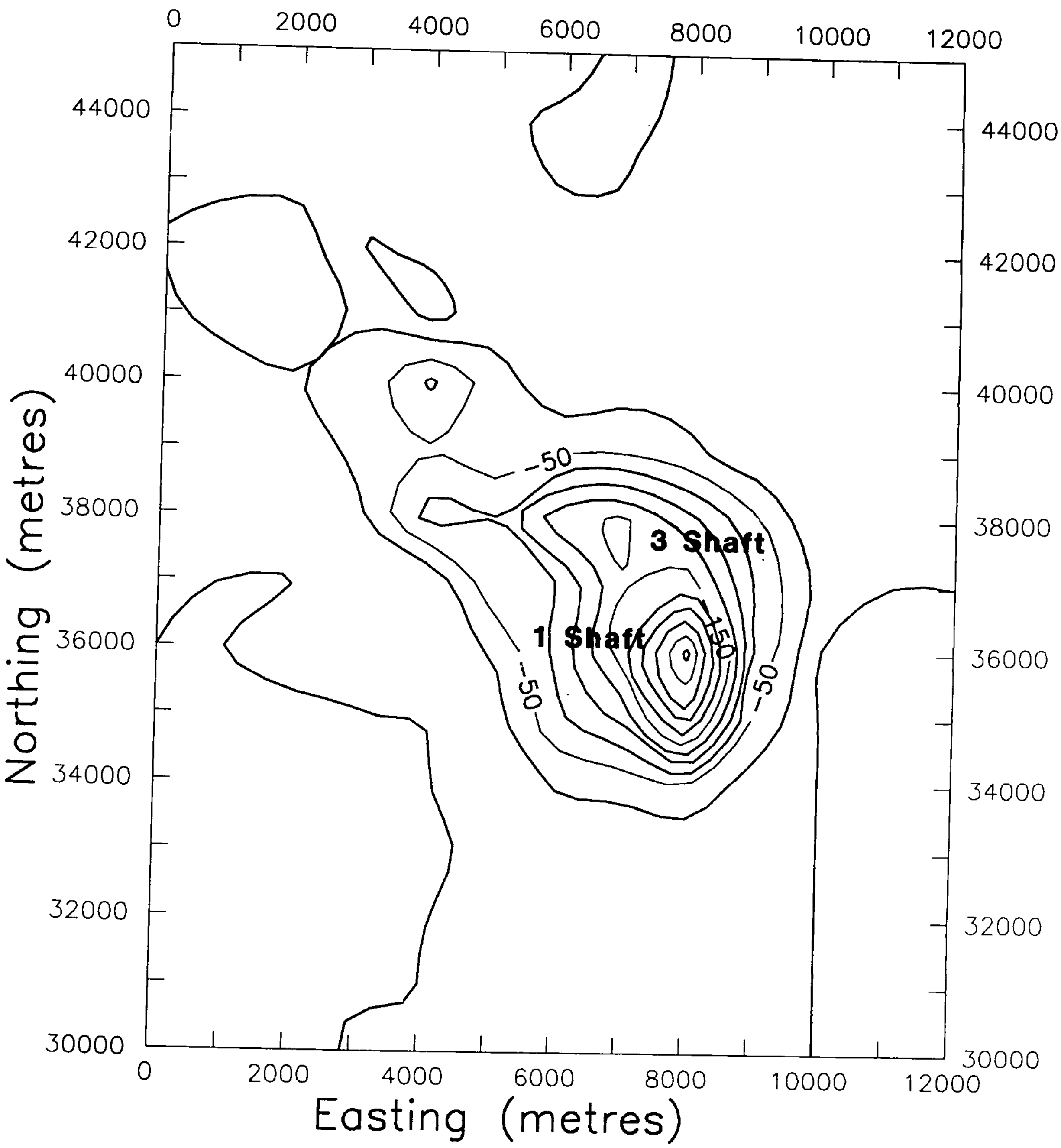


Figure 4.7: KONKOLA GROUNDWATER LEVEL CHANGE

1970 - 1987



Scale

0 1000 metres

4.3.1 Correlation Between Mine Pumping, Water Levels and Rainfall

Daily rainfall and mine pumping data for September 1989 to May 1990, seasonal rainfall and mine pumping records for 1986 to 1990 and groundwater level records for selected boreholes, were analysed in an attempt to determine whether there is a seasonal variation of mine water inflows and groundwater levels.

The results, shown in Figures 4.8 to 4.11, demonstrate quite clearly that there is a seasonal variation in the volumes of water pumped from the mine. However, the variation is not significant. This may be mainly due to the fact that because excessively large volumes of water is pumped from the mine at all times, the time lag effect smooths any potential significant fluctuations in volume pumped, to surface. In contrast to this, the behaviour of groundwater levels as demonstrated in Figures 4.10 and 4.11, show a significant sensitivity to rainfall recharge. There is a rise in water levels following rainfall recharge.

Figure 4.8 : KONKOLA MINE DAILY RAINFALL AND DAILY MINE PUMPING SEPT.1989 - MAY 1990

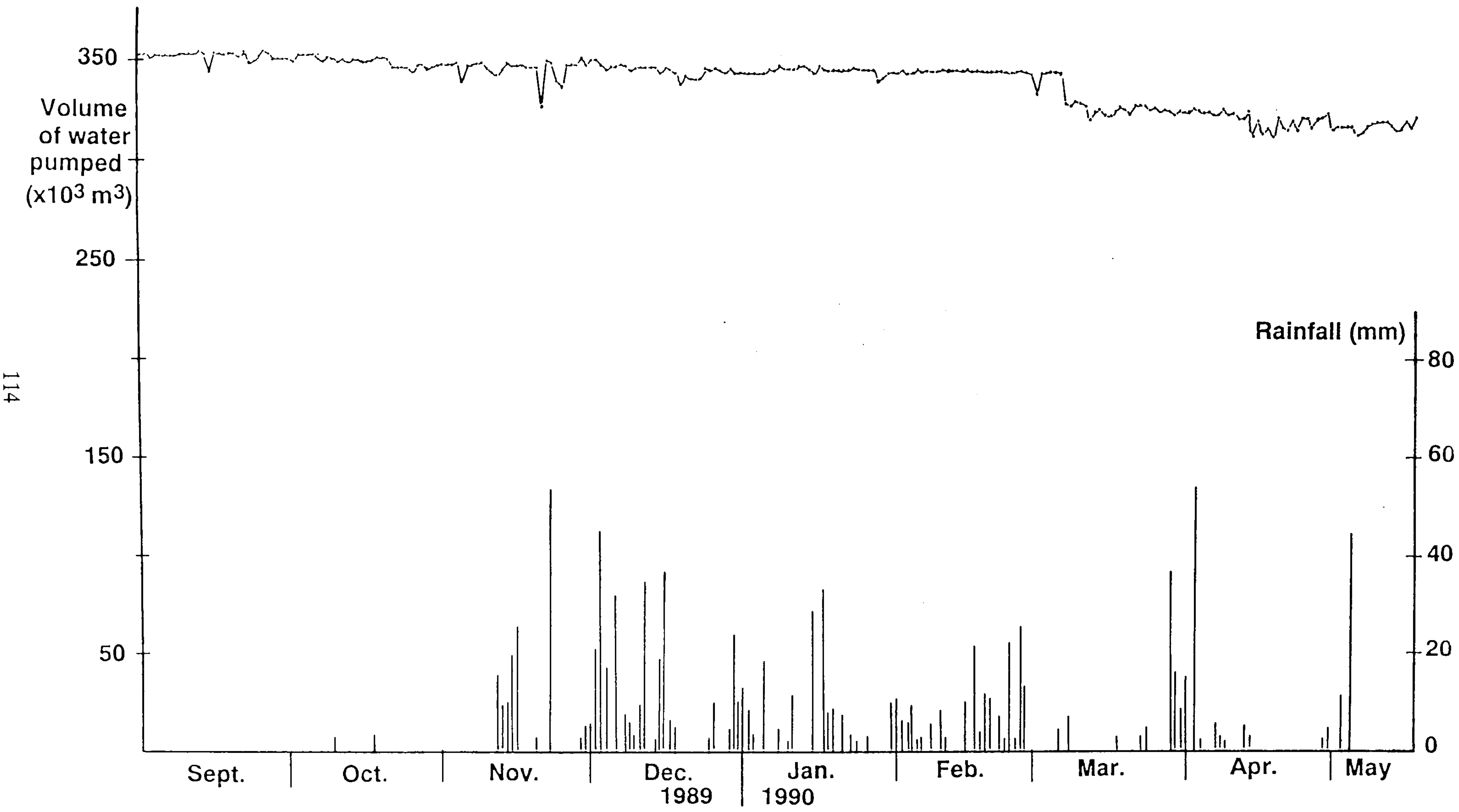


Figure 4.9 : KONKOLA MINE MONTHLY RAINFALL AND MINE PUMPING 1986 - 1990

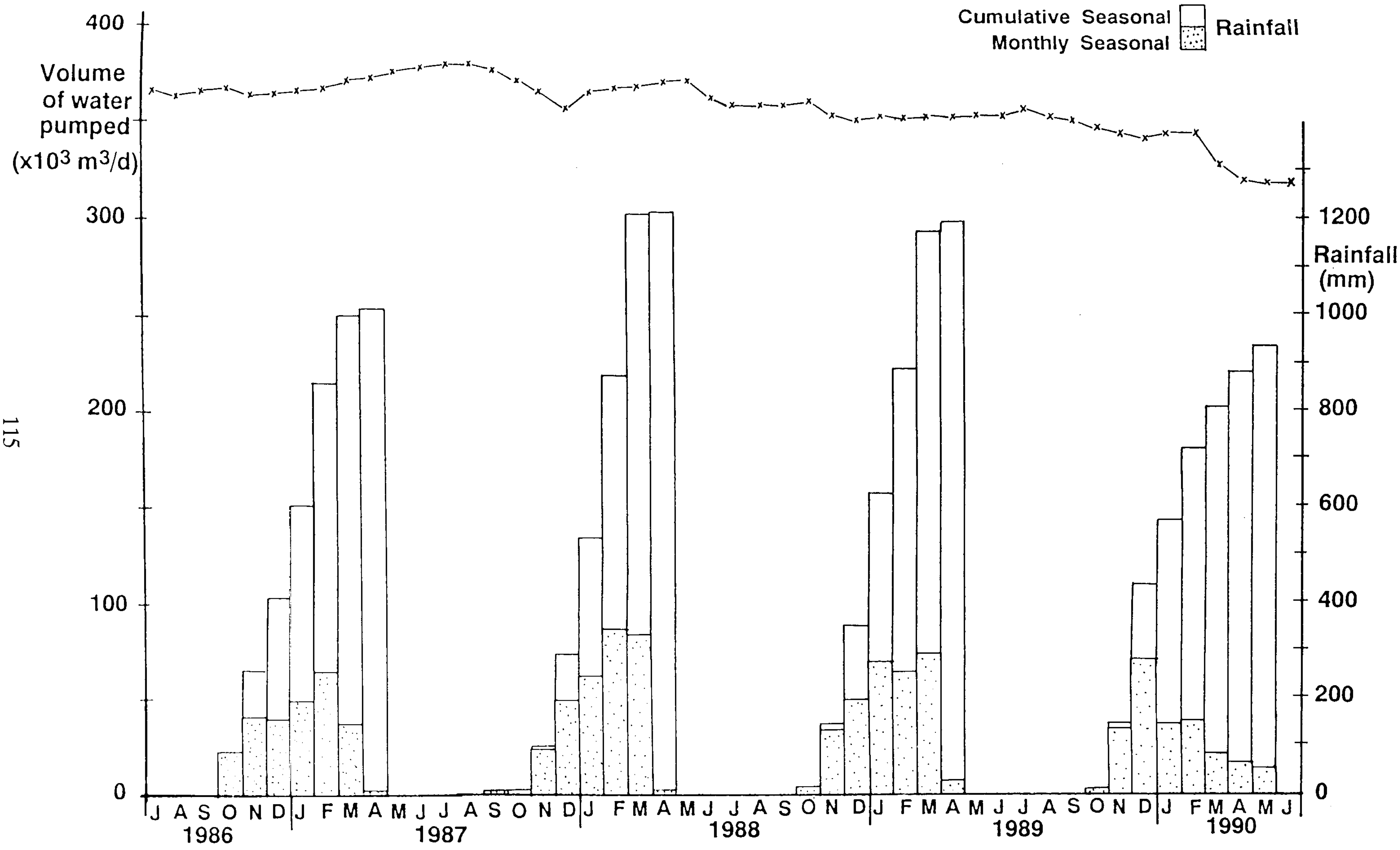


Figure 4.10 : GROUNDWATER LEVEL VS RAINFALL

KLB104

LUBENGELE FAULT ZONE

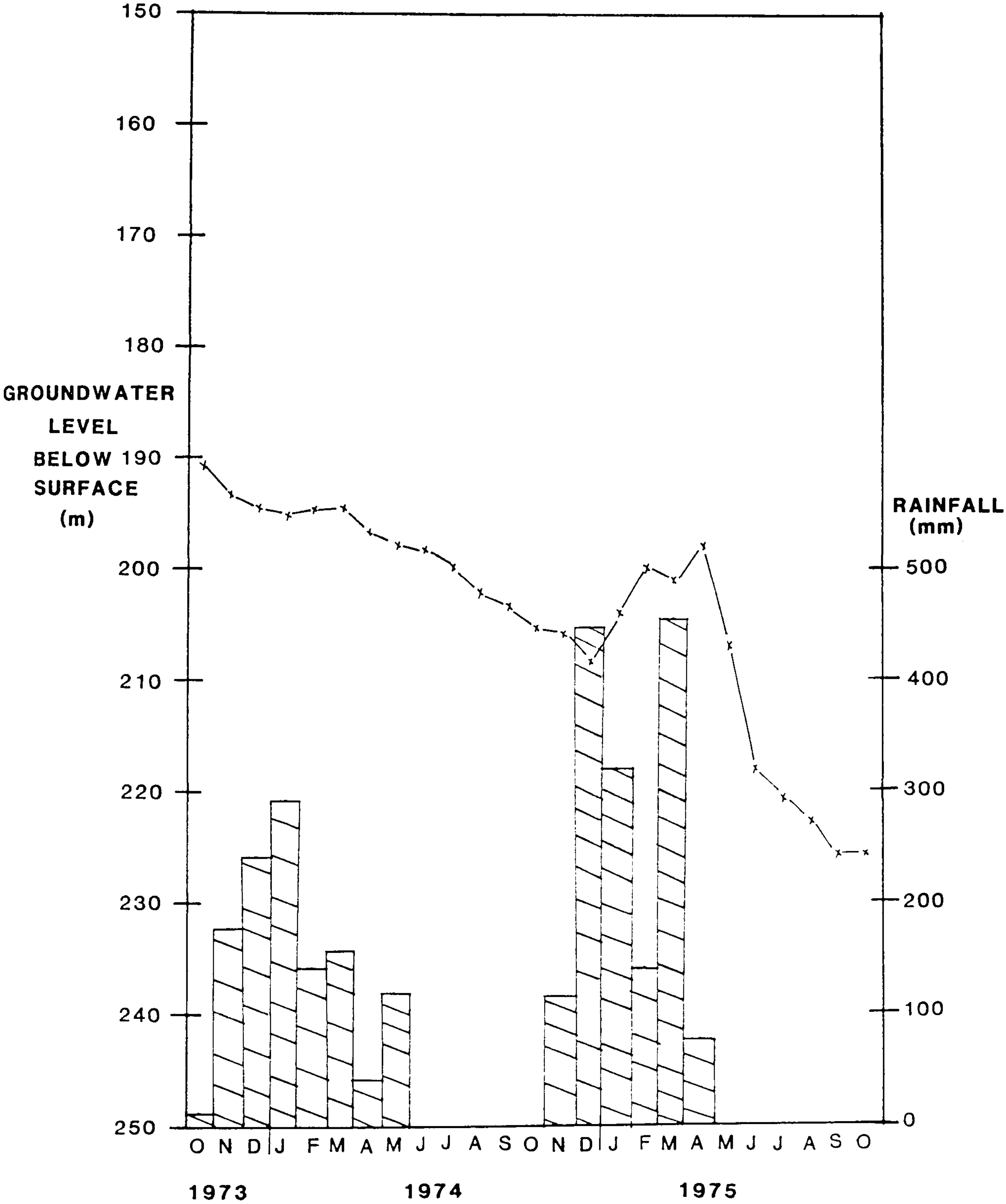
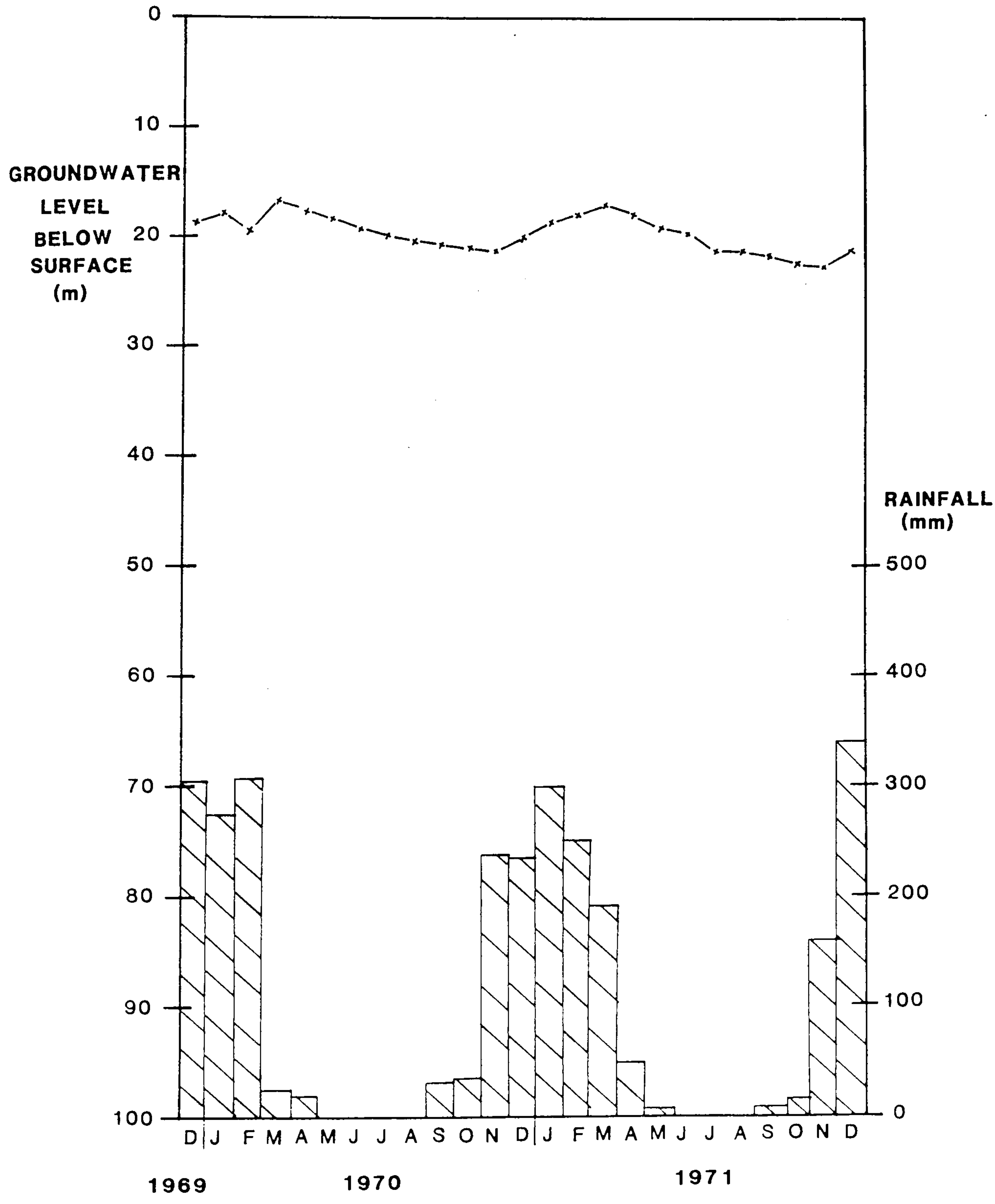


Figure 4.11 : GROUNDWATER LEVEL VS RAINFALL
BOREHOLE KLB88
UNFAULTED ZONE



4.3.2 Hangingwall Aquifer Water Levels

The water levels have fallen as a consequence of mine dewatering. The water levels have fallen to about 2650 ft Level (808m) at Number 1 Shaft and 1480 Level (451m) at the north limb of Number 3 Shaft.

Figure 4.12 shows a maximum drawdown of about 650 ft (198m) at Number 1 Shaft has been achieved in the ten-year period June 1980 to June 1990. This gives an average annual drawdown of 19.8m. However, the average for the whole mining area is around the 10 m/yr mark. The total volume of water pumped from the mine during this ten-year period was 1,348,679,970 m^3 and estimated total volume of rock dewatered was 19,830,870,000 m^3. This gives a theoretical storage value of about 0.07.

As can be clearly seen this figure does not make much hydrogeological sense. This is because, due to the presence of large recharge, disproportionately high volumes of water are pumped in order to achieve any drawdown. As discussed in the preceding sections, the disproportionality between volume of water pumped from the mine and drawdowns achieved is evidence for the existence of a more complex hydrogeological regime (Figure 4.13b) and not a simple one (Figure 4.13a). The recharge keeps drawdowns low.

Figure 4.12 : KONKOLA MINE - NUMBER 1 SHAFT LONGITUDINAL PROFILE SHOWING HANGINGWALL AQUIFER WATER LEVELS JUNE 1980 - JUNE 1990

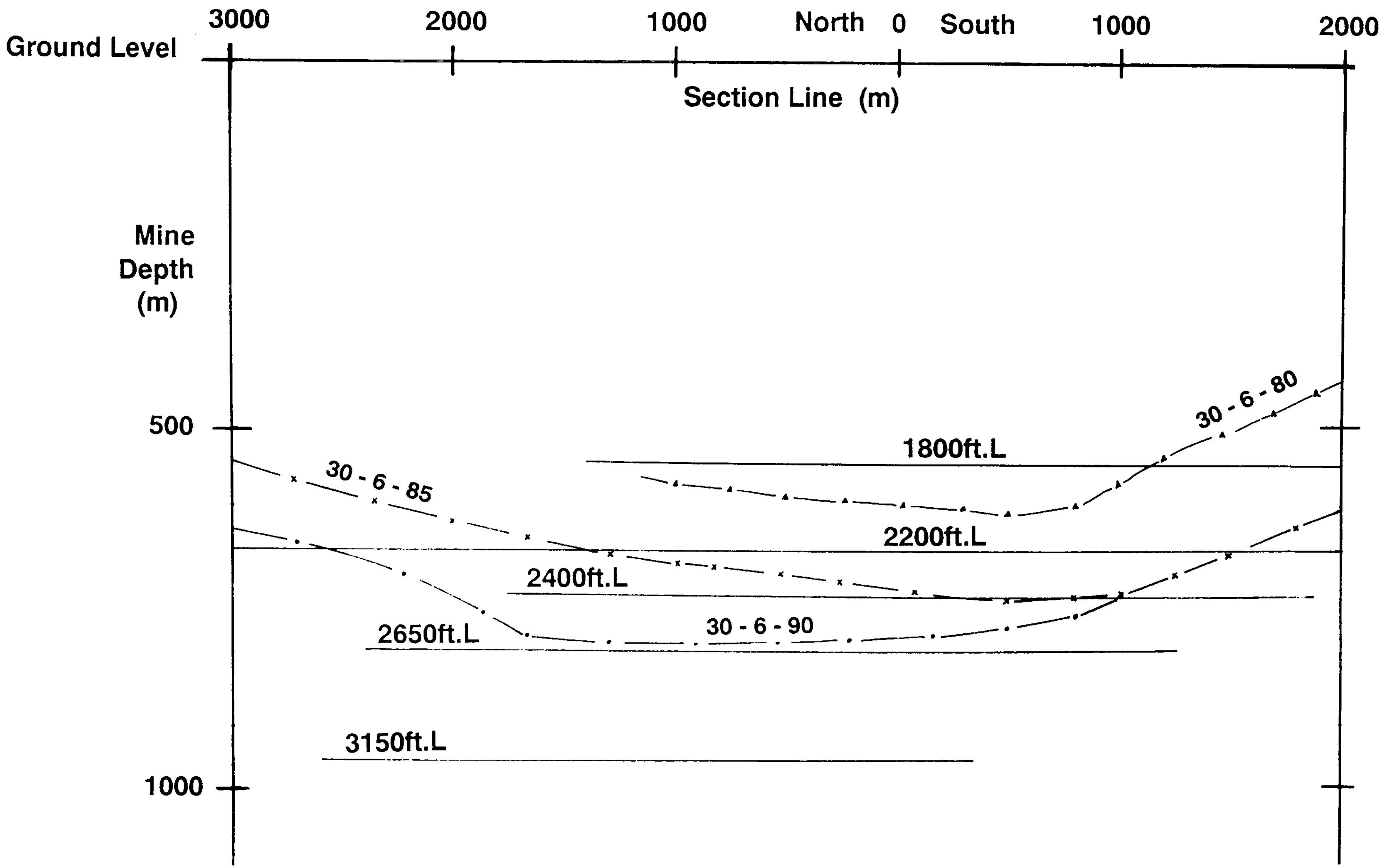
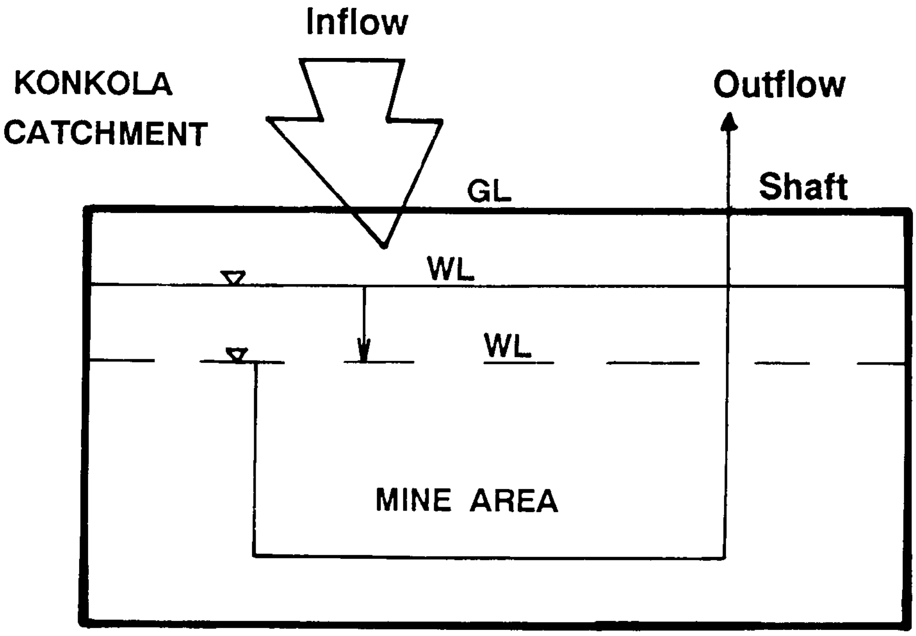


Figure 4.13:

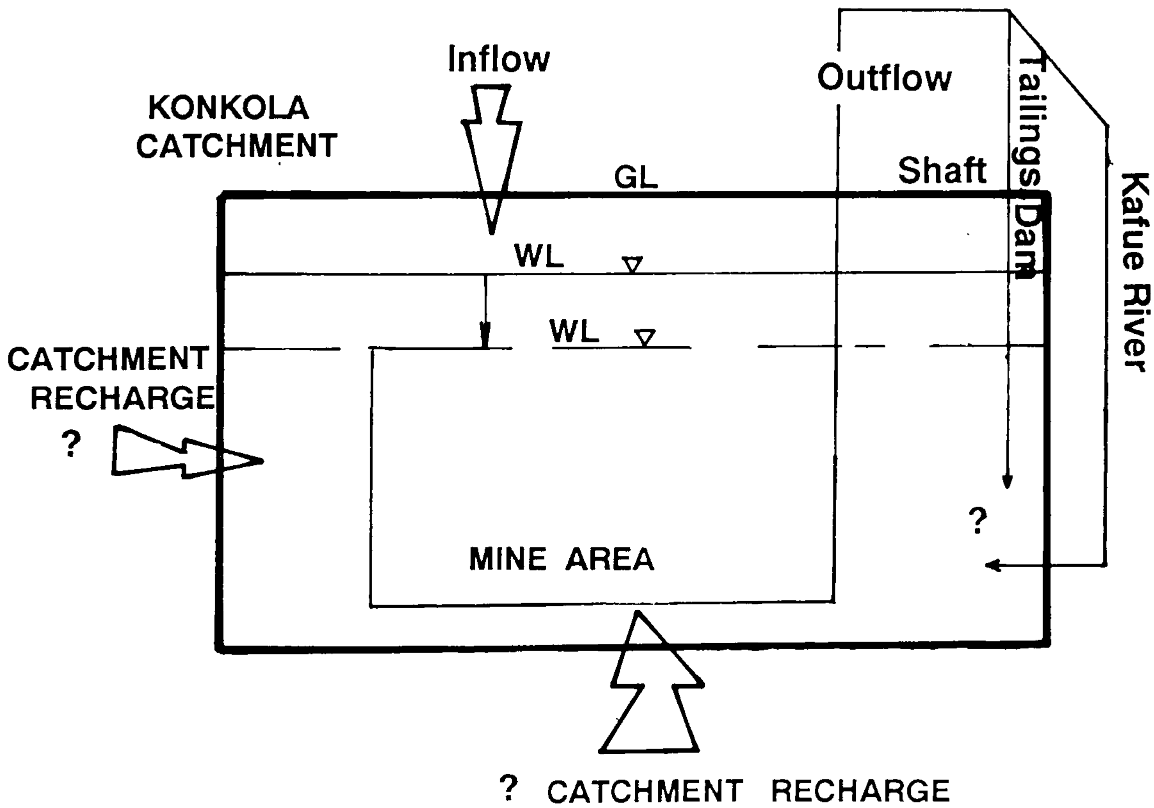
KONKOLA MINE - POSSIBLE HYDROGEOLOGICAL SETTINGS

(a) Single Recharge Source



OR

(b) Multiple Recharge Sources



4.4 CONCLUSIONS

It has been clearly demonstrated that more water is pumped from the mine than can be accounted for by rainfall from the mine topographic catchment. The drawdowns achieved are far much less than expected from a knowledge of the volumes of mine water pumped and reasonable values for storage.

As stated earlier, there is plenty of evidence to suggest that water could come from a number of sources, viz. the regional aquifer at depth, and surface waters of the Kafue River and Mine Tailings Dam.

The dam itself is founded directly on the superficial deposits which cover bedrock as a thin veneer. These include a former river deposit which occupies the bottom of a shallow valley across which the dam is built, and in which the Tailings reservoir is impounded. The Lubengele and Mingomba streams still flow into the Mine Tailings Dam. Although attempts have been made to seal the base of the reservoir using tailings there is no guarantee that a hydrogeological seal exists between the reservoir and the drift upon which it rests nor is there any guarantee that leakage beneath the Tailings Dam is prevented.

The inflow boundary is unlikely to be just ground level. This means sub-surface recharge to the mining area via aquifers connected to distant catchments must be considered. This later water we can think of as regional groundwater as distinct from recharge from the catchment of the mine which can be thought of as circulating meteoric water.

To study this further a programme of field and laboratory work had to be initiated, that would enable these two sources, of water to be distinguished and their relative contributions measured.

As this involved the use of geochemical data, it became important to support these studies with an understanding of the speed of entry of water to the mine via pores and fissures. Such a study had to be related to the identification of the preferred routes used by groundwater in its journey towards the mine. The programme of laboratory and fieldwork which therefore emerged had to include independent methods of verifying the conclusions drawn from mine mapping of structural geology, historical records and the study of surface hydrology as shown in Figure 1.7. As shown in Figure 1.7 this independent verification involved measurements of hydraulic conductivity, transmissivity, storage and the introduction of groundwater chemistry to the inquiry.

CHAPTER 5

FIELD AND LABORATORY INVESTIGATIONS OF HYDROCHEMISTRY

5.0 INTRODUCTION

Fundamental questions had now been identified and needed to be resolved if the groundwater flow problem at Konkola mine was to be properly understood and solved.

The origin of the source of recharge to the mine needed to be established so that a conceptual model of Konkola mine, groundwater flow system, based on sound scientific understanding, could be developed. This would then facilitate the formulation of a cost-effective long-term groundwater management solution. The solution so evolved would be unique to Konkola mine as it would have been tailored to tackle the prevailing groundwater conditions at the mine. In addition, the conceptual model would be needed to plan further stages, of investigation, to forecast system behaviour and to provide the necessary base for mathematical modelling of mine water discharge and water level drawdown, following the implementation of a cost-effective groundwater management programme.

Thus, source(s) of groundwater recharge to the mine needed to be identified and, if there were more than one source, an estimate of their relative contributions had to be made. Furthermore, the groundwater flow pattern, major flow routes and flow velocities had to be established. The Kafue river flows along the Hangingwall Aquifer, and crosses the Luansobe fault in the vicinity of the mine. The Mine Tailings Dam is located on the Lubengele fault which is close to Number 3 Shaft. All this strongly suggested a possible hydraulic interconnection between these surface waters and the aquifers in the mine. Possible leakage of the surface waters into the mine had to be confirmed.

5.1 TRACING THE SOURCE(S) OF WATER INFLOW TO THE MINE

The source of water inflow to the mine was traced by studying the rock chemistry and the chemical composition of the mine water. Williams *et al* (1986) have demonstrated that water chemistry is a very powerful tool in determining sources of water inflow in mines. This is further evidenced in work done by other researchers, namely: Mazor (1991), Fernandez-Rubio *et al* (1988), Todd (1980), Fetter Jr. (1980), and Freeze and Cherry (1979), to name just a few.

5.1.1 Konkola Mine Rock Chemistry

The mineral composition of the rock is almost always reflected in the chemical composition of the water that has passed through it.

As water flows through the rock it dissolves part of the rock and from these dissolved constituents, one is able to determine, to some extent, the groundwater geologic history, the rock masses through which it has passed and the mode of origin within the hydrologic cycle (Mazor 1991, Singh 1989, Mazor *et al* 1985, and Davis and De Wiest 1966).

At Konkola mine the aquifers fall into two mineralogical groups. The Hangingwall Aquifer are predominantly calcareous. They are composed of limestone, dolomite and calcareous siltstones. The Footwall Aquifers are siliceous. They are composed of feldspathic sandstones, conglomerates and quartzites. Therefore, water intersected in the Hangingwall Aquifers would be expected to have high values of dissolved ions of Calcium (Ca^{2+}), Magnesium (Mg^{2+}), Bicarbonate (HCO_3^-), Carbonate (CO_3^{2-}) and any other secondary elements uniquely associated with calcareous rocks. On the other hand, water intersected in the Footwall Aquifers would be expected to have lower concentrations of Calcium, Magnesium, Bicarbonate and Carbonate ions but higher concentrations of Silica and elements uniquely associated with siliceous rocks.

5.1.2. Tracing the Groundwater Composition

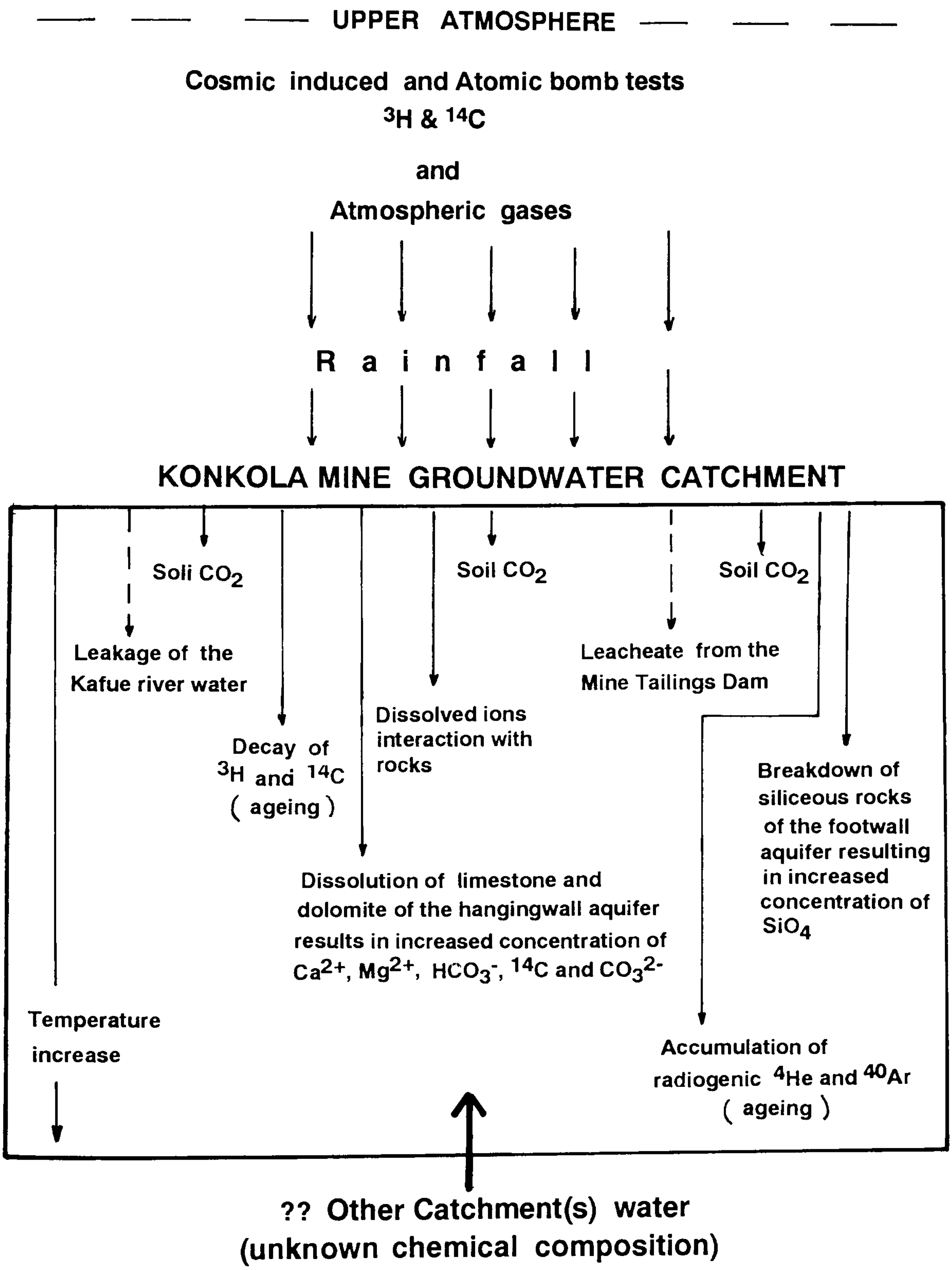
A clear appreciation of the significance of the Konkola mine water chemical composition has to be viewed from a background of the mine hydrogeologic setting.

Figure 5.1 is an attempt to represent the Konkola hydrologic setting in a schematic form. Before rain reaches the ground, it is tagged by: both man-made and cosmic ray induced Tritium (^{3}H) and Carbon-14 (^{14}C), Deuterium (^{18}O), dissolved atmospheric gases which include Helium (He), Neon (Ne), Argon (Ar), Krypton (Kr) and Xenon (Xe). Upon entering the ground more information is coded into the water in form of dissolved ions as a consequence of water-rock interaction. Significantly more carbon dioxide (CO_2) is added to water percolating through the soil layer, forming carbonic acid, as soil air contains around one hundred times more carbon dioxide than free air (Mazor 1991). Soil carbon dioxide is produced by biological action such as root respiration and decay of plant material. Once water reaches the saturated zone of an aquifer it is isolated from the atmosphere and its Tritium (^{3}H) and Carbon-14 (^{14}C) decays.

As the water flows through the Hangingwall Aquifer the dissolution of limestone and dolomite results in increased concentration of Calcium (Ca^{2+}), Magnesium (Mg^{2+}) and other associated ions. In Footwall Aquifers, more silica is added to the water. As the water flows deeper the temperature increases, there is an accumulation of radiogenic Helium and Argon -40 and decay of Tritium (^{3}H) and Carbon-14 (^{14}C).

If the Mine Tailings Dam is leaking into the mine, the leacheate is added to the groundwater system which is also mixed with surface waters of the Kafue river, should this also be leaking into the mine. The groundwater chemistry could be further complicated if there is recharge from a different catchment with dissimilar chemistry.

Figure 5.1 : Schematic Diagram of a Possible Groundwater Hydrologic Setting at Konkola Mine - Basis for Geochemical Investigations



Therefore, based on this conceptualization, in July 1988 a programme of geochemical water sampling was undertaken at Konkola mine. The sampling was undertaken at Konkola mine. The sampling was carried out over a limited period of time of ten days, so as to have hydrologically related samples (Mazor 1976) and thus obtain a geochemical snap-shot of the mine surface and groundwater systems. Samples were collected from the Kafue river and its tributaries, the Mine Tailings Dam and from the seven underground main working mine levels; 450m (1480 ft) to 960m (3150 ft) levels. A total of eighty-eight samples were collected, covering sixty-six sites. The samples were analysed both on site, in the mine laboratories, and in London at Imperial College Geology Department Geochemistry Laboratory.

Standard Industrial laboratory analytical techniques were used to conduct the analyses at the mine laboratories (Standard Laboratory Analytical Techniques: Zambia Consolidated Copper Mines Limited). At Imperial College, the inductively coupled Plasma Atomic-Emission Spectrometry technique was used to analyse the samples for dissolved elements (Thompson and Ramsey 1985, Thompson and Walsh 1988, Thompson *et al* 1982).

Forty parameters were measured. Temperature, pH and total dissolved solids (TDS) were measured on site as the water sample was being collected. Cations, anions and bacteria were analysed for Sodium (Na), Potassium (K), Silicon (Si), Copper (Cu), Iron (Fe), Lead (Pb), Zinc (Zn), Nickel (Ni), Cobalt (Co), Manganese (Mn), Sulphur (S), Lithium (Li), Silver (Ag), Rubidium (Rb), Beryllium (Be), Strontium (Sr), Barium (Ba), Lanthanum (La), titanium (Ti), Vanadium (V), Chromium (Cr), Molybdenum (Mo), Cadmium (Cd), Boron (B), Aluminium (Al), Phosphorous (P), and Arsenic (As). Anions were as follows: Sulphate (SO_4^{2-}), Chloride (Cl^{-1}), Silica (SiO_2^{-}), Bicarbonate (HCO_3^{-1}), and Nitrate (NO_3^{-1}). In addition, residual xanthates and bacteria (Eschericia Coli) concentrations were analysed at the mine in an attempt to confirm whether or not the Mine Tailings Leacheate and the Kafue river waters were leaking into the mine groundwater system.

The field investigation was repeated a year later in July 1989, for the same period, to verify the scientific conclusions drawn from the first study and resolve any questions that had arisen from the first investigation. This time sixty-three water samples were collected. Furthermore, water samples for Tritium age-dating were collected and analysed independently at the Harwell Isotope Laboratory of the United Kingdom Atomic Energy Authority.

The geochemical data obtained from the field studies are tabulated in Tables 5.1 to 5.17 in Appendix 2. Tables 5.1 to 5.9 list results of July 1988, and Tables 5.10 to 5.17 those of July 1989. Table 5.18 is a comparison of July 1988 and July 1989 major ions concentrations of the aquifers. The results show clearly that there are distinctive differences between the water from the Footwall and the Hangingwall Aquifers.

Figures 5.2 to 5.7 clearly demonstrate that water in the Footwall Quartzite and Footwall Aquifers is warmer than the Hangingwall Aquifer water by an average of 2°C and is slightly acidic, low in total dissolved solids, Calcium, Magnesium, Potassium, Sulphur, bicarbonates and sulphates. The Hangingwall Aquifer water is colder, slightly alkaline, high in total dissolved solids, Calcium, Magnesium, Potassium, Sulphur, bicarbonates and sulphates.

Furthermore, the Footwall Aquifer water has concentrations of dissolved ions, temperature and pH values that are intermediate between the Hangingwall and Footwall Quartzite Aquifers. This seem to suggest that the Footwall Aquifer water is a mixture of the Footwall Quartzite and Hangingwall Aquifers water. *

These simple differences in both physical and chemical characteristics of the aquifers water strongly indicate different origins of the waters. However, these differences in ionic chemistry are not pronounced in fault zones, indicating mixing of the waters. This also implies that the fault zones are major conduits for water from both basic sources, viz, below and above the ore zone. Correlation analyses and chemical composition plots (Figures 5.8 to 5.11) confirm the existence of two

*Footnote : See Appendix 2

Table 5.18: Comparison of Konkola Mine Aquifers' Water Chemistry - July 1988 and 1989; Dissolved Ion Concentration in mg/l

Variable:		Temperature ^{o}C			pH			TDS			Bicarbonate			Mg			Ca		
		FWQ	FWA	HWA	FWQ	FWA	HWA	FWQ	FWA	HWA	FWQ	FWA	HWA	FWQ	FWA	HWA	FWQ	FWA	HWA
(a) July 1988	Average	28.6	28.2	25.4	6.2	6.7	7.2	97.1	151	276	102	152	218	7.1	16.2	1.2	23.6	42.7	54.6
	Minimum	27.1	26.0	24.4	5.9	5.9	6.9	40	60	170	71	16	62	4.4	4.4	0.3	14.6	4.4	28.7
	Maximum	29.9	29.8	27.4	6.5	7.4	7.8	210	330	430	159	366	317	9.7	38.5	2.6	44.5	98.4	83.8
	Range	2.8	3.8	3.0	0.6	1.5	0.9	170	270	260	88	350	255	5.3	34.1	2.3	29.8	94	54.1
(b) July 1989	Average	29.1	28.2	25.9	6.5	6.9	7.3	98	203	292	68	93	132	8.53	18.04	35.98	26.8	43.1	54.0
	Minimum	28.1	25.6	24.7	6.3	6.1	6.6	10	30	25	39	27	57	6.33	4.39	11.55	14.7	4.3	23.63
	Maximum	30.3	30.5	29.8	6.7	7.4	7.7	210	500	460	126	135	162	10.44	35.66	102.00	46.4	97.8	65.4
	Range	2.2	4.9	5.1	0.4	1.3	1.1	200	470	435	87	108	105	4.11	31.27	90.45	31.7	93.5	41.77

Variable:		Na			K			Sulphate			S			Sr			Chloride		
		FWQ	FWA	HWA	FWQ	FWA	HWA	FWQ	FWA	HWA	FWQ	FWA	HWA	FWQ	FWA	HWA	FWQ	FWA	HWA
(a) July 1988	Average	3.1	3.6	4.1	7.1	7.0	6.5	34.1	75.6	80.9	9.3	18.2	25.6	0.09	0.29	0.51	1.3	1.6	1.2
	Minimum	2.1	1.4	2.2	4.9	4.5	4.8	2	16	13	4.3	2.1	3.2	0.04	0.01	0.15	0.1	0.1	0.3
	Maximum	6.2	8.1	7.9	9.2	9.8	7.4	85	199	178	19.1	66.7	60.5	0.17	0.81	3.84	2.2	8.7	2.6
	Range	4.1	6.7	5.7	4.3	5.3	2.6	83	183	165	14.8	64.6	57.3	0.13	0.80	3.69	2.1	8.6	2.3
(b) July 1989	Average	4.8	4.96	5.55	8.3	7.37	10.88	39.8	76.3	108	22.73	34.5	40.1	0.097	0.207	0.30	3.23	3.36	2.54
	Minimum	2.11	1.47	3.24	6.5	4.89	4.72	22.0	6.0	42	6.94	2.0	4.6	0.045	0.011	0.14	1.4	0.5	0.9
	Maximum	10.23	10.93	11.53	9.89	10.67	47.00	67.0	379.0	357	50.1	88.2	150.5	0.178	0.893	0.46	6.1	9.2	4.6
	Range	8.12	9.46	8.29	3.39	5.78	42.28	45.0	373.0	315	43.16	86.2	145.9	0.133	0.881	0.31	4.7	8.7	3.7

FWQ = Footwall Quartzite Aquifer FWA = Footwall Aquifer HWA = Hangingwall Aquifer

Figure 5.2 : KONKOLA MINE AQUIFER WATER AVERAGE CHEMISTRY - JULY 1989

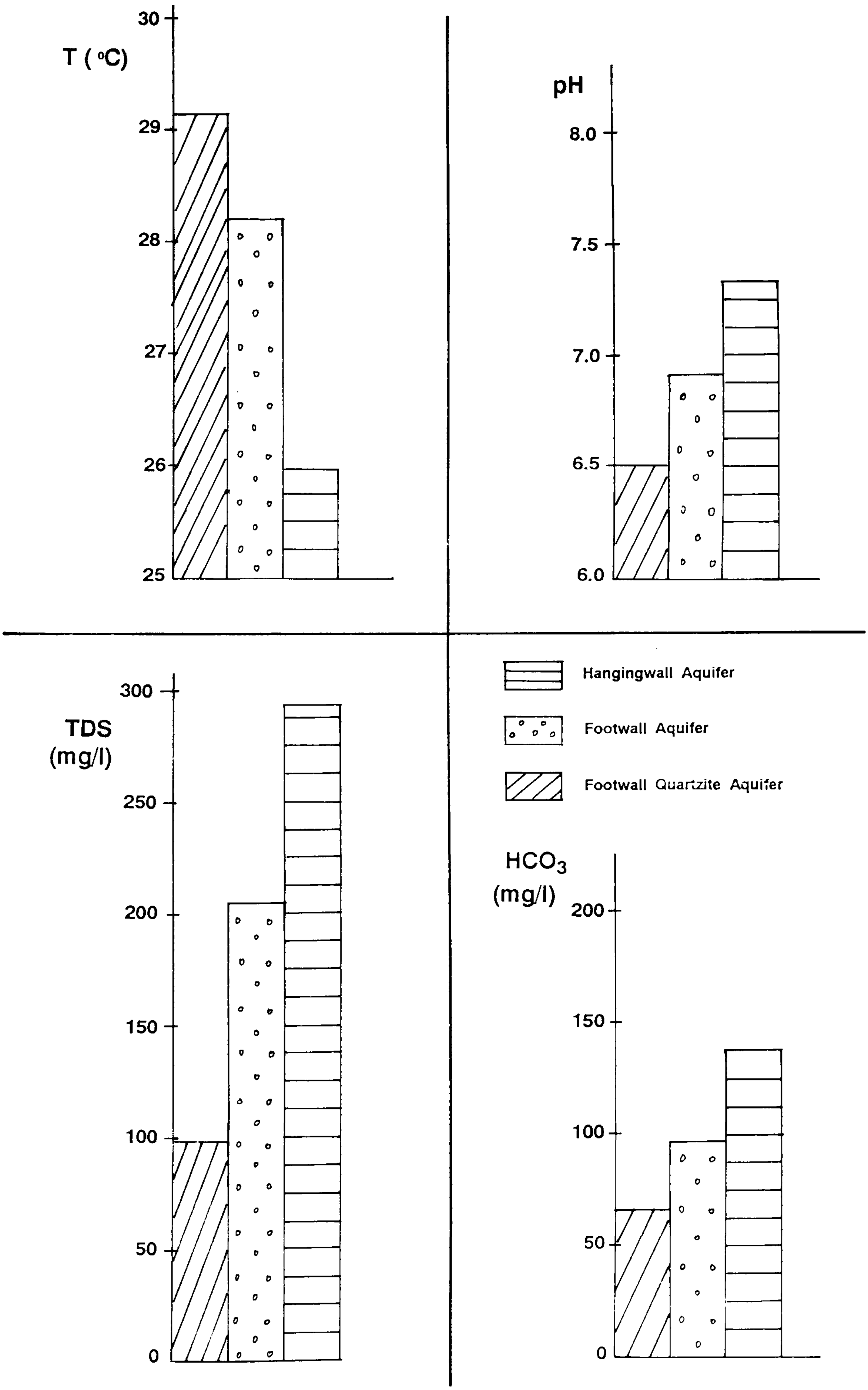


Figure 5.3 : KONKOLA MINE AQUIFER WATER AVERAGE CHEMISTRY - JULY 1989

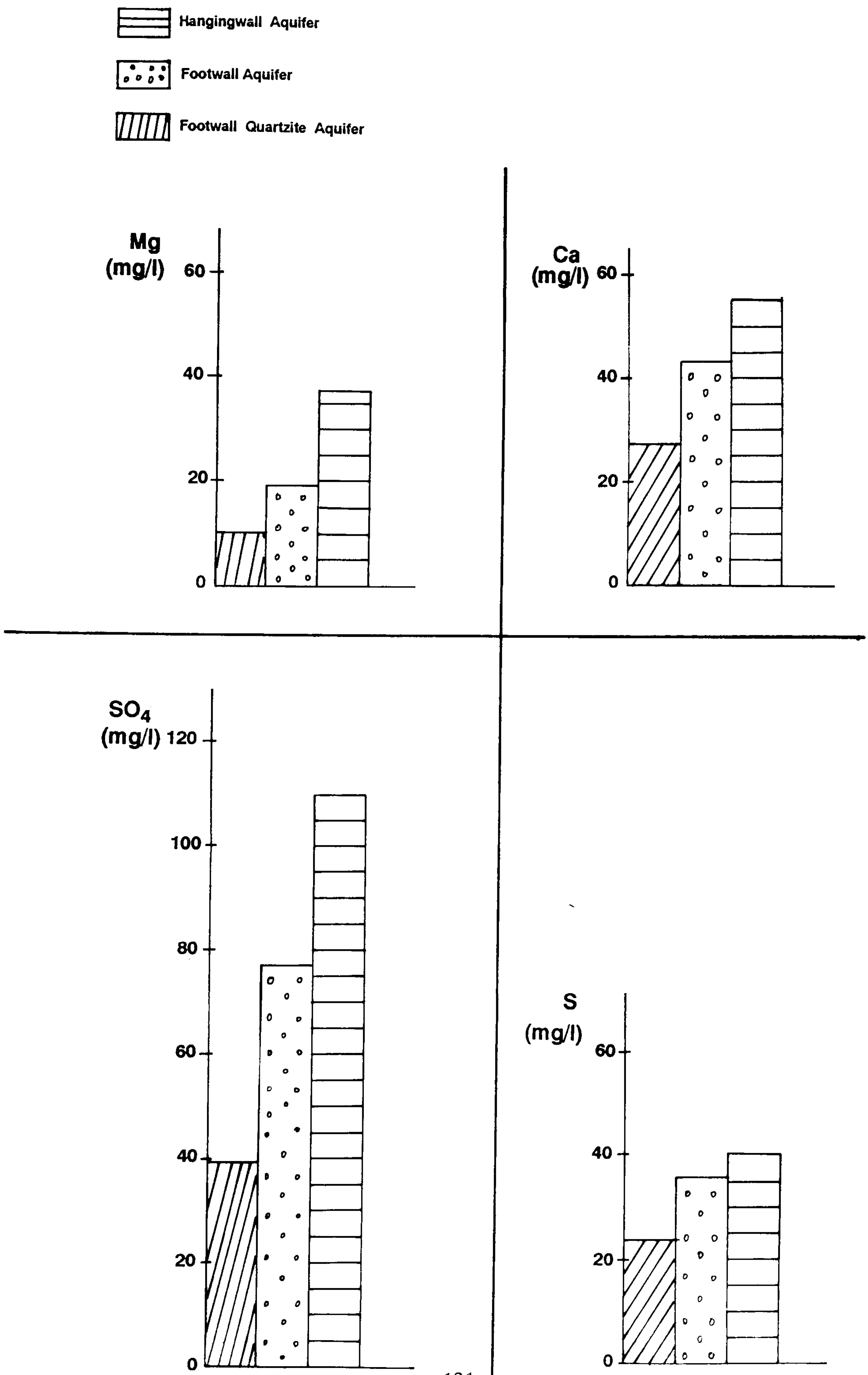


Figure 5.4 : KONKOLA MINE AQUIFER WATER AVERAGE CHEMISTRY - JULY 1988

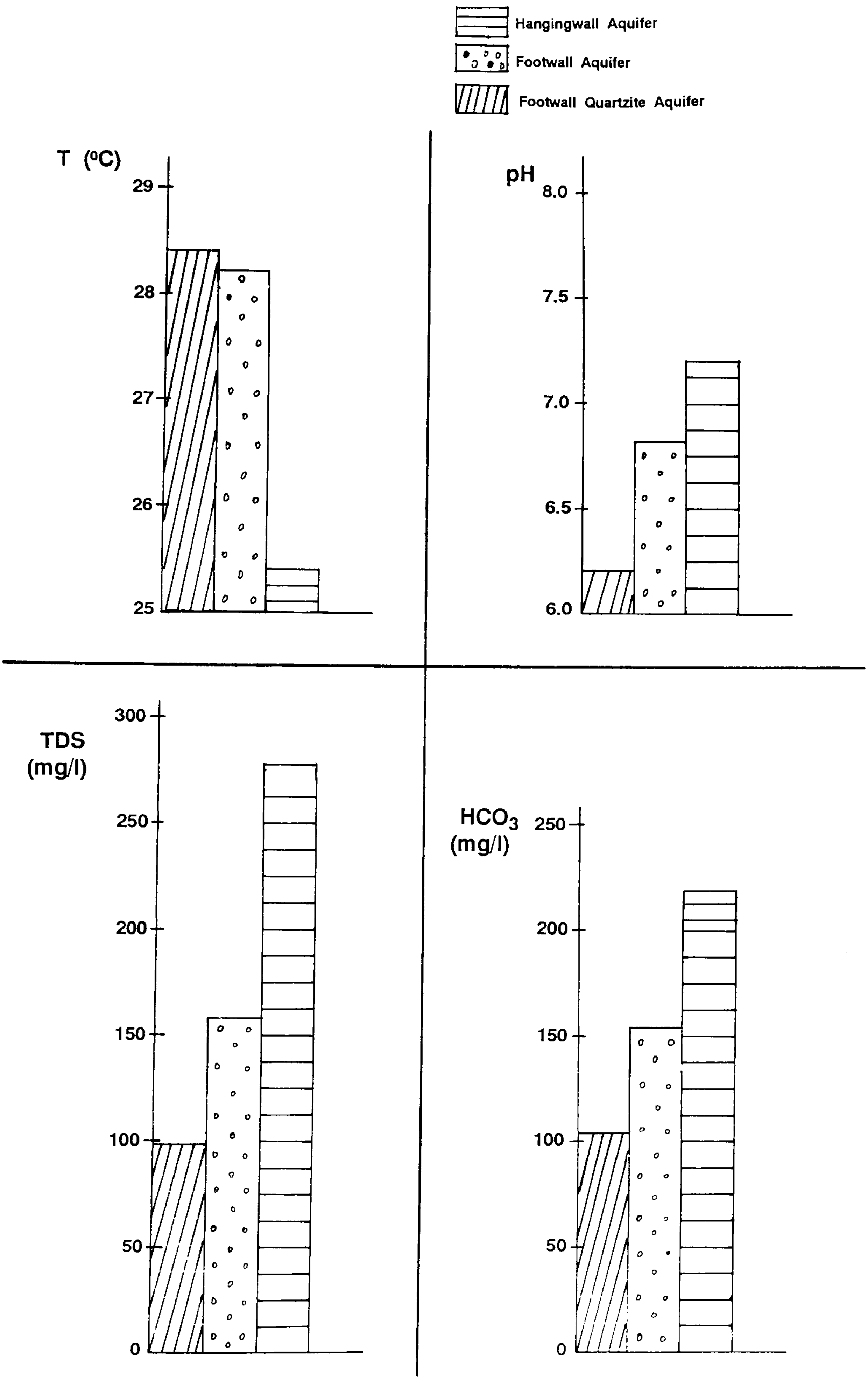


Figure 5.5 : KONKOLA MINE AQUIFER WATER AVERAGE CHEMISTRY - JULY 1988

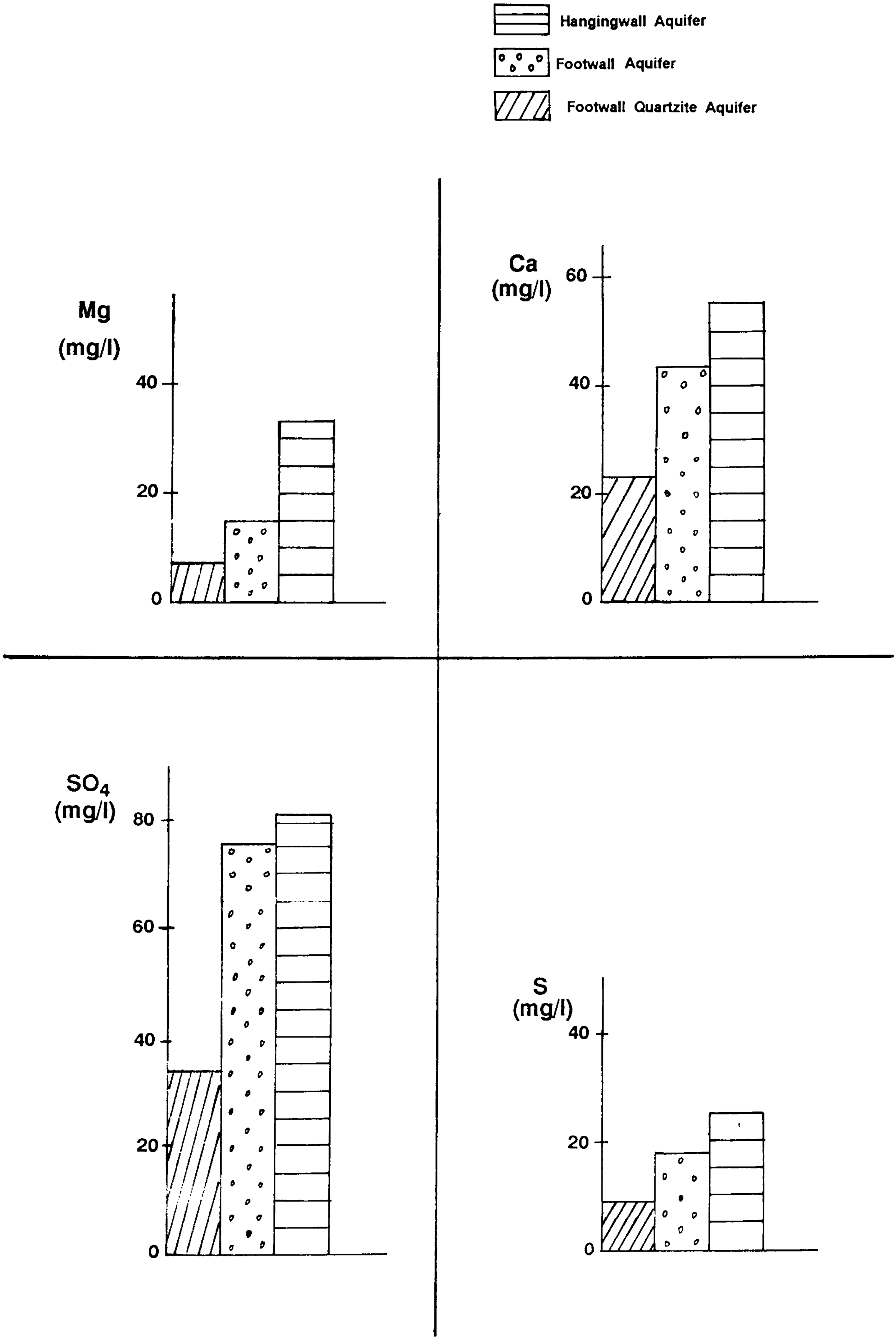


Figure 5.6 : 810m (2650ft.) Level Groundwater pH - July 1988 & 1989

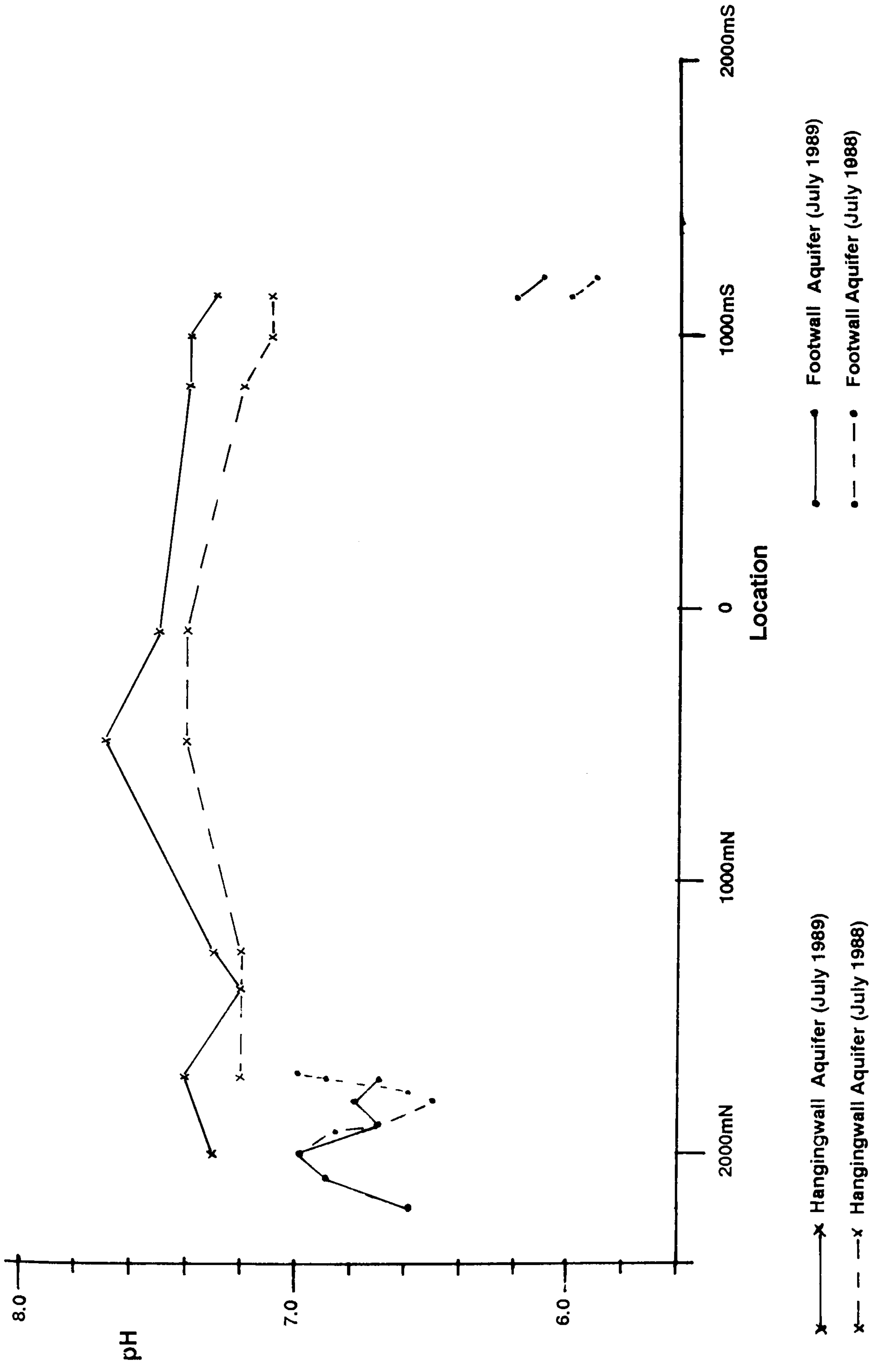
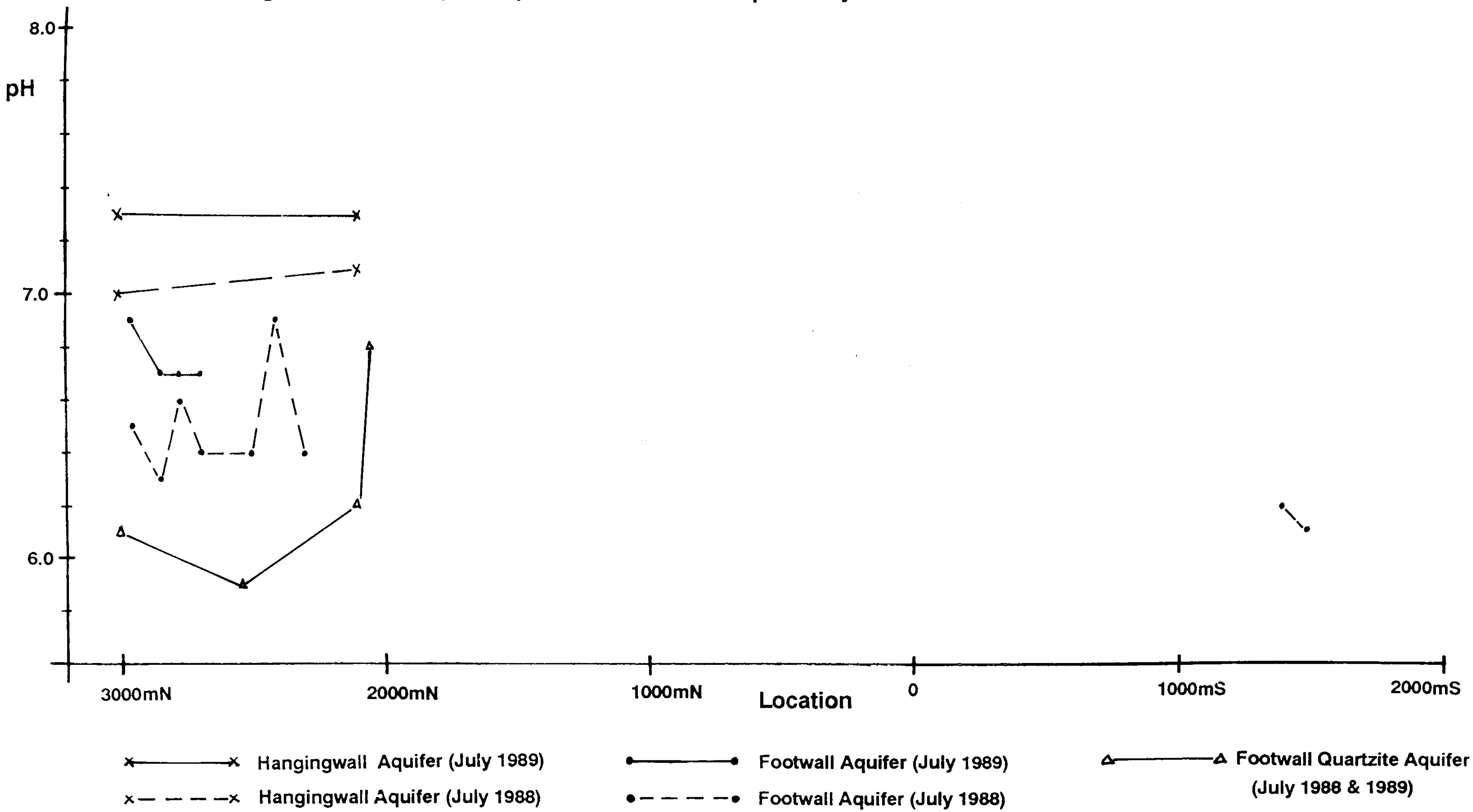


Figure 5.7 : 670m (2200ft.) Level Groundwater pH - July 1988 & 1989



KONKOLA MINE GROUNDWATER CHEMICAL COMPOSITION DIAGRAMS
JULY 1989

Figure 5.8 : Calcium vs Total Dissolved Solids (mg/l)

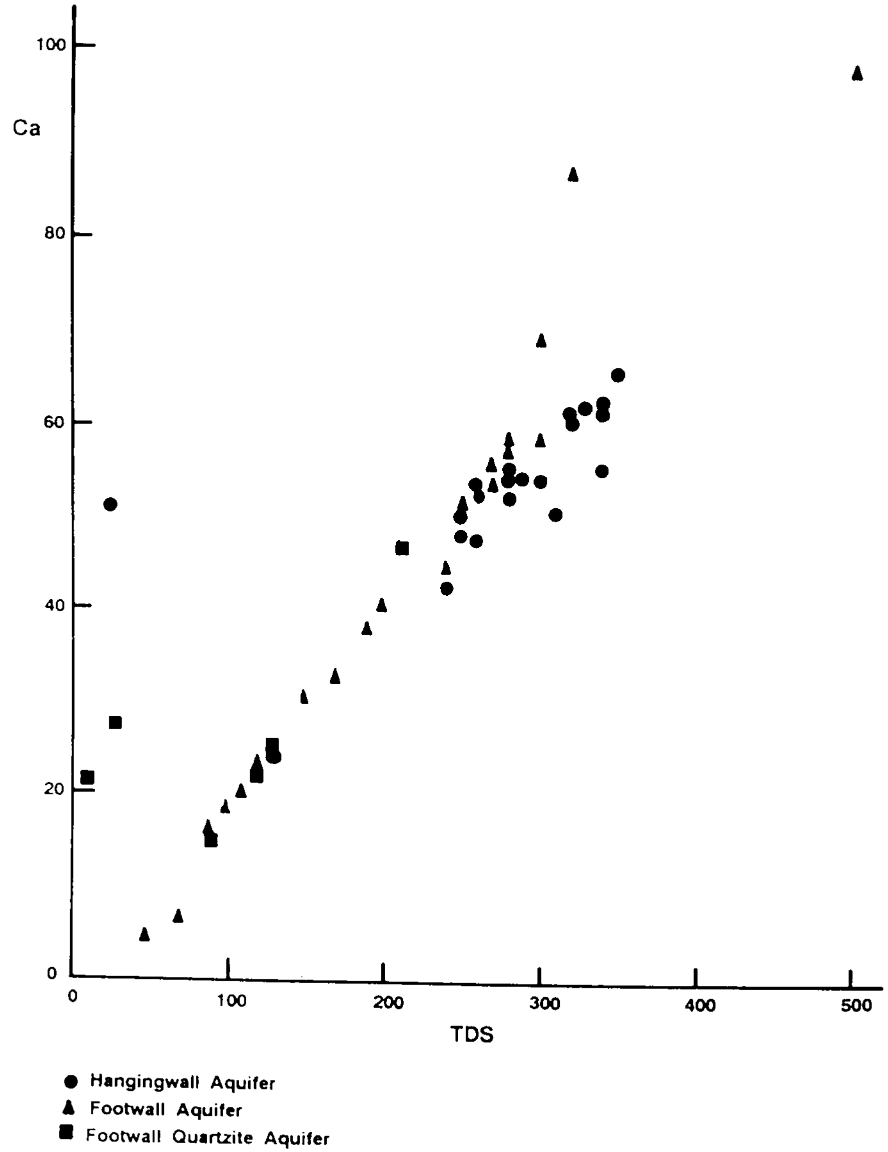
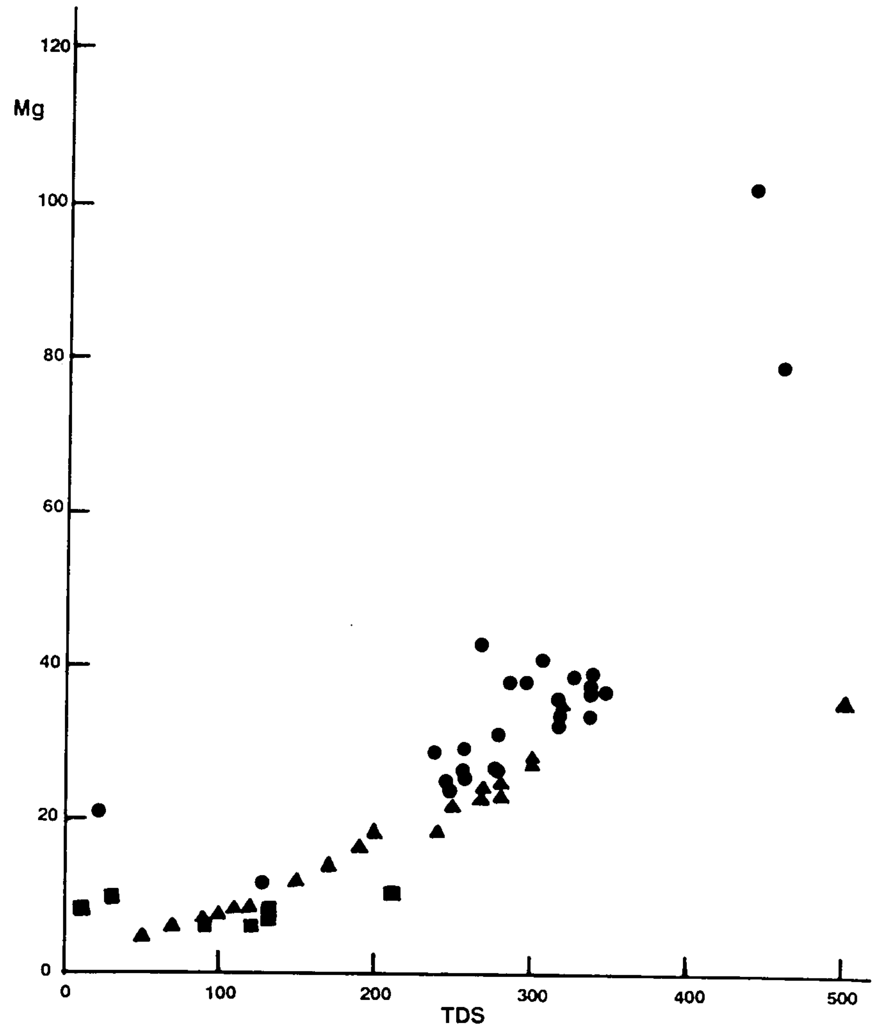


Figure 5.9 : Magnesium vs Total Dissolved Solids (mg/l)



KONKOLA MINE GROUNDWATER CHEMICAL COMPOSITION DIAGRAMS
JULY 1989

Figure 5.10 Magnesium vs Bicarbonate (mg/l)

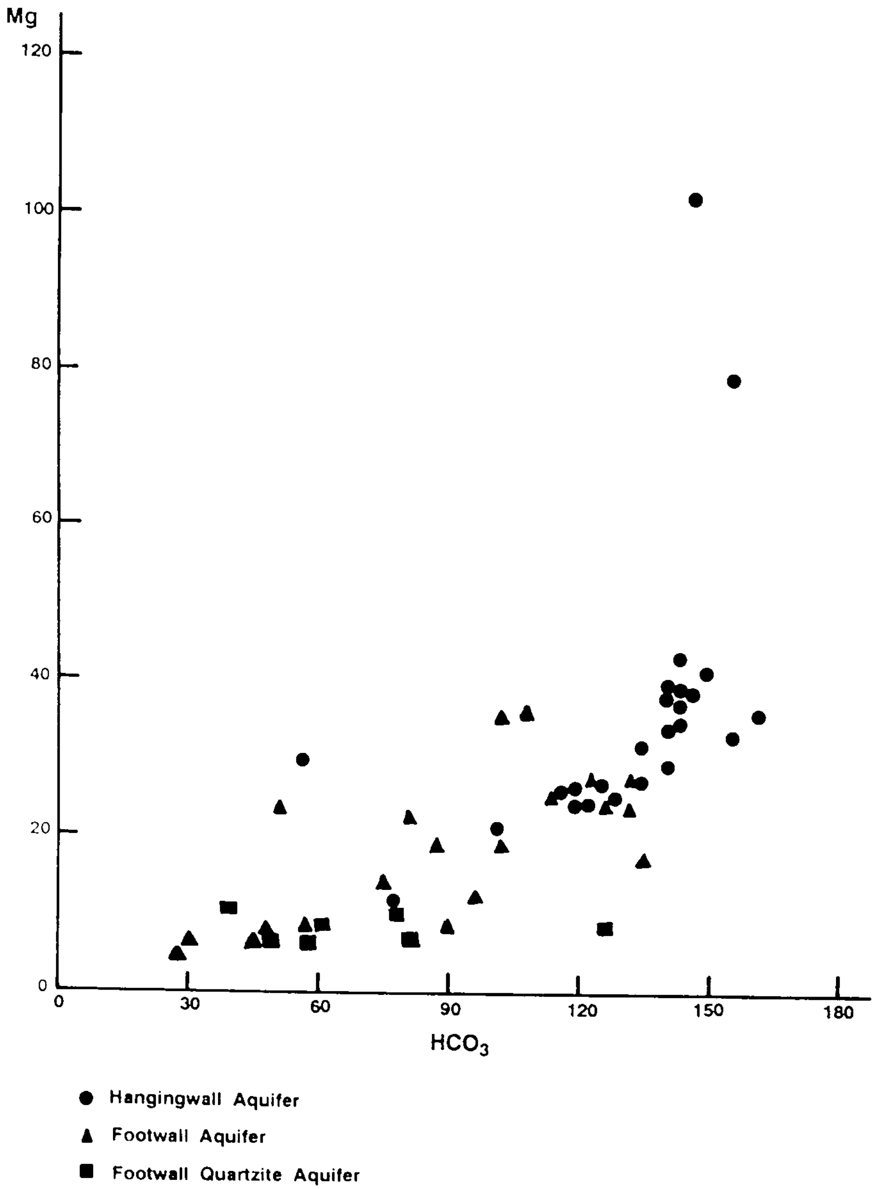
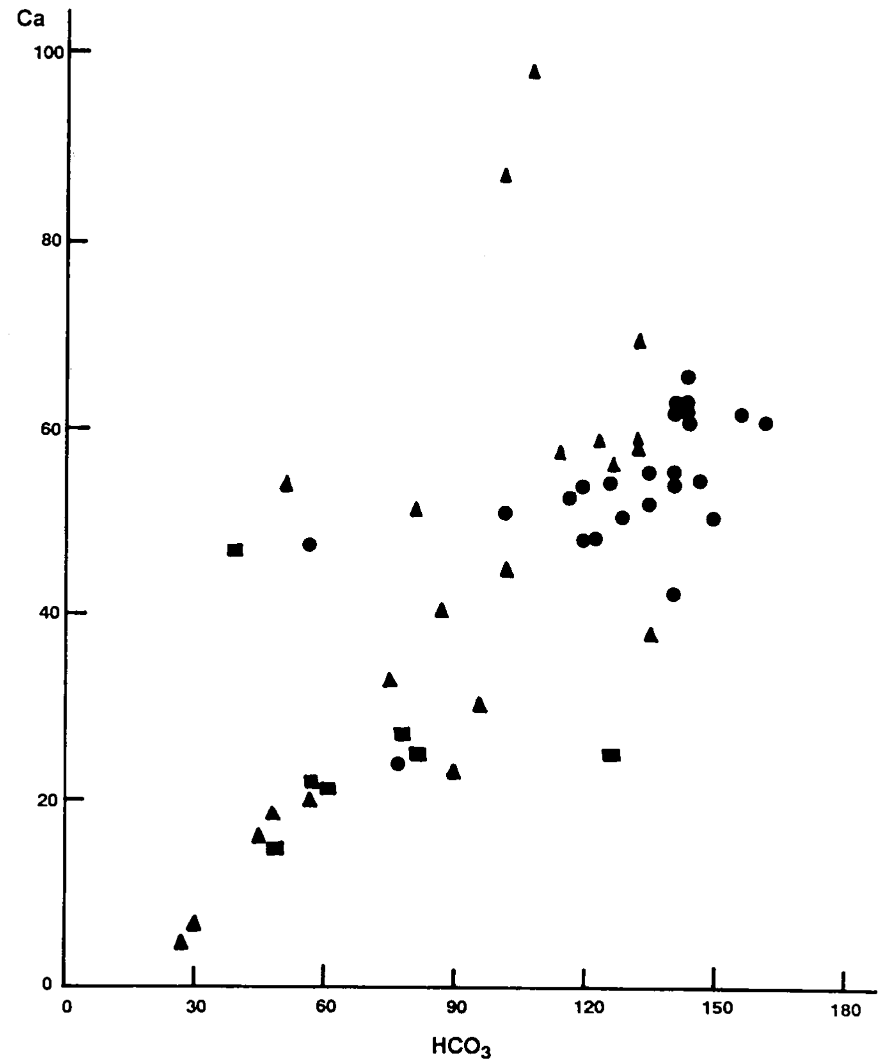


Figure 5.11 : Calcium vs Bicarbonate



different types of water at Konkola. On a plot of calcium versus total dissolved solids, points of different aquifer cluster into two main separate groups, with the clusters lying along a straight line. Hangingwall Aquifer points cluster in the upper parts of the straight line, while those of the Footwall Quartzite Aquifer lie in the lower parts. The Footwall Aquifer points lie in the middle and spreads in the other aquifer zones. According to Mazor (1991) and Mazor et al (1985), this is strong evidence for the existence of two different types of water which are mixing. For the reason explained earlier, it is strongly believed that the mixing takes place mainly in fault and fissure zones where there is direct interconnection between aquifers.

5.2 TRACING GROUNDWATER FLOW PATTERN

Having now established that there exists at Konkola mine essentially two types of water, it was necessary to establish the flow pattern as this had a direct link with identifying the sources of recharge and delineating the major flow routes.

In order to achieve this an analysis of both surface water and groundwater temperature distribution pattern was carried out.

The groundwater temperature is controlled by the flow of heat from the interior of the earth; the geothermal gradient (Fernandez-Rubio *et al* 1988, Mazor and Verhagen 1976, and Stevens Jr *et al* 1975). High groundwater temperatures usually indicate the presence of deep regional flow systems. If the source of recharge is surface water, groundwater temperature will increase in the direction of flow while if the source of recharge is at greater depth temperature will decrease in the direction of flow. Groundwater at shallow depths has nearly constant temperature that approximates the mean annual air temperature of the region. Seasonal variations in temperature is usually damped out at a formation depth of about 9 metres (Domenico 1972).

Therefore, in light of the importance of the geothermal gradient with respect to determining the overall groundwater flow pattern, it was essential to establish the geothermal gradient at Konkola mine.

5.2.1 Rock Temperature and Geothermal Gradient

Dry virgin rock temperatures were measured on different levels of underground workings of the mine in order to establish the geothermal gradient of the Konkola mine area.

Following the South African Chamber of mines recommended procedures (Transvaal and Orange Free State Chamber of Mines, 1965), the dry rock temperature at each site was measured by drilling a 15mm-diameter 40m deep borehole, perpendicular to haulage direction. Holes were drilled in the Footwall Quartzite formation. Six clinical thermometers were then mounted in special grooves, cut in blasting-charging rods, at equal intervals and then left in the hole for two hours to allow the thermometer to attain rock temperature. Holes were sealed off from the effect of haulage ventilation air current by plugging them with cotton wool.

Graphs of temperature versus length of borehole were plotted for each site, as shown in Figures 5.12a to 5.12d. The representative temperature was that at which the curve flattened.

From these representative temperatures for each mine level, a graph of temperature versus mine depth was plotted and the geothermal gradient of the area determined as demonstrated in Figure 5.13. The graph gives a geothermal gradient for Konkola mine of 1°C for every 435m.

Figure 5.12 : DETERMINATION OF DRY VIRGIN ROCK TEMPERATURE

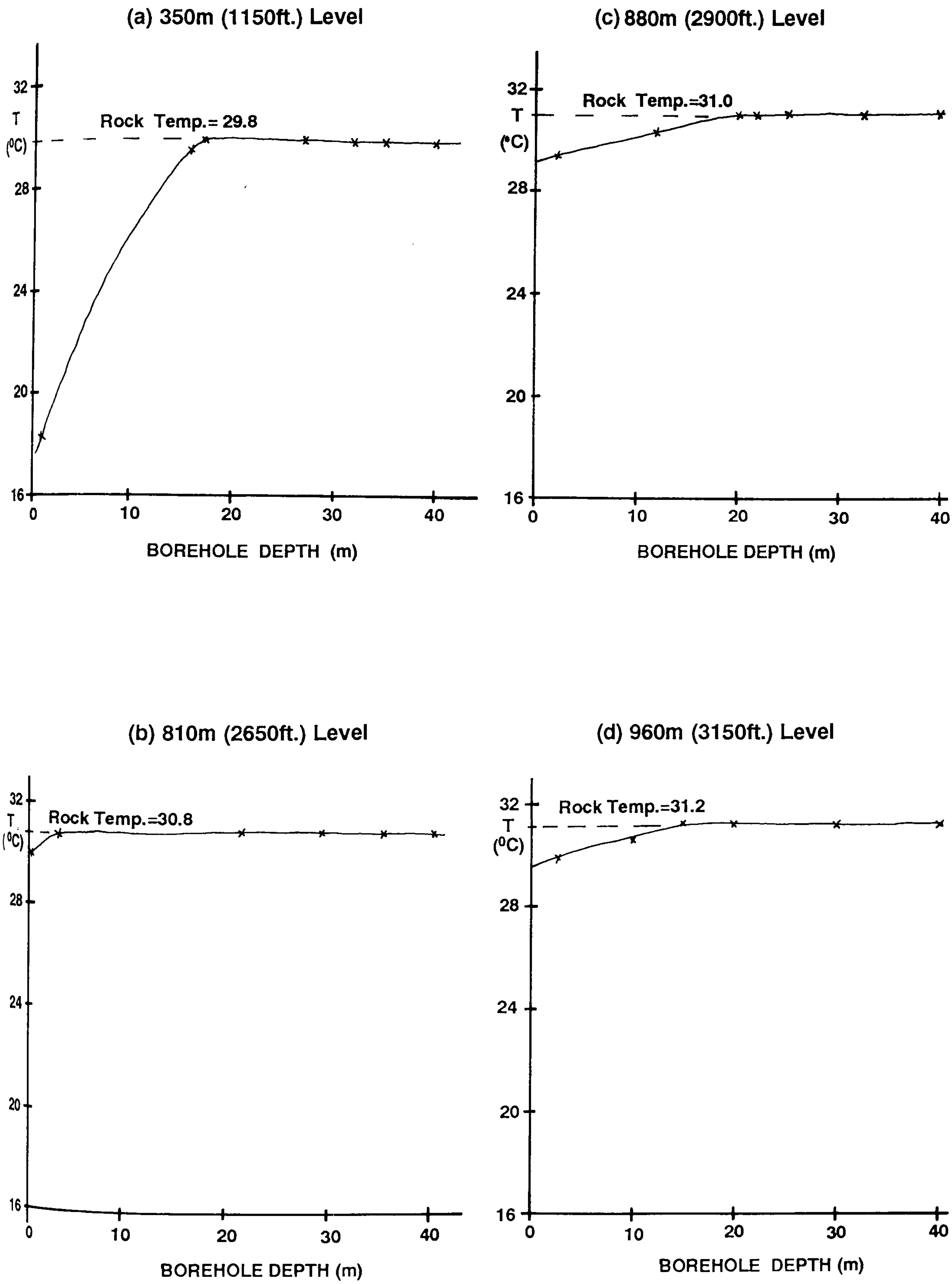
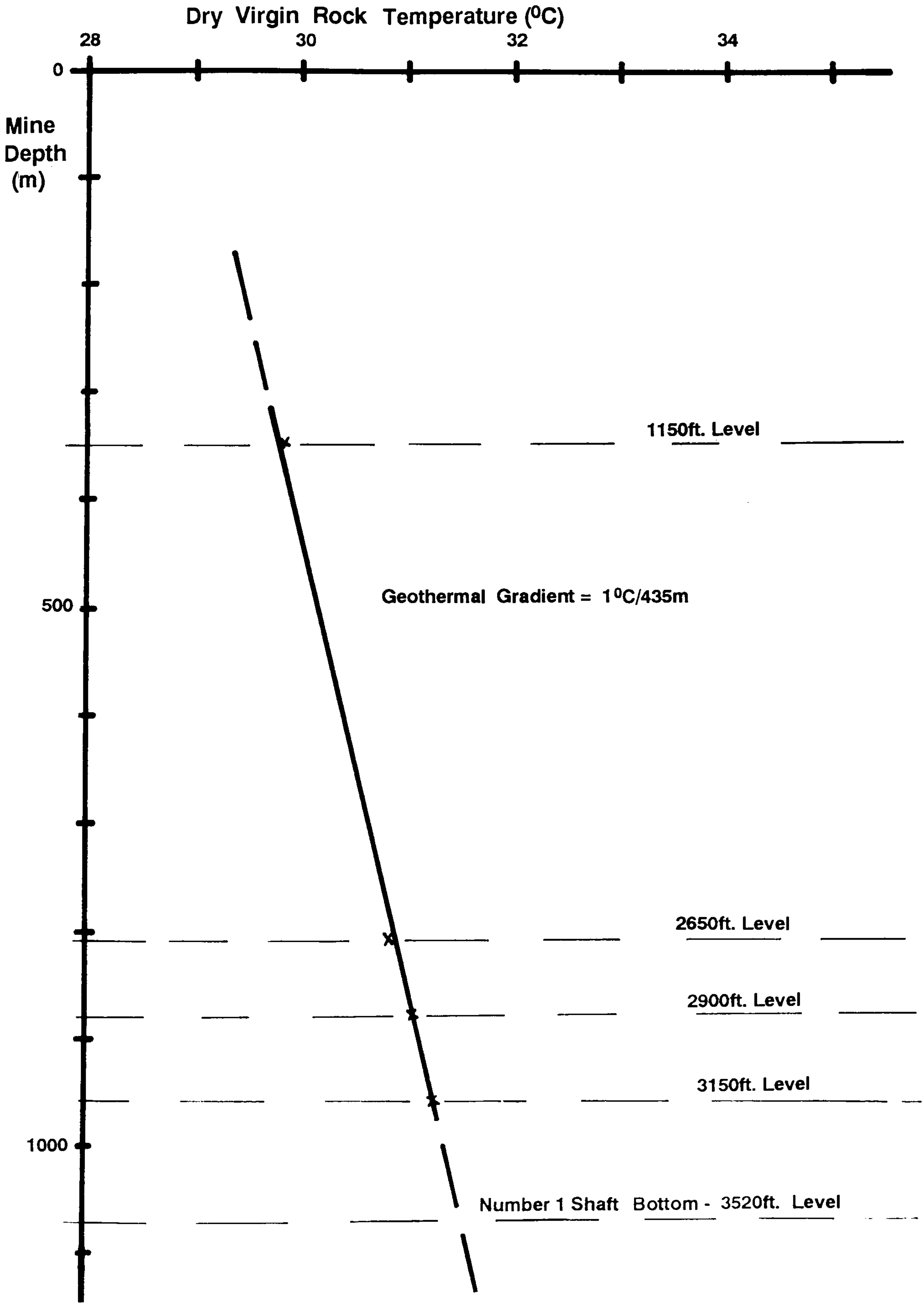


Figure 5.13 KONKOLA MINE LEVELS
ROCK TEMPERATURE & GEOTHERMAL GRADIENT



The dry virgin rock temperatures and geothermal gradient are shown in Table 5.19 below.

Table 5.19 Konkola Mine Dry Virgin Rock Temperature

Mine Level	Temperature (°C)	Geothermal Gradient
350m (1150 ft)	29.8	1°C/435m
450m (1480 ft)	30.0	
565m (1850 ft)	30.3	
670m (2200 ft)	30.5	
810m (2650 ft)	30.8	
880m (2900 ft)	31.0	
960m (3150 ft)	31.2	

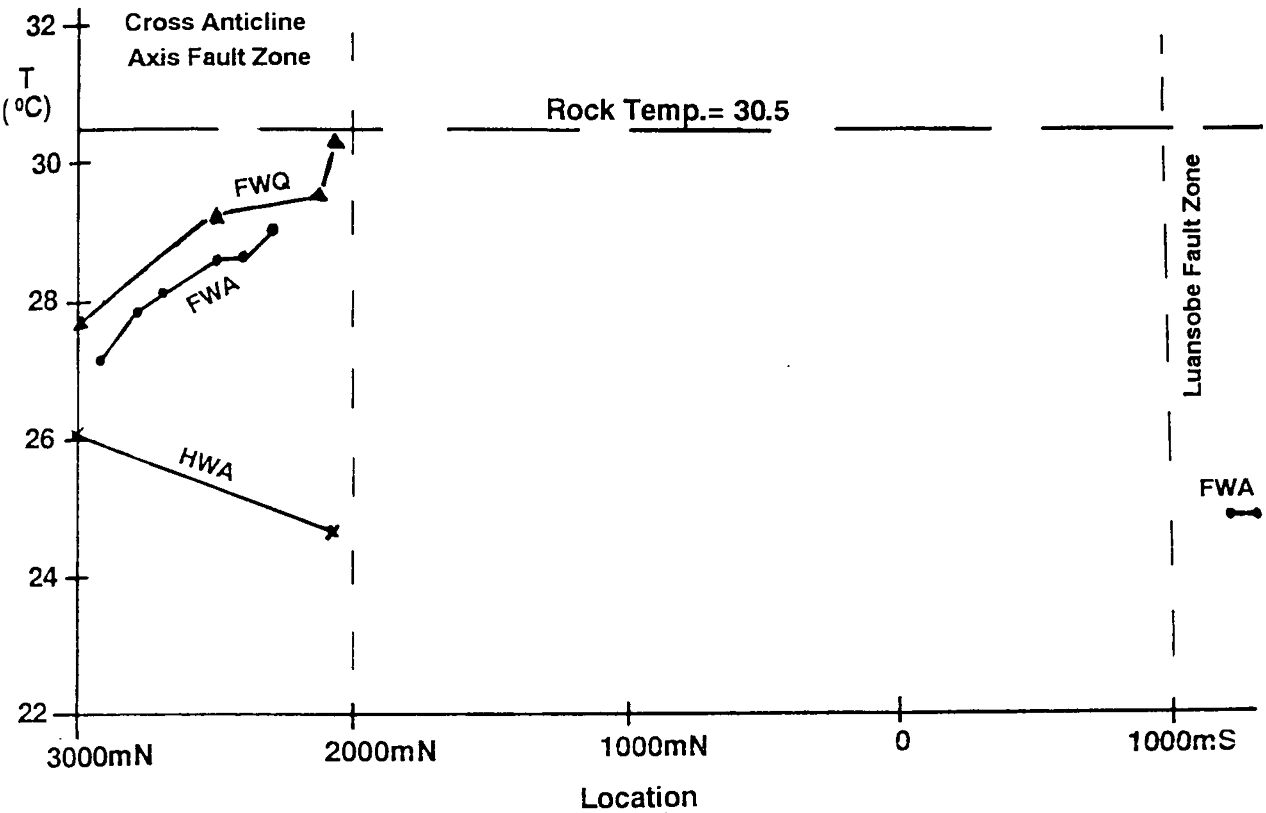
The geothemal gradient for Konkola mine is the lowest of all the Zambian Copperbelt mines. The values for other mines range from 1°C/30m to 1°C/60m. This low value of geothermal gradient at Konkola is positive evidence for the presence of a massive body of cool water that is recharging the mine and consequently lowering the rock temperature far below the range of the regional geothermal gradient. This cool water can only originate from the surface water system.

5.2.2. Groundwater Flow Pattern as Deduced from Water Temperatures

The groundwater temperatures were measured at source rock, before water had much contact with the surrounding air. The thermometer used had a specified range of -15°C to 170°C and accurate to 0.1°C (Temperature Sensor PT154). Groundwater temperature cross-sections (Figures 5.14 to 5.17) show that Hangingwall Aquifer water is essentially downward flow from higher (cooler) levels near surface, and Footwall Aquifer water is upward flow from deeper (hotter) levels.

Figure 5.14 : KONKOLA MINE : NUMBER 1 SHAFT GROUNDWATER TEMPERATURE CROSS SECTIONS - JULY 1988

(a) 670m (2200ft.) Level



HWA=Hangingwall Aquifer
FWA=Footwall Aquifer
FWQ=Footwall Quartzite Aquifer

(b) 810m (2650ft.) Level

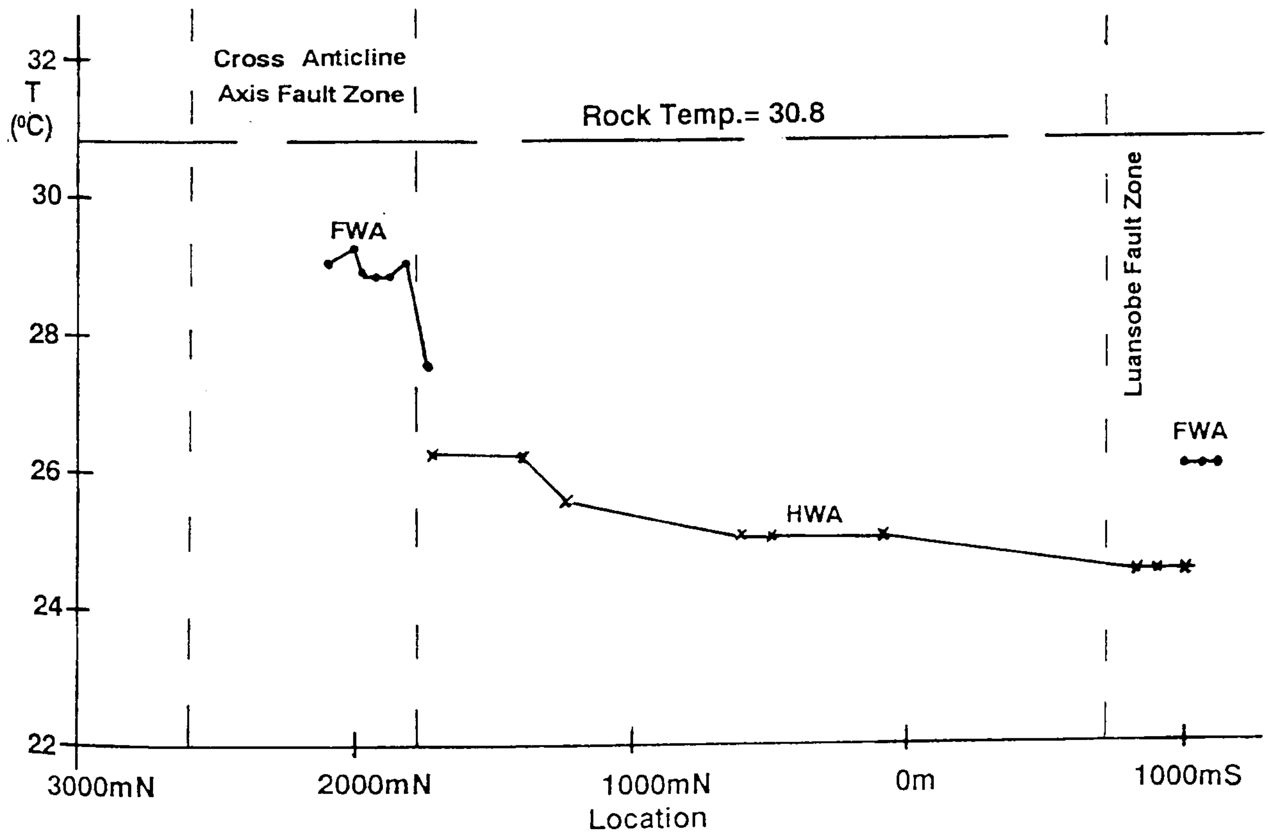
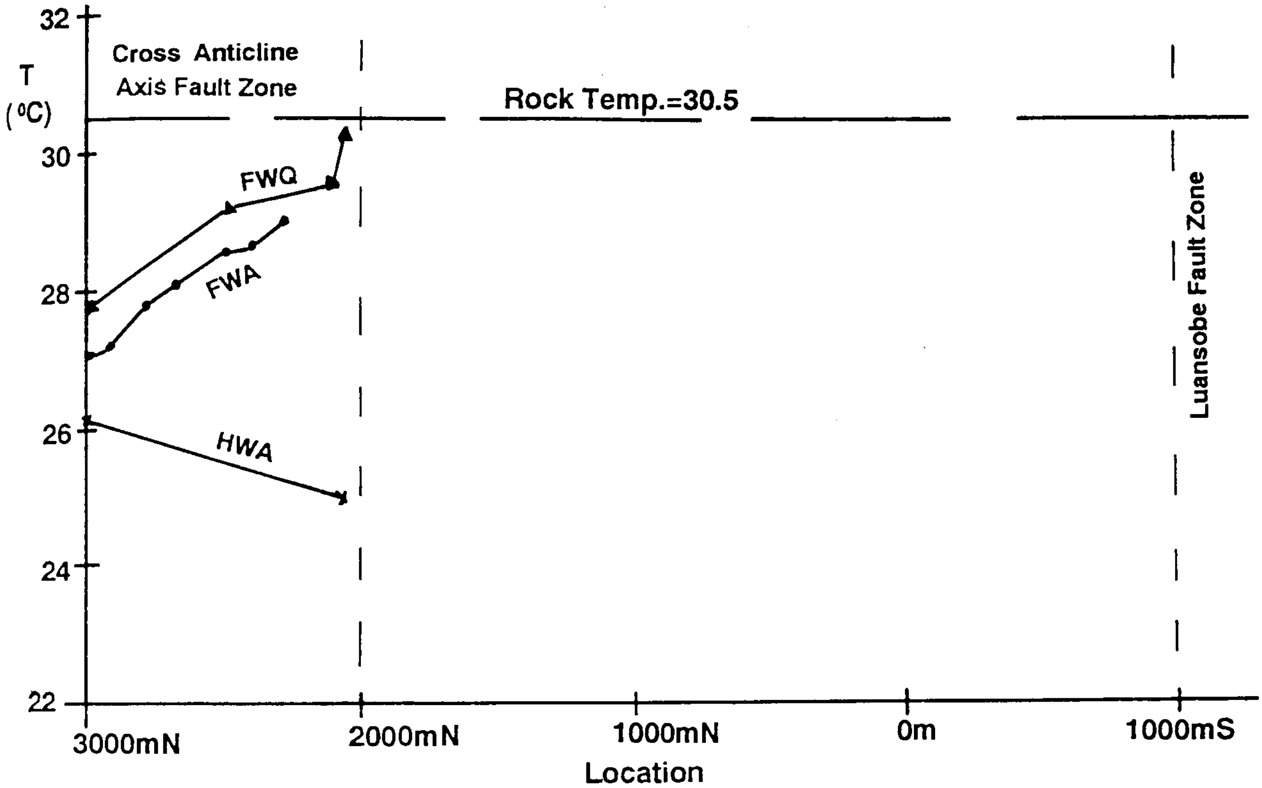


Figure 5.15 : KONKOLA MINE : NUMBER 1 SHAFT

GROUNDWATER TEMPERATURE CROSS SECTIONS - JULY 1989

(a) 670m (2200ft.) Level



HWA= Hangingwall Aquifer

FWA= Footwall Aquifer

FWQ= Footwall Quartzite Aquifer

(b) 810m (2650ft.) Level

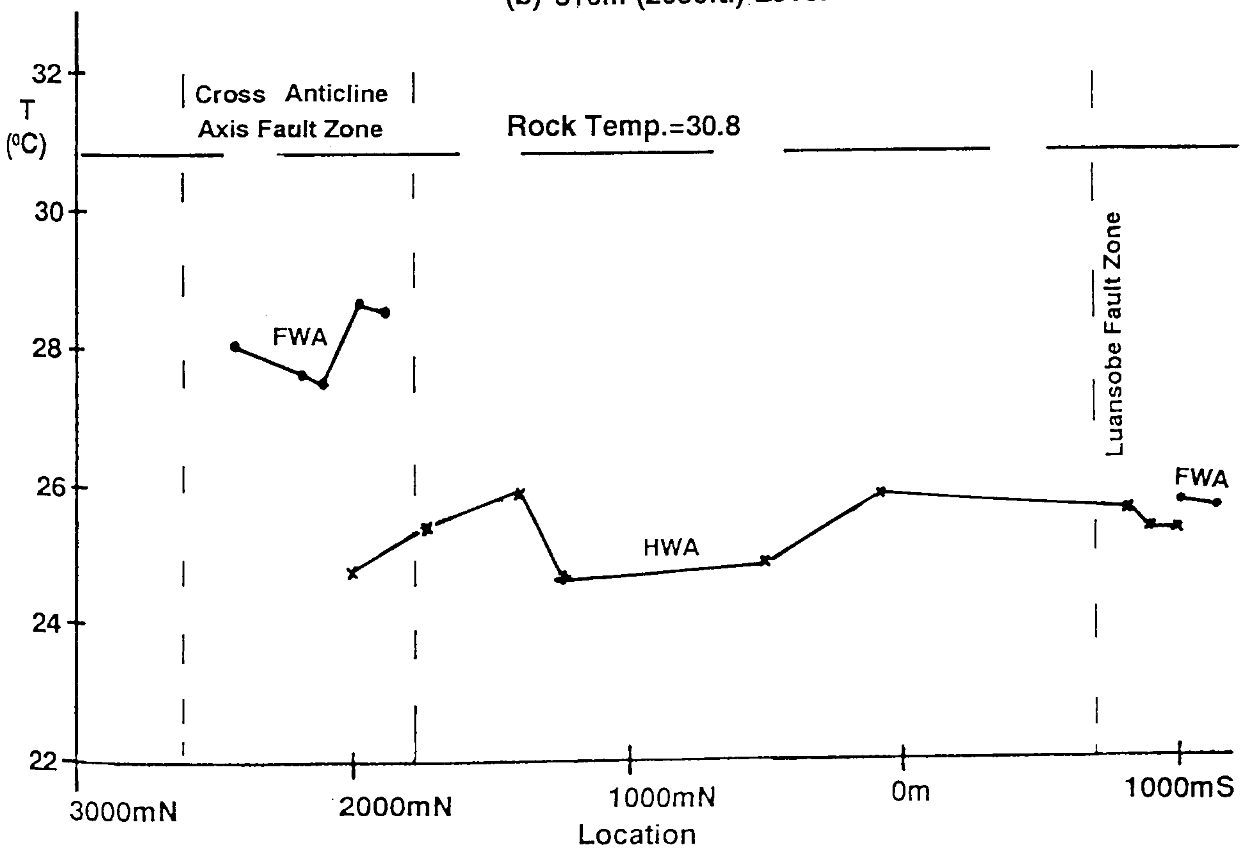


Figure 5.16 : COMPARISON OF JULY 1988 & 1989
GROUNDWATER TEMPERATURE AT NUMBER 1 SHAFT

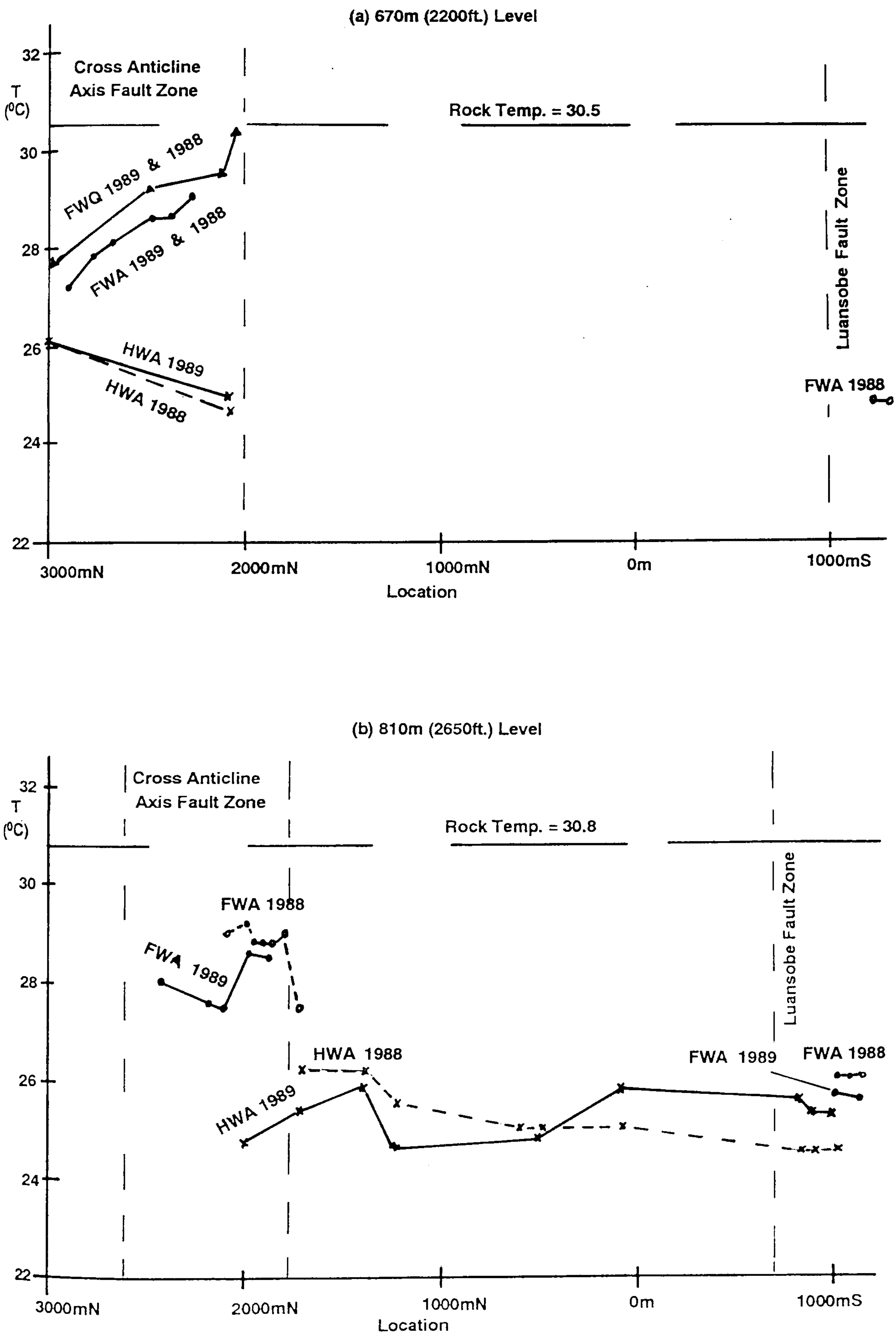
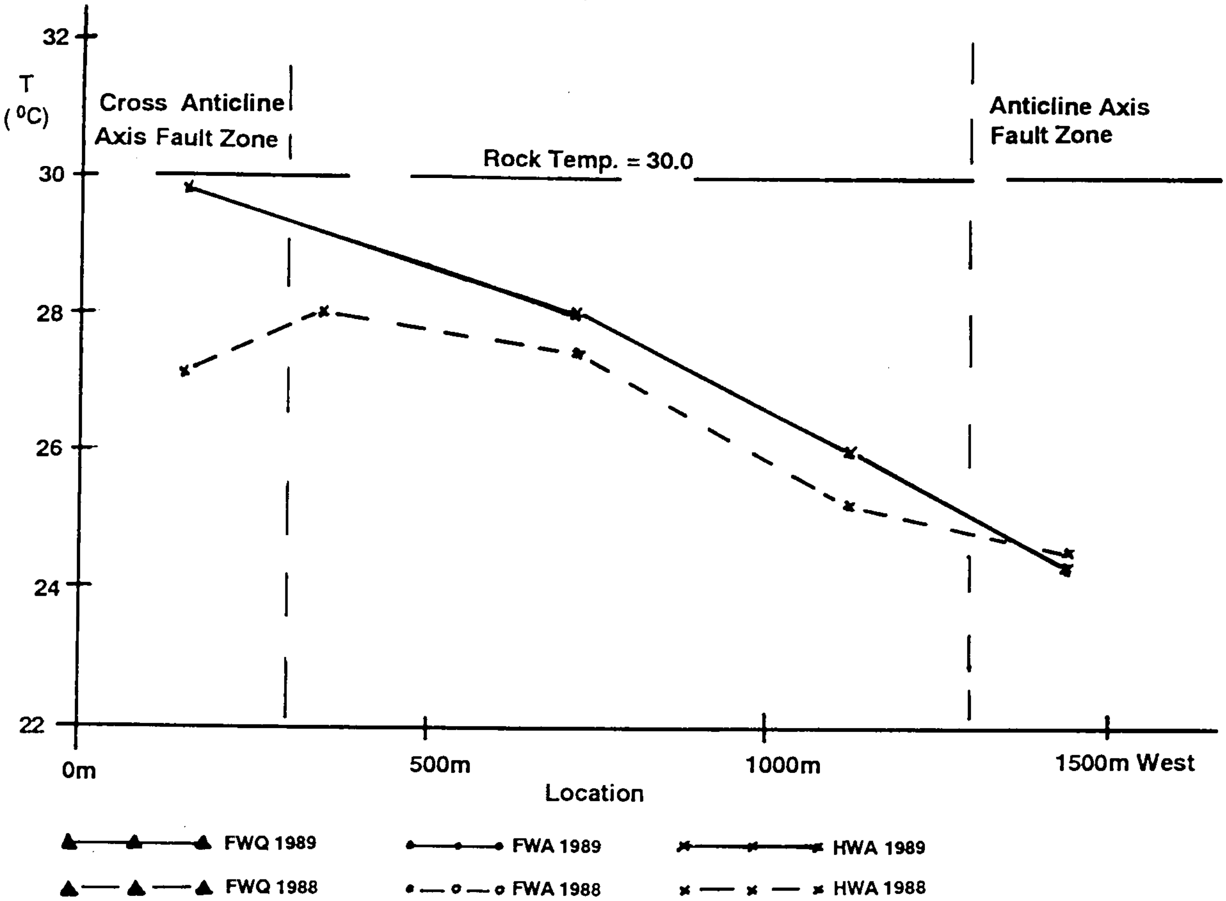
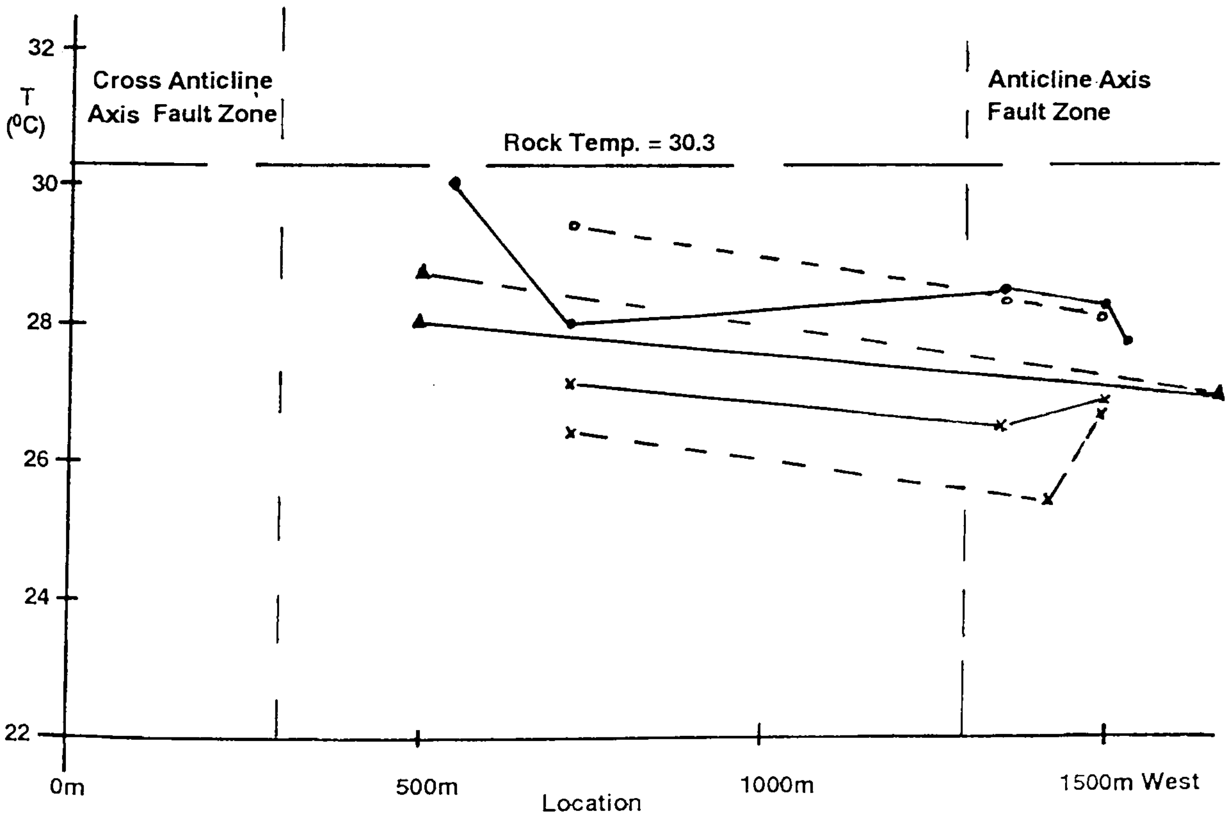


Figure 5.17 : COMPARISON OF JULY 1988 & 1989 GROUNDWATER TEMPERATURE AT NUMBER 3 SHAFT

(a) 450m (1480ft.) Level



(b) 565m (1850ft.) Level



The fault zones are again seen as acting as major conduits of water inflow to the mine. These faults act as both mixing and drainage zones.

As seen in Figures 5.15b and 5.16b, in the Luansobe Fault zone, there is very little difference in temperature between the Footwall and Hangingwall Aquifers' waters. For example, temperature difference on 2650 ft level, in July 1988 was 1.5°C and dropped to 0.4°C in July 1989. This strongly suggests mixing of the two waters. The Luansobe Fault zone is a Mixing Zone. The same pattern of behaviour was observed in the Anticline Axis Fault zone.

On the other hand, in the Cross-Fold Axis Fault Zone, there was a very significant temperature difference between the Footwall and Hangingwall Aquifers' waters. The temperature differences were as follows. On 2200ft level; July 1988 = 4.5°C, July 1989 = 3.5°C and 2650ft Level; July 1988 = 3.2°C, July 1989 = 3.8°C. This strongly demonstrates that in this zone, groundwater from the Footwall and Hangingwall Aquifers moves so rapidly that it does not have time to mix and equilibriate in temperature. This is suggestive of a Drainage Zone. Footwall Aquifer water upwells from deep levels and Hangingwall Aquifer water flows downwards from higher levels near surface.

5.3 TESTING FOR HYDRAULIC INTERCONNECTION BETWEEN SURFACE WATER AND GROUNDWATER SYSTEMS

All analyses of data and hydrogeological setting of the mine strongly suggested that there was hydraulic interconnection between the aquifer water in the mine and the surface waters of the Kafue River and its tributaries, and the Mine Tailings Dam.

In order to confirm this, two tracer tests were conducted using a bacteria that resides only in sewerage environment, and a chemical which is a by-product of the copper ore concentration process.

5.3.1 Inflow from the Kafue River and the Tributaries

There are sewerage ponds located on the southern periphery of Number 1 Shaft which discharge into the Kafue River downstream of the Kafue Pump House.

Escherichia Coli bacteria is a marker in water contaminated with sewage. Samples of water from the Kafue River in the vicinity of the sewage ponds, and from all the aquifers in all underground mine workings both at Number 1 and 3 Shafts, were analysed for the presence of this bacteria (Cruickshank 1975). The exercise was carried out in both July 1988 and July 1989. A total of thirty eight samples were analysed in July 1988 and fifty seven samples in July 1989, as shown in Tables 5.20 and 5.21 respectively in Appendix 2.

The Escherichia Coli bacilli was found present in both the Kafue River and its tributaries, and underground mine workings closest to these areas (Figure 5.18). This proved the existence of direct hydraulic interconnection between the surface river water system and the groundwater in the mine. It was noteworthy that the concentration of the bacilli in both waters, especially in July 1988, was similar. This implied that leakage of the Kafue river water into the mine is vey significant.

5.3.2 Inflow from the Lubengele Mine Tailings Dam

Xanthate is a chemical compound used in the recovery process of copper in the concentrator plant. The effluent from the concentrator is discharged into the Mine Tailings Dam, which as stated in the preceeding chapters, is located on the Lubengele Fault.

A total of thirty six samples were collected from the Mine Tailings Dam area, Kafue River and underground mine workings (Table 5.22 in Appendix 2).

Figure 5.18 : PLAN OF 565m (1850ft.) AND 810m (2650ft.) LEVELS SHOWING DISTRIBUTION OF E. COLI BACILLI IN GROUNDWATER

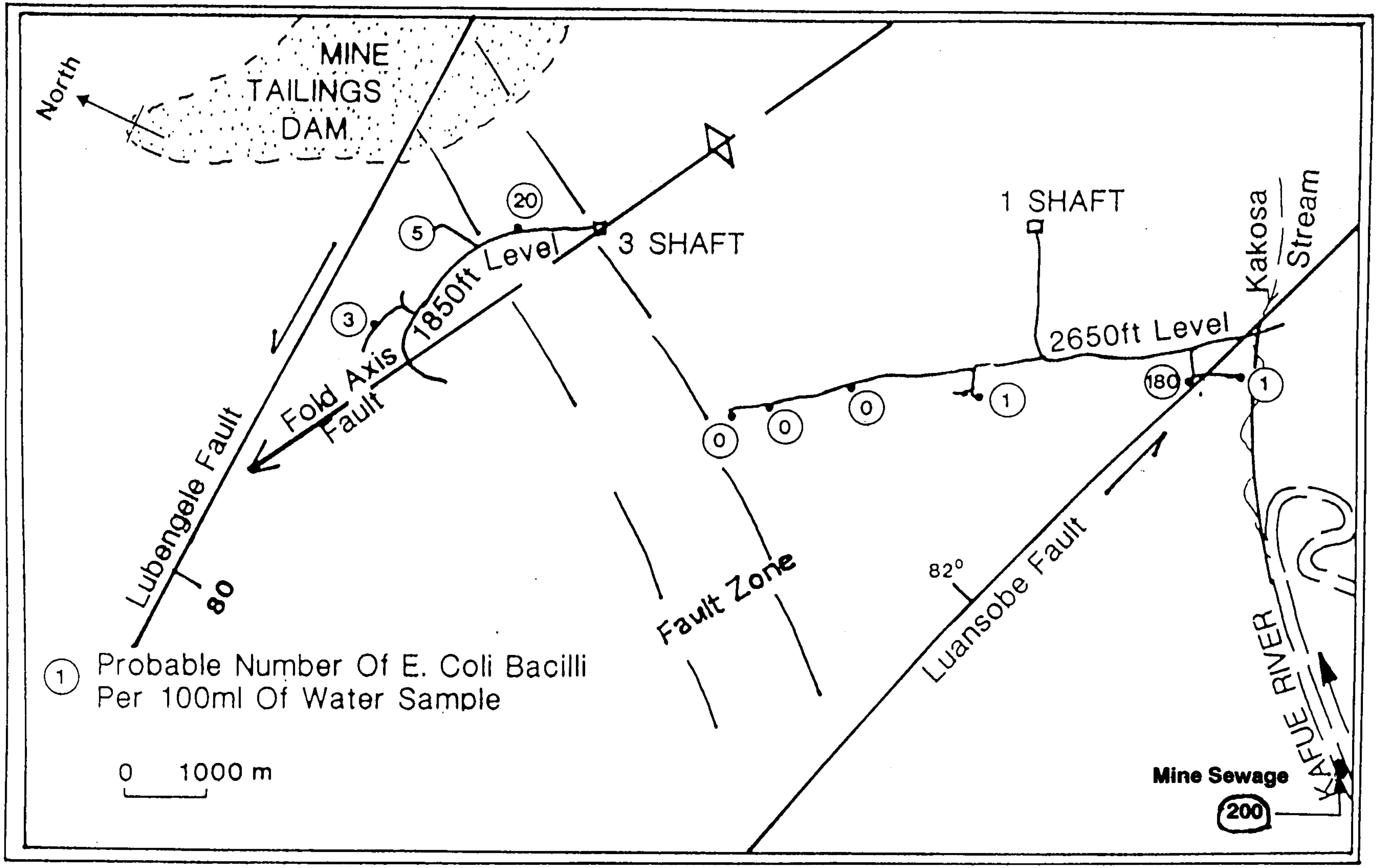
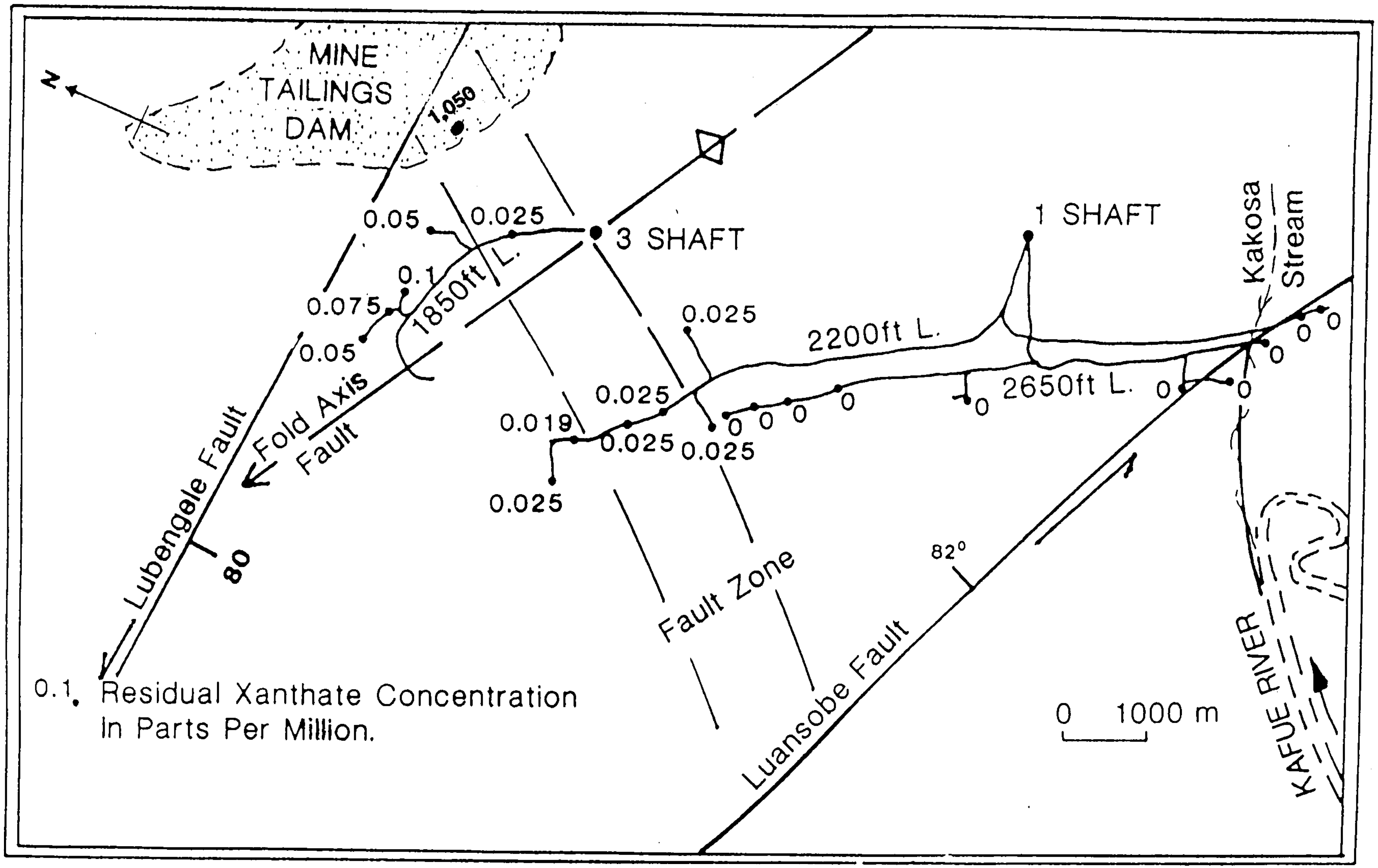


Figure 5.19 : PLAN OF 565m (1850ft.) AND 810m (2650ft.) LEVELS SHOWING THE DISTRIBUTION OF RESIDUAL XANTHATE IN GROUNDWATER



Residual Xanthates were found to be present in the Tailings Dam waters as well as in underground mine workings in the area directly close to the dam, as shown in Figure 5.19. More significant is the fact that the Xanthate was concentrated in the area lying within the Cross Fold Axis Fault zone and the north limb of Number 3 Shaft which is close to the Lubengele Fault. A direct link between Tailings Dam leacheate and the mine was thus proved. The Mine Tailings Dam water is leaking into the mine through the Lubengele and Cross Fold Axis Fault zones.

5.4 ISOTOPIC DATING OF MINE WATERS

In order to determine the duration of groundwater residence and further confirm the link between surface waters of Kafue River system and the mine groundwater, studies of tritium content of water were undertaken.

Tritium is a radioactive isotope of hydrogen. It allows the determination of effective age of the water to be determined as pre- and post-1952, the time hydrogen bomb tests begun. It has a half life of 12.3 years (Mazor 1991, Fernandes-Rubio *et al* 1988, Mazor 1982, Verhagen *et al* 1979 and 1974, and Vogel *et al* 1974). Thus, after 12.3 years half the initial concentration of Tritium atoms is left, after 24.6 years only a quarter is left and so on. A radioactive decay curve of Tritium is given in Figure 5.20 (Mazor 1991).

Twelve samples were collected at the mine from the Kafue River and underground mine workings, in July 1989. These samples were independently analysed by the Isotope Measurements Laboratory at Harwell - United Kingdom Atomic Energy Authority (Otlet 1989).

The results are tabulated in Table 5.23.

Table 5.23: Konkola Mine Water Tritium Content - July 1989

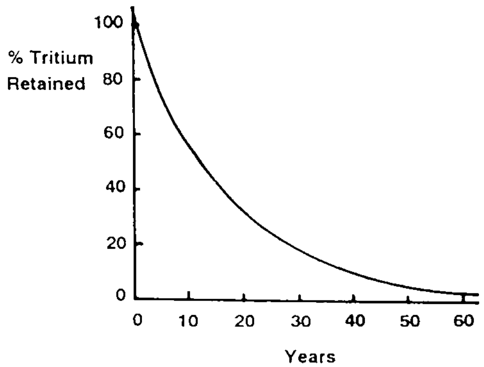
Sampling Site			Water Source	Tritium Activity (TU)±1
Kafue River - Pump Station			River	5.3
Mingomba River - Pump Station			River	4.6
3 Shaft	- 565m (1850ft) Level	- Fissure - 725mW	FWA	0.3
	- " (1850ft) Level	- Borehole BPN247 - 725mW	HWA	3.0
	- " (1850ft) Level	- Borehole BPN237 - Anticline Axis	FWQ	5.6
1 Shaft	- 670m (2200ft) Level	- Borehole CP394 - 2950mN	FWA	0.6
	- " (2200ft) Level	- Borehole AD951 - 3000mN	HWA	0.1
	- 810m (2650ft) Level	- Borehole CP387 - 1190mS	FWA	0.6
	- " (2650ft) Level	- Borehole AD895 - 880mS	HWA	4.1
	- " (2650ft) Level	- Borehole CP438 - 1915mN	FWA	0.0
	- " (2650ft) Level	- Borehole AD956 - 2000mN	HWA	4.6
	- 950m (3120ft) Level	- Borehole AD925 - 10mN	FWQ	0.7

HWA - Hangingwall Aquifer
FWA - Footwall Aquifer
FWQ - Footwall Quartzite Aquifer

These results (Figure 5.21) confirm the evidence already gathered from ionic chemistry and temperature that there are two types of groundwater recharging the mine. There is old and young water. Furthermore and most significant, these results show beyond any doubt that the young water is of the same age as the river water. River water is recharging the Hangingwall Aquifer or the Hangingwall Aquifer water originates from the same source that feeds the river (precipitation). The old water is pre-1952 and, as already demonstrated, originates from the regional aquifer at depth.

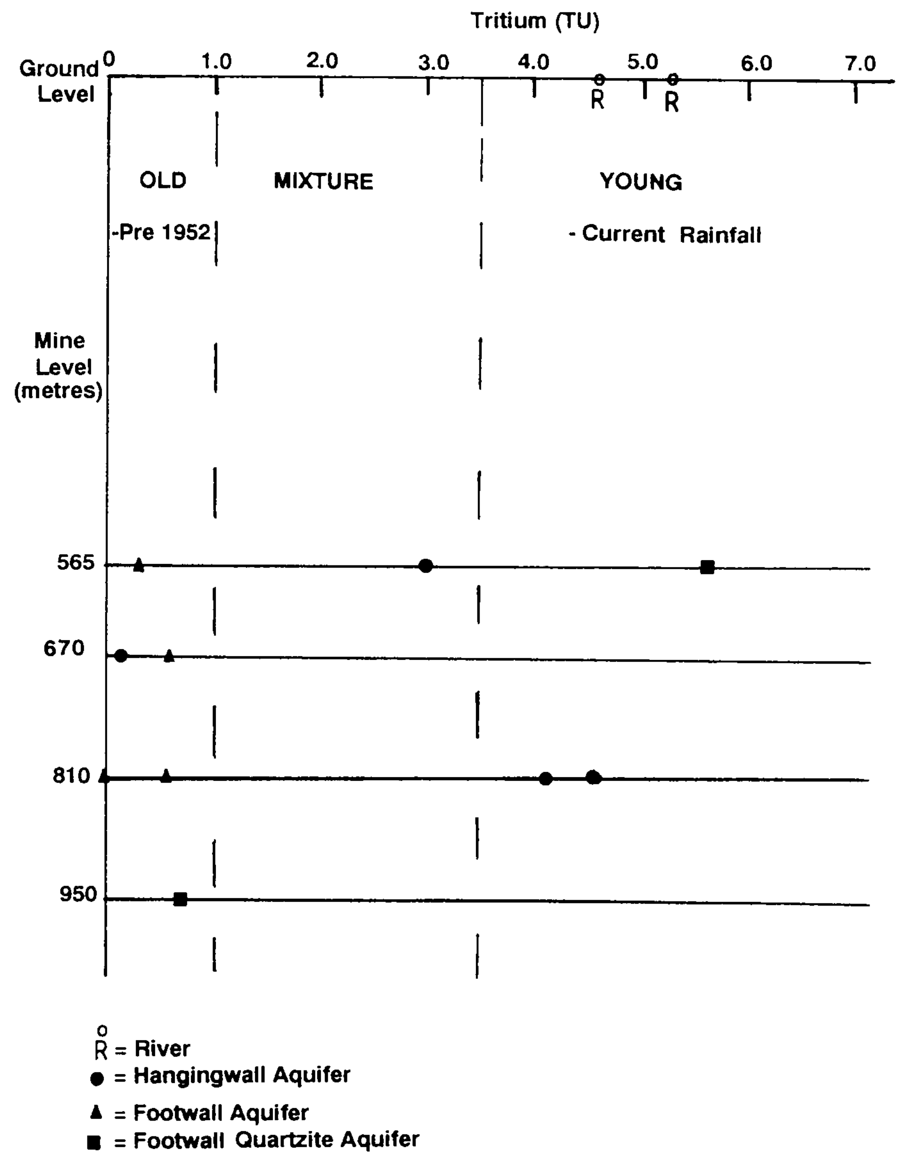
The three highest values of Tritium concentration in groundwater samples were obtained from boreholes located in the Luansobe Fault zone, the Cross-Fold Axis Fault zone and the Anticline Axis Fault zone. This is further evidence to confirm that these faults are the major routes through which water is entering the mine.

Figure 5.20 : **RADIOACTIVE DECAY OURVE OF TRITIUM** (After Mazor 1991)



After 12.3 years 50% of an initial concentration are left, after 24.6 years 25% are left, etc.

Figure 5.21 : KONKOLA MINE GROUNDWATER AND RIVER WATER TRITIUM CONCENTRATION AND RELATIVE AGE - JULY 1989



5.5 QUANTIFICATION OF MIXING PROPORTIONS OF KONKOLA MINE GROUNDWATER

Having established that Footwall Aquifer water is a mixture of Footwall Quartzite Aquifer and Hangingwall Aquifer waters, an attempt was made to estimate the proportions of Hangingwall Aquifer and Footwall Quartzite Aquifer water in the Footwall Aquifer water. This would then enable one to estimate the proportions of Hangingwall and Footwall Quartzite Aquifers' waters in the mine total water inflow.

Using the major dissolved ions (Mg, Ca, HCO_3 and TDS), average values of elemental concentrations were determined for all three aquifers. These data were then used to estimate the mixing proportions in the Footwall Aquifer water. In locations where borehole data for all three aquifers was available, single borehole values were used to estimate the mixing proportions in the Footwall Aquifer water.

This analysis is based on the assumptions that the Hangingwall Aquifer and Footwall Quartzite waters are the end-members of the Konkola Mine groundwater regime and can be separated as such, and still retain a reasonable degree of separation in certain areas unaffected by mining activity and fracturing. *

If two groundwaters A and B mix to produce C, then the proportion of B in C can be computed as follows:

$$Bx + A(1-x) = C \qquad \text{(Mazor 1991)}$$

The July 1989 data in Tables 5.11 and 5.12 was used to work out the estimates. End-member values for ions used were those of 565m Level in Table 5.11 and 670m, 810m and 950m Levels in Table 5.12. Average aquifer water ion concentrations are tabulated in Table 5.24 and estimates of proportions of Hangingwall Aquifer water in Footwall Aquifer water are listed in Table 5.25. The mathematical computations are in Appendix 2.

* Footnote: See Appendix 2

TABLE 5.24: AVERAGE DISSOLVED ION CONCENTRATION (MG/L) - JULY 1989

Aquifer	Mg	Ca	HCO_3	TDS
565m (1850ft) Level: 100mW - 1800mW (North Limb Fracture Zone Between Lubengele Fault and the Anticline Axis)				
HWA	24.3	37.0	125.3	262.0
FWA	22.6	40.0	103.2	272.0
FWQ	8.0	29.5	60.0	170.0
670M (2200ft) Level: 2000mN - 3000mN (Cross Anticline Axis Fault Zone)				
HWA	33.5	47.0	120.0	302.5
FWA	14.0	30.4	96.6	166.0
FWQ	7.0	20.5	91.5	125.0
810m (2650ft) Level: 100mN - 2100mN (Cross Anticline Axis Fault Zone and Fringe)				
HWA	32.6	52.9	145.7	335.7
FWA	5.0	5.0	28.5	60.0
FWQ	9.0	16.0	37.0	110.0

TABLE 5.25: ESTIMATES OF PROPORTIONS OF HANGINGWALL AQUIFER WATER IN FOOTWALL AQUIFER WATER USING AVERAGE BOREHOLE WATER ION CONCENTRATIONS

Variable	565m Level: 100mW - 1800mW	670m Level: 200mN - 3000mN	810m Level: 100mN - 2000mN
Mg	90%	30%	20%
Ca	140% (over dilution)	40%	30%
HCO_3	70%	20%	2.5%
TDS	110% (over dilution)	20%	30%
Average	100%	30%	20%

These results indicate that the proportion of Hangingwall Aquifer water in Footwall Aquifer water is highest in the upper levels of the mine and falls at deeper levels in the mine. The proportions falls from about 100% at the 565m level to about 20% on 810m level. This could be due to the depth of circulation of the Hangingwall Aquifer at depth. This critical depth may be for example ≤500m≥ below current deepest working mine level, and changes with time as the mine gets deeper and the groundwater level falls.

Based on this analysis the July 1989 mine water inflow proportions were as tabulated in Table 5.26.

TABLE 5.26: ESTIMATED AQUIFER PROPORTIONS OF KONKOLA MINE GROUNDWATER - JULY 1989

Aquifer	Number 1 Shaft Volume (m^3/d)	Number 3 Shaft Volume (m^3/d)	Mine Total Volume (m^3/d)	Aquifer Water Proportion of Mine Total
HWA	77439	68442	145881	41.3%
FWQ	178902	28489	207391	58.7%
Total	256341	96931	353272	

These figures show that as at July 1989, 41.3% and 58.7% of the total mine groundwater flow came from the Hangingwall and Footwall Quartzite Aquifers respectively. Of more significance is the fact that the proportions of each aquifer contribution differ at each shaft. At Number 3 Shaft, about 70% of the water intersected came from the Hangingwall Aquifer whilst at Number 1 Shaft the Hangingwall Aquifer contributed about 30%. Number 3 Shaft lies in the Lubengele Fault zone above which the Mine Tailings Dam is located.

Although the figures given here use average values for dissolved ion concentrations they do give an insight into the kind of mixing proportions of aquifer waters that can be expected, and serve to give a feel for the orders of magnitude involved. The theoretically ideal situation would be to sample at each location all the aquifers and use these values to estimate the proportions. The Footwall Quartzite Aquifer samples should be taken at depths lying below the depth of ciruculation of Hangingwall Aquifer water to establish a 'true' end-member value for this source.

5.6 CONCEPTUAL MODEL OF KONKOLA MINE HYDROGEOLOGICAL SETTING AND GROUNDWATER FLOW PATTERN

As clearly shown in the geological maps of both surface and underground workings, Konkola Mine is wedged between two major faults; the Lubengele in the north and the Luansobe in the south. In between these faults are the Anticline (Fold) Axis Fault which abuts into the Lubengele Fault and the Cross-Anticline (Fold) Axis Fault which link up the Lubengele and the Luansobe Fault. These faults form the main hydrological boundaries and major channels of water flow into the mine.

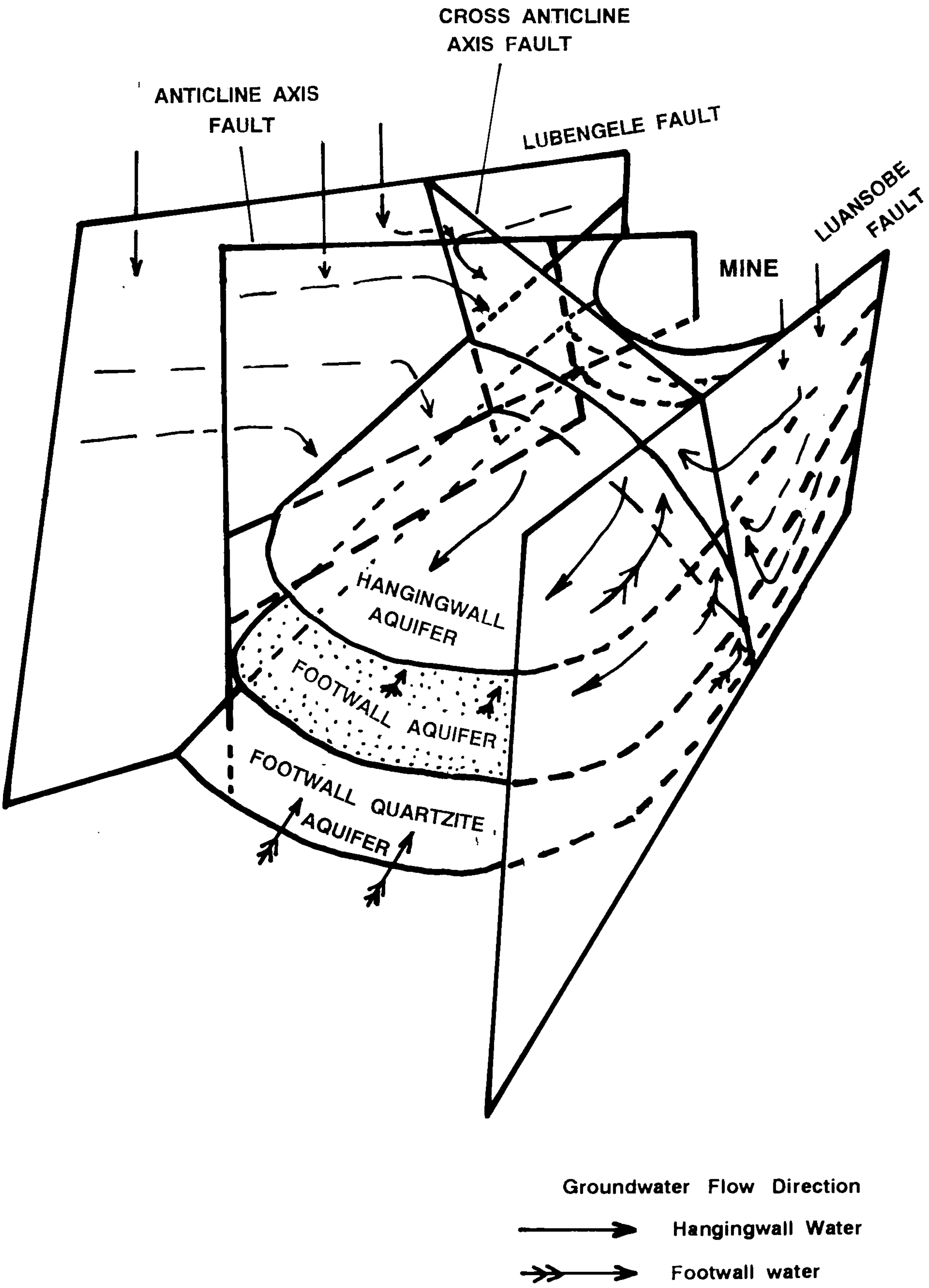
There is overwhelming evidence in the mine to show that the main mode of groundwater movement is fissure-flow. Stratigraphic control of flow through pores and vugs plays a significant part especially in the unfaulted regions of the mine.

There are two distinct bodies of water at the mine ; recent and ancient. The young (recent) waters of the Hangingwall Aquifer (cold) ingresses the mine from surface water system and flows mainly downwards. The old (ancient) waters of the Footwall (hot) originates at depth from the regional aquifer and moves upwards mainly through fissure and faults zones. These waters mix in the fault zones.

The surface waters of the Kafue River and its tributaries and the Lubengele Mine Tailings Dam, are leaking directly into the mine and recharging the Hangingwall Aquifer; mainly via the fault system. Figure 5.22 is a conceptual model of the mine hydrogeological setting.

In all, the evidence strongly suggests that a significant proportion of water pumped from the mine and discharged into the Kafue River maybe is being recycled back into the mine via the Luansobe fault. However, although we now have a feel of the order of magnitude of the proportions of Footwall Quartzite and Hangingwall Aquifers waters in the mine water inflow, the relative contributions of river water to the mine inflow has yet to be established.

Figure 5.22 : CONCEPTUAL MODEL OF KONKOLA MINE HYDROGEOLOGICAL SETTING



PART III

NUMERICAL MODELLING OF KONKOLA MINE GROUNDWATER FLOW AND CONTROL OF RECHARGE

This part of the thesis demonstrates that the conceptual model of groundwater flow developed from the knowledge gained from this research can be expressed into a numerical groundwater flow model. The numerical model can be used not only for predicting future mine groundwater discharge and water level drawdowns needed for planning mine development and ore-production but even more significant, to investigate various groundwater management options and facilitate formulation of a much needed long-term cost-effective solution to the groundwater flow problem. Only then will the economic viability of the mine be enhanced and full ore-production potential of the mine be realised. For this numerical model to develop into an invaluable tool, good representative data on hydraulic parameters of the aquifers and other hydrogeological aspects should be obtained so that with time an excellent data base can be developed, and model uncertainties reduced to insignificant levels.

This part is composed of two chapters; Chapters 6 and 7. Chapter 6 addresses the subject of numerical modelling of the Konkola Mine groundwater flow and Chapter 7 deals with the control of groundwater recharge into the mine.

CHAPTER 6

KONKOLA MINE GROUNDWATER FLOW NUMERICAL MODEL

6.1 INTRODUCTION

The conceptual model of Konkola mine hydrogeological regime (Figure 5.22) clearly demonstrates the complexity of the system. Groundwater flow is primarily controlled by the fault-fracture system, which act as the main groundwater flow channels. However, the dynamics of the groundwater flow system is yet to be clearly understood. The main hydrogeological boundaries should be identified and flow velocities established.

Historically, systematic records of underground dewatering boreholes discharges, pressure heads and surface borehole water levels have been maintained but there is no data on storage coefficient, and transmissivity. Values of hydraulic conductivity are only poorly known.

Thus in order to gain a better understanding of the groundwater flow dynamics in the mine, a numerical model was developed which took into account the three-dimensional nature of the problem. Measured values of aquifer pressure heads and discharges were input into the model to produce matching values of hydraulic conductivity. Having obtained the required representative data of hydraulic parameters, simulations of mine groundwater flow were then carried out.

The model is to serve as an invaluable tool in the development of a clear understanding of the mine groundwater flow system. It will be of great value in the formulation and evaluation of results of hydrogeological investigations that would be carried out at the mine. Pumping tests would produce the necessary data on hydraulic conductivity, transmissivity, storage coefficient and hydrogeological boundary conditions.

Through this approach it was envisaged that a sound, scientific framework would be built, upon which the formulation of a long term cost-effective groundwater management solution would be based. This would enable one to make meaningful simulations of the Konkola mine groundwater flow and predictions of mine discharge and groundwater level drawdown (Mulenga and De Freitas, 1991).

The main steps in building the model were as follows.

(i) Choice of model and modelling approach.
(ii) Representation of the mine geology, hydrology and hydrogeology.
(iii) Input parameters to be used.
(iv) Calibration of the model.
(v) Simulation of groundwater flow.
(vi) Preliminary results and model uncertainties.

6.2 Choice of Model and Modelling Approach

The simulation of any groundwater flow problem involves the solution of the fluid mass balance equation over a specified volume of ground, subject to the internal parameters of hydraulic conductivity and storage coefficient, and to boundary conditions (Fawcett et al 1984, Singh and Atkins 1984, Wang and Anderson 1982, Fetter Jr. 1980, Freeze and Cherry 1979, and Toth 1963 & 1962).

The potential flow which describes the change in flux in response to a change in potential is given by the Laplace Equation, which for

(a) Steady State Flow

$$\Delta Q = [K_x\frac{\delta^2 h}{\delta x^2} + K_y\frac{\delta^2 h}{\delta y^2} + K_z\frac{\delta^2 h}{\delta z^2}]\, dx.dy.dz = 0$$

and for

(b) Transient State Flow

$$\Delta Q = [K_x\frac{\delta^2 h}{\delta x^2} + K_y\frac{\delta^2 h}{\delta y^2} + K_z\frac{\delta^2 h}{\delta z^2}]dx.dy.dz = \nabla^2 h = (\frac{S}{Kb})\frac{\delta h}{\delta t} \neq 0$$

where:

Q	=	change in discharge
K	=	hydraulic conductivity
S	=	coefficient of aquifer storage
b	=	saturated thickness
h	=	total head
t	=	time
dx.dy.dz	=	volume of ground
∇h	=	$\left(\frac{\delta h}{\delta x}, \frac{\delta h}{\delta y}, \frac{\delta h}{\delta z}\right)$ = hydraulic gradient

Historically being in a mine, almost all the data available on ground conditions had been gathered with mineral exploration and exploitation in mind, and very little in the way of systematically gathered information for hydrogeological purposes had been obtained other than the measurement of groundwater levels in open holes that crossed the aquifers. The mine has abundant data on water levels and borehole discharge but no values on storage coefficient and hydraulic conductivity. Therefore, in light of this factor, one of the major considerations in the choice of type of model, was to go for one that could make the best use of the available data of water levels and boreholes discharge.

Measurements of groundwater levels and mine discharge have shown repeatedly that a transient groundwater flow system exists at Konkola mine. However, taken over the short term, i.e. one or two months, the situation could be approximated to steady state. Thus, in view of the paucity of information on storage at present, it was decided to start by using the model to simulate steady state flow. From this exploratory work a knowledge of the order of magnitude and global view of values and distribution of hydraulic conductivity and flow velocities operating at the mine, would be created.

Groundwater clearly flows mainly through the fracture system at Konkola mine, although some porous flow also exists. However, the distribution, number, orientation and conductivity of these fractures is beyond prediction with the quality of data presently at hand. As shown by many researchers such as Priest (1986), Samaneigo (1985) and Sharp (1970) just to mention a few, modelling of groundwater flow through fractured rock as fracture-flow requires more detailed data than has been collected at Konkola mine at present. Thus, it was decided to model the ground as if it were a continuum rather than to attempt a representation of fracture flow. Bearing in mind the large area of the mine and its great depth, the relative scales of the real situation and its model implies that representing the ground as a continuum would not significantly affect the quality of predictions obtained.

These considerations led to the selection of MODFLOW, a three-dimensional finite difference groundwater flow model developed by McDonald and Harbaugh of the United States of America Geological Survey (McDonald and Harbaugh, 1988).

The model simulates groundwater flow in three-dimensions and structurally consists of modules. The modular structure consists of the main program and a series of highly independent subroutines called modules. The modules are grouped into packages. Each package deals with a specific feature of the hydrologic system which is to be simulated. The major options that are presently available are; procedures to simulate the effects of wells, rivers, drains, evapotranspiration, recharge and general head boundaries. It has two specific methods of solving the equations which describe the flow system and these are the strongly Implicit Procedure (SIP) and the Slice-Successive Over-relaxation (SSOR) methods.

Groundwater flow within an aquifer is simulated using a block-centred finite-difference approach. Layers can be simulated as confined, unconfined, or a combination of confined and unconfined and flow can either be steady state or transient state. The program is written in FORTRAN 77 and will run without modifications on most computers that have a FORTRAN 77 compiler.

The development of the groundwater flow equation in finite-difference form follows from the application of the continuity equation which states that: the sum of all flows into and out of the cell must be equal to the rate of change in storage within the cell. Under the assumption that the density of groundwater is constant, the continuity equation expressing the balance of flow for a cell is:

$$\Sigma Q_i = SS^{\Delta h} \times \Delta V$$

where:

Q_i = Flow rate into cell (L^3t^{-1}).

SS = Specific storage in the finite-difference formulation, it is the volume of water which can be injected per unit volume of aquifer material per unit change in head (L^{-1}).

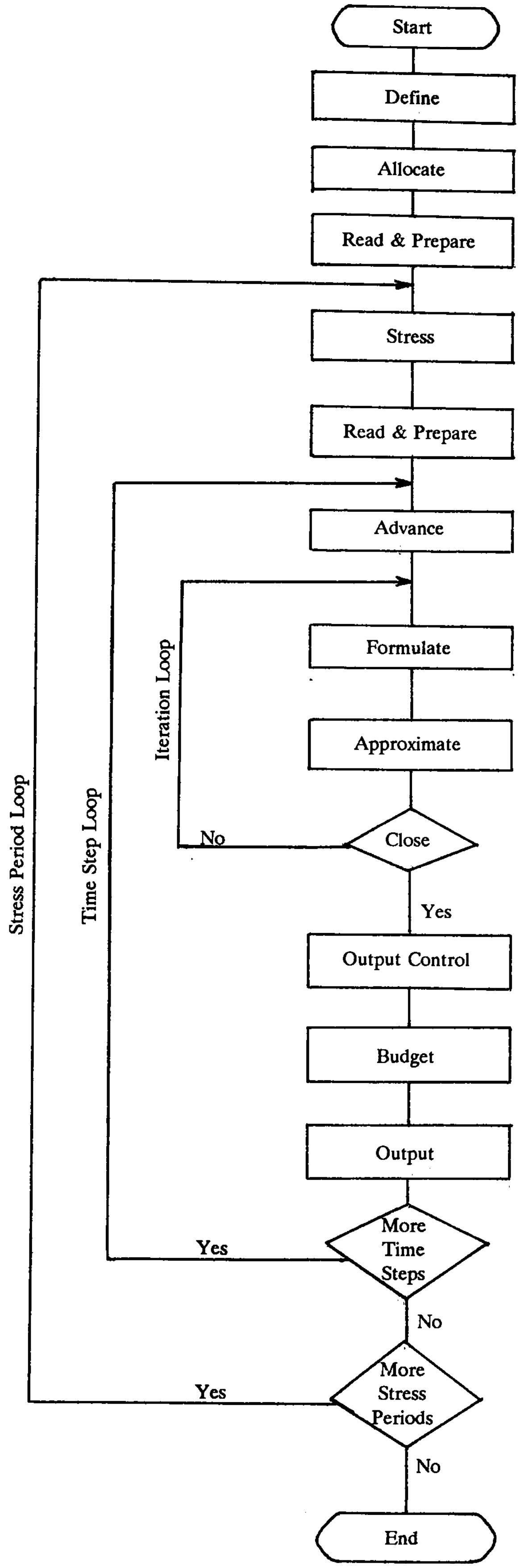
ΔV = The volume of the cell (L^3).

Δh = Change in head over a time interval of length Δt.

The term on the right hand side is equivalent to the volume of water taken into storage over a time interval Δt given a change in head of Δh.

An outline description of the program is given below. The program listing and mathematical formulations are described in the program manual to which the reader is referred.

Figure 6.1: Modflow Program Structure



DEFINE - Read data specifying number of rows, columns, layers, stress periods, and major program options.

ALLOCATE - Allocate space in the computer to store data.

READ AND PREPARE - Read data which is constant throughout the simulation. Prepare the data by performing whatever calculations can be made at this stage.

STRESS - Determine the length of a stress period and calculate terms to divide stress periods into time steps.

READ AND PREPARE - Read data which changes from one stress period to the next. Prepare the data by performing whatever calculations can be made at this stage.

ADVANCE - Calculate length of time step and set heads at beginning of a new time step equal to heads calculated for the end of the previous time step.

FORMULATE - Calculate the coefficients of the finite difference equations for each cell.

APPROXIMATE - Make one cut at approximating a solution to the system of finite difference equations.

OUTPUT CONTROL - Determine whether results should be written or saved on disk for this time step. Send signals to the BUDGET and OUTPUT procedures to indicate exactly what information should be put out.

BUDGET - Calculate terms for the overall volumetric budget and calculate and save cell-by-cell flow terms for each components of flow.

OUTPUT - Print and save heads, drawdown and overall volumetric budgets in accordance with signals from OUTPUT CONTROL procedure.

Table 6.1: Modflow Program Modules

Module (Package) Name	System Name	Description		
Basic	MODBAS	Handles those tasks that are part of the model as a whole. Among those tasks are specification of boundaries, determination of time-step length, establishment of initial conditions, and printing of results.		
Block-Centred	MODBCF	Calculates terms of finite-difference equations which represent flow within porous medium; specifically, flow from cell to cell and flow into storage.		Flow Component Packages
Well	MODWEL	Adds terms representing flow to wells to the finite-difference equations.	Stress Packages	Flow Component Packages
Recharge	MODRCH	Adds terms representing areally distributed recharge to the finite-difference equations.	Stress Packages	Flow Component Packages
River	MODRIV	Adds terms representing flow to rivers to the finite-difference equations.	Stress Packages	Flow Component Packages
Drain	MODDRN	Adds terms representing flow to drains to the finite-difference equations.	Stress Packages	Flow Component Packages
Evapotrans-piration	MODEVT	Adds terms representing ET to the finite-difference equations.	Stress Packages	Flow Component Packages
General-Head Boundaries	MODGHB	Adds terms representing general-head boundaries to the finite-difference equations.	Stress Packages	Flow Component Packages
Strongly Implicit Procedure	MODSIP	Iteratively solves the system of finite-difference equations using the Strongly Implicit Procedure		Solver Packages
Slice-Successive Overrelaxation	MODSOR	Iteratively solves the system of finite-difference equations using the Slice-Successive Overrelaxation.		Solver Packages

Figure 6.1 shows the overall program structure and Table 6.1 the modules that make up the program. The period of simulation is divided into a series of "stress periods" within which specified stress parameters are constant. Each stress period, in turn, is divided into a series of time steps. The system of finite-difference equations is used to yield the head at each node at the end of each time step. Iterative solution methods are used to solve for the heads for each time step. Thus, within a simulation, there are three nested loops: a stress-period loop, within which there is a time-step loop, which in turn contains an iteration loop.

Each rectangle in Figure 6.1 is termed a "procedure". For example prior to entering the stress loop, the program executes three procedures which pertain to the simulation as a whole. In the Define Procedure, the problem to be simulated is defined: the size of the model, the type of simulation (transient or steady state), the number of stress periods, the hydrologic options, and the solution scheme to be used are specified. In the Allocate Procedure, memory space required by the program is allocated. In the Read and Prepare Procedure, all data that are not functions of time are read. These data may include all or some of the following: boundary conditions, initial heads (starting heads), transmissivity, hydraulic conductivity, specific yield, storage coefficients, elevations of layer tops and bottoms, and parameters required by the specified solution scheme. Certain preliminary calculations are also made in this procedure to prepare data for further processing.

Within the stress period loop the first procedure is termed the stress Procedure. In this procedure the number of time steps (NSTP) in the stress period and certain information to calculate the length of each time step are read. In a second Read and Prepare Procedure, all data that pertain to a stress period, such as pumping rates and areal recharge, are read and processed. The time-step loop is then entered (Figure 6.1); in the Advance Procedure, the length of the time step is calculated and the heads for the start of the time step are initialized. The iteration loop contains the Formulate Procedure which determines the conductances and coefficients for each node as required by the finite-difference equation, and the Approximate Procedure which approximates a solution to the system of linear equations for head. Iteration proceeds until closure is achieved or until a specified maximum number of allowable iterations is reached. At the end of the iteration loop, the Output Control Procedure determines the disposition of the computed heads, budget terms, and cell-by-cell flow terms. In the Budget Procedure, budget entries are calculated and cell-by-cell flow terms are printed or recorded as explained in the manual. In the Output Procedure, heads, drawdown and the volumetric budget are printed or recorded as specified by the user.

6.3 REPRESENTATION OF MINE GEOLOGY, HYDROGEOLOGY AND HYDROLOGY

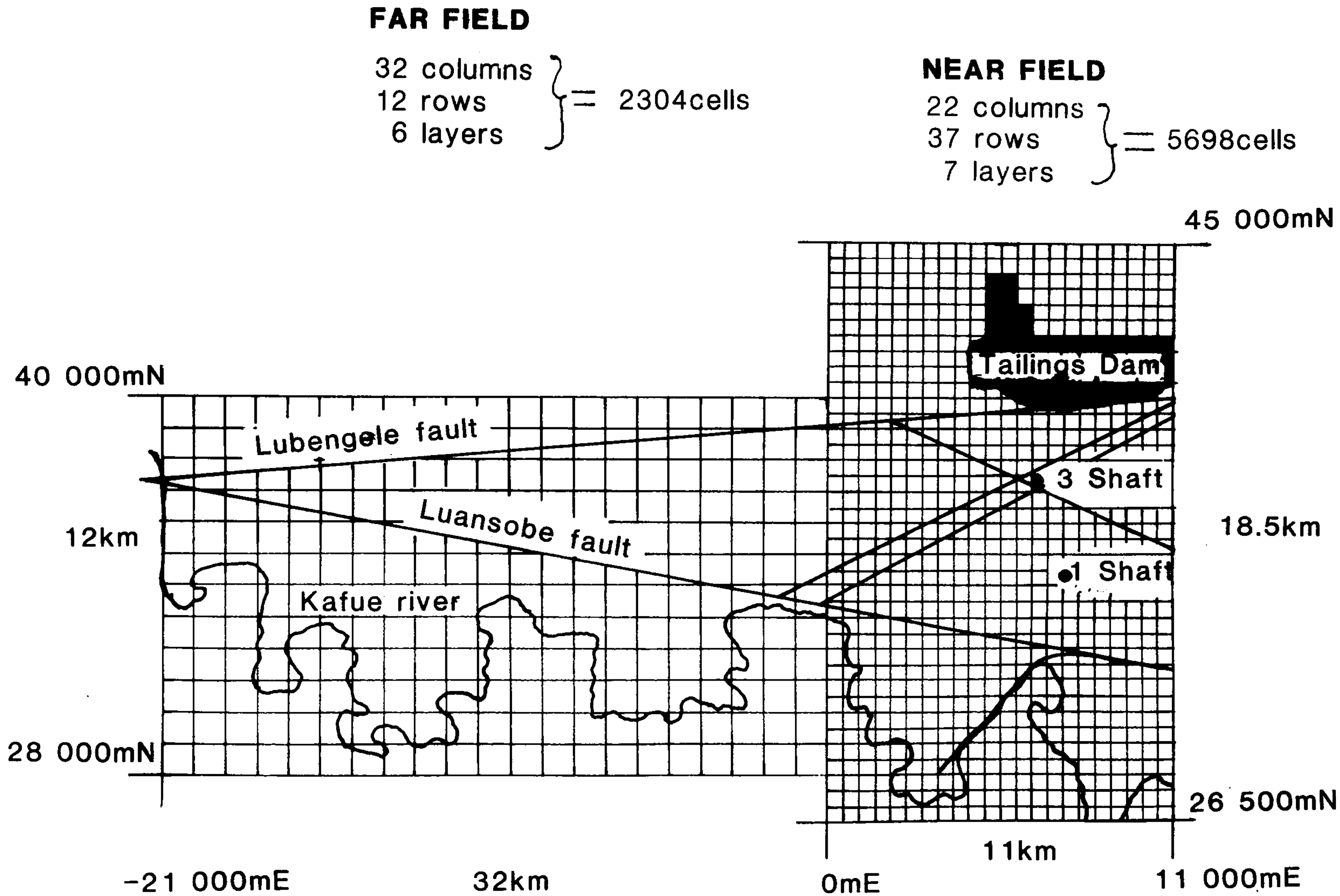
6.3.1 Modelling Scale

Simulation of the mine groundwater flow was achieved by modelling the problem on two separate scales; a Far Field (Regional) and Near Field (Local) as shown in Figure 6.2. The mine was represented in the numerical model by seven horizontal layers from surface to 1320m level based on mine level maps (Table 6.2).

TABLE 6.2: REPRESENTATION OF KONKOLA MINE LEVELS IN THE NUMERICAL MODEL LAYERS

Model Layer	Mine Level	Elevation (m)		Model Layer Thickness (m)
		Top	Bottom	
1	Surface	1320.0	1215.0	105.0
2	200m (650 ft)	1215.0	990.0	225.0
3	450m (1480 ft)	990.0	792.5	197.5
4	565m (1850 ft)	792.5	680.0	112.5
5	670m (2200 ft)	680.0	577.5	102.5
6	810m (2650 ft)	577.5	435.0	142.5
7	960m (3150 ft)	435.0	0.0	435.0

Figure 6.2 : SCHEMATIC PLAN OF THE FAR FIELD AND NEAR FIELD MODELS



(a) Far Field Model

This is on a regional scale and governed by the natural hydrogeological boundaries which surround the mine. It provides a useful basis for understanding the overall behaviour of the problem and guidance for parameter values that have to be selected.

Boundary conditions were defined as follows: The Basement Complex was taken as defining the lower impermeable boundary at the bottom of the model and the eastern boundary where it outcrops. The northern boundary was defined by the Lubengele fault and the Mine Tailings Dam. The western boundary was defined at the point where the Lubengele and Luansobe faults intersect. At this point the Kafue river is intersected and its extension south was used as part of the western boundary. The southern boundary was defined by the Kafue river and the Luansobe fault.

The eastern boundary as defined by the Basement Complex was designated a no-flow boundary except where it is crossed by the fault-fracture system. The western and southern boundaries defined by the Kafue river and the northern boundary defined by the Mine Tailings Dam, were designated constant-head recharge boundaries based on field investigation results. Everywhere else the boundaries were defined as variable-head.

The dimensions of this model are 32 km by 12 km and 1.32 km deep, with a model grid of 1 km square. It consists of 6 layers each having 32 columns, and 12 rows. Total number of model grid cells is 2304.

(b) Near Field Model

This is on a local scale and is used to simulate in more detail the groundwater levels within the mine area and provide predictions of mine discharge and water level drawdown. It covers the area of past, present and future mining activity. Boundary conditions were obtained from a combination of results from the Far Field model and observed groundwater levels.

Boundary conditions were defined as follows: The area north of the Lubengele fault and the Mine Tailings Dam defined the northern boundary and the limit of mining defined the western boundary. The southern boundary was defined by the Luansobe fault and the sections of the Kafue river which lie along the Luansobe fault. The eastern boundary was defined by the outcrop of the Basement complex.

The Kafue River and Mine Tailings Dam were designated constant-head recharge boundaries based on results obtained from field investigations. The Basement Complex was defined a no-flow boundary and everywhere else, it was variable-head boundary.

The dimensions are 11 km by 18.5 km by 1.32 km deep. There are 7 layers each having a model grid of 500 m square, 22 columns and 37 rows. The total number of model grid cells is 5698.

6.3.2 Basic Geology

The mine is located in the nose of the Kirilabombwe anticline and wedged between two major faults; the Lubengele and the Luansobe as shown in the preceding chapters. The ground between these faults is broken and blocky and, in some ways can be likened to a box of sugar cubes. This means that groundwater flows through the rock mass in the hangingwall and in the footwall close to the orebody is dominated by the transmission of water through fissures, even though the rock is porous and can consequently transmit groundwater by porous flow. Porosity tests done on the mine rock core samples showed that in general porosity does not exceed 20%. My twelve years experience of working at the mine has shown that groundwater flow in the mine is predominantly fissure controlled.

The Ore Shale formation is sandwiched between two major aquifers; those of the hangingwall and those of the footwall. The Hangingwall Aquifer is composed mainly of dolomites, limestones and interbedded dolomitic siltstones and shales. The Footwall Aquifer consists of feldspathic sandstones, conglomerates and quartzites which overlie the granites and the gneisses of the Basement Complex.

The geology of the mine was represented in the numerical model by seven horizontal layers from surface to 1320 m level based on mine level maps. The mine levels represented were as follows: 450m (1480 ft), 565m (1850 ft), 670m (2200 ft), 810m (2650 ft), 960m (3150 ft) and 1320m (4000 ft) as shown in Table 6.2. Each layer had the ability to be given different values of hydraulic conductivity. Hydraulic conductivity was assumed to decrease with depth to reflect the effect of increase in the confining stress (Hoek and Brown 1980). Superimposed upon this general structure were zones of much higher conductivity that represented the Luansobe and Lubengele faults and the zones of internal deformation associated with the Kirilabombwe Anticline Axis fault zone and the Cross-Axis fault zone.

6.3.3 Hydrology and Hydrogeology

The Kafue river controls the drainage system of the Konkola Mine area. It flows over the Hangingwall Aquifer and along faults in the vicinity of the mine. The Tailings Dam is located on the Lubengele fault which is close to and dips towards Number 3 Shaft. The dam itself is founded directly on the superficial deposits which cover bedrock as a thin veneer. These include the alluvium of the Lubengele and the Mingomba streams which occupies the bottom of the valley across which the dam is built and in which the tailings reservoir is impounded. The Lubengele and Mingomba streams still flow into the Dam.

Although attempts were made to seal the base of the reservoir using tailings, there is no guarantee that a hydrologic seal exists between the reservoir and the drift upon which it rests nor is there any guarantee that leakage beneath the tailings is prevented.

As discussed in Chapter 5, field tests of groundwater flow showed that the dam and the Kafue river are both leaking into the mine. For these reasons, the Kafue river and the Tailings dam were designated as constant-head recharge boundaries.

6.4 INPUT PARAMETERS

The mine has abundant records of underground dewatering boreholes discharge and pressure heads as well as groundwater level elevations in mineral exploration boreholes drilled from the surface. There is no data at present on storage coefficient and transmissivity. Values of hydraulic conductivity are poorly known. Borehole pressure build-up tests were conducted but only gave sparse data.

With these restrictions the approach adopted was to model a steady-state flow by inputting the measured values of discharge and head in the model, and use the model to obtain matching values of hydraulic conductivity, which could then be used for the initial simulations.

From the knowledge of the mine geology, it is clear that substantial differences in values of hydraulic parameters exists, especially in areas of faulting and in fracture zones. Groundwater modelling in a mine poses other problems too. Mining activity disturbs the ground and, unlike the situation with most models for water supply, the model for the mine must not only be provided with data on these hydraulic parameters as they exist in undisturbed ground but also with data on how these values can change as the effects of stoping are felt in the Hangingwall aquifer, and the effects of stress relief are felt in the footwall aquifer. This difficulty would be resolved by conducting field tests.

There was no data on storage coefficient and for this reason it was decided to start modelling by simulating steady-state flow which does not need storage coefficient input. For the steady-state simulation, known values of mine discharge and borehole pressure-heads were used to produce matching values of hydraulic conductivities.

Laboratory tests done on rock core samples (Mineral Resources Development 1990), gave the following results. The horizontal hydraulic conductivity ranged from 2.22×10^{-5} m/d to 7.48×10^{-2} m/d, and the vertical values ranged from 1.48×10^{-5} m/d to 20.38 m/d. These values are too low to represent the in situ values of mine hydraulic conductivity and fissure-flow that exists at Konkola.

Mine records of 1970 to 1990, and the Kafue river and Mine Tailings Dam ground level elevations were used as the data set. The data is listed in Tables 6.3 to 6.6.

Table 6.3 Surface and Underground Boreholes Water Levels Elevation 1970 to 1987

BHID	MINE COORDINATES Northing (m)	Easting (m)	BOREHOLE COLLAR ELEVATION (m)	GROUNDWATER LEVEL ELEVATION (m) June 1970	June 1975	June 1980	June 1985	June 1987
KLB24	35332.7	6547.7	1307.4	1267	1225	1229	1126	1125
KLB25	34998.4	6996.4	1310.0	1183	1182	1190	1196	1203
KLB38	38696.9	4600.5	1285.4	1214	1201	1213	1201	1216
KLB39	37734.5	5058.9	1262.7	1045	1033	1096		
KLB81	39281.9	4771.3	1302.7	1157	1129	1133	1129	1130
KLB84	36728.2	5437.6	1296.0	1208	1172	1163	1165	1173
KLB86	39596.2	2802.7	1319.5	1275	1251	1264	1276	1247
KLB87	38315.8	4306.9	1297.1	1071	996	953	910	874
KLB88	37471.8	4700.7	1275.7	1257	1248	1244	1240	
KLB91	41150.5	4086.1	1321.9	1287	1280	1309	1299	1298
KLB92	36535.9	4731.8	1276.0	1263	1247	1256	1256	1256
KLB94	33637.8	5392.9	1277.2	1264	1260	1267	1270	1270
KLB95	42079.5	3142.8	1330.5	1319	1323	1328	1322	1322
KLB96	41659.4	1895.5	1333.3	1188	1126	1209	1167	1147
KLB98	39013.9	4341.2	1289.2	1134	1108	1109	1108	1107
KLB99	37166.2	5071.2	1284.8	1121	1054	1014	969	989
KLB100	38907.7	5219.4	1295.7	1212	1135	1174	1195	1211
KLB102	39708.0	4360.6	1306.3	1153	1082	1067	1014	1011
KLB103	37910.9	4499.9	1291.0	1287	1287	1285	1285	1284
KLB104	39919.3	5230.1	1313.8	1156	1096	1081		
KLB105	38664.9	3920.0	1303.2	1299	1301	1301	1296	1292
KLB106	36882.0	4849.2	1278.5	1268	1266	1267	1264	1263
KLB107	39934.2	3069.6	1313.0	1230	1170	1170	1199	1196
SDH15	43500.0	1700.0	1343.1	1318	1310	1312		
SDH17	39615.0	6292.5	1292.1	1288	1289	1291	1289	1289
SDH18	39613.1	8986.9	1312.1	1307	1308	1309	1307	1307
SDH19	40270.0	5380.0	1317.0	1312	1312	1315	1311	1311
SDH24	34230.0	7260.0	1301.9	1182	1095			
SDH26	34896.2	6808.0	1306.1	1187	1113	1089	1090	1089
SDH40	34153.5	7079.9	1299.7	1270	1235	1241	1231	
SDH43	41630.8	6877.4	1307.4	1302	1301	1300	1300	1300
UNDERGROUND DEWATERING POINTS								
Mine Level			Level Elevation					
810m	34256.0	7230.0	530.0		778	703	640	608
"	35418.0	6554.0	"		790	708	638	612
"	36136.0	6156.0	"		710	703	665	626
670m	36685.0	5710.0	675.0			775	733	720
"	37095.0	6175.0	675.0				758	720
950m	35340.0	7322.0	385.0				509	428
450m	38522.0	6600.0	850.0		988	958	915	888
"	38682.0	6040.0	850.0		998	922	898	882
MAIN MINE SHAFTS AREAS								
Shaft	Shaft Collar Coordinates		Shaft Collar Elevation					
No. 1	35540.0	7627.0	1331.0		875	875	509	428
No. 3	37867.0	6453.0	1300.0		958	942	780	773
No. 2	44530.0	980.0	1372.0		1110	1060	1013	995

Table 6.4 Kafue River and Mine Tailing Dam Ground Level Elevation

AREA	MINE COORDINATES		RIVER/DAM GROUND LEVEL ELEVATION (m)
	Northing (m)	Easting (m)	
Kafue river	35500	-21000	1264.90
	33500	-20500	1264.80
	34000	-19000	1264.76
	32000	-18000	1264.70
	31000	-17500	1264.62
	31000	-17000	1264.58
	32500	-15000	1264.52
	30000	-15000	1264.43
	29000	-13000	1264.34
	29500	-12000	1264.22
	33000	-11000	1264.00
	33000	-7500	1263.64
	30000	-7500	1263.46
	30000	-3000	1262.90
	33500	-3000	1262.47
	33500	0	1262.11
	32000	500	1262.02
	29500	500	1261.57
	29000	1500	1261.48
	28000	3000	1261.30
	28000	4000	1261.00
	29000	2000	1260.30
	31500	5500	1258.20
	29500	6000	1255.00
	28000	7000	1254.00
	28500	8500	1253.50
	28500	9500	1253.4
Mine Tailings Dam	40000	6500	1302.0
	40000	7500	1302.0
	40000	8500	1302.0
	40000	9500	1302.0
	40000	10500	1302.0

Table 6.5: March 1988 Mine Records of Measured Discharge and Water Level Elevation

GRID CELL LOCATION			MINE COORDINATES OF CELL (m)			MEASURED DISCHARGE (m^3/d)
Layer	Row	Column	Water Level Elevation	Northing	Easting	
3	13	11	890	38750	5250	92
3	13	12	890	38750	5750	11893
3	13	13	890	38750	6250	11893
3	15	10	890	37750	4750	262
4	13	11	735	38750	5250	38336
4	13	12	735	38750	5750	30393
4	14	11	735	38250	5250	18445
5	16	10	630	37250	4750	27805
5	16	12	630	37250	5750	23194
5	16	13	630	37250	6250	46511
5	17	12	630	36750	5750	1823
5	23	16	630	33750	7750	5000
6	17	12	506	36750	5750	18000
6	18	12	506	36250	5750	15063
6	18	13	506	36250	6250	14400
6	19	13	506	35750	6250	2120
6	20	14	506	35250	6750	5000
6	22	14	506	34250	6750	1000
6	22	15	506	34250	7250	25600
6	22	16	506	34250	7750	8766
7	20	14	340	35250	6750	27772
7	20	15	340	35250	7250	38275
7	21	15	340	34750	7250	3158
					TOTAL MINE DISCHARGE =	363164

Table 6.6: Mine Average Daily Discharge From 1970 to 1990

Date	Average Discharge m^3/d
June 1970	400000
June 1975	377000
June 1980	395000
June 1985	355000
June 1987	372443
March 1988	363164
March 1989	357000
March 1990	274260
June 1990	269455

6.5 MODEL CALIBRATION

The reliability of a model can be gauged by the accuracy with which it can reproduce past events. In order to reproduce the observed groundwater behaviour, the model was calibrated by using data from June 1970 to March 1988. The data comprised total volume of mine pumping, dewatering-borehole discharges and pressure heads, values of hydraulic conductivities obtained from borehole pressure build-up tests and flow tracers, and the Kafue river and Tailings Dam ground level elevations.

The Far Field boundaries were used to generate the total heads for the Near Field boundaries.

In the Near Field model, the boundary conditions were set at values obtained from the Far Field simulation coupled with measured water levels from borehole data. 1970 records were used as a starting point. As stated earlier, simulation of faults and fracture systems was achieved by introducing high hydraulic conductivity zones. Constant-head values were set at various block nodes relevant to the Kafue river and the Tailings dam. The recharge into the system from the river and dam were simulated using the MODRIV module, and mine drainage was simulated using the MODDRN module.

In the MODRIV module, discharge (Q_r) is treated as a linear relationship between the difference in the river water evaluation (Z_r) and either the water table (Z_w) or the river bed (Z_b), depending on which is the greatest, hence:

$$Q_r = C_r(Z_r - Z_w) \text{ if } Z_w > Z_b$$

$$\text{or} \quad Q_r = C_r(Z_r - Z_b) \text{ if } Z_w < Z_b$$

The river recharge parameter C_r can be calculated from combining the river bed thickness (M) and hydraulic conductivities (K) combined with the physical characteristics of each length (L) and width (W) over the relevant grid elements, thus;

$$C_r = \frac{KLW}{M}$$

Owing to a lack of all the required information, in order to calculate the parameters C_r, a degree of experimentation was used in finding suitable values. The eventual conclusion was to set the recharge parameters to 10,000 for those recharge areas situated in fault and fracture zones. This is not unreasonable given the Local Field Model grid length of 500m with an estimated river width of 20m, a river bed thickness of 0.2m and hydraulic conductivity, of 0.1 m/d. In the case of the Tailings Dam the width is much larger but this would be offset against an increased bed thickness.

The MODWEL module was used in the initial calibration of the model to provide the means through which appropriate values of hydraulic conductivity could be estimated. The discharges from the mine were used as input data to the problem so as to obtain matching values of hydraulic conductivity for steady state conditions. In other words, the model was forced to produce the measured discharge in order to give a feel of the appropriate values of hydraulic conductivity, that would produce a groundwater level map similar to that produced from measured borehole water level data. These values of hydraulic conductivity were then used in the subsequent simulations.

Hydraulic conductivity values obtained ranged from 0.125 m/d in unfractured ground to 0.875 m/d in fractured ground. The variation with depth was very little. In the unfractured ground, the value changed from 0.125 m/d in the upper levels to 0.10 m/d below the 810m mine level. In the fractured levels to 0.70 m/d below the 810m mine level.

These absolute values for hydraulic conductivity appear at first sight to be somewhat low for the type of problem under consideration. This is because the volume of fractures through which the water is transmitted is substantially smaller than the volume of rock in which they are situated, and because a "continuum" model is being used in which parameter values must apply to entire model blocks. Therefore, although the individual fractures have high conductivities when these are averaged over the volumes occupied by a model block, the resulting value to ascribe to fracture conductivity is substantially smaller.

In the MODDRN module, the discharge was calculated as follows. Mine dewatering drives were represented as drains in the model where a linear relationship between discharge (Q_d) and the height of the water table (Z_w) above the drain's elevation (Z_d) is expressed in the following form:

$$Q_d = C_d(Z_w - Z_d)$$

Where C_d is termed the drainage parameter. Its value is related to a combination of the effects of the hydraulic conductivity of the rock formation in which the drain is situated coupled with a structure of the drain itself (i.e. the size and number of entry points for water to enter the system),

Drainage parameters were set at each of the discharge locations. Their values were then tuned against the known discharge values. The results of this calibration are given in Table 6.7 in Section 6.6.

As stated earlier, the initial calibration was done at the Far Field scale and then followed by the Local Field Scale. The regional simulations were used to investigate the important major elements of the groundwater flow problem on the mine.

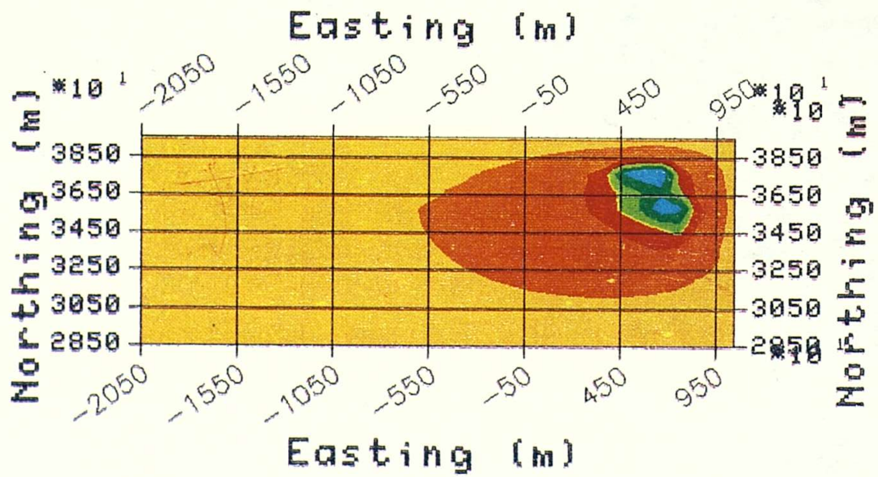
Boundary conditions for the Far Field scale were as follows. Initially all the boundaries were set at constant head and later modified to represent conditions appropriate to the mine. These were as follows. The eastern boundary was set as no-flow boundary in the area where the Basement Complex Granite outcrops and everywhere else, it was set as flow-boundary. The northern, southern and western boundaries were set as flow-boundaries. In the areas where the aquifers are in direct contact with the major surface water features such as the Kafue river and Mine Tailings Dam, the areas were designated constant-head zones. Everywhere else it was all variable-head.

Firstly, isotropic homogeneous conditions were modelled and then changed to anisotropic heterogeneous conditions.

Figure 6.3 shows the effect of mine discharges in an isotropic and homogeneous unconfined aquifer with constant-head boundaries where the constant-head boundaries are held at 1280m total head. The value of hydraulic conductivity was set at 0.2 m/d as obtained from earlier work done at the mine in the seventies on groundwater flow rate using fluorescein dye applied to a borehole. This result shows the beginnings of a formation of cone of dewatering depression.

Figure 6.4a and 6.4b illustrate the situation when the same boundary conditions as in Figure 6.3 apply except that in this case the aquifer hydraulic conductivities have been modified to represent the presence of the various fault and fracture systems (Figure 6.4a). The result as illustrated in Figure 4b shows a noticeable reduction of the extent of the cone of depression due to the effective means of transporting the water along the fracture and fault zones.

Figure 6.3 : REGIONAL MODEL SCALE : SIMULATION ONE
- Isotropic Homogeneous Conditions



Groundwater
level elevation (m)

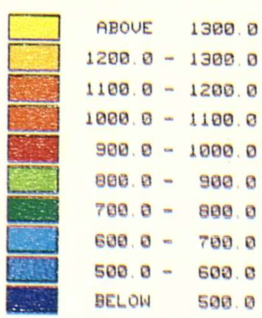
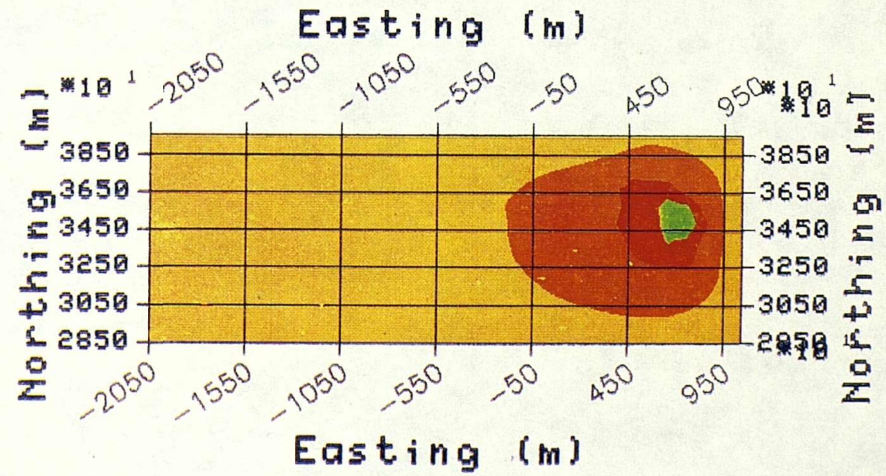


Figure 6.4 : REGIONAL MODEL SCALE : SIMULATION TWO - Anisotropic
Heterogeneous Conditions (fault and fracture systems included)

(b) Map of Groundwater Levels



(a) Map of Hydraulic Conductivity

K=m/d

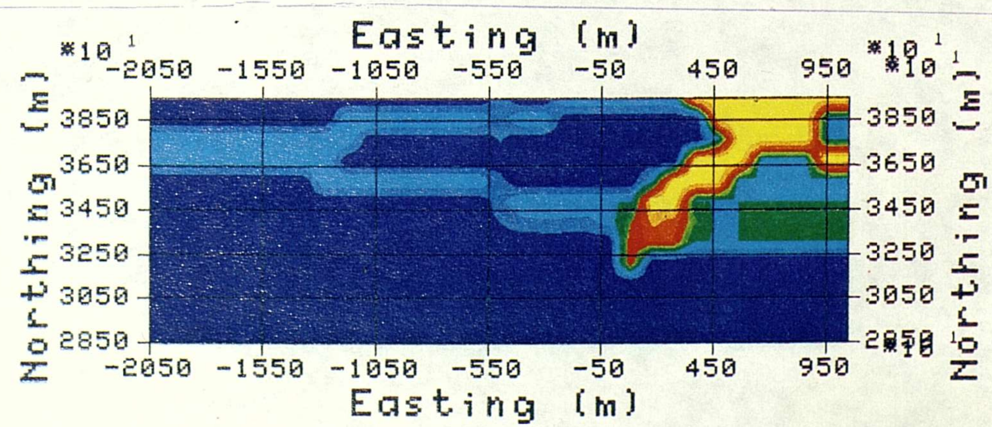
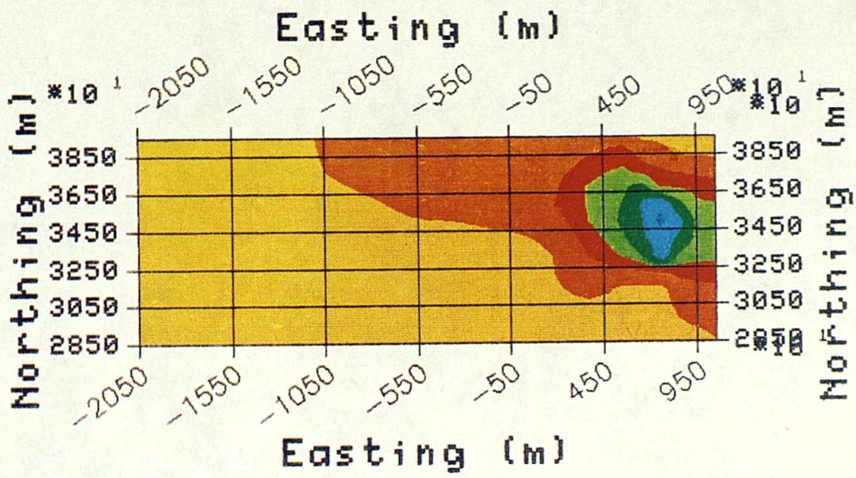


Figure 6.5 shows the effects of boundary conditions which are appropriate to those around the mine.

Figure 6.6a shows the representation of hydraulic conductivity at the Local Field scale which produced a groundwater level map (Figure 6.6b) that matched one produced from measured borehole data (Figure 6.7a). Cones of dewatering depressions as produced from both measured and simulated groundwater levels are shown in Figures 6.7b and 6.6c respectively.

Figure 6.5 : REGIONAL MODEL SCALE : SIMULATION THREE
- Anisotropic heterogeneous conditions with Kafue River and Mine Tailings Dam waters leaking into the mine



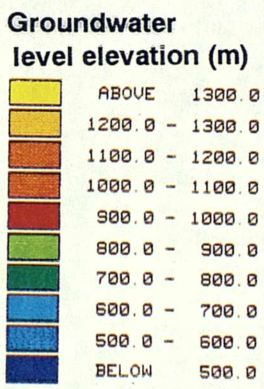


Figure 6.4b : (Simulation Two)

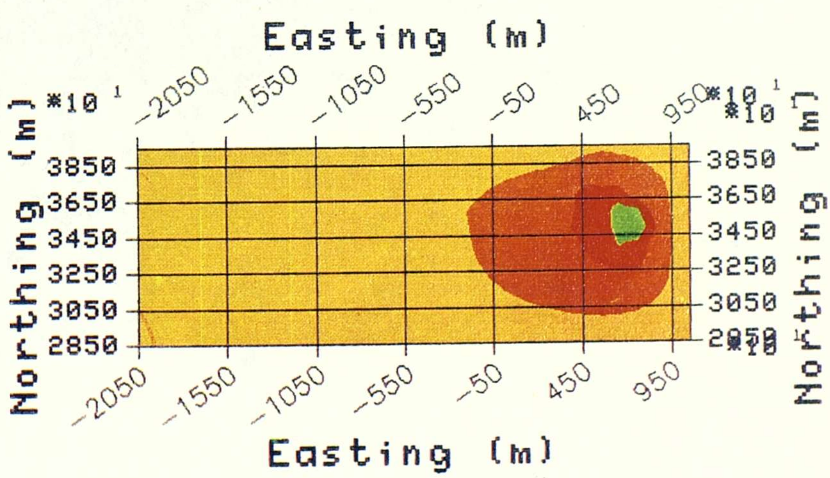


Figure 6.6a : SIMULATION OF REPRESENTATIVE HYDRAULIC CONDUCTIVITY AT THE LOCAL FIELD SCALE

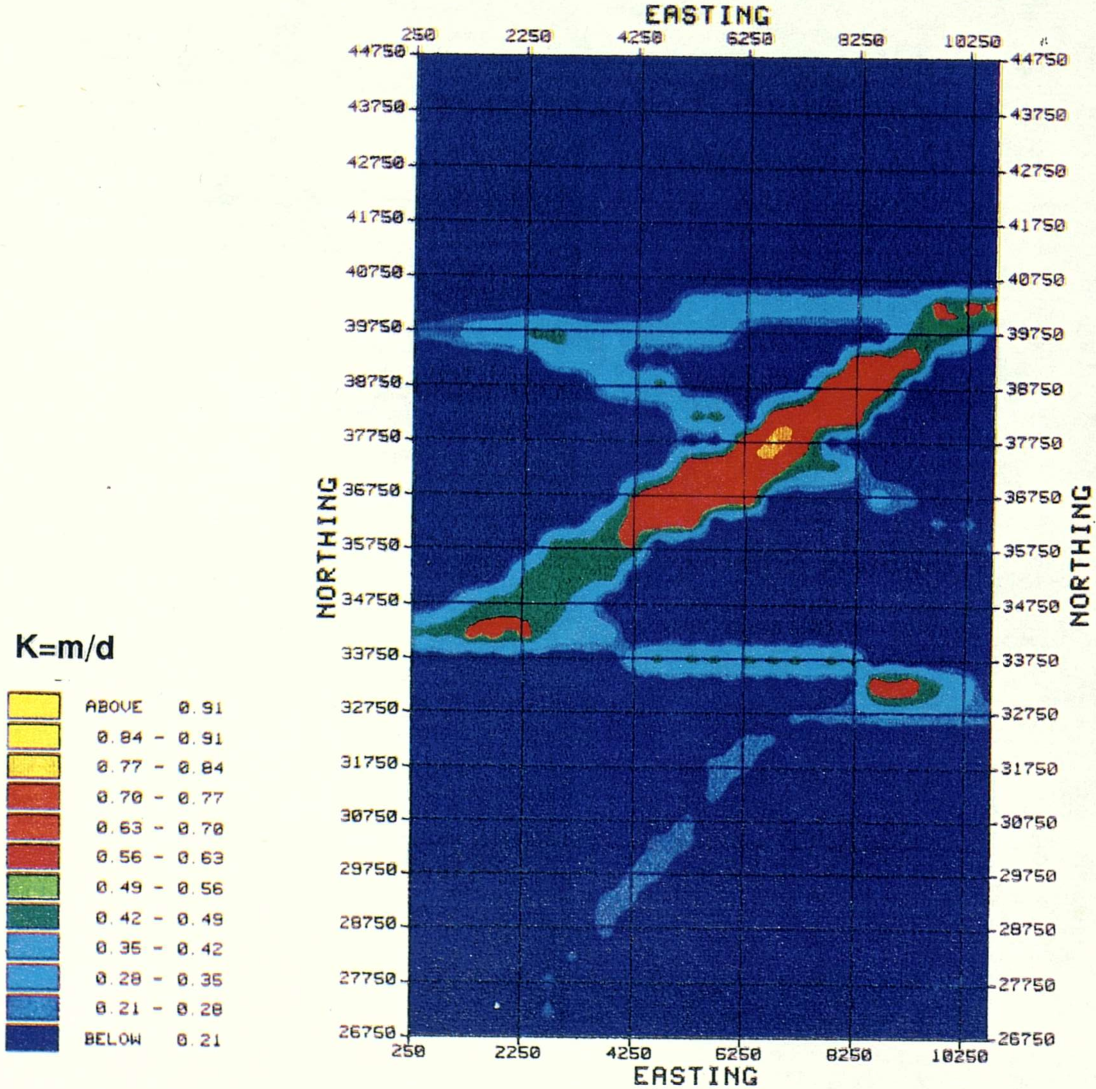


Figure 6.6b : MAP OF SIMULATED GROUNDWATER CONTOURS FOR MARCH 1988

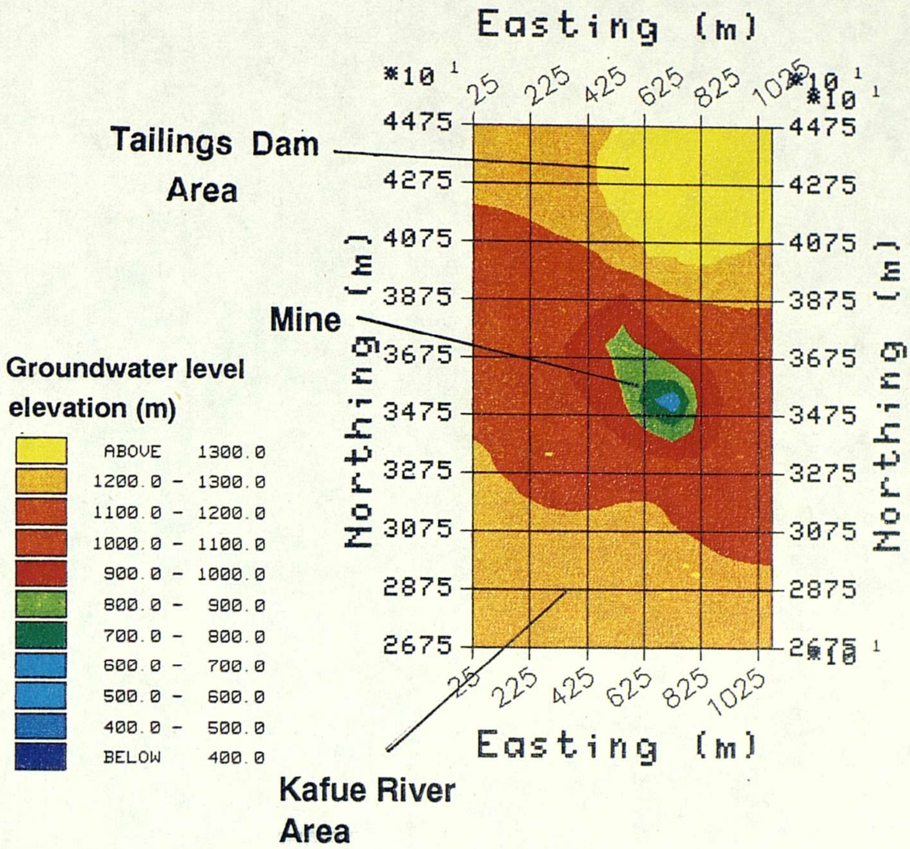


Figure 6.7a : MAP OF MEASURED GROUNDWATER CONTOURS FOR MARCH 1988

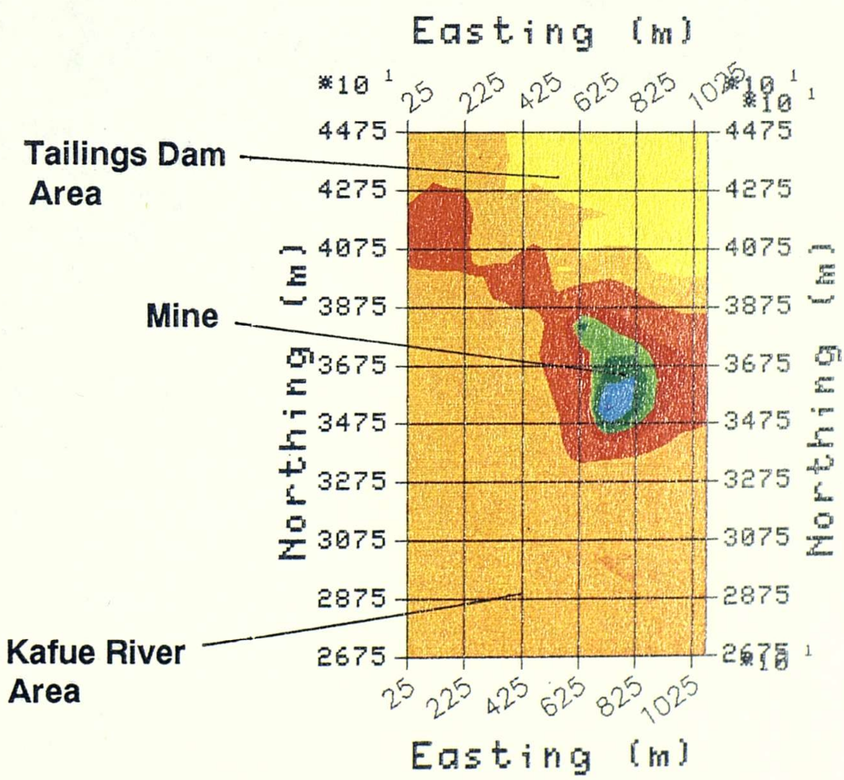


Figure 6.6c : THREE - DIMENSIONAL REPRESENTATION OF SIMULATED WATER LEVELS (MARCH 1988)

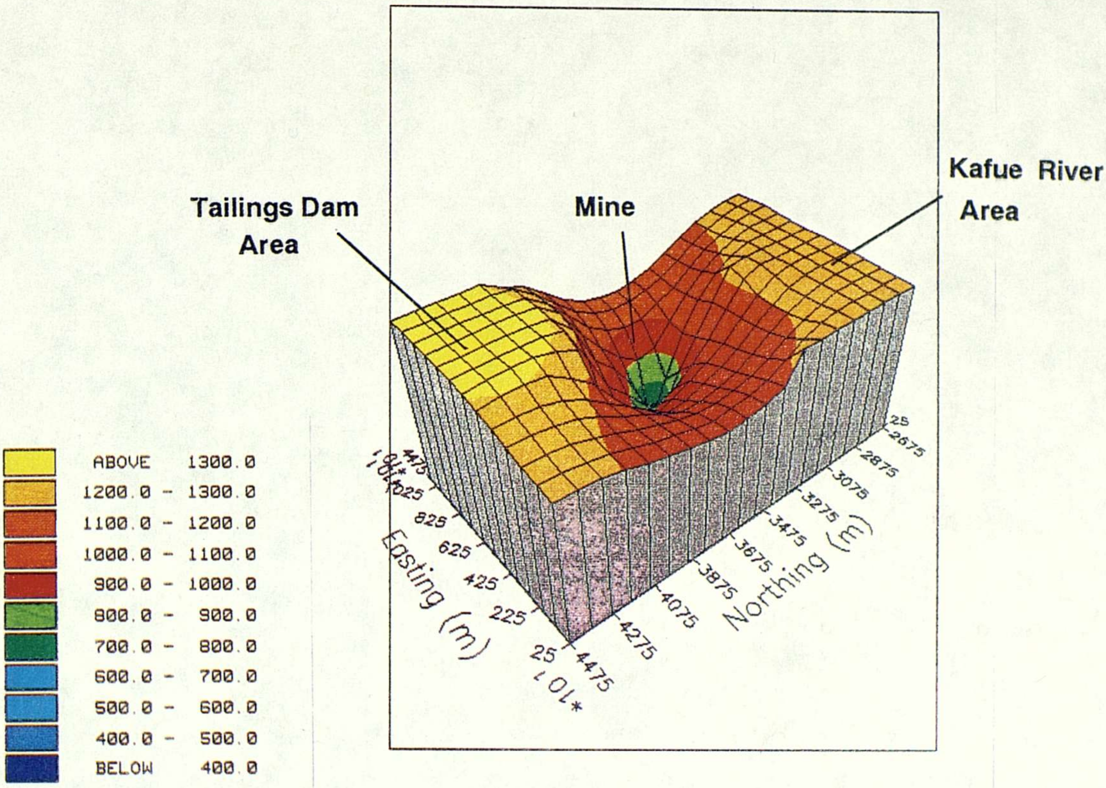
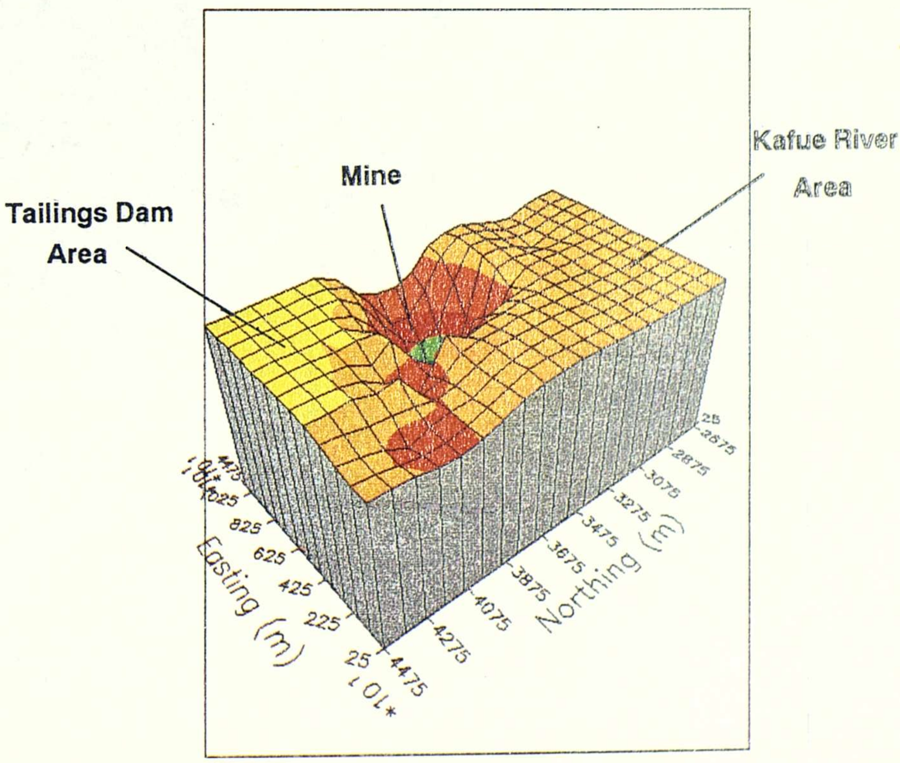


Figure 6.7b : THREE - DIMENSIONAL REPRESENTATION OF MEASURED WATER LEVELS (MARCH 1988)



6.6 RESULTS

The model has been able to reproduce the shape of groundwater level map and of cone of depression as shown by comparing Figure 6.7a with 6.7b constructed using borehole water levels around the mine. The importance of the Luansobe and Lubengele faults and their associated fracture zones, as the main channel ways of groundwater flow, and of the Kafue river and Tailings Dam as the major surface recharge sources, is well demonstrated because the shape and magnitude of the cone of depression could not be reproduced by the model until these sources of recharge had been introduced, as shown in Figures 6.6c and 6.7b.

The model predicted the lowest water level of 521m. The observed lowest water level was 428m. This is a discrepancy of 21.7%.

Table 6.7 shows that there is good agreement between the model simulation of total mine discharge and the real situation. The model has over-estimated the total mine discharge by 6.0%, which given the limited data available, is a very good prediction. However, it should be noted that there is a wide variation in degrees of accuracy attained at individual block nodes. 87% of the total points lay in the -16% to +64% discrepancy range between measured and simulated discharge. This clearly shows that representative field data is essential if meaningful simulations are to be achieved at a scale equivalent to the ground represented by either a single model block or small groups of model blocks.

Table 6.7: Comparison between Measured Discharge at Dewatering Locations in the Mine and the Discharge predicted by simulation at grid locations where these dewatering points were sited

GRID CELL LOCATION			DISCHARGE (m^3/d)		
Layer	Row	Column	Measured	Simulated	Percent Discrepancy (%)
3	13	11	92	381	+ 314.1
3	13	12	216	354	+ 63.9
3	13	13	11893	14284	+ 20.1
3	15	10	262	402	+ 53.4
4	13	11	38336	38521	+ 0.5
4	13	12	30393	31785	+ 4.6
4	14	11	18445	17765	- 3.7
5	16	10	27805	25622	- 7.9
5	16	12	23194	22964	- 1.0
5	16	13	46511	45326	- 2.6
5	17	12	1823	2203	+ 20.9
5	23	16	5000	6090	+ 21.8
6	17	12	18000	21986	+ 22.1
6	18	12	15063	18394	+ 16.2
6	19	13	2121	4790	+ 125.8
6	20	14	5000	4220	- 15.6
6	22	14	1000	3299	+ 229.9
6	22	15	25600	27936	+ 9.1
6	22	16	8766	12201	+ 39.2
7	20	14	27772	28789	+ 3.7
7	20	15	38275	36873	- 3.7
7	21	15	3158	3920	+ 24.1
TOTAL MINE DISCHARGE =			**363164**	**384880**	**+ 6.0**

6.7 MODEL UNCERTAINTIES

Although the model is proving a great help in building a better understanding of the Mine groundwater flow problem, at this stage it can only simulate steady-state conditions and thus only give a broad picture. In order to make accurate simulations, transient-state conditions must be simulated. As stated earlier, the data required to simulate transient-state conditions is presently not available.

The broad picture obtained has been of great value in understanding the problem. The simulations have shown that boundaries have an over-riding influence on the hydrogeology of the mine and very little field data is available. Hydraulic properties such as hydraulic conductivity, transmissivity and storage are absolutely crucial, but there are no adequate field values for these parameters to cover the whole area. To remove these uncertainties, field investigations to obtain representative values of hydraulic parameters would have to be carried out.

6.8 CONCLUSIONS

The model has greatly helped to enhance the current understanding of the Konkola Mine groundwater flow problem, and highlighted the problems involved in understanding groundwater flow in a mining environment. A knowledge of the basic geology of the mine and the region in which the mine is located has proved to be of great value in creating the numerical model, especially when hydrogeological data is sparse and values have to be assumed. The insertion of zones of high conductivity had a profound effect upon the response of the model and their incorporation into the model was one of the most important decisions that was made in order to match water levels to mine discharge.

Calibration runs that link hydraulic gradients to mine discharge provide an indication of the values for hydraulic conductivity but do not reveal the flows in any particular fissure. The model serves to compliment field studies and will therefore never be a substitute for ground investigation and monitoring. It is quite clear that a numerical model for a mine, once created, has to be supported by a parallel programme of field studies during the life of the mine in order that the time dependent values for hydraulic conductivity, storage coefficient, discharge and boundary conditions may be updated as the relaxation of ground around the mine evolves.

CHAPTER 7

THE CONTROL OF GROUNDWATER

7.1 INTRODUCTION

Groundwater inflow into a mine is traditionally controlled through methods of groundwater exclusion, abstraction, or a combination of exclusion and abstraction (Whittaker and Frith 1990 and Loofbourow 1973).

Groundwater exclusion involves grouting of major flow routes to reduce hydraulic conductivity and make the rock less permeable to flow or sealing off the major flow zones. Abstraction or removal is achieved by dewatering of the aquifers and pumping the water to surface. This is usually achieved by drilling boreholes from surface into the aquifers or as normally is the case in underground mines, by dewatering - drilling from underground sites to lower the groundwater level.

The current practice of groundwater control at Konkola Mine, is a combination of dewatering-drilling of the aquifers from underground locations and grouting of mine headings to facilitate safe mining.

In this Chapter various groundwater control by water exclusion options are investigated using the numerical model described in Chapter 6 with a view to recommend the most long term cost-effective option within the limits of the knowledge of the groundwater flow regime gained from this research and available hydrogeological data. In all the options the basic assumption is that the present groundwater control practices at Konkola Mine continue, the only change being that a means of reducing groundwater flow by exclusion, into the mine is added to the overall strategy.

7.2 NUMERICAL MODEL GROUNDWATER CONTROL SIMULATIONS

Three main groundwater control options were considered.

(i) The first option involved simulating the effect of stopping leakage of the Mine Tailings Dam into the mine underground workings, by draining the existing dam and identifying a new location for continued disposal of tailings, where leakage of water into the mine would not occur.

(ii) The second simulation was to investigate the effect of stopping or significantly reducing leakage of the Kafue River waters into the mine underground workings. This would be achieved by either a system of grouting or diverting the river depending on which ever method would prove cost-effective.

(iii) The third option was to investigate the effect of reducing hydraulic conductivity of the Luansobe fault by grouting so that it ceases to be a major groundwater flow channel for mine water inflow. The same was not done for the Lubengele fault because there is a dam over it.

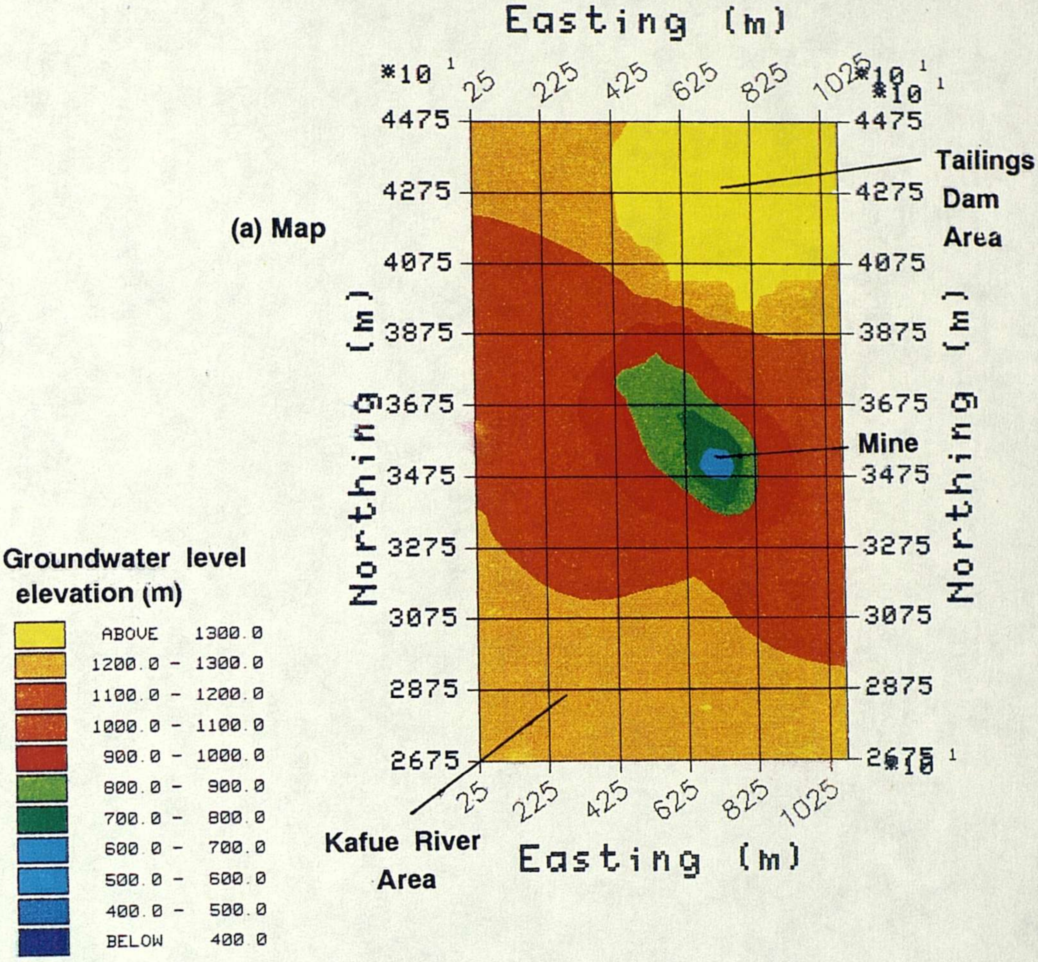
In all these simulations, boundary conditions of the mine hydrogeological regime were kept constant throughout. All boundaries were flow-boundaries with the exception of part of the eastern boundary where the Basement Complex granite outcrops, which was designated a no-flow boundary (see Chapter 6: Section 6.3).

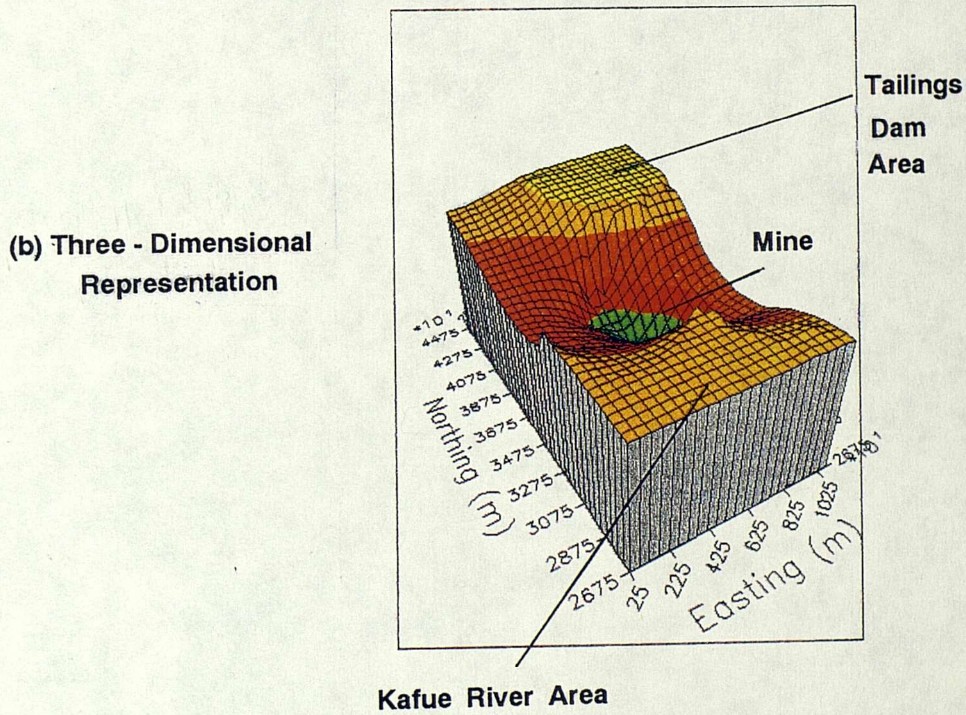
Grouting was simulated by reducing the values of hydraulic conductivity in the areas of interest. Stopping of surface-water leakage into the mine aquifers was achieved by changing the constant-head nodes in the model to variable-head nodes and reducing the level of head.

All the simulations were done using the steady-state conditions, for the reasons already stated in Chapter 6: Section 6.2. It should be remembered that the mine dewatering is simulated by gravitational drainage via the Drainage Module (MODDRN), thus permitting drawdown under gravitational drainage to be compared with discharge created by that drainage. This relationship between the gravitational flow to the mine and the groundwater levels resulting from it is critical to the valuation of possible options for groundwater control. The data used in the simulation was that of March 1988.

As shown in Chapter 6: Section 6.6, the model simulation of the March 1988 setting, gave a figure of 384880 m^3/d as total volume of water pumped from the mine during that month, and the lowest groundwater level elevation being 570m. Groundwater level contour map and the three-dimensional representation are shown in Figures 7.1a and 7.1b. These figures show a cone of dewatering depression elongated in the northwest-southeast direction, with highest groundwater levels in the northern part of the mine where the Mine Tailings Dam is located and the southern part of the mine where the Kafue River crosses the mine aquifers. The lowest groundwater levels were in the Number 1 Shaft area (570m). In the Mine Tailings Dam area, the groundwater level elevation was in the range of 1300m to 1320m and in the Kafue River area 1235m - 1270m range.

Figure 7.1a - 7.1b : MAP AND THREE - DIMENSIONAL REPRESENTATION OF SIMULATED GROUNDWATER LEVEL CONTOURS FOR MARCH 1988





7.2.1. Simulation One

Figures 7.2a and 7.2b show the effect of stopping leakage of Mine Tailings Dam water into the mine.

The results show that there is a very significant drop in both mine discharge and groundwater level elevation. The total mine discharge falls from 384880 m^3/d to 139560 m^3/d which represents a reduction in mine discharge of 63.7%. The groundwater level elevation in the Mine Tailings Dam and adjacent areas, falls from a high of about 1300m to a low of about 600m. The lowest groundwater level elevation achieved was 498m. The cone of dewatering depression broadens and flattens in the northerly direction as it is no longer constrained by the Tailings Dam. The consequence of this is that for the first time ever, all the Kirilabombwe North Orebody (Number 3 Shaft) and over 90% of currently mineable Kirilabombwe South Orebody (Number 1 Shaft), becomes easily dewatered.

*(Note)

This simulation employs the Drain Module which adjusts heads to balance out flow for a given conductivity. The Head for the Mine Tailings Dam required to provide the measured mine discharge has to recharge the Lubengele fault across which it sits. Thus inflow from the fault, whether it is from the tailings dam or from the ground to the north, is combined as one effect. Thus 'removing' the tailings dam also 'removes' water from the Lubengele fault whatever its source from the north. Lack of field evidence means this requires further study.

Figure 7.1a

Figure 7.2a : Map of simulating the effect of stopping leakage of Mine Tailings Dam water into the mine workings on groundwater level

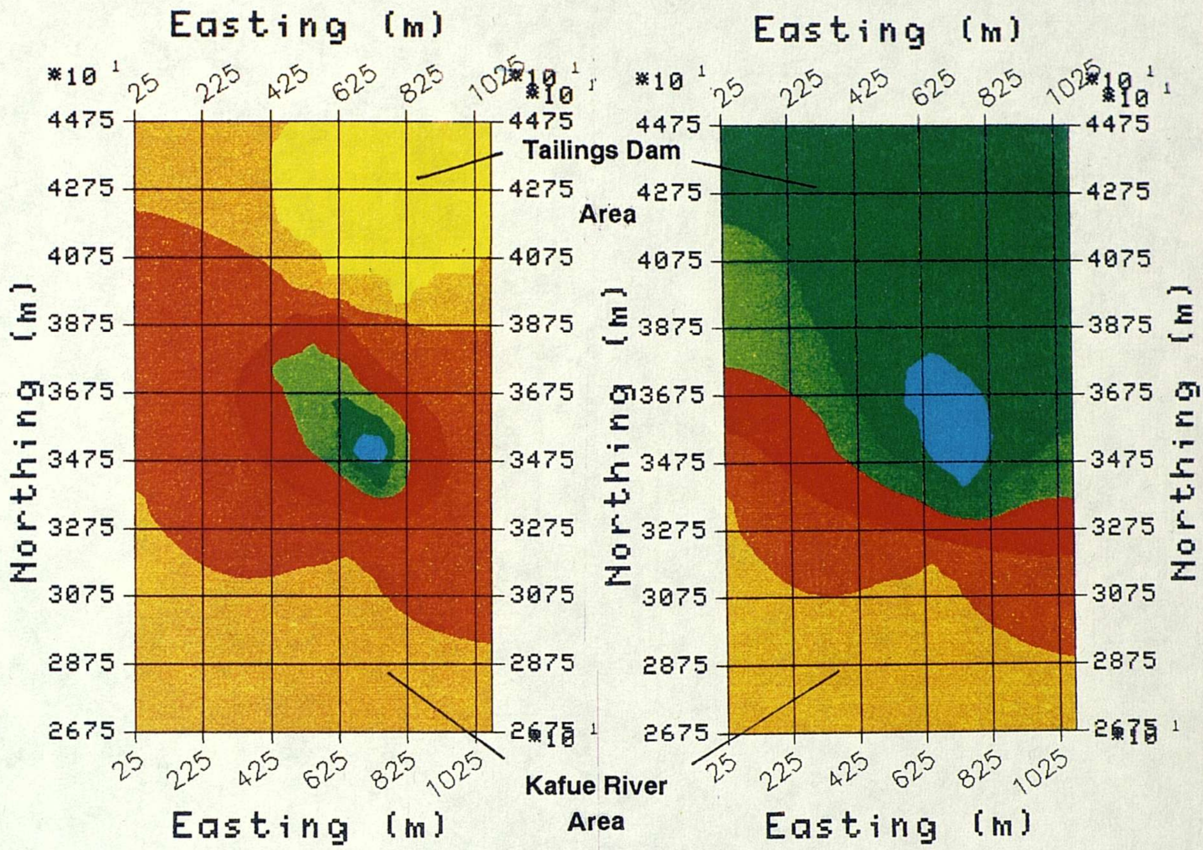
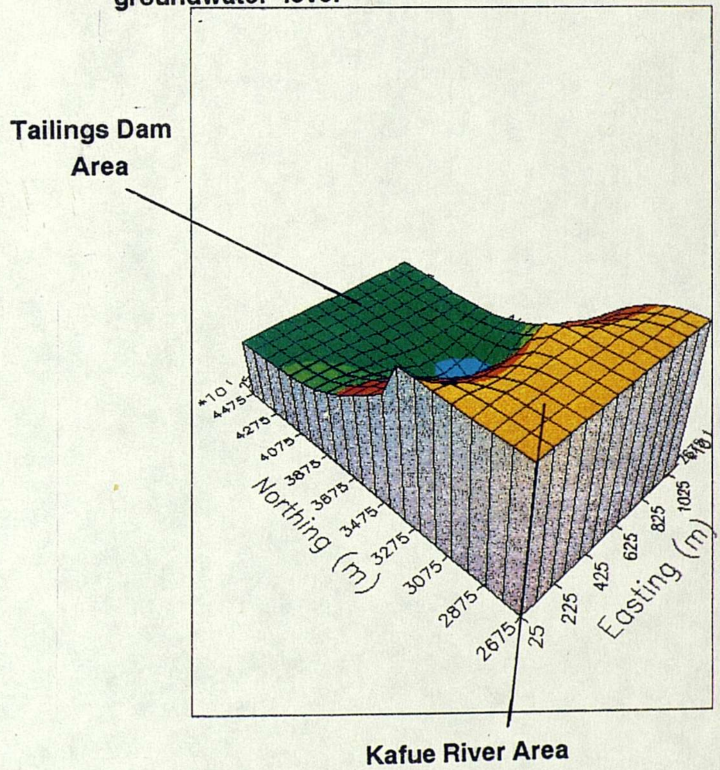
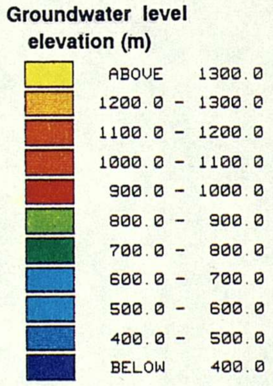


Figure 7.2b : Three - Dimensional Representation of simulating the effect of stopping leakage of Mine Tailings Dam water into the mine workings on groundwater level





7.2.2 Simulation Two

Figures 7.3a and 7.3b show the effect of stopping leakage of the Kafue River water into the mine underground workings by grouting or rechannelling the river. The Tailings Dam is still in place; leaking into the mine and the Lubengele and Luansobe faults and associated fracture systems remaining the major groundwater flow routes in the mine area.

The results show a fall in mine discharge from 384880 m^3/d to 289900 m^3/d; a reduction of 24.7%. The lowest water level dropped from to 570m to 5.21m. The cone of dewatering depression broadens and flattens in the Kafue River area, tending to extend in the southern area. However, although there is a significant lowering of groundwater levels, the greater part of the areas affected by this measure lie outside the orebody zones and thus would contribute very little to facilitating easy dewatering of the mine.

Figure 7.1a

Figure 7.3a : Map of simulating the effect of stopping leakage of the Kafue River water into the mine workings on groundwater level

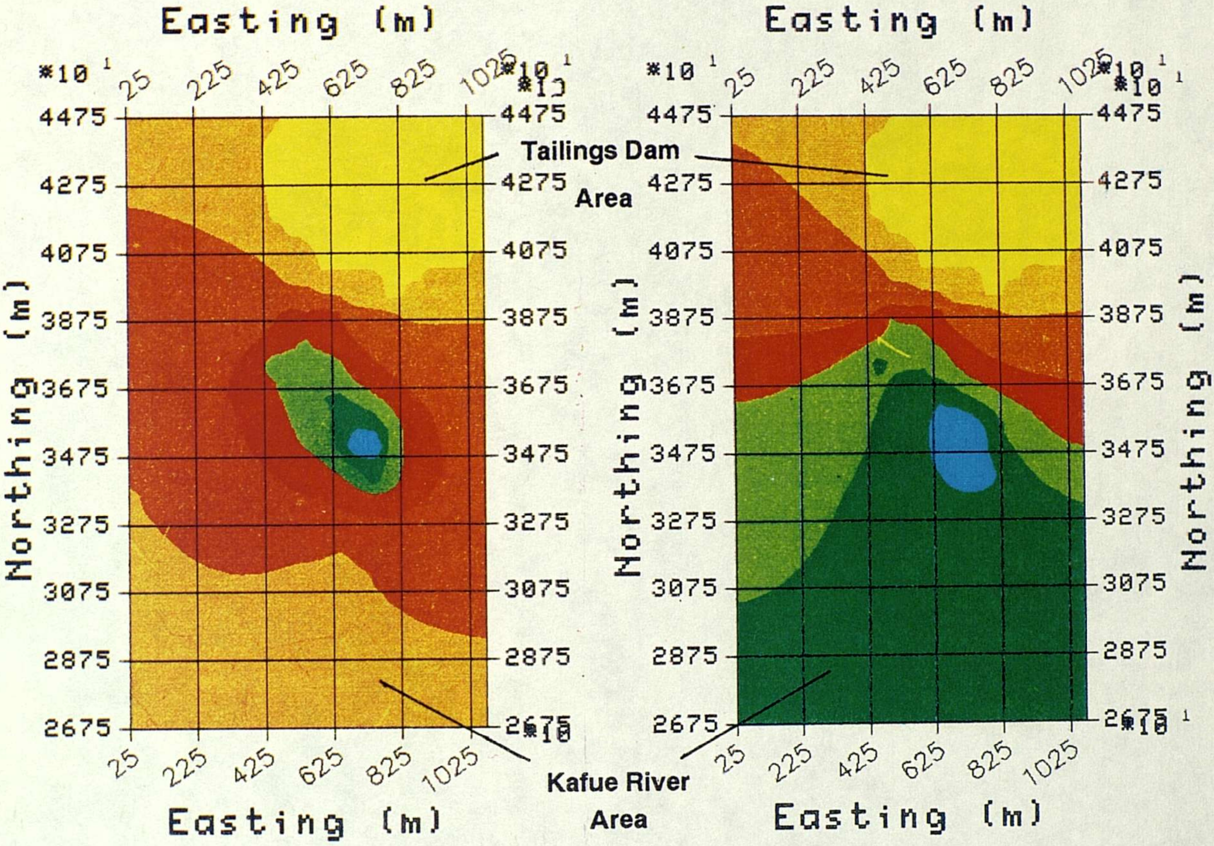
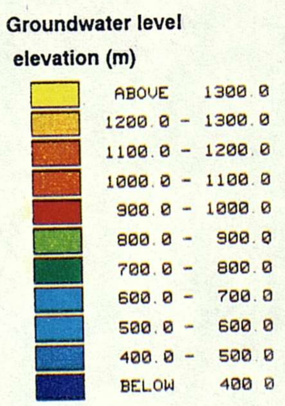
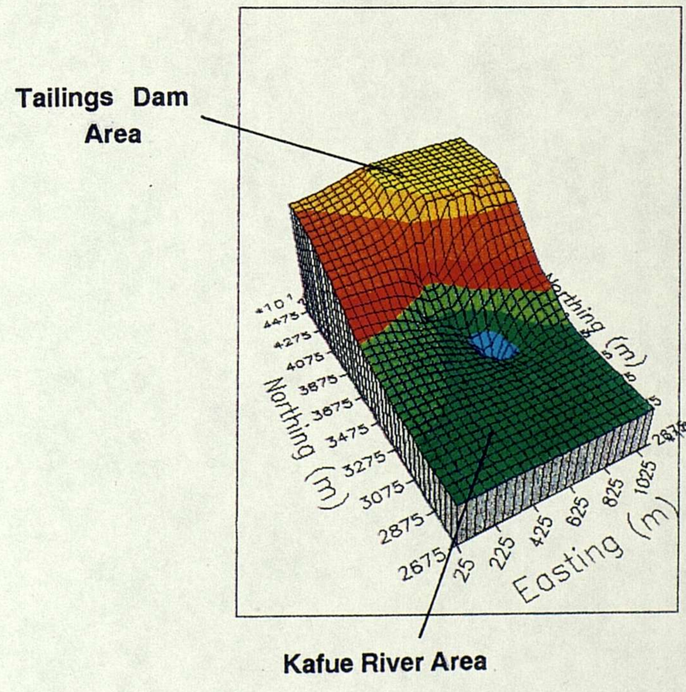


Figure 7.3b : Three - Dimensional Representation of simulating the effect of stopping leakage of the Kafue River water into the mine workings on groundwater level





7.2.3 Simulation Three

Figures 7.4a and 7.4b show the effect of grouting the Luansobe fault only. The Mine Tailings Dam and Kafue River waters still leaking into the mine. Grouting of the fault was simulated by reducing the hydraulic conductivity to the same values as those used to simulate the surrounding rock. The values of hydraulic conductivity in the Luansobe fault zone were reduced from 0.88 m/d to 0.13 m/d.

The results show a fall in mine discharge from 384880 m^3/d to 373230 m^3/d; representing a reduction of 3% in mine pumping. There was very little change in the shape of the cone of dewatering depression and the groundwater level elevation.

Figure 7.4a : Map of simulating the effect of grouting the Luansobe Fault on groundwater level within the mine area

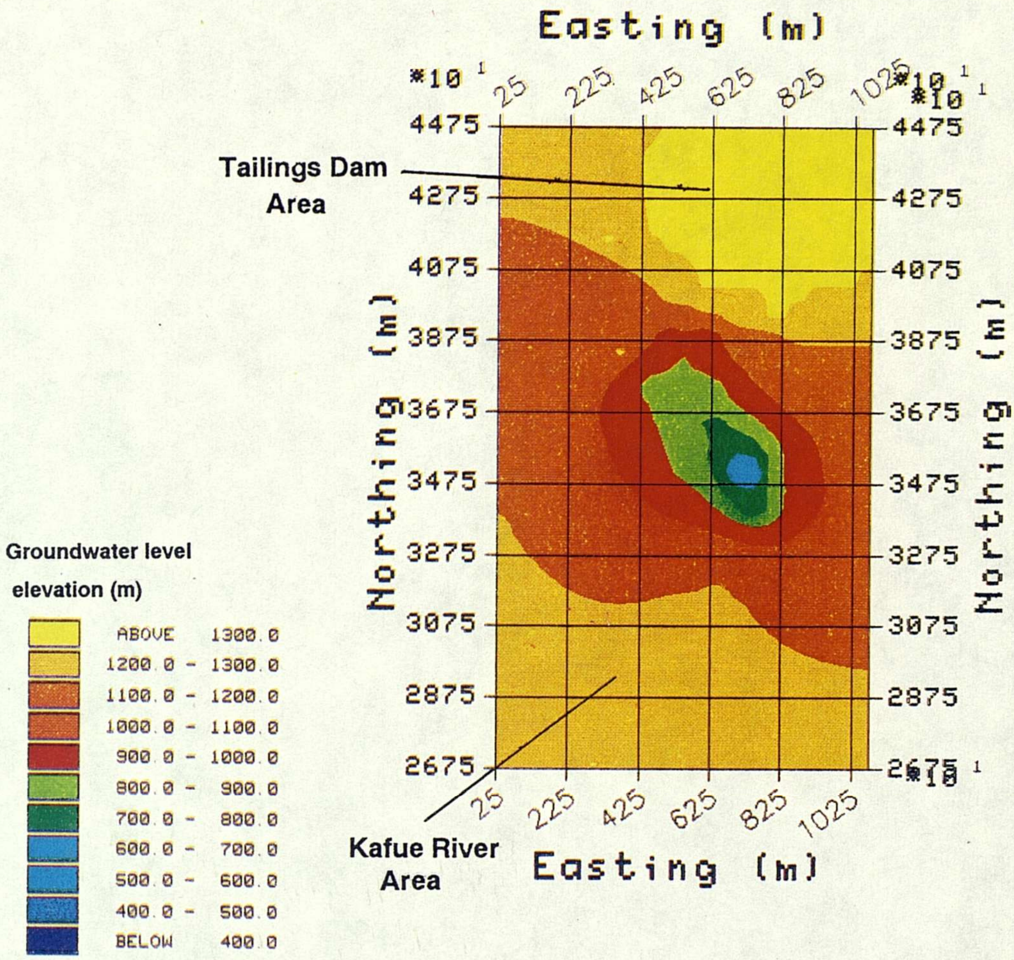
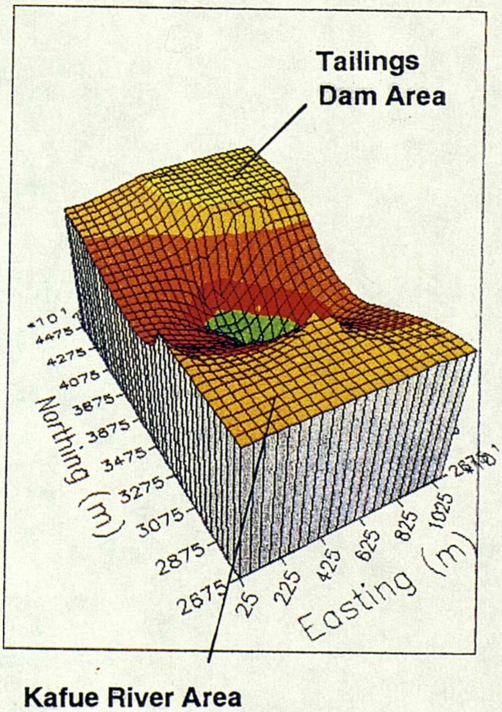


Figure 7.4b : Three - Dimensional Representation of simulating the effect of grouting the Luansobe Fault on groundwater level within the mine area



7.3 FINANCIAL DECISIONS

Pumping of water from the mine to surface alone, that is excluding the cost of mining and equipping pump chamber complexes, mine dewatering development and drilling, and mine development cementation cost, accounts for about 10% to 15% of the annual total mine operational cost as shown in the Mine financial figures tabulated in Table 7.1.

Table 7.1: KONKOLA MINE OPERATIONAL COST FIGURES FOR 1989-90 AND 1990-91 FINANCIAL YEARS

Item	Cost in Zambian Currency (Kwacha) Financial Year 1989-1990	1990-1991
Total Mine Operational Cost	675 760 000	1 287 201 000
Annual Mine Pumping Cost	84 956 000	127 380 000
Total Mine Dewatering Development and Drilling Cost:		
(a) All Aquifers	18 371 000	41 754 000
(b) Hangingwall Aquifers only	4 704 000	10 173 000
Mine Development Cementation Cost	4 729 000	5 069 000

Current Dewatering Drilling Cost:

(a) 4.5 inch borehole = K2 500 per metre
(b) NXC borehole = K1 800 per meter

NOTE: These figures do not include the cost of mining and equipping pump chamber complexes.

EXCHANGE RATE:

1989-1990:	British Sterling	£1 = K42.98	Zambian Kwacha
	US Dollar	$1 = K25.79	" "
1990-1991:		£1 = K55.00	
		$1 = K33.00	

The mine pumping cost is bound to significantly rise as the mine expands. In the current 20-year mine plan (1987-2007), ore production is scheduled to more than double the current output. Production is scheduled to increase from the current 1.1 million to 2.24 million tonnes of ore per year. This has to be achieved by the year 2000, if the shortfall from the other mines is to be offset. To achieve this target, a massive expansion programme is being planned and sinking of shaft(s) is to commence soon. This will require substantial lowering of groundwater levels. In the Hangingwall Aquifer, drawdowns of at least 20 metres per year at Number 1 Shaft (Kirilabombwe South Orebody) and 15 metres per year at Number 3 Shaft (Kirilabombwe North Orebody) will be necessary. Pumping costs alone could increase from their current levels of 10% to 15% to approach 20% of the annual total mine operational costs.

Although the simulations completed indicate that great savings can be achieved by removing recharge from the Mine Tailings Dam, it must be remembered that these simulations are based on very sparse data and the primary task to be now completed is to verify the data used. Thus, in order to formulate a long-term cost-effective groundwater management solution, field investigations would need to be carried out in order to remove current uncertainties in the hydrogeological boundaries and obtain representative data on hydraulic parameters such as hydraulic conductivity, transmissivity and in particular storage coefficient values essential for transient-state groundwater flow simulations.

Verification of geological parameters especially fault zones and aquifer boundaries is estimated to cost about 0.07% of the annual pumping cost. Hydrogeological investigations directed to securing values of transmissivity and storage coefficient within the key areas of the mine are estimated to cost approximately 7% of the annual pumping cost. Investigations required to define the rate of flow through the mine and its surrounding hydrogeological boundaries are estimated as costing about 8% of the annual pumping cost. On-going monitoring of instrumented sites to obtain values of head and appropriate values of transmissivity and storage coefficient which operate during mining are estimated as costing in the region of 0.2% of the annual pumping cost.

From this preliminary analysis, it is evident that an investment equal to about 15.3% of the annual pumping cost, which is equal to approximately 2% of the annual mine total operational cost, would be required to provide the factual data needed to assess groundwater control in the mine where pumping alone accounts for 10% to 15% of the annual mine total operational cost.

Since the anticipated working life of Konkola Mine extends beyond year 2020, this figure should be viewed as a good investment in providing a scientific basis upon which the formulation of a permanent and cost-effective groundwater management solution could be based.

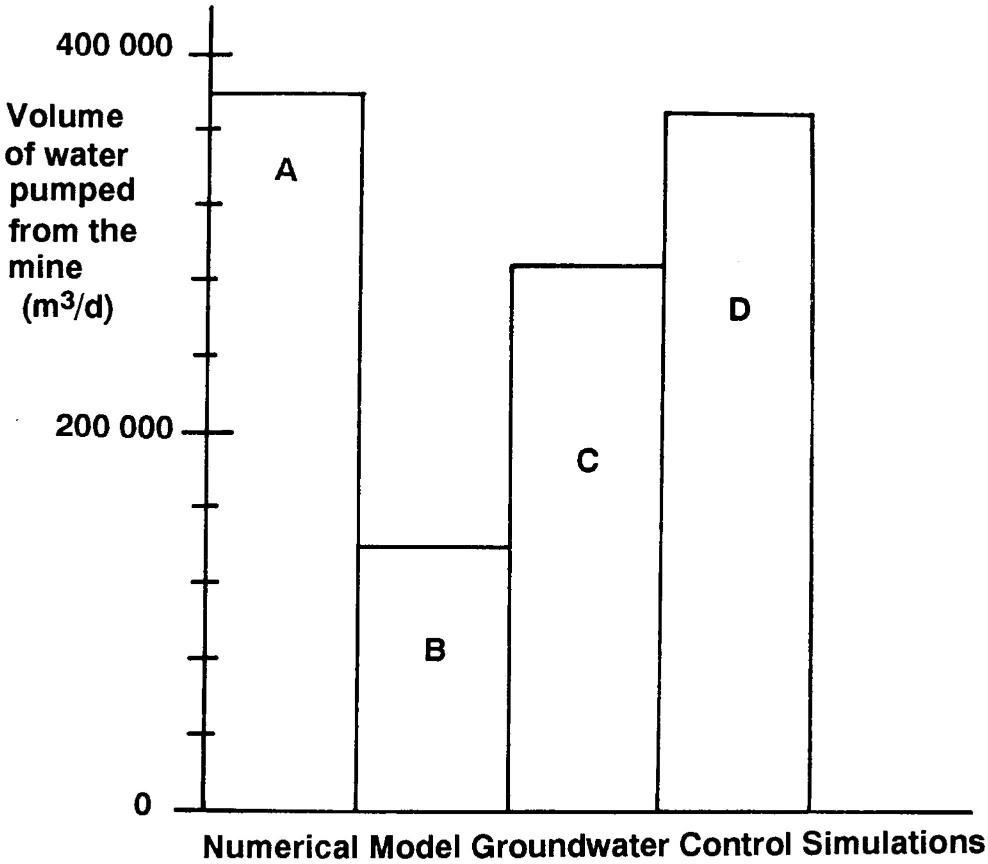
7.4 CONCLUSIONS

These preliminary steady-state groundwater flow simulations have clearly demonstrated that a cost-effective method of groundwater control other than control by pumping can be evolved which would result in reducing significantly the volume of groundwater entering the mine. This approach offers the best long-term cost-effective permanent solution to the Konkola problem.

Figure 7.5 shows that the stopping of leakage of Mine Tailings Dam water into the mine workings offers the best initial step towards reducing inflow into the mine. This preliminary analysis shows that the effect of this measure would reduce significantly mine discharge. The volume of water pumped from the mine to surface would drop by as much as 63%. This would have the desired effect of greatly reducing pumping cost.

Therefore, in the light of this evidence and background knowledge evolved from this research, it is recommended that the first step towards finding a permanent cost-effective groundwater management solution for Konkola should be the draining of the existing Tailings Dam and identifying a new location for continued disposal where leakage of water into the mine would not occur. The need for other measures would be assessed following evaluation of field investigations results recommended in Section 7.3.

Figure 7.5 : COMPARISON OF KONKOLA MINE PUMPING FIGURES WHEN VARIOUS GROUNDWATER CONTROL OPTIONS ARE IMPLEMENTED



A = Original setting. No groundwater control by exclusion implemented. Current Mine setting.

B = Mine Tailings Dam drained and resitted. No leakage of dam water into the mine.

C = Leakage of the Kafue River water into the mine stopped by either grouting or rechannelling the river.

D = Luansobe Fault grouted. Mine Tailings Dam and Kafue River waters continue to recharge the mine groundwater regime.

PART IV

CONCLUSIONS AND RECOMMENDATIONS
REFERENCES AND APPENDICES

This research has clearly demonstrated how a thorough systematic scientific study of a problem can yield good results. A better understanding of the mine hydrogeological regime enabled the indentification of causes of the groundwater flow problem. Having identified the causes meant that, for the first time ever in the 35 years or so that Konkola Mine has been in operation, the problem of large groundwater inflow into the mine was amenable to a long-term cost-effective groundwater management solution.

There is one Chapter in this part, Chapter 8, where the conclusions not only record the essential aspects of findings made during this research but also reflect the lessons that have been learnt during this research.

CHAPTER 8

CONCLUSIONS AND RECOMMENDATIONS

The main findings of this research have already been summarized at the end of each of the preceding chapters. Here in conclusion to this thesis, the following major points are highlighted:

1. The mine lies on the nose of an anticline which is wedged between two major faults; the Lubengele in the north and the Luansobe in the south. The rock in the Konkola mine area is fractured and broken as exemplified by the discontinuity pattern. The ground can be likened to a box of sugar cubes. These faults with their associated fracture zones, the Anticline-Axis and Cross-Axis fault zones, are the major channels of groundwater flow into and through the mine.

2. The surface hydrology pattern is controlled by the structural geology of the area. In particular, the river/stream pattern is in conformity with the discontinuity pattern.

3. The effective groundwater catchment for Konkola Mine is much greater than that exhibited by the topographic catchment.

4. There is more water pumped from the mine than can be accounted for by topographic catchment rainfall. About 70% to 80% of the water pumped from the mine comes from sources other than the Konkola Mine topographic catchment rainfall. Therefore, the main groundwater problem is that of recharge.

5. The mine groundwater flow system is recharged from two sources; the regional aquifer at depth and adjacent catchments via the regional faults, and from surface waters close to the mine. The major recharge source is the surface water system which accounts for about 88% of the total recharge.

6. Surface waters of the Kafue River and its tributaries, the Lubengele Fault and Mine Tailings Dam are the main sources of recharge to the hangingwall aquifers. About 72% of surface water recharge to the mine comes from the Lubengele Fault and Mine Tailings Dam, and about 28% from the Kafue River. The implications of this observation is that future remedial measures to reduce inflow into the mine should start by focussing on the Lubengele Fault and Mine Tailings Dam.

7. There are two chemically and physically distinct bodies of water at Konkola Mine; young (post-1952) and old (pre-1952). Hanginingwall water (cold) is the young water and has the same age as the Kafue River system water. Hangingwall water ingresses the mine from precipitation and leakage of surface water systems close to the mine, and flows mainly downwards. Footwall water (hot) is the old water and originates at depth from the regional aquifer and moves upwards mainly through fissure and fault zones.

8. Bearing in mind the uncertainties noted in the Appendix 2 Footnotes to pages 128 and 154, it seems fairly clear that Hangingwall Aquifers account for a substantial proportion (i.e. more than 40%) of total mine discharge, being greatest towards Number 3 Shaft (i.e. probably approaching 70%) where the contribution from the Lubengele Fault and Mine Tailings Dam is most strongly felt.

9. Flow to the mine is dominated by flow through fractures and fissures which form zones of high conductivity. The Hangingwall and Footwall aquifers water mix in these transmissive zones. The Cross-Axis fault zone is the main groundwater drainage zone within the mine. The flow in these zones can be modelled by finite-difference networks. This has enabled the numerical model to be produced which can be used to make groundwater discharge and water level drawdown predictions for future groundwater management decisions.

10. At Number 1 Shaft, the Kafue River water enters the mine mainly through the southern extremity of the orebody in the Luansobe fault zone and flows northwards into the rest of the mine mainly through fissures. At Number 3 Shaft, the Mine Tailings Dam water enters the mine through the Lubengele fault zone on the north limb of the orebody, and flows into the rest of the mine via the Cross-Axis fault zone. This is why the forementioned areas are the wettest parts of the mine and are very difficult to dewater.

11. The long-term cost-effective solution to the groundwater problem at Konkola Mine lies primarily in reducing the volume of water entering the mine. Computer model simulations of inflow from the north, via the Lubengele Fault, and its constraint (Simulation One) suggest that a reduction in mine discharge of as much as 63.7% could be achieved; an obvious start would be to resite the Tailings Dam. This would have a positive effect of making all the Number 3 Shaft and over 90% of currently mineable Number 1 Shaft orebodies, amenable to easy and less costly dewatering. As repeatedly stated throughout this thesis, these figures have yet to be confirmed even though they are based on the best data available at present.

12. This research has clearly demonstrated that in order to formulate an appropriate cost-effective groundwater management solution in a mine , particularly in underground mines, which Konkola is, it is essential that a correct understanding of the groundwater flow regime be obtained. This involves identifying and defining the scientific groundwater flow problem pertaining to the mine. In other words, finding the causes of groundwater ingress into the mine.

    Contrary to the conventional wisdom held at the mine for the past 30 years or so, the groundwater flow problem is not merely a problem of dewatering and water disposal, which can be dealt with by installing bigger pumps. My ten years of active involvement in mine dewatering at Konkola, coupled with scientific knowledge gained from the three years of this research, show that the way forward is to tackle the causes of the problem. Addressing the symptoms is a wrong approach to solving a problem.

3. The realisation that the Konkola Mine groundwater flow problem was primarily a scientific problem led to the adoption of an integrated approach in seeking the solution. An appropriate long-term cost-effective groundwater management solution had to be evolved in order to sustain the economic viability of the mine. This integrated approach included the use of appropriate field investigation methods and pertinent experimental techniques.

The strategy used the following six basic components:

(i) Historical dewatering and mining records
(ii) Current dewatering and mining records
(iii) Structural geology
(iv) Surface hydrology
(v) Hydrogeology
(vi) Groundwater and rock chemistry

A review of dewatering and mining data was done in order to sort out the good data from the bad data. Using the good data one could then ascertain what was already known and what was unknown and consequently define the nature of the scientific problem. Using this framework, a strategy for tackling the problem was formulated and a decision made on what sort of data needed to be collected.

Mine records of borehole water levels and discharge were combined to produce maps of water levels and water level change for given periods so that volumes of ground dewatered could be compared with volumes of water discharged. From these studies it became clear that the major hydrogeological problem at Konkola Mine was that of groundwater recharge. The water levels contained anomalies in its elevation, with highs and lows existing instead of a smooth drawdown surface. These maps were superimposed upon maps of basic geology of the mine, at similar scale and revealed that the basic geological controls existed for the movement of water in the mine area. This conclusion agreed with the experience of working in the mine and prompted a study of the drainage patterns in the area in an attempt to reveal any fundamental controls exerted by geological structures upon drainage.

Sorting out the good from the bad data in the mine records was very difficult and tedious. It took nine months to sort out the data because most of the data was corrupt. Borehole records consisted of water levels and discharge. A lot of these boreholes were open, uncased and crossed more than one aquifer. Therefore, the data they gave had unknown hydrogeological significance and thus had to be discarded.

Study of structural geology and hydrogeology were of critical importance to finding the reasons why the groundwater flow problem exists at all. The fractured and broken nature of ground in the Konkola mine area, analogous to a box of sugar cubes, coupled with the effects of mining on the rock, implies that there is enormous potential of not only surface water leaking into the mine but also adjacent groundwater catchments being hydraulically connected to the mine via the extensive fracture zones.

Thus, a usable map of geological structure had to be created in order to identify the location of potential major transmissive zones and delineate the effective groundwater catchment area. Boundary conditions that appeared to be pertinent and aquifer characteristics that seemed to be operating could then be identified. This would facilitate the prediction of the order of magnitude of the variation in transmissivity and storage coefficient to be expected.

The review of surface hydrology provided an opportunity to study the relationship between drainage patterns and structural geology with a view to identifying potential sources of recharge to the mine groundwater flow regime.

Study of groundwater chemistry provided the means of establishing the source of water and its flow path into and through the mine.

Groundwater chemistry has to be done properly. The water samples have to be related to the geology from which they came if correct analysis is to be achieved. Going down a mine and conducting water surveys is simply an incorrect way of tackling the problem. During this research one had to work against the incorrect use of chemistry in the past, particularly chemical surveys dating the water at the mine. Corrupt data was collected and corrupt analyses were entered into the mine records used by management.

One other important point this research has shown and has also been found to be true by other scientific researchers, is that groundwater can be partitioned into distinct areas, such that one aquifer can be different from the other. This is one of the problems of using open holes and different aquifer water mixes.

In other words, groundwater chemistry, groundwater pressure requires properly designed tests and monitoring points. Open boreholes and crude insitu solutions simply produce corrupt data which causes problems later on when it is used in the analysis.

With this strategy, the results of all previous investigations records were incorporated for the first time into an overall picture for the mine hydrogeology, thus permitting proper understanding of how groundwater is moving into and through the mine, and the creation of an adequate data base from which the numerical model was developed, and a programme of drying the mine formulated.

14. The successful completion of this research could not have been possible without the excellent communication existing between the researchers and mine management. This relationship enabled the mine to benefit from the fruits of the work even before its completion, for example, the resiting of the proposed new shaft to a location away from major groundwater flow routes.

Suggestions for Future Work

1. During the evaluation and review of mine dewatering borehole data, it was observed that the bulk of the records collected were those of borehole discharge and groundwater pressure-heads. There was very little if any data on hydraulic parameters of the aquifers. In order to accurately calculate groundwater flow velocities and relate mine pumping to mine water inflow, it is necessary to obtain representative aquifer values for hydraulic conductivity and storage coefficient over the area of mining interest. Only when this has been done will it be possible to accurately predict mine discharge and groundwater level drawdown. It is thus recommended that a programme of field investigations in the form of single-borehole pressure build-up tests be carried out at the mine to obtain these values. The numerical groundwater flow model needs representative data input in order to be a reliable tool.

2. Permanent river flow stations should be established on the Kafue River, one on the upstream and the other on the downstream of the mine. This will provide data for long-term monitoring of the river flow and provide a means of quantifying river water leakage into the mine.

3. Research should be established into use of horizontal dewatering boreholes drilled from underground mine workings as a means of obtaining aquifer pressure-heads and relevant hydraulic parameters. The conventional methods of analysis may not be appropriate because they are based on vertically drilled boreholes. Groundwater flow to a borehole is gravitational. Groundwater flow in vertical boreholes is horizontal and radial whilst in horizontal boreholes it is primarily vertical in nature. Thus using analyses designed for radial flow may not be appropriate.

4. A detailed structural map showing in particular the discontinuities should be produced to facilitate siting a new tailings dam in a location where leakage of dam water into the mine will not occur. Geophysics methods of delineating rock

fracture zones and groundwater transmissive zones, should be used and data on the groundwater regime collected to ensure that the best site is chosen. This map should be at a scale similar to the one used to design the present mine tailings dam.

5. An integrated team of personnel should be formed to implement the fruits of this research on site. Their main task would be to implement the program of reducing inflow of water into the mine in light of the knowledge gained from this research and solve problems pertinent to the task. For the long term, the team should develop into an in-house mining-hydrogeology consultancy for all the mines in Zambia Consolidated Copper Mines Limited.

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APPENDICES

APPENDIX 1

Appendix 1 contains topographic map of Konkola Mine area (Figure 3.2), geological map (Figure 3.3) and the geological plans of underground mine levels (Figures 3.7a to 3.7d).

Figures 3.7a to 3.7d are overleaf.

Figures 3.2 and 3.3 are in the pocket at the back of the thesis.

Figure 3.7a : KONKOLA MINE 450m (1480ft.) LEVEL GEOLOGICAL PLAN

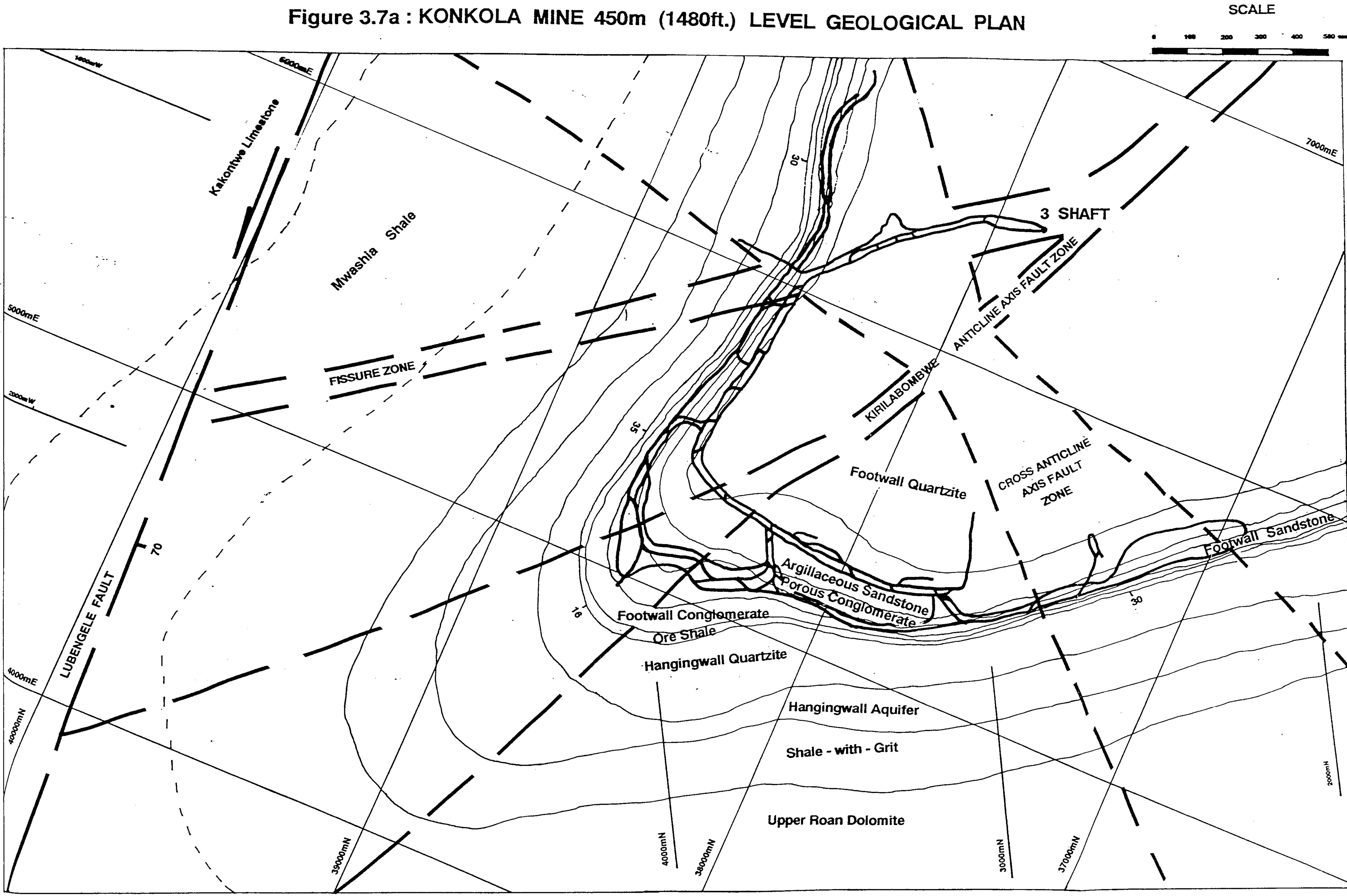


Figure 3.7b : KONKOLA MINE 565m (1850ft.) LEVEL GEOLOGICAL PLAN

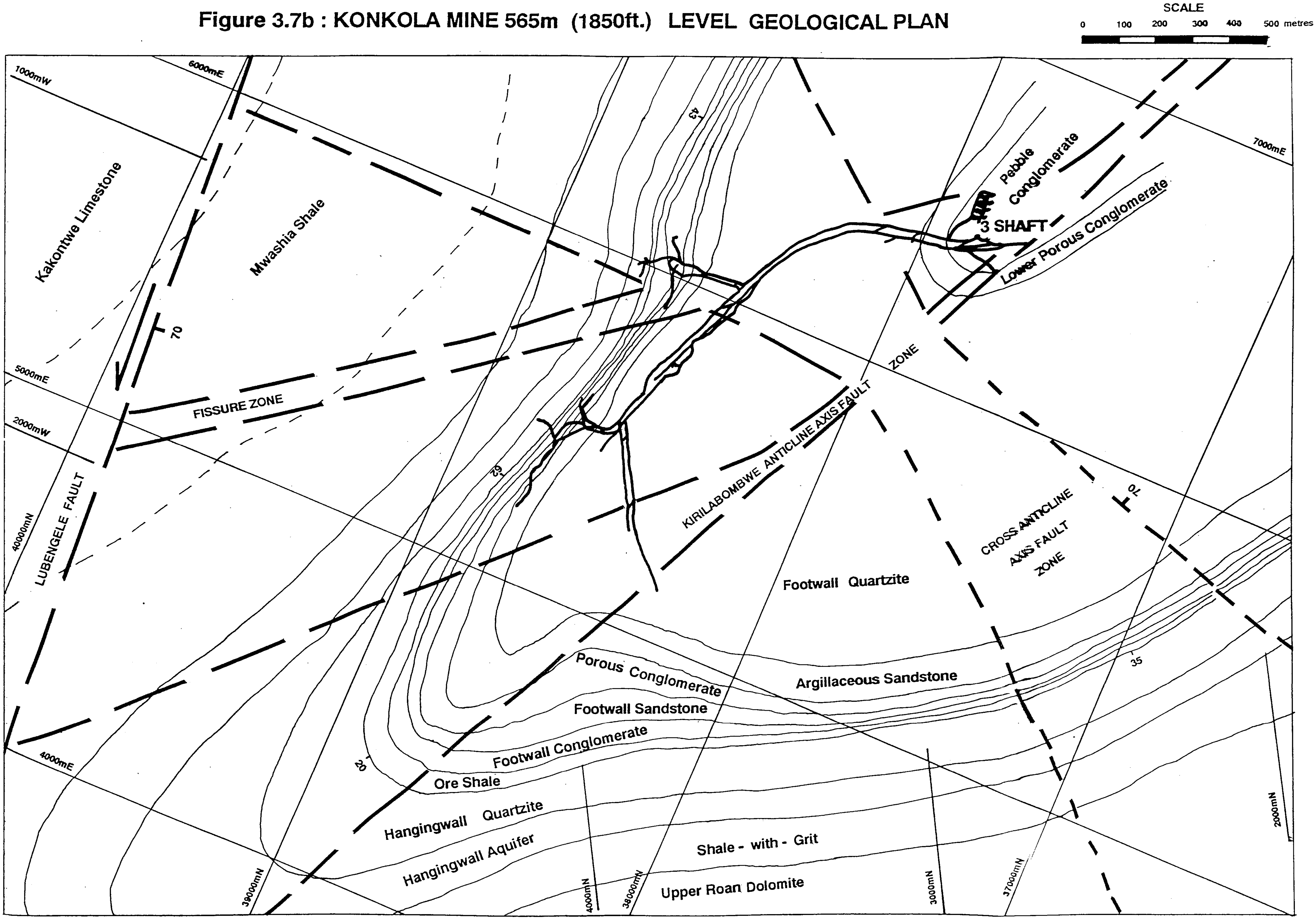


Figure 3.7c : KONKOLA MINE 610m (2200ft.) LEVEL GEOLOGICAL PLAN

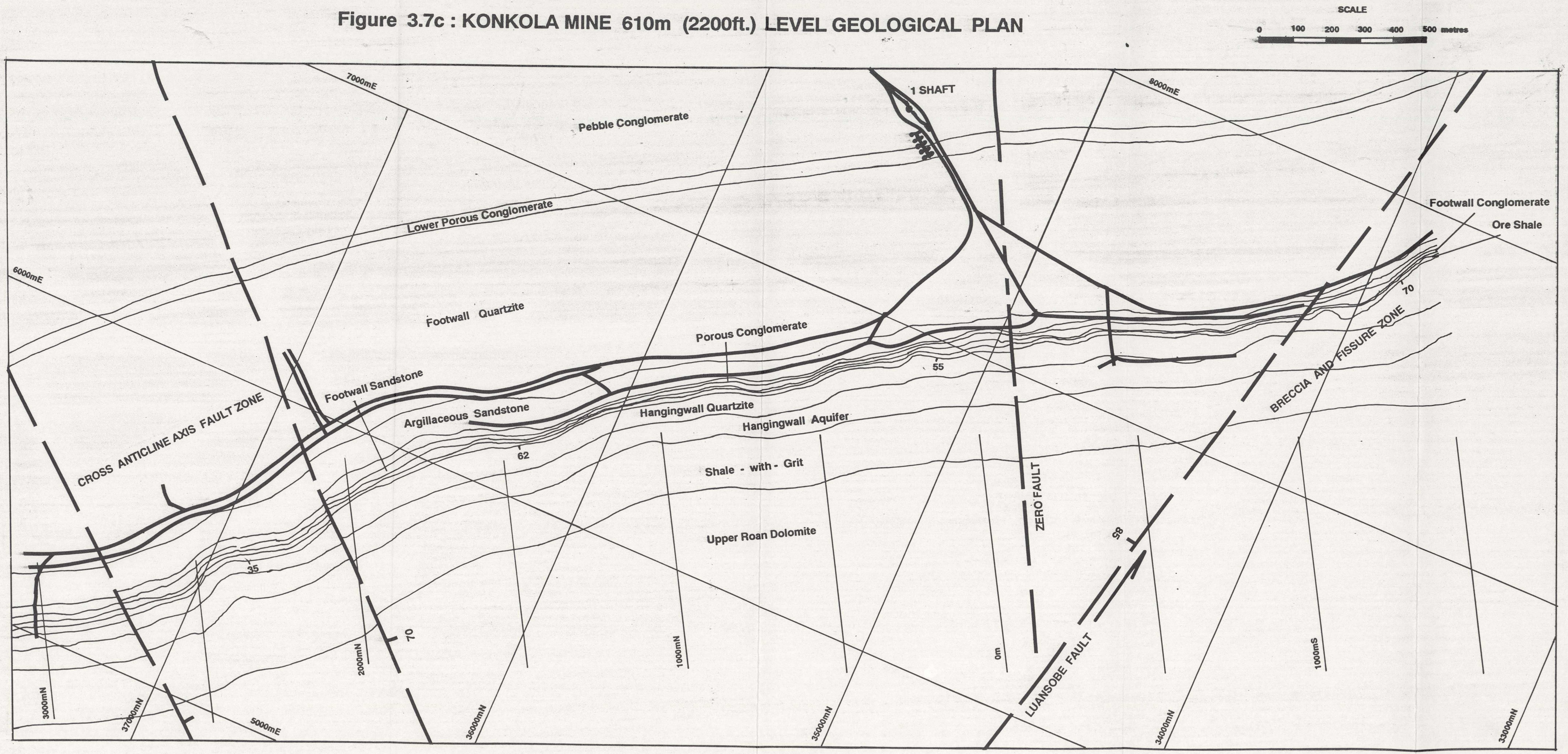
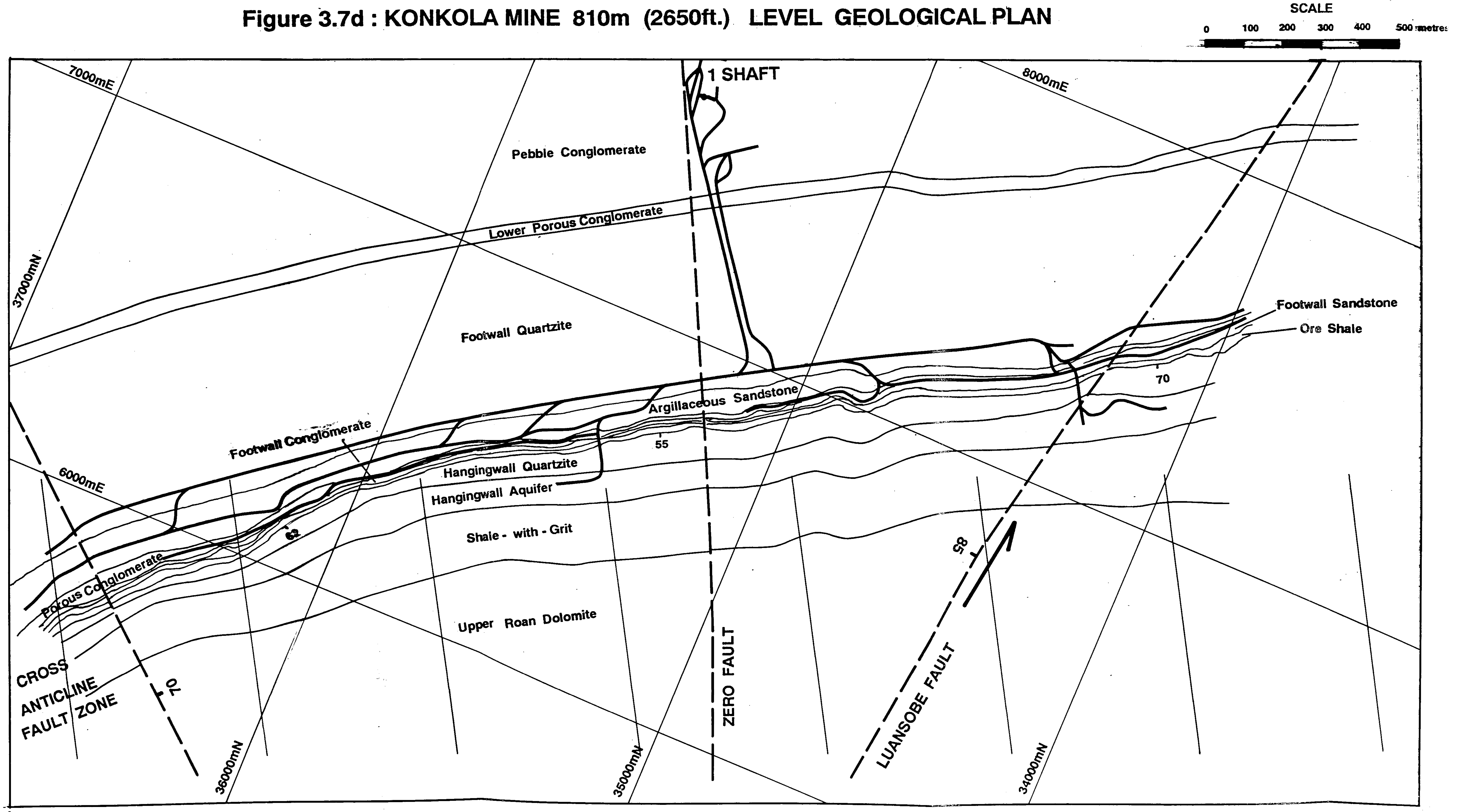


Figure 3.7d : KONKOLA MINE 810m (2650ft.) LEVEL GEOLOGICAL PLAN



APPENDIX 2

This Appendix contains geochemical and groundwater flow tracer tests data obtained from the field studies carried out at Konkola Mine in July 1988 and July 1989. Tables 5.1 to 5.17 contain geochemical data, Tables 5.20 and 5.21 data of Escherichia Coli bacteria concentration in mine waters, and Table 5.22 data of Residual Xanthate concentration in mine waters. Table 5.24 contains data of Average Concentration of Dissolved Ions in Mine Water in July 1989.

The Appendix also contains mathematical computations of mixing proportions of aquifers water.

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* Footnotes on pages 128 and 154

All these calculations have been based on the assumption that the Hangingwall Aquifers and Footwall Quartzite waters are two distinct end-members of the groundwater system and the Footwall Aquifer water is a mixture of the end-members. It is possible that this assumption may not be accurate, especially if the lithologies separating the Footwall Quartzite from the Hangingwall limestones and dolomites are transitional in chemical character, and thus have a groundwater chemistry that is transitional between the assumed end-members, and not a product of mixing. This has yet to be verified and negative numbers (see page 259) suggest the assumption may not be entirely correct. The relative proportions calculated on the basis of mixing thus provide a first assessment of the relative contribution from the Hangingwall and Footwall aquifers. This is further evidenced by the fact that if different ions are used, different ratios emerge.

Table 5.1: July 1988 Konkola Mine Area Surface Water Chemical Composition in Milligrams Per Litre (Mine Analytical Laboratory Results)

SOURCE	Temp (^oC)	pH	TDS	Mg	Ca	Na	K	Al	Cu	Fe	Co	NO_3	Cl	SO_4	HCO_3	SiO_2	Total Hardness
Kafue River																	
- Mine Water Works Pump Station	19.6	7.7	70	14	15	2.8	3.4	<0.1	<0.1	<0.1	<0.1	<0.25	1.8	4	68	2	94
- Chimfunshi	19.4	7.3	80	14	16	3.2	1.4	<0.1	<0.1	<0.1	<0.1	<0.25	0.1	<1.0	68	<1.0	96
Mine Tailings Dam																	
- Dam Spillway (Drainage Canal)	20.3	7.7	360	15	21	4.2	6.2	1.0	<0.1	0.1	<0.1	<0.25	3.6	30	98	6	112
Lubengele Stream - Upstream Mine Tailings Dam	19.7	6.3	10	2	3.2	0.4	0.8	<0.1	<0.1	<0.1	<0.1	<0.25	2.1	<1.0	15	2	16
Mingomba Stream - Muwa Farm	16.5	7.6	70	18	17	1.0	4.2	<0.1	<0.1	<0.1	<0.1	<0.25	1.6	<1.0	85	2	61
Konkola Stream - Chimfunshi Rd	15.7	7.6	10	1.2	2.6	1.2	1.4	<0.1	<0.1	<0.1	<0.1	<0.25	1.6	<1.0	27	4	11
Kakosa Stream - Chingola Rd	18.9	7.7	20	3.6	6.2	2.6	2.2	<0.1	<0.1	0.3	<0.1	<0.25	3.0	30	29	4	30

Table 5.2: July 1988 Konkola Mine Number 3 Shaft Groundwater Chemical Composition in Milligrams Per Litre (Mine Analytical Laboratory Results)

Shaft No.	Mine Level	Location	BH No.	Source (Aquifer)	Temp (^oC)	pH	TDS	Mg	Ca	Na	K	Al	Cu	Fe	Co	NO_3	Cl	SO_4	HCO_3	SiO_2	Total Hardness
3	450m	150m W	ADN320	HWA	27.1	6.4	80	6.6	14	1.4	5.2	<0.1	<0.1	<0.1	<0.1	<0.25	2.8	12	85	22	62
	(1480 ft)	350m W	ADN293	HWA	28.0	6.5	120	10	22	1.0	5.2	<0.1	<0.1	<0.1	<0.1	<0.25	0.9	72	159	18	95
		720m W	ADN220	HWA	27.4	6.9	200	18	38	3.0	6.8	<0.1	<0.1	<0.1	<0.1	<0.25	1.2	53	220	28	167
		1120m W	ADN206	HWA	25.2	7.0	250	25	48	4.0	7.6	<0.1	<0.1	<0.1	<0.1	<0.25	0.9	13	234	24	223
		1440m W	AD286	HWA	24.5	7.4	240	29	42	3.0	6.6	<0.1	<0.1	<0.1	<0.1	<0.25	0.8	66	281	18	224
		3275m N	ADN265	HWA	24.4	7.0	250	24	42	3.6	7.4	<0.1	<0.1	<0.1	<0.1	<0.25	1.4	13	244	20	204
	488m	720m W	ADN358	HWA	26.6	7.2	260	26	54	4.6	7.8	<0.1	<0.1	<0.1	<0.1	<0.25	1.2	58	256	16	239
	(1600 ft)	960m W	ADN352	HWA	25.8	7.1	240	24	48	3.6	7.6	<0.1	<0.1	<0.1	<0.1	<0.25	1.3	20	256	20	216
	565m	720m W	BPN247	HWA	26.4	7.1	260	37	54	5.0	7.2	<0.1	<0.1	<0.1	<0.1	<0.25	2.2	92	256	12	283
	(1850 ft)		Fissure	FWA	29.4	7.0	240	29	47	5.8	7.2	<0.1	<0.1	<0.1	<0.1	<0.25	0.1	73	195	16	235
		500m W	Fissure	FWQ	28.7	6.5	210	16	44	6.6	9.8	<0.1	<0.1	<0.1	<0.1	<0.25	0.1	85	159	26	174
		1340m W	ADN362	FWA	28.1	6.9	260	37	54	5.2	7.8	<0.1	<0.1	<0.1	<0.1	<0.25	2.4	199	232	18	283
		1520m W	Fissure	FWA	27.8	7.0	260	31	51	4.8	8.0	<0.1	<0.1	<0.1	<0.1	0.50	2.5	109	220	24	251
		1800m W	Fissure	FWQ	27.1	6.5	90	9.8	19	2.6	7.8	<0.1	<0.1	<0.1	0.1	<0.25	3.4	46	85	22	86
			BPN237	FWQ	27.1	6.3	110	11	23	2.8	7.4	<0.1	<0.1	<0.1	<0.1	<0.25	1.6	54	98	18	101

HWA = Hangingwall Aquifer

FWA = Footwall Aquifer

FWQ = Footwall Quartzite

Table 5.3: July 1988 Konkola Mine Number 1 Shaft Groundwater Chemical Composition in Milligrams Per Litre (Mine Analytical Laboratory Results)

Shaft No.	Mine Level	Location	BH No.	Source (Aquifer)	Temp (^oC)	pH	TDS	Mg	Ca	Na	K	Al	Cu	Fe	Co	NO_3	Cl	SO_4	HCO_3	SiO_2	Total Hardness
1	670m (2200 ft)	3000m N	AD947	HWA	26.1	7.0	190	35	59	3.8	8.6	1.0	<0.1	<0.1	<0.1	<0.25	0.9	55	264	10	288
			CP951	FWA	27.0	6.8	140	40	61	3.2	7.2	0.6	0.1	<0.1	<0.1	0.50	1.4	16	200	10	312
			Fissure	FWQ	27.8	6.1	50	6.9	51	3.0	7.0	0.4	<0.1	<0.1	<0.1	<0.25	2.2	2	84	10	156
		2950m N	CP394	FWA	27.4	6.5	80	26	51	9.2	13.0	<0.1	<0.1	<0.1	<0.1	<0.25	2.1	24	157	8	166
		2850m N	CP384	FWA	28.0	6.3	90	22	43	8.6	12.0	<0.1	<0.1	<0.1	<0.1	<0.25	2.8	57	137	4	104
		2782m N	CP363	FWA	27.9	6.6	80	17	25	3.0	6.4	1.2	<0.1	<0.1	<0.1	<0.25	2.8	20	159	4	123
		2685m N	CP356	FWA	28.1	6.4	90	12	34	2.4	6.6	0.6	<0.1	<0.1	<0.1	<0.25	3.0	23	112	8	133
		2685m N	CP354	HWA/FWA	27.5	6.6	110	29	56	10.2	12.0	<0.1	<0.1	<0.1	<0.1	<0.25	3.1	148	163	8	204
		2550m N	AD931	FWQ	29.2	5.9	40	6.4	23	2.8	5.6	0.6	<0.1	<0.1	<0.1	0.75	2.2	32	71	10	83
		2515m N	CP320	FWA	28.5	6.4	90	10	29	2.4	5.8	0.2	<0.1	<0.1	<0.1	0.25	0.9	37	93	28	113
		2515m N	CP324	FWA	28.6	6.5	70	8.8	29	2.4	9.0	0.2	<0.1	<0.1	<0.1	<0.25	0.9	5	96	6	106
		2410m N	CP309	FWA	28.6	6.9	90	18	38	7.6	10.4	<0.1	<0.1	<0.1	<0.1	<0.25	8.7	24	97	10	108
		2310m N	CP293	FWA	29.0	6.4	90	11	27	2.8	6.0	0.6	<0.1	<0.1	<0.1	<0.25	8.7	28	103	10	111
		2100m N	AD839	HWA	24.5	7.1	180	33	53	3.4	7.4	1.6	<0.1	<0.1	<0.1	<0.25	2.6	49	252	8	265
			AD943	FWQ	29.5	6.2	80	8.3	25	3.0	6.8	0.4	<0.1	<0.1	<0.1	<0.25	2.2	24	96	10	175
		1400m S	CP164	FWA	24.9	6.2	80	2.6	7.5	2.0	9.0	1.8	<0.1	<0.1	<0.1	<0.25	2.2	8	61	12	448
		1475m S	AD798	FWA	24.6	6.1	40	<0.1	7.2	1.2	6.6	1.2	<0.1	<0.1	<0.1	0.25	0.4	2	37	10	19
	730m (2400 ft)	1400m N	AD855	HWA	24.8	7.4	170	30	60	3.2	6.0	1.8	<0.1	<0.1	<0.1	<0.25		59	240	6	268
	810m (2650 ft)	2060m N	CP453	FWA	28.5	7.0	110	31	54	3.2	8.4	<0.1	0.1	<0.1	0.4	0.25	1.4	57	205	10	86
		1950m N	CP442	FWA	29.2	6.9	100	17	33	7.6	9.9	<0.1	<0.1	<0.1	<0.1	0.25	1.5	152	92	9	84
		1918m N	CP438	FWA	28.8	6.5	120	10	24	2.8	6.0	<0.1	<0.1	<0.1	<0.1	0.25	1.6	8	107	10	100
		1830m N	CP433	FWA	28.8	6.6	150	11	22	8.4	6.4	<0.1	<0.1	<0.1	0.2	0.25	0.5	18	131	8	99
		1800m N	AD949	HWA/FWA	27.9	6.7	200	17	44	3.2	7.4	<0.1	<0.1	<0.1	0.3	<0.25	1.1	87	131	6	178
			CP430	FWA	28.8	6.6	140	25	64	8.0	12.0	<0.1	<0.1	<0.1	<0.1	0.25	1.0	85	150		164
		1755m N	CP426	FWA	29.0	6.9	100	42	78	13.6	11.9	<0.1	<0.1	<0.1	<0.1	<0.25	1.2	253	264		300
		1715m N	AD948	HWA/FWA	26.1	7.2	310	42	81	15.0	12.0	<0.1	<0.1	<0.1	<0.1	<0.25	1.0	262	281		290
			AD945	HWA/FWA	26.1	7.1	308	19	37	7.2	10.0	<0.1	<0.1	<0.1	<0.1	<0.25		31	106		100
			CP424	FWA	27.5	7.0	190	22	41	7.2	9.8	<0.1	<0.1	<0.1	<0.1	<0.25		5	137		162
		1400m N	AD933	HWA	26.2	7.2	360	31	98	6.2	7.2	<0.1	<0.1	<0.1	<0.1	<0.25	1.1	91	273	8	369
		1260m N	AD928	HWA	25.5	7.2	270	26	54	4.4	6.0	<0.1	<0.1	<0.1	0.2	<0.25	1.2	66	259	8	239
		600m N	AD857	HWA	25.0	7.4	220	44	72	2.6	8.0	<0.1	<0.1	<0.1	0.3	<0.25	0.9	143	293	6	356
		500m N	AD865	HWA	25.0	7.4	300	28	88	7.2	7.4	<0.1	<0.1	<0.1	0.1	<0.25	0.8	178	244	6	132
			AD870	HWA	24.5	7.9	230	20	52	4.4	6.6	<0.1	<0.1	<0.1	<0.1	<0.25	0.8	160	254	8	310
		100m N	AD835	HWA	25.0	7.4	430	40	72	7.6	7.6	<0.1	<0.1	<0.1	<0.1	<0.25	1.1	172	317	4	340

Table 5.3: July 1988 Konkola Mine Number 1 Shaft Groundwater Chemical Composition in Milligrams Per Litre (Mine Analytical Laboratory Results)

Shaft No.	Mine Level	Location	BH No.	Source (Aquifer)	Temp (°C)	pH	TDS	Mg	Ca	Na	K	Al	Cu	Fe	Co	NO_3	Cl	SO_4	HCO_3	SiO_2	Total Hardness
1	670m (2200 ft)	3000m N	AD947	HWA	26.1	7.0	190	35	59	3.8	8.6	1.0	<0.1	<0.1	<0.1	<0.25	0.9	55	264	10	288
			CP951	FWA	27.0	6.8	140	40	61	3.2	7.2	0.6	0.1	<0.1	<0.1	0.50	1.4	16	200	10	312
			Fissure	FWQ	27.8	6.1	50	6.9	51	3.0	7.0	0.4	<0.1	<0.1	<0.1	<0.25	2.2	2	84	10	156
		2950m N	CP394	FWA	27.4	6.5	80	26	51	9.2	13.0	<0.1	<0.1	<0.1	<0.1	<0.25	2.1	24	157	8	166
		2850m N	CP384	FWA	28.0	6.3	90	22	43	8.6	12.0	<0.1	<0.1	<0.1	<0.1	<0.25	2.8	57	137	4	104
		2782m N	CP363	FWA	27.9	6.6	80	17	25	3.0	6.4	1.2	<0.1	<0.1	<0.1	<0.25	2.8	20	159	4	123
		2685m N	CP356	FWA	28.1	6.4	90	12	34	2.4	6.6	0.6	<0.1	<0.1	<0.1	<0.25	3.0	23	112	8	133
		2685m N	CP354	HWA/FWA	27.5	6.6	110	29	56	10.2	12.0	<0.1	<0.1	<0.1	<0.1	<0.25	3.1	148	163	8	204
		2550m N	AD931	FWQ	29.2	5.9	40	6.4	23	2.8	5.6	0.6	<0.1	<0.1	<0.1	0.75	2.2	32	71	10	83
		2515m N	CP320	FWA	28.5	6.4	90	10	29	2.4	5.8	0.2	<0.1	<0.1	<0.1	0.25	0.9	37	93	28	113
		2515m N	CP324	FWA	28.6	6.5	70	8.8	29	2.4	9.0	0.2	<0.1	<0.1	<0.1	<0.25	0.9	5	96	6	106
		2410m N	CP309	FWA	28.6	6.9	90	18	38	7.6	10.4	<0.1	<0.1	<0.1	<0.1	<0.25	8.7	24	97	10	108
		2310m N	CP293	FWA	29.0	6.4	90	11	27	2.8	6.0	0.6	<0.1	<0.1	<0.1	<0.25	8.7	28	103	10	111
		2100m N	AD839	HWA	24.5	7.1	180	33	53	3.4	7.4	1.6	<0.1	<0.1	<0.1	<0.25	2.6	49	252	8	265
			AD943	FWQ	29.5	6.2	80	8.3	25	3.0	6.8	0.4	<0.1	<0.1	<0.1	<0.25	2.2	24	96	10	175
		1400m S	CP164	FWA	24.9	6.2	80	2.6	7.5	2.0	9.0	1.8	<0.1	<0.1	<0.1	<0.25	2.2	8	61	12	448
		1475m S	AD798	FWA	24.6	6.1	40	<0.1	7.2	1.2	6.6	1.2	<0.1	<0.1	<0.1	0.25	0.4	2	37	10	19
	730m (2400 ft)	1400m N	AD855	HWA	24.8	7.4	170	30	60	3.2	6.0	1.8	<0.1	<0.1	<0.1	<0.25		59	240	6	268
	810m (2650 ft)	2060m N	CP453	FWA	28.5	7.0	110	31	54	3.2	8.4	<0.1	0.1	<0.1	0.4	0.25	1.4	57	205	10	86
		1950m N	CP442	FWA	29.2	6.9	100	17	33	7.6	9.9	<0.1	<0.1	<0.1	<0.1	0.25	1.5	152	92	9	84
		1918m N	CP438	FWA	28.8	6.5	120	10	24	2.8	6.0	<0.1	<0.1	<0.1	<0.1	0.25	1.6	8	107	10	100
		1830m N	CP433	FWA	28.8	6.6	150	11	22	8.4	6.4	<0.1	<0.1	<0.1	0.2	0.25	0.5	18	131	8	99
		1800m N	AD949	HWA/FWA	27.9	6.7	200	17	44	3.2	7.4	<0.1	<0.1	<0.1	0.3	<0.25	1.1	87	131	6	178
			CP430	FWA	28.8	6.6	140	25	64	8.0	12.0	<0.1	<0.1	<0.1	<0.1	0.25	1.0	85	150		164
		1755m N	CP426	FWA	29.0	6.9	100	42	78	13.6	11.9	<0.1	<0.1	<0.1	<0.1	<0.25	1.2	253	264		300
		1715m N	AD948	HWA/FWA	26.1	7.2	310	42	81	15.0	12.0	<0.1	<0.1	<0.1	<0.1	<0.25	1.0	262	281		290
			AD945	HWA/FWA	26.1	7.1	308	19	37	7.2	10.0	<0.1	<0.1	<0.1	<0.1	<0.25		31	106		100
			CP424	FWA	27.5	7.0	190	22	41	7.2	9.8	<0.1	<0.1	<0.1	<0.1	<0.25		5	137		162
		1400m N	AD933	HWA	26.2	7.2	360	31	98	6.2	7.2	<0.1	<0.1	<0.1	<0.1	<0.25	1.1	91	273	8	369
		1260m N	AD928	HWA	25.5	7.2	270	26	54	4.4	6.0	<0.1	<0.1	<0.1	0.2	<0.25	1.2	66	259	8	239
		600m N	AD857	HWA	25.0	7.4	220	44	72	2.6	8.0	<0.1	<0.1	<0.1	0.3	<0.25	0.9	143	293	6	356
		500m N	AD865	HWA	25.0	7.4	300	28	88	7.2	7.4	<0.1	<0.1	<0.1	0.1	<0.25	0.8	178	244	6	132
			AD870	HWA	24.5	7.9	230	20	52	4.4	6.6	<0.1	<0.1	<0.1	<0.1	<0.25	0.8	160	254	8	310
		100m N	AD835	HWA	25.0	7.4	430	40	72	7.6	7.6	<0.1	<0.1	<0.1	<0.1	<0.25	1.1	172	317	4	340

Table 5.4: July 1988 Konkola Mine Area Surface Water Chemical Composition in Milligrams Per Litre (Imperial College Geochemistry Laboratory Results)

Source	Mg	Ca	Na	K	Al	Cu	Fe	Ba	Sr	Si	P	S
Kafue River												
- Mine Water Works Pump Station	8.83	16.40	4.67	0.22	<0.06	0.29	0.22	0.02	0.05	3.33	<0.04	1.50
- Mine Sewage Confluence	9.28	17.60	4.92	2.89	"	0.16	0.16	0.05	0.06	3.50	0.04	4.50
- Weir Downstream Golf Club	8.72	16.53	4.44	0.89	"	0.07	0.04	0.03	0.06	3.30	<0.04	2.17
Mine Tailings Dam												
- Dam Spillway	12.89	23.53	7.55	10.39	0.11	0.20	0.27	0.11	0.11	3.89	"	16.66
Lubengele Stream												
- Upstream Tailings Dam	1.27	53.6	1.11	<0.11	<0.056	0.28	7.20	0.02	0.01	2.75	"	0.61
- Downstream Tailings Dam	17.61	37.76	18.83	9.61	"	0.26	0.27	0.10	0.16	4.11	"	49.16
Mingomba Stream												
- Pump Station	8.72	14.50	3.78	1.44	0.86	1.21	1.24	0.33	0.04	4.59	3.60	4.67
Concentrator Tailings Effluent Pipe	21.05	37.76	15.44	22.72	0.50	0.67	0.33	0.11	0.21	4.67	5.49	32.61
Kakosa Stream												
- Chingola Rd	2.50	5.57	3.78	2.28	<0.06	0.14	0.53	0.05	0.02	5.25	0.07	0.94
- Downstream Drainage Canal	20.0	41.83	5.44	9.72	0.58	0.96	0.73	0.15	0.22	10.75	<0.04	19.78

Source	Li	Rb	Be	La	Ti	V	Cr	Mo	Mn	Ni	Ag	Zn
Kafue River												
- Mine Water Works Pump Station	<0.004	<0.22	<0.001	<0.01	<0.07	<0.006	0.02	<0.01	<0.02	0.11	0.02	0.84
- Mine Sewage Confluence	"	"	"	"	"	"	<0.01	"	"	0.05	<0.01	0.28
- Weir Downstream Golf Club	"	"	"	"	"	"	"	"	"	<0.01	"	0.02
Mine Tailing Dam												
- Dam Spillway	"	"	"	"	"	"	"	0.02	0.05	"	"	0.01
Lubengele Stream												
- Upstream Tailings Dam	0.004	"	"	"	"	"	"	<0.01	0.12	0.83	0.02	0.13
- Downstream Tailings Dam	<0.004	"	"	0.02	"	"	"	"	0.10	0.03	0.02	0.07
Mingomba Stream												
- Pump Station	"	"	"	<0.01	"	"	"	"	0.06	<0.01	<0.01	0.33
Concentrator Tailings Effluent Pipe	0.01	"	0.001	0.01	"	"	"	0.06	0.72	0.10	"	0.13
Kakosa Stream												
- Chingola Rd												
- Downstream Drainage Canal												

Table 5.4: **July 1988 Konkola Mine Area Surface Water Chemical Composition in Milligrams Per Litre (Imperial College Geochemistry Laboratory Results) (Contd)**

Source	Cd	B	Pb	As	Co
Kafue River					
- Mine Water Works Station Pump	<0.006	<0.01	<0.06	<0.06	<0.01
- Mine Sewage Confluence	"	"	"	"	"
- Weir Downstream Golf Club	"	0.11	"	"	"
Mine Tailings Dam					
- Dam Spillway	"	"	"	"	0.02
Lubengele Stream					
- Upstream Tailings Dam	"	"	"	0.14	<0.01
- Downstream Tailings Dam	"	"	"	<0.06	"
Mingomba Steam					
- Pump Station	"	0.46	"	0.08	0.06
Concentrator Tailings Effluent Pipe	"	0.07	"	<0.06	0.08
Kakosa Stream					
- Chingola Rd	"	<0.01	"	"	<0.01
- Downstream Drainage Canal	"	"	"	"	0.04

Table 5.5: **July 1988 Konkola Mine Number 3 Shaft Groundwater Chemical Composition in Milligrams Per Litre (Imperial College Geochemical Laboratory Results)**

Shaft	Mine Level	Location	BHID	Source (Aquifer)	Mg	Ca	Na	K	Al	Cu	Fe	Ba	Sr	Si
3	450m	150m W	ADN309	HWA	24.00	59.70	6.72	7.50	<0.06	0.04	<0.04	0.08	0.33	11.36
	(1480ft)		ADN320	HWA	6.72	14.17	1.47	4.34	"	<0.01	"	0.04	0.08	11.50
			ADN323	HWA	21.89	53.80	6.47	7.28	"	0.01	"	0.08	0.31	11.47
		350m W	AD293	HWA	12.11	24.70	2.10	4.89	"	0.01	"	0.06	0.14	11.80
		720m W	AD270	HWA	12.28	28.66	2.17	4.83	0.17	0.01	"	0.07	0.15	12.11
			AD218	HWA	23.61	63.10	3.94	6.11	0.69	0.01	"	0.10	0.29	10.80
			AD209	HWA	17.55	36.60	2.61	5.33	<0.06	<0.01	"	0.08	0.21	11.97
		1120m W	ADN206	HWA	24.44	50.40	3.31	6.33	"	<0.01	"	0.11	0.28	10.75
		1440m W	AD286	HWA	27.94	41.93	2.50	5.17	"	<0.01	"	0.08	0.33	11.17
		3275m N	ADN265	HWA	24.55	44.36	3.22	6.67	"	<0.01	"	0.09	0.18	8.80

Location	BHID	Source	P	S	Li	Rb	Be	La	Ti	V	Cr	Mo
150m W	ADN309	HWA	0.11	31.22	0.02	0.33	<0.001	0.03	<0.07	0.01	<0.01	0.03
	ADN320	HWA	0.40	0.90	<0.004	<0.22	"	<0.01	"	<0.01	"	<0.01
	ADN323	HWA	0.06	24.83	0.01	"	"	0.02	"	"	"	0.02
350m W	AD293	HWA	0.33	3.05	0.01	"	"	0.02	"	"	"	"
720m W	AD270	HWA	0.18	3.22	0.02	"	"	0.02	"	"	"	"
	AD218	HWA	0.09	14.44	0.07	"	"	0.02	"	"	"	"
	AD209	HWA	0.11	6.50	<0.004	"	"	<0.01	"	"	"	"
1120m W	ADN206	HWA	0.09	16.67	"	"	"	0.02	"	"	"	0.03
1440m W	AD286	HWA	<0.04	5.44	"	"	"	0.02	"	"	"	0.03
3275m N	ADN265	HWA	0.09	14.94	"	"	"	<0.01	"	"	"	0.02

Location	BHID	Source	Ni	Ag	Zn	Cd	B	Pb	As	Co
150m W	ADN309	HWA	0.03	<0.01	0.03	<0.01	<0.01	<0.06	<0.06	<0.01
	ADN320	HWA	<0.01	"	<0.01	"	"	"	"	"
	ADN323	HWA	0.01	"	0.01	"	"	"	"	"
350m W	AD293	HWA	0.02	"	<0.01	"	"	"	"	"
720m W	AD270	HWA	<0.01	"	0.02	"	"	"	"	"
	AD218	HWA	"	"	<0.01	"	0.11	"	"	"
	AD209	HWA	"	"	"	"	<0.01	"	"	"
1120m W	ADN206	HWA	"	"	"	"	"	"	"	"
1440m W	AD286	HWA	"	"	"	"	"	"	"	"
3275m N	ADN265	HWA	"	"	"	"	"	"	"	"

Table 5.6: July 1988 Konkola Mine Number 3 Shaft Groundwater Chemical Composition in Milligrams Per Litre (Imperial College Geochemical Laboratory Results)

Shaft	Mine Level	Location	BHID	Source (Aquifer)	Mg	Ca	Na	K	Al	Cu	Fe	Ba	Sr	Si
3	488m	720m W	ADN358	HWA	25.89	53.80	4.22	6.78	<0.06	<0.01	<0.04	0.11	0.27	11.00
	(1600ft)	960m W	ADN352	HWA	23.00	48.60	3.27	6.28	"	"	"	0.10	0.29	10.30
	565m	1440m W	BPN250	FWA	23.00	47.23	3.28	6.28	<0.06	<0.01	<0.04	0.10	0.27	10.53
	(1850ft)	1400m W	BPN245	FWA	21.55	49.83	4.58	7.89	"	0.05	"	0.15	0.23	10.53
		720m W	Fissure	FWA	22.89	57.20	5.61	7.61	"	0.04	"	0.13	0.27	10.64
			Fissure	FWA	21.83	58.90	5.39	7.17	0.44	0.03	"	0.10	0.27	10.36
			BPN247	HWA	25.66	53.10	4.17	6.61	<0.06	<0.01	"	0.11	0.27	10.80
			BPN365	HWA	27.05	56.20	5.11	7.22	0.11	0.08	"	0.12	0.29	11.25
		500m W	Fissure	FWQ	9.72	44.46	6.22	9.22	<0.06	0.03	"	0.15	0.27	10.44
		1250m W	AD362	FWA	23.50	56.70	5.42	7.44	"	0.09	"	0.13	0.16	10.47
		1800m W	BPN237	FWQ	6.22	23.16	3.00	7.22	"	0.01	"	0.16	0.09	9.00
			Fissure	FWQ	6.55	21.16	2.41	6.44	"	0.01	"	0.13	0.84	8.55

Mine Level	Location	BHID	Source (Aquifer)	P	S	Li	Rb	Be	La	Ti	V	Cr	Mo
488m	720m W	ADN358	HWA	0.04	19.33	<0.004	<0.02	<0.001	0.02	<0.07	<0.01	<0.01	0.02
(1600ft)	960m W	ADN352	HWA	<0.04	15.00	"	"	"	<0.01	"	"	"	<0.01
565m	1440m W	BPN250	FWA	0.07	21.11	<0.004	<0.02	<0.001	0.01	<0.07	<0.01	<0.01	"
(1850ft)	1400m W	BPN245	FWA	<0.04	25.72	"	"	"	0.02	"	"	"	"
	720m W	Fissure	FWA	0.07	32.16	0.02	"	"	0.02	"	"	"	0.02
		Fissure	FWA	0.04	21.94	<0.004	"	"	0.01	"	"	"	<0.01
		BPN247	HWA	0.07	19.50	"	"	"	"	"	"	"	0.02
		BPN365	HWA	0.07	23.39	"	"	"	"	"	"	"	0.03
	500m W	Fissure	FWQ	0.22	19.05	"	"	"	"	"	"	"	0.01
	1250m W	AD362	FWA	<0.04	22.33	0.01	"	"	<0.01	"	"	"	0.02
	1800m W	BPN237	FWQ	0.13	10.17	<0.004	"	"	0.01	"	"	"	0.02
		Fissure	FWQ	0.15	6.67	"	"	"	"	"	"	"	0.01

Mine Level	Location	BHID	Source (Aquifer)	Mn	Ni	Ag	Zn	Cd	B	Pb	As	Co
488m	720m W	ADN358	HWA	<0.02	<0.01	<0.01	<0.01	<0.01	<0.01	<0.06	<0.06	<0.01
(1600ft)	960m W	ADN352	HWA	"	"	"	"	"	"	"	"	"
565m	1440m W	BPN250	FWA	"	"	"	"	"	"	"	"	"
(1850ft)	1400m W	BPN245	FWA	"	"	"	"	"	"	"	"	"
	720m W	Fissure	FWA	"	"	"	0.02	"	0.05	"	0.08	"
		Fissure	FWA	"	"	"	<0.01	"	<0.01	"	<0.06	"
		BPN247	HWA	"	"	"	"	"	"	"	"	"
		BPN365	HWA	"	"	"	0.05	"	"	"	"	"
	500m W	Fissure	FWQ	"	"	"	0.01	"	"	"	"	"
	1250m W	AD362	FWA	"	"	"	0.26	"	"	"	"	"
	1800m W	BPN237	FWQ	"	"	"	<0.01	"	"	"	"	"
		Fissure	FWQ	"	"	"	"	"	"	"	"	"

Table 5.7: **July 1988 Konkola Mine Number 1 Shaft 670m Level Groundwater Chemical Composition in Milligrams Per Litre (Imperial College Geochemical Laboratory Results)**

Shaft	Mine Level	Location	BHID	Source (Aquifer)	Mg	Ca	Na	K	Al	Cu	Fe	Ba	Sr	Si
1	670m (2200ft)	3000m N	AD947	HWA	32.50	57.30	4.06	7.22	0.22	<0.01	0.22	0.15	0.25	10.08
			CP394	FWA	7.22	20.10	2.25	5.78	<0.06	0.01	<0.04	0.11	0.14	8.55
			Fissure	FWQ	16.22	37.63	3.14	7.28	"	0.02	"	0.18	0.08	9.14
		2850m N	CP384	FWA	10.89	28.00	2.33	6.05	"	0.06	"	0.13	0.13	8.19
		2782m N	CP363	FWA	13.28	21.60	2.50	6.50	"	<0.01	"	0.13	0.10	8.44
		2685m N	CP354	FWA	8.94	40.30	3.19	6.94	"	0.02	"	0.15	0.11	8.80
			CP356	FWA	8.50	23.26	2.33	5.83	"	0.01	"	0.10	0.14	8.80
		2515m N	CP324	FWA	6.89	21.70	2.25	5.55	"	<0.01	"	0.09	0.08	8.72
		2515m N	CP320	FWA	5.50	18.70	2.25	5.33	"	"	"	0.07	0.08	8.55
		2410m N	CP309	FWA	8.33	25.70	2.28	6.11	"	"	"	0.10	0.06	8.69
		2310m N	CP293	FWA	7.30	24.33	2.50	6.11	"	"	"	0.09	0.10	8.78
		2550m N	AD931	FWQ	4.44	17.16	2.33	4.94	"	"	"	0.06	0.06	9.14
		2100m N	AD846	HWA	30.55	50.70	3.64	6.39	"	"	"	0.10	0.23	10.47
			AD842	HWA	32.61	54.20	4.00	6.67	"	"	"	0.10	0.25	10.67
			AD839	HWA	30.39	50.10	3.50	5.83	"	"	"	0.08	0.22	10.75
			AD943	FWQ	6.83	23.50	3.19	6.72	"	"	"	0.10	0.09	9.44
			AD880	FWQ	7.05	23.96	2.97	6.55	"	"	"	0.10	0.09	9.47
			AD869	FWQ	5.11	15.80	3.31	5.94	"	"	"	0.08	0.06	11.19

Shaft	Mine Level	Location	BHID	Source (Aquifer)	P	S	Li	Rb	Be	La	Ti	V	Cr	Mo
1	670m (2200ft)	3000m N	AD947	HWA	<0.04	27.16	0.01	<0.22	<0.001	0.02	<0.07	<0.01	<0.01	0.04
			CP394	FWA	0.22	4.28	<0.004	"	"	<0.01	"	"	"	0.01
			Fissure	FWQ	0.15	11.39	"	"	0.001	"	"	"	"	0.02
		2850m N	CP384	FWA	0.06	6.89	"	"	<0.001	"	"	"	"	0.01
		2782m N	CP363	FWA	0.09	8.50	"	"	"	"	"	"	"	<0.01
		2685m N	CP354	FWA	0.13	17.33	"	"	"	"	"	"	"	"
			CP356	FWA	0.09	6.28	"	"	"	"	"	"	"	"
		2515m N	CP324	FWA	0.09	5.33	"	"	"	"	"	"	"	"
		2515m N	CP320	FWA	0.07	6.44	"	"	"	"	"	"	"	"
		2410m N	CP309	FWA	0.16	5.78	"	"	"	"	"	"	"	"
		2310m N	CP293	FWA	0.07	7.44	"	"	"	"	"	"	"	"
		2550m N	AD931	FWQ	0.13	5.89	"	"	"	"	"	"	"	"
		2100m N	AD846	HWA	0.07	24.28	"	"	"	"	"	"	"	"
			AD842	HWA	0.07	27.50	"	"	"	0.02	"	"	"	0.04
			AD839	HWA	0.09	18.94	"	"	"	<0.01	"	"	"	<0.1
			AD943	FWQ	0.16	9.89	"	"	"	"	"	"	"	0.02
			AD880	FWQ	0.16	9.61	"	"	"	"	"	"	"	0.01
			AD869	FWQ	0.20	6.39	"	"	"	"	"	"	"	0.02

Shaft	Mine Level	Location	BHID	Source (Aquifer)	Mn	Ni	Ag	Zn	Cd	B	Pb	As	Co
1	670m (2200ft)	3000m N	AD947	HWA	<0.02	<0.01	<0.01	0.01	<0.01	<0.01	<0.06	<0.06	<0.01
			CP394	FWA	"	"	"	<0.01	"	"	"	"	"
			Fissure	FWQ	"	"	"	0.02	"	"	"	"	"
		2850m N	CP384	FWA	"	"	"	0.07	"	"	"	"	"
		2782m N	CP363	FWA	"	"	"	<0.01	"	"	"	"	"
		2685m N	CP354	FWA	"	"	"	0.02	"	"	"	"	"
			CP356	FWA	"	"	"	<0.01	"	"	"	"	"
		2515m N	CP324	FWA	"	"	"	"	"	0.02	"	"	"
		2515m N	CP320	FWA	"	"	"	"	"	<0.01	"	"	"
		2410m N	CP309	FWA	"	"	"	"	"	"	"	"	"
		2310m N	CP293	FWA	"	"	"	"	"	"	"	"	"
		2550m N	AD931	FWQ	"	"	"	"	"	"	"	"	"
		2100m N	AD846	HWA	"	"	"	"	"	"	"	"	0.03
			AD842	HWA	"	"	"	"	"	"	"	"	<0.01
			AD839	HWA	"	"	"	"	"	"	"	"	"
			AD943	FWQ	"	"	"	"	"	"	"	"	"
			AD880	FWQ	"	"	"	"	"	"	"	"	"
			AD869	FWQ	"	"	"	"	"	"	"	"	"

Table 5.8: **July 1988 Konkola Mine Number 1 Shaft 810m Level Groundwater Chemical Composition in Milligrams Per Litre (Imperial College Geochemical Laboratory Results)**

Shaft	Mine Level	Location	BHID	Source (Aquifer)	Mg	Ca	Na	K	Al	Cu	Fe	Ba	Sr	Si
1	810M (2650ft)	2060m N	CP453	FWA	21.89	52.30	3.50	7.28	<0.06	0.05	0.27	0.16	0.17	8.53
		1950m N	CP442	FWA	7.22	18.73	2.06	5.39	"	<0.01	0.73	0.09	0.06	10.08
		1918m N	CP438	FWA	7.67	20.53	2.06	5.22	"	0.02	<0.04	0.09	0.06	9.92
		1800m N	AD949	HWA/ FWA	15.67	39.63	2.61	6.83	"	0.01	"	0.15	0.13	8.86
			CP430	FWA	9.94	24.90	2.03	5.61	"	<0.01	<0.04	0.11	0.82	9.19
		1755m N	CP426	FWA	9.28	23.06	2.03	5.44	"	<0.01	"	0.11	0.08	9.64
		1715m N	CP424	FWA	12.94	29.00	2.31	5.28	"	<0.01	"	0.10	0.10	10.17
			AD948	HWA	36.11	67.20	5.25	6.67	"	0.02	"	0.14	0.30	9.33
		1400m N	AD933	HWA	36.89	67.20	5.75	6.55	"	0.01	"	0.12	0.32	9.72
		1260m N	AD928	HWA	31.16	50.30	4.11	6.00	"	<0.01	"	0.09	0.26	10.28
		600m N	AD857	HWA	49.66	63.70	9.05	8.17	"	"	"	0.08	1.19	9.47
		500m N	AD865	HWA	36.11	83.80	6.97	7.11	"	"	"	0.08	1.04	10.08
		100m N	AD835	HWA	53.40	73.00	7.92	7.17	"	0.01	0.13	0.10	3.84	8.86
		800m S	AD919	HWA	36.22	62.10	3.08	7.39	"	<0.01	<0.04	0.10	0.31	9.78
			AD879	HWA	37.33	58.90	3.25	7.28	"	"	"	0.10	0.32	9.30
		880m S	AD895	HWA	36.66	52.40	3.44	6.94	"	0.01	"	0.09	0.31	8.78
		1020m S	AD899	HWA	32.22	54.40	2.58	7.17	"	<0.01	"	0.09	0.24	9.36
		1050m S	Fissure	FWA	4.39	11.33	4.56	4.50	"	<0.01	"	0.07	0.02	15.75
		1150m S	CP386	FWA	5.78	6.40	1.61	8.94	"	0.85	0.07	0.26	0.02	11.64
		1190m S	CP387	FWA	4.39	4.40	1.39	8.17	"	0.67	0.13	0.20	0.01	12.39

					P	S	Co	Pb	Zn	Li	Rb	Be	La	Ti
1	810M (2650ft)	2060m N	CP453	FWA	0.13	22.66	<0.01	<0.06	0.03	<0.04	0.33	<0.001	0.03	<0.07
		1950m N	CP442	FWA	0.24	3.67	"	"	<0.01	"	<0.22	0.001	0.01	"
		1918m N	CP438	FWA	0.13	3.44	"	"	0.02	"	"	<0.001	<0.01	"
		1800m N	AD949	HWA/ FWA	0.20	13.11	"	"	<0.01	"	"	"	0.02	"
			CP430	FWA	0.13	6.17	"	"	"	"	"	"	<0.01	"
		1755m N	CP426	FWA	0.22	4.56	"	"	"	"	"	"	"	"
		1715m N	CP424	FWA	0.15	3.83	"	"	0.02	"	"	"	0.01	"
			AD948	HWA	<0.04	35.89	"	"	<0.01	"	"	"	0.02	"
		1400m N	AD933	HWA	"	41.94	"	"	"	0.13	"	"	0.02	"
		1260m N	AD928	HWA	0.07	21.72	"	"	"	<0.04	"	"	0.02	"
		600m N	AD857	HWA	0.09	52.7	"	"	0.07	0.01	"	"	0.02	"
		500m N	AD865	HWA	0.13	60.5	"	"	<0.01	0.01	"	"	0.02	"
		100m N	AD835	HWA	0.07	60.5	"	"	"	<0.04	"	"	0.02	"
		800m S	AD919	HWA	<0.04	33.05	"	"	"	"	"	"	0.02	"
			AD879	HWA	"	33.44	"	"	"	"	"	"	0.02	"
		880m S	AD895	HWA	"	28.00	"	"	"	"	"	"	0.02	"
		1020m S	AD899	HWA	"	21.39	"	"	"	"	"	"	0.02	"
		1050m S	Fissure	FWA	0.53	2.11	"	"	"	"	"	0.001	<0.01	"
		1150m S	CP386	FWA	0.18	4.39	0.05	"	"					
		1190m S	CP387	FWA	0.16	2.22	0.03	"	"	"	"	<0.001	"	"

Table 5.8: July 1988 Konkola Mine Number 1 Shaft 810m Level Groundwater Chemical Composition in Milligrams Per Litre (Imperial College Geochemical Laboratory Results) (Contd)

Shaft	Mine Level	Location	BHID	Source (Aquifer)	V	Cr	Mo	Mn	Ni	Ag	Cd	B	As
1	810M	2060m N	CP453	FWA	<0.01	<0.01	0.03	<0.02	0.02	<0.01	<0.01	<0.01	<0.06
	(2650ft)	1950m N	CP442	FWA	"	"	(0.01	"	<0.01	"	"	"	"
		1918m N	CP438	FWA	"	"	"	"	"	"	"	"	"
		1800m N	AD949	HWA/ FWA	"	"	0.02	"	"	"	"	"	"
			CP430	FWA	"	"	<0.01	"	"	"	"	"	"
		1755m N	CP426	FWA	"	"	"	"	"	"	"	"	"
		1715m N	CP424	FWA	"	"	0.01	"	"	"	"	"	"
			AD948	HWA	"	"	0.02	"	"	"	"	"	"
		1400m N	AD933	HWA	"	"	0.03	"	"	"	"	"	"
		1260m N	AD928	HWA	"	"	0.04	"	"	"	"	"	"
		600m N	AD857	HWA	"	"	0.04	"	"	"	"	"	"
		500m N	AD865	HWA	"	"	0.04	"	"	"	"	"	"
		100m N	AD835	HWA	"	"	0.05	"	"	"	"	"	"
		800m S	AD919	HWA	"	"	0.02	"	"	"	"	"	"
			AD879	HWA	"	"	<0.01	"	"	"	"	"	"
		880m S	AD895	HWA	"	"	0.02	"	"	"	"	"	"
		1020m S	AD899	HWA	"	"	0.02	"	"	"	"	"	"
		1050m S	Fissure	FWA	"	"	<0.01	"	"	"	"	"	"
		1150m S	CP386	FWA	"	"	"	"	"	"	"	"	"
		1190m S	CP387	FWA	"	"	"	"	"	"	"	"	"

Table 5.9: **July 1988 Konkola Mine Number 1 Shaft 730m, 885m, 950m and 960m Level Groundwater Chemical Composition in Milligrams Per Litre**
(Imperial College Geochemical Laboratory Results)

Shaft	Mine Level	Location	BHID	Source (Aquifer)	Mg	Ca	Na	K	Al	Cu	Fe	Ba	Sr	Si
1	730m (2400ft)	1400m N	AD855	HWA	27.89	45.13	3.31	5.94	<0.06	0.01	0.31	0.08	0.21	10.83
	885m (2900ft)	482m N	CP445	FWA	28.77	68.70	6.03	7.89	"	0.03	<0.04	0.06	0.33	10.44
		440m N	CP441	FWA	28.39	62.90	5.36	8.05	"	0.02	"	0.06	0.34	9.92
		300m N	AD946	HWA	37.77	70.50	6.22	6.94	"	<0.01	"	0.07	0.66	10.33
		100m N	CP452	FWA	38.50	98.4	8.08	9.22	"	0.02	"	0.08	0.45	11.14
		25m N	CP455	FWA	43.11	106.2	13.58	9.67	"	0.07	"	0.12	0.51	11.08
		250m S	CP429	FWA	29.11	71.3	5.44	9.33	"	0.12	"	0.14	0.50	10.50
	950m (3120ft)	150m N	AD917	FWQ	9.78	27.76	3.42	8.89	"	0.02	0.11	0.14	0.10	10.11
		105m N	AD916	FWQ	7.61	20.53	2.58	8.22	"	0.01	<0.04	0.13	0.06	9.89
		10m N	AD925	FWQ	6.11	14.63	2.14	7.72	"	0.08	0.11	0.11	0.04	10.00
	960m (3150ft)	457m N	CP444	FWA	26.00	84.40	5.72	8.22	0.67	0.02	<0.04	0.06	0.34	11.30
		365m N	CP431	FWA	29.72	70.10	5.83	9.83	<0.06	<0.01	"	0.10	0.35	10.50
		255m S	CP443	FWA	16.28	36.43	3.08	7.44	"	0.24	"	0.19	0.17	10.69

					P	S	Co	Pb	Zn	Li	Rb	Be	La	Ti
1	730m (2400ft)	1400m N	AD855	HWA	0.11	15.17	<0.01	<0.06	0.02	0.01	<0.22	<0.001	0.02	<0.07
	885m (2900ft)	482m N	CP445	FWA	0.07	38.66	"	"	<0.01	<0.004	"	"	0.03	"
		440m N	CP441	FWA	0.07	33.44	"	"	0.02	0.01	"	"	0.02	"
		300m N	AD946	HWA	0.07	46.16	"	"	<0.01	<0.004	"	"	0.02	"
		100m N	CP452	FWA	0.04	66.7	"	"	"	"	"	"	0.03	"
		25m N	CP455	FWA	0.09	88.7	"	"	"	"	"	"	0.03	"
		250m S	CP429	FWA	0.11	56.7	"	"	"	"	"	"	0.02	"
	950m (3120ft)	150m N	AD917	FWQ	0.22	12.33	"	"	0.03	0.01	"	0.001	0.02	"
		105m N	AD916	FWQ	0.20	10.22	"	"	0.01	<0.004	"	<0.001	<0.01	"
		10m N	AD925	FWQ	0.16	7.05	"	"	0.04	"	"	"	"	"
	960m (3150ft)	457m N	CP444	FWA	<0.04	36.22	"	"	0.06	0.08	"	"	0.02	"
		365m N	CP431	FWA	"	37.83	"	"	<0.01	<0.004	"	"	<0.01	"
		255m S	CP443	FWA	0.20	16.94	"	"	0.02	"	"	"	"	"

					V	Cr	Mo	Mn	Ni	Ag	Cd	B	As
1	730m (2400ft)	1400m N	AD855	HWA	<0.01	<0.01	0.03	<0.02	<0.01	<0.01	<0.01	<0.01	<0.06
	885m (2900ft)	482m N	CP445	FWA	"	"	0.03	"	"	"	"	"	"
		440m N	CP441	FWA	"	"	0.02	"	"	"	"	"	"
		300m N	AD946	HWA	"	"	0.03	"	"	"	"	"	"
		100m N	CP452	FWA	"	"	0.04	0.03	"	"	"	"	"
		25m N	CP455	FWA	"	"	0.06	<0.02	"	"	"	"	"
		250m S	CP429	FWA	"	"	0.03	"	"	"	"	"	"
	950m (3120ft)	150m N	AD917	FWQ	"	"	<0.01	"	"	"	"	"	"
		105m N	AD916	FWQ	"	"	0.02	"	"	"	"	"	"
		10m N	AD925	FWQ	"	"	0.01	"	"	"	"	"	"
	960m (3150ft)	457m N	CP444	FWA	"	"	0.02	"	"	"	"	0.11	"
		365m N	CP431	FWA	"	"	0.03	"	"	"	"	<0.01	"
		255m S	CP443	FWA	"	"	0.02	"	"	"	"	"	"

Table 5.10: July 1989 Konkola Mine area Surface Water Chemical Composition in Milligrams Per Litre

(Mine Analytical Laboratory Results)

Source	Temp (^{o}C)	pH	TDS	Mg	Ca	Na	K	Al	Cu	Co	Cl	SO_4	HCO_3	SiO_2
Kafu River - Mine Water Works Pump Station	17.7	7.8	70	7	15	3.8	<0.1	<0.5	<0.1	<0.1	11.8	12	48	2
Mine Tailings Dam - Dam Spillway (Drainage Canal)	14.8	7.9	70	10	19	3.3	3.5	<0.5	<0.1	<0.1	9.9	30	57	4

Table 5.11: July 1989 Konkola Mine Number 3 Shaft Groundwater Chemical Composition in Milligrams Per Litre

(Mine Analytical Laboratory Results)

Shaft No.	Mine Level	Location	BH No.	Source (Aquifer)	Temp (°C)	pH	TDS	Mg	Ca	Na	K	Al	Cu	Co	Cl	SO_4	HCO_3	SiO_2
3	450m	150m W	ADN320	HWA	29.8	7.0	250	18	42	7.3	7.0	<0.5	<0.1	<0.1	2.4	55	102	2
	(1480ft)	720m W	ADN220	HWA	28.0	6.6	130	10	19	2.5	5.0	<0.5	<0.1	0.3	1.6	50	78	2
		1120m W	ADN206	HWA	26.0	7.0	250	23	39	3.5	7.5	<0.5	<0.1	<0.1	4.6	42	123	2
		1440m W	ADN286	HWA	24.3	7.5	240	28	31	3.3	5.0	<0.5	<0.1	<0.1	3.9	12	141	4
	488m	720m W	ADN358	HWA	26.2	7.3	280	26	42	5.5	8.0	<0.5	<0.1	<0.1	3.0	58	126	6
	(1600ft)	960m W	ADN358	HWA	26.0	7.1	250	23	35	4.0	7.0	<0.5	<0.1	0.2	2.8	57	120	2
	565m	500m W	Fissure	HWQ	28.1	6.6	210	9	41	8.0	10.0	<0.5	0.2	0.1	1.4	67	39	2
	(1850ft)	550m W	Fissure	FWA	30.0	7.2	240	17	35	7.5	7.0	<0.5	0.2	<0.1	3.4	107	102	2
		720m W	BPN247	HWA	27.1	7.3	280	26	42	6.5	7.5	<0.5	0.1	<0.1	4.3	70	135	2
			Fissure	FWA	28.0	7.4	300	26	37	8.0	8.0	<0.5	0.1	<0.1	4.2	96	123	4
		1340m W	ADN366	HWA	26.4	7.2	260	23	33	5.0	7.0	<0.5	<0.1	<0.1	3.4	53	120	2
			ADN363	HWA	26.4	7.2	250	24	34	4.2	7.0	<0.5	<0.1	<0.1	3.4	47	129	2
			AD362	FWA	28.5	7.1	280	25	45	7.0	8.0	<0.5	<0.1	0.1	3.2	74	114	2
		1500m W	BPN253	HWA	26.9	7.2	260	24	39	4.5	7.5	<0.5	<0.1	0.1	2.9	253	117	2
			BPN258	FWA	28.3	7.1	270	23	39	6.0	10.0	<0.5	0.1	<0.1	3.2	71	126	2
		1550m W	Fissure	FWA	27.7	7.4	270	22	44	6.0	10	<0.5	<0.1	0.2	4.9	70	51	2
		1800m W	BPN237	FWQ	27.1	6.2	130	7	18	3.5	9.0	<0.5	0.1	<0.1	2.6	35	81	2

Table 5.12: July 1989 Konkola Mine Number 1 Shaft Groundwater Chemical Composition in Milligrams Per Litre

(Mine Analytical Laboratory Results)

Shaft No.	Mine Level	Location	BH No.	Source (Aquifer)	Temp (°C)	pH	TDS	Mg	Ca	Na	K	Al	Cu	Co	Cl	SO_4	HCO_3	SiO_2
1	670m	3000m N	AD951	HWA	25.9	7.2	320	36	50	3.5	5.5	<0.5	<0.1	<0.1	0.9	89	144	2
	(2200ft)		AD952	HWA	26.0	7.3	350	36	52	2.5	5.5	<0.5	<0.1	0.1	1.1	124	144	2
		2950m N	CP394	FWA	27.1	6.9	190	17	31	3.5	5.3	<0.5	<0.1	<0.1	0.5	31	135	2
		2850m N	CP384	FWA	27.9	6.7	150	12	28	3.0	4.5	<0.5	<0.1	<0.1	1.4	21	96	2
		2782m N	CP363	FWA	28.0	6.7	170	14	29	2.0	4.5	<0.5	<0.1	<0.1	1.2	21	75	2
		2685m N	CP356	FWA	28.1	6.6	120	9	24	3.0	4.2	<0.5	<0.1	<0.1	0.9	12	90	2
			CP354	HWA/FWA	28.0	6.9	200	18	40	4.5	4.8	<0.5	<0.1	<0.1	0.9	43	87	2
		2100m N	AD839	HWA	24.9	7.3	260	30	39	5.0	4	<0.5	<0.1	<0.1	1.1	63	57	2
			AD842	HWA	25.2	7.3	280	32	47	4.0	5.0	<0.5	<0.1	<0.1	1.6	71	135	2
			AD869	FWQ	30.3	6.7	130	8	22	3.0	4.2	<0.5	<0.1	<0.1	1.7	22	126	6
			AD880	FWQ	29.0	6.4	120	6	19	4.5	4.2	<0.5	<0.1	<0.1	1.8	33	57	2
	810m	2220m N	CP470	FWA	28.0	6.6	230	22	44	3.5	6.0	<0.5	<0.1	<0.1	2.9	81	105	6
	(2650ft)	2100m N	CP457	FWA	27.9	6.9	250	23	46	4.0	4.5	<0.5	<0.1	<0.1	2.2	120	81	2
		2060m N	CP453	FWA	27.7	7.0	250	23	52	3.3	5.0	<0.5	<0.1	<0.1	2.4	61	114	2
		2000m N	AD956	HWA	24.7	7.4	340	44	65	5.9	4.0	<0.5	<0.1	0.2	1.8	104	141	4
			AD957	HWA	24.8	7.1	330	44	64	5.6	4.0	<0.5	<0.1	<0.1	2.0	97	144	6
			CP438	FWA	28.6	6.7	100	9	18	2.0	3.0	<0.5	<0.1	<0.1	2.6	14	48	4
		1875m N	CP436	FWA	28.5	6.8	90	7	15	2.0	3.0	<0.5	<0.1	<0.1	2.2	7	45	2
		1800m N	AD949	HWA/FWA	28.0	6.8	160	14	32	2,5	4,3	<0.5	<0.1	<0.1	4.6	22	78	6
			CP430	FWA	28.5	6.7	110	9	19	2.0	3.5	<0.5	<0.1	<0.1	2.9	6	57	2
		1715m N	AD948	HWA/FWA	25.4	7.4	320	39	51	5.2	4.5	<0.5	<0.1	<0.1	2.6	95	162	6
		1400m N	AD933	HWA	25.9	7.2	320	35	65	5.3	4.0	<0.5	<0.1	<0.1	2.2	86	156	4
		1260m N	AD928	HWA	24.7	7.3	270	34	43	4.6	4.0	<0.5	<0.1	<0.1	2.1	112	144	4
		500m N	AD865	HWA	24.8	7.7	440	47	102	8.6	4.5	<0.5	<0.1	<0.1	2.5	224	147	4
		100m N	AD835	HWA	25.8	7.5	460	46	79	8.0	5.9	<0.5	<0.1	<0.1	3.8	196	156	2
		800m S	AD879	HWA	26.1	7.3	340	35	51	3.5	6.0	<0.5	<0.1	<0.1	2.4	98	141	2
			AD906	HWA	25.3	7.4	290	34	43	4.0	5.0	<0.5	<0.1	<0.1	2.9	357	147	2
			AD910	HWA	25.6	7.4	310	24	40	4.0	5.5	<0.5	<0.1	<0.1	3.0	73	150	2
			AD919	HWA	25.8	7.3	340	24	56	3.0	6.0	<0.5	<0.1	<0.1	2.6	104	144	2
		880m S	AD895	HWA	25.3	7.4	310	34	50	3.5	5.8	<0.5	<0.1	<0.1	3.0	84	141	2
		1020m S	AD899	HWA	25.3	7.3	300	31	51	3.0	6.0	<0.5	<0.1	<0.1	2.1	146	141	2
		1150m S	CP386	FWA	25.7	6.2	70	6	6	1.5	7.0	<0.5	0.6	0.1	3.0	10	30	2
		1190m S	CP387	FWA	25.6	6.1	50	4	4	1.5	5.5	<0.5	0.3	0.1	2.1	14	27	2

Table 5.12: July 1989 Konkola Mine Number 1 Shaft Groundwater Chemical Composition in Milligrams Per Litre (Contd)

(Mine Analytical Laboratory Results)

Shaft No.	Mine Level	Location	BH No.	Source (Aquifer)	Temp (°C)	pH	TDS	Mg	Ca	Na	K	Al	Cu	Co	Cl	SO_4	HCO_3	SiO_2
1	885m	700m N	CP468	FWA	30.0	7.1	280	24	55	6.0	8.0	<0.5	<0.1	<0.1	6.1	88	132	2
	(2900ft)	650m N	CP464	FWA	30.2	7.1	280	24	50	10.0	9.0	<0.5	0.3	<0.1	6.2	108	132	2
		250m S	CP429	FWA	27.0	7.2	500	48	136	10.0	12.0	<0.5	<0.1	<0.1	7.6	379	108	2
	950m	150m N	AD917	FWQ	30.0	6.3	30	10	23	5.0	9.0	<0.5	<0.1	<0.1	6.1	46	78	2
	(3120ft)	105m N	AD916	FWQ	29.2	6.4	10	8	14	4.0	9.0	<0.5	<0.1	<0.1	4.2	35	60	8
		10m N	AD925	FWQ	28.5	6.3	90	7	13	3.0	8.0	<0.5	<0.1	0.2	4.2	36	48	4
	960m	457m N	CP444	FWA	30.5	7.3	30	29	62	7.0	9.0	<0.5	<0.1	<0.1	5.7	123	132	2
	(3150ft)	365m N	CP431	FWA	29.2	7.1	320	39	68	9.0	11.0	<0.5	<0.1	0.3	9.2	192	102	2

HWA = Hangingwall Aquifer
FWA = Footwall Aquifer
FWQ = Footwall Quartz Aquifer

Table 5.13: July 1989 Konkola Mine Surface Water Chemical Composition in Milligrams Per Litre (Imperial College Geochemistry Laboratory Results)

Source	Mg	Ca	Na	K	Al	Fe	Cu	Ba	Si	S
Kafue River										
- Mine Water Works Pump Station	7.39	13.57	7.05	0.33	<0.06	0.22	0.01	0.01	3.37	22.22
- Weir Downstream Golf Club	7.33	13.40	11.84	0.50	"	0.24	0.02	0.02	3.37	22.22
Lubengele Stream										
- Downstream Mine Tailings Dam	17.55	55.9	19.42	10.83	"	0.20	0.01	0.09	2.52	22.22
Mingomba Stream										
- Pump Station	7.50	10.97	0.94	0.33	"	<0.04	0.01	0.02	3.77	<0.11

	P	Sr	Li	Rb	Be	La	Ti	V	Cr	Mo
Kafue River										
- Mine Water Works Pump Station	<0.06	0.05	<0.004	<0.22	<0.001	0.01	0.01	<0.01	<0.01	<0.01
- Weir Downstream Golf Club	"	0.05	"	"	"	0.02	0.01	"	"	"
Lubengele Stream										
- Downstream Mine Tailings Dam	0.08	0.18	"	"	"	0.09	0.02	"	"	0.03
Mingomba Stream										
- Pump Station	<0.06	0.04	"	"	"	0.02	<0.01	"	"	<0.01

	Mn	Ni	Ag	Zn	Cd	B	Pb	As	Co
Kafue River									
- Mine Water Works Pump Station	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.06	<0.06	<0.01
- Weir Downstream Golf Club	"	"	"	"	"	"	"	"	"
Lubengele Stream									
- Downstream Mine Tailings Dam	0.08	"	"	0.01	"	"	"	"	"
Mingomba Stream									
- Pump Station	<0.01	"	"	<0.01	"	"	"	"	"

Table 5.14: **July 1989 Konkola Mine Number 3 Shaft 450m, 488m and 565m Level Groundwater Chemical Composition in Milligrams Per Litre**

(Imperial College Geochemical Laboratory Results)

Shaft	Mine Level	Location	BHID	Source (Aquifer)	Mg	Ca	Na	K	Al	Fe	Cu	Ba	Sr	Si
3	450m	1440m W	ADN286	HWA	28.94	42.16	2.91	5.72	<0.06	<0.04	<0.01	0.09	0.36	9.19
	(1480ft)	1120m W	ADN206	HWA	23.94	47.90	3.38	6.67	"	"	0.01	0.11	0.28	10.44
		720m W	ADN220	HWA	11.55	23.63	11.53	4.72	"	"	<0.01	0.06	0.15	11.97
		150m W	ADN320	HWA	20.89	50.70	6.27	6.94	"	"	0.01	0.08	0.31	11.40
	488m	960m W	ADN352	HWA	23.66	47.76	5.84	6.55	<0.06	<0.04	0.01	0.11	0.28	10.84
	(1600ft)	720m W	ADN358	HWA	26.33	53.90	7.43	7.00	"	"	0.01	0.11	0.28	11.06
	565m	1800m W	BPN237	FWQ	6.72	24.86	4.97	7.61	<0.06	<0.04	0.02	0.18	0.10	8.83
	(1850ft)	1550m W	Fissure	FWA	23.05	53.30	5.04	8.05	"	0.11	0.08	0.16	0.25	10.28
		1500m W	BPN253	HWA /FWA	25.39	52.20	4.43	7.50	"	<0.04	0.03	0.14	0.22	10.80
			BPN258	FWA	23.78	55.50	5.48	8.05	"	"	0.06	0.15	0.27	10.50
		1340m W	ADN362	FWA	24.89	57.00	10.93	7.67	"	"	0.04	0.13	0.29	10.59
			ADN366	HWA	26.11	53.50	9.89	6.83	"	"	0.01	0.11	0.29	10.76
			ADN363	HWA	24.83	50.30	8.10	6.72	"	"	0.01	0.11	0.30	10.74
		720m W	BPN247	HWA	18.39	44.43	9.30	6.89	"	"	0.04	0.11	0.24	11.16
			Fissure	FWA	26.83	55.10	11.11	7.00	"	"	0.02	0.12	0.30	10.87
		500m W	Fissure	FWQ	10.44	46.36	10.23	9.89	"	"	0.04	0.16	0.12	10.85

Shaft	Mine Level	Location	BHID	Source (Aquifer)	P	S	Co	Pb	Zn	Li	Rb	Be	La	Ti
3	450m	1440m W	ADN286	HWA	<0.06	4.72	<0.01	<0.06	0.02	0.01	<0.22	<0.001	0.02	0.01
	(1480ft)	1120m W	ADN206	HWA	0.06	14.05	"	"	<0.01	0.01	"	"	0.02	0.01
		720m W	ADN220	HWA	0.17	22.22	"	"	"	0.01	"	"	0.02	0.01
		150m W	ADN320	HWA	<0.06	19.33	"	"	0.01	0.02	"	"	0.02	0.01
	488m	960m W	ADN352	HWA	<0.06	22.22	<0.01	<0.06	0.01	0.01	<0.22	<0.001	0.02	0.01
	(1600ft)	720m W	ADN358	HWA	0.06	22.22	"	"	0.02	0.01	"	"	0.03	0.01
	565m	1800m W	BPN237	FWQ	0.14	22.22	<0.01	<0.06	<0.01	<0.01	<0.22	<0.001	0.02	0.01
	(1850ft)	1550m W	Fissure	FWA	0.11	21.16	"	"	"	0.01	"	"	0.02	0.01
		1500m W	BPN253	HWA /FWA	0.11	17.33	"	"	"	0.01	"	"	0.02	0.01
			BPN258	FWA	0.08	22.22	"	"	"	0.01	"	"	0.02	0.01
		1340m W	ADN362	FWA	0.08	22.22	"	"	"	0.01	"	"	0.02	0.01
			ADN366	HWA	<0.06	22.22	"	"	"	0.01	"	"	0.02	0.01
			ADN363	HWA	0.06	22.22	"	"	"	0.01	"	"	0.02	0.01
		720m W	BPN247	HWA	0.11	22.22	"	"	0.02	0.01	"	"	0.02	0.01
			Fissure	FWA	<0.06	22.22	"	"	0.02	0.01	"	"	0.02	0.01
		500m W	Fissure	FWQ	0.22	22.22	"	"	0.02	<0.01	"	"	0.02	0.01

Shaft	Mine Level	Location	BHID	Source (Aquifer)	V	Cr	Mo	Mn	Ni	Ag	Cd	B	As
3	450m	1440m W	ADN286	HWA	<0.01	<0.01	0.03	<0.01	<0.01	<0.01	<0.01	<0.01	<0.06
	(1480ft)	1120m W	ADN206	HWA	"	"	0.03	"	"	"	"	"	"
		720m W	ADN220	HWA	"	"	0.11	"	"	"	"	"	"
		150m W	ADN320	HWA	"	"	0.03	"	"	"	"	"	0.17
	488m	960m W	ADN352	HWA	<0.01	<0.01	0.03	<0.01	<0.01	<0.01	<0.01	<0.01	0.17
	(1600ft)	720m W	ADN358	HWA	"	"	0.03	"	"	"	"	"	"
	565m	1800m W	BPN237	FWQ	<0.01	<0.01	0.02	<0.01	<0.01	<0.01	<0.01	<0.01	0.17
	(1850ft)	1550m W	Fissure	FWA	"	0.01	0.03	"	"	"	"	"	0.36
		1500m W	BPN253	HWA /FWA	"	<0.01	0.03	"	"	"	"	"	0.33
			BPN258	FWA	"	"	0.03	"	"	"	"	"	0.36
		1340m W	ADN362	FWA	"	"	0.03	"	"	"	"	"	0.33
			ADN366	HWA	"	0.01	0.03	"	"	"	"	"	0.33
			ADN363	HWA	"	<0.01	0.03	"	"	"	"	"	0.36
		720m W	BPN247	HWA	"	"	0.02	"	"	"	"	"	<0.06
			Fissure	FWA	"	"	0.04	"	"	"	"	"	0.17
		500m W	Fissure	FWQ	"	"	0.22	"	0.01	"	"	"	0.37

Table 5.15: **July 1989 Konkola Mine Number 1 Shaft 670m Level Groundwater Chemical Composition in Milligrams Per Litre**

(Imperial College Geochemical Laboratory Results)

Shaft	Mine Level	Location	BHID	Source (Aquifer)	Mg	Ca	Na	K	Al	Fe	Cu	Ba	Sr	Si
1	670m (2200ft)	3000m N	AD951	HWA	34.16	60.40	4.18	7.17	<0.06	<0.04	<0.01	0.15	0.26	9.65
			AD952	HWA	36.66	65.40	4.92	7.11	"	"	"	0.11	0.32	9.48
			CP394	FWA	16.44	37.66	2.98	7.28	"	"	0.02	0.17	0.12	9.19
		2850m N	CP384	FWA	11.78	29.96	7.00	6.61	"	"	0.02	0.13	0.11	8.78
		2782m N	CP363	FWA	13.67	32.43	2.51	6.55	"	"	0.01	0.13	0.11	8.74
		2685n N	CP354	FWA	18.28	39.96	3.09	6.89	"	"	0.01	0.14	0.14	9.09
			CP356	FWA	8.44	22.73	2.41	5.83	"	"	0.01	0.10	0.09	8.42
		2100m N	AD842	HWA	31.27	51.70	3.87	6.61	"	"	<0.01	0.10	0.25	10.23
			AD839	FWA	29.39	47.20	3.51	5.83	"	"	"	0.09	0.23	10.43
			AD880	FWQ	6.00	21.63	4.16	6.39	"	"	"	0.09	0.10	10.09
			AD869	FWQ	8.11	24.83	2.87	6.50	"	"	0.01	0.09	0.10	9.23

					P	S	Co	Pb	Zn	Li	Rb	Be	La	Ti
1	670m (2200ft)	3000m N	AD951	HWA	<0.06	22.22	<0.04	<0.06	<0.01	0.01	<0.20	<0.001	0.03	0.02
			AD952	HWA	"	22.22	"	"	0.01	0.01	"	"	0.03	0.01
			CP394	FWA	0.14	9.89	"	"	<0.01	0.01	"	"	0.02	0.02
		2850m N	CP384	FWA	0.14	22.22	"	"	0.01	0.01	"	"	0.02	0.01
		2782m N	CP363	FWA	0.11	7.78	"	"	<0.01	0.01	"	"	0.02	0.01
		2685n N	CP354	FWA	0.11	14.94	"	"	"	0.01	"	"	0.03	0.01
			CP356	FWA	0.11	5.61	"	"	"	<0.004	"	"	0.01	0.01
		2100m N	AD842	HWA	0.10	22.22	"	"	0.01	0.01	"	"	0.02	0.01
			AD839	FWA	<0.06	15.67	"	"	0.01	0.01	"	"	0.02	0.01
			AD880	FWQ	0.14	10.83	"	"	"	0.01	"	"	0.02	0.01
			AD869	FWQ	0.11	7.28	"	"	"	0.01	"	"	0.02	0.01

					V	Cr	Mo	Mn	Ni	Ag	Cd	B	As
1	670m (2200ft)	3000m N	AD951	HWA	<0.01	<0.01	0.04	<0.01	<0.01	<0.01	<0.01	<0.01	<0.06
			AD952	HWA	"	"	0.04	"	"	"	"	"	"
			CP394	FWA	"	"	0.02	"	"	"	"	"	"
		2850m N	CP384	FWA	"	"	0.02	"	"	"	"	"	"
		2782m N	CP363	FWA	"	"	0.02	"	"	"	"	"	"
		2685n N	CP354	FWA	"	"	0.03	"	"	"	"	"	"
			CP356	FWA	"	"	0.01	"	0.01	"	"	"	"
		2100m N	AD842	HWA	"	0.01	0.04	"	0.02	"	"	"	"
			AD839	FWA	"	<0.01	0.03	"	<0.01	"	"	"	"
			AD880	FWQ	"	"	<0.01	"	"	"	"	"	"
			AD869	FWQ	"	"	0.01	"	"	"	"	"	"

Table 5.16: July 1989 Konkola Mine Number 1 Shaft 810m Level Groundwater Chemical Composition in Milligrams Per Litre

(Imperial College Geochemical Laboratory Results)

Shaft	Mine Level	Location	BHID	Source (Aquifer)	Mg	Ca	Na	K	Al	Fe	Cu	Ba	Sr	Si
1	810m	2100m N	CP457	FWA	21.89	50.70	7.07	7.22	<0.06	<0.04	0.02	0.15	0.16	8.37
	(2650ft)	2000m N	AD956	HWA	37.39	62.60	5.85	6.44	"	"	0.01	0.13	0.41	9.80
			AD957	HWA	38.61	61.90	5.70	6.44	"	"	0.01	0.11	0.37	10.02
		1915m N	CP438	FWA	7.39	18.00	6.51	5.17	"	"	0.02	0.08	0.06	9.88
		1875m N	CP436	FWA	6.44	15.77	1.84	4.89	"	"	0.01	0.08	0.05	9.84
		1800m N	CP430	FWA	8.28	19.96	1.91	5.06	"	"	0.02	0.09	0.06	9.89
			AD949	HWA/ FWA	13.00	32.16	2.36	6.22	"	"	0.01	0.13	0.11	9.38
		1715m N	AD948	HWA	35.55	60.40	5.06	6.72	"	"	0.01	0.14	0.29	9.78
			CP424	FWA	6.72	16.63	1.80	4.94	"	"	0.01	0.09	0.06	9.85
			AD945	HWA/ FWA	35.72	60.90	5.10	6.83	"	0.18	0.02	0.15	0.30	9.54
		1400m N	AD933	HWA	32.55	61.3	5.11	6.44	"	<0.04	0.02	0.11	0.29	10.15
		800m S	AD919	HWA	36.89	62.60	3.24	7.40	"	"	0.02	0.10	0.33	9.60
			AD906	HWA	38.00	54.10	3.62	7.11	"	"	0.02	0.10	0.34	9.01
			AD910	HWA	40.88	50.30	4.00	6.83	"	"	0.02	0.10	0.46	8.62
			AD879	HWA	39.11	61.30	3.53	7.55	"	"	0.02	0.10	0.35	9.36
		880m S	AD895	HWA	33.66	55.10	5.83	7.28	"	"	0.03	0.10	0.26	9.10
		1020m S	AD899	HWA	37.89	53.70	3.71	7.11	"	"	0.02	0.10	0.33	8.70
		1150m S	CP386	FWA	5.89	6.23	2.11	9.17	"	"	0.65	0.27	0.02	11.41
		1190m S	CP387	FWA	4.39	4.30	1.47	8.28	"	"	0.59	0.20	0.01	12.02

Shaft	Mine Level	Location	BHID	Source (Aquifer)	P	S	Co	Pb	Zn	Li	Rb	Be	La	Ti
1	810m	2100m N	CP457	FWA	0.08	22.22	<0.01	<0.06	0.02	<0.01	<0.22	<0.001	0.02	0.01
	(2650ft)	2000m N	AD956	HWA	<0.06	22.22	"	"	<0.01	0.01	"	"	0.03	0.01
			AD957	HWA	"	22.22	"	"	0.01	0.01	"	"	0.03	0.02
		1915m N	CP438	FWA	0.22	22.22	"	"	<0.01	0.01	"	"	0.01	0.01
		1875m N	CP436	FWA	0.19	2.79	"	"	"	0.01	"	"	0.01	<0.01
		1800m N	CP430	FWA	0.19	2.94	"	"	"	0.01	"	"	0.02	"
			AD949	HWA/ FWA	0.14	8.50	"	"	"	0.01	"	"	0.02	0.01
		1715m N	AD948	HWA	<0.06	22.22	"	"	0.01	0.01	"	"	0.03	0.02
			CP424	FWA	0.19	1.94	"	"	<0.01	0.01	"	"	0.01	<0.01
			AD945	HWA/ FWA	0.06	22.22	"	"	0.01	0.01	"	"	0.02	0.02
		1400m N	AD933	HWA	<0.06	22.22	"	"	<0.01	0.01	"	"	0.03	0.02
		800m S	AD919	HWA	"	22.22	"	"	<0.01	0.01	"	"	0.03	0.01
			AD906	HWA	"	22.22	"	"	"	0.01	"	"	0.03	0.01
			AD910	HWA	"	22.22	"	"	"	0.01	"	"	0.02	0.01
			AD879	HWA	"	22.22	"	"	"	0.01	"	"	0.03	0.01
		880m S	AD895	HWA	"	22.22	"	"	"	0.01	"	"	0.02	0.01
		1020m S	AD899	HWA	"	22.22	"	"	"	0.01	"	"	0.02	0.01
		1150m S	CP386	FWA	0.11	4.22	0.06	"	0.01	<0.01	"	"	<0.01	<0.01
		1190m S	CP387	FWA	<0.06	2.00	0.04	"	<0.01	"	"	"	"	"

Shaft	Mine Level	Location	BHID	Source (Aquifer)	V	Cr	Mo	Mn	Ni	Ag	Cd	B	As
1	810m	2100m N	CP457	FWA	<0.01	<0.01	0.02	<0.01	<0.01	<0.01	<0.01	<0.01	<0.06
	(2650ft)	2000m N	AD956	HWA	"	"	0.03	"	"	"	"	"	"
			AD957	HWA	"	0.01	0.04	"	0.01	"	"	"	"
		1915m N	CP438	FWA	"	<0.01	<0.01	"	<0.01	"	"	"	"
		1875m N	CP436	FWA	"	"	"	"	"	"	"	"	"
		1800m N	CP430	FWA	"	0.01	0.01	"		"	"	"	"
			AD949	HWA/ FWA	"	<0.01	0.01	"	<0.01	"	"	"	"
		1715m N	AD948	HWA	"	0.01	0.03	"	"	"	"	0.02	"
			CP424	FWA	"	<0.01	<0.01	"	"	"	"	<0.01	"
			AD945	HWA/ FWA	"	"	0.04	"	0.01	"	"	"	"
		1400m N	AD933	HWA	"	0.01	0.04	"	0.02	"	"	"	"
		800m S	AD919	HWA	"	<0.01	0.04	"	0.01	"	"	"	"
			AD906	HWA	"	0.01	0.04	"	<0.01	"	"	"	"
			AD910	HWA	"	0.01	0.04	"	"	"	"	"	"
			AD879	HWA	"	0.01	0.04	"	"	"	"	"	"
		880m S	AD895	HWA	"	<0.01	0.04	"	"	"	"	"	"
		1020m S	AD899	HWA	"	"	0.03	"	"	"	"	"	"
		1150m S	CP386	FWA	"	"	<0.01	"	"	"	"	"	"
		1190m S	CP387	FWA	"	"	"	"	"	"	"	"	"

Table 5.17: **July 1989 Konkola Mine Number 1 Shaft 885m, 950m and 960m Level Groundwater Chemical Composition in Milligrams Per Litre**

(Imperial College Geochemical Laboratory Results)

Shaft	Mine Level	Location	BHID	Source (Aquifer)	Mg	Ca	Na	K	Al	Fe	Cu	Ba	Sr	Si
1	885m (2900ft)	700m N	CP468	FWA	23.28	58.2	4.18	7.55	<0.06	<0.04	0.02	0.06	0.24	9.04
		650m N	CP464	FWA	23.22	57.9	4.00	7.50	"	"	0.01	0.06	0.25	8.93
		250m N	CP429	FWA	35.66	97.8	6.99	10.28	"	0.20	<0.01	0.13	0.89	9.82
	950m (3120ft)	150m N	AD917	FWQ	9.72	26.93	3.39	8.50	<0.06	<0.04	0.01	0.14	0.10	9.70
		105m N	AD916	FWQ	8.05	21.16	5.41	8.78	"	"	0.01	0.14	0.07	9.83
		10m N	AD925	FWQ	6.33	14.70	2.11	7.83	"	"	<0.01	0.12	0.05	9.70
	960m (3150ft)	457m N	CP444	FWA	27.11	69.10	6.03	8.55	<0.06	<0.04	0.01	0.07	0.34	10.28
		365m N	CP431	FWA	34.77	86.70	7.67	10.67	"	"	0.01	0.12	0.38	10.04

Shaft	Mine Level	Location	BHID	Source (Aquifer)	P	S	Co	Pb	Zn	Li	Rb	Be	La	Ti
1	885m (2900ft)	700m N	CP468	FWA	0.83	22.22	<0.01	<0.06	0.01	0.01	<0.22	<0.001	0.03	0.01
		650m N	CP464	FWA	<0.06	22.22	"	"	<0.01	0.01	"	"	0.03	0.01
		250m N	CP429	FWA	0.17	22.22	0.01	"	<0.01	0.01	"	"	0.04	0.02
	950m (3120ft)	150m N	AD917	FWQ	0.14	11.50	<0.01	<0.06	<0.01	<0.01	<0.22	<0.001	0.02	0.01
		105m N	AD916	FWQ	<0.06	22.22	0.01	"	"	<0.01	"	"	0.01	0.01
		10m N	AD925	FWQ	0.17	6.94	<0.01	"	"	0.01	"	"	<0.01	0.01
	960m (3150ft)	457m N	CP444	FWA	<0.06	22.22	<0.01	<0.06	0.01	0.01	<0.22	<0.001	0.03	0.02
		365m N	CP431	FWA	0.08	22.22	"	"	<0.01	0.01	"	"	0.03	0.02

Shaft	Mine Level	Location	BHID	Source (Aquifer)	V	Cr	Mo	Mn	Ni	Ag	Cd	B	As
1	885m (2900ft)	700m N	CP468	FWA	<0.01	<0.01	0.03	<0.01	0.01	<0.01	<0.01	<0.01	<0.06
		650m N	CP464	FWA	"	"	0.03	"	0.01	"	"	"	"
		250m N	CP429	FWA	"	"	0.05	0.07	0.02	"	"	"	"
	950m (3120ft)	150m N	AD917	FWQ	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.06
		105m N	AD916	FWQ	"	"	0.01	"	"	"	"		
		10m N	AD925	FWQ	"	"	<0.01	"	"	"	"		
	960m (3150ft)	457m N	CP444	FWA	<0.01	<0.01	0.04	<0.01	0.02	<0.01	<0.01	<0.01	<0.06
		365m N	CP431	FWA	0.01	0.01	0.04	"	0.02	"	"	"	"

Table 5.20: Escherichia Coli Concentration in Konkola Mine Waters: July 1988

Location		Probable Number of E. Coli Bacilli per 100ml of Water
Kafue River	- Pump Station	40
	- Weir	180+
	- downstream Kakosa confluence	35
Lubengele Stream	- Upstream Kawama Township	13
Mingomba Stream	- Muwa Farm	35
Kafue River	- Chimfunshi	13
Konkola Stream	- Chimfunshi Bridge	40

Number 1 Shaft Underground Workings:

Level	Location	Borehole	Aquifer	Probable Number of E. Coli Bacilli per 100ml of Water
670m (2200ft)	1475m S	AD798	FWA/HWA	0
	1400m S	CP164	FWA/HWA	0
	2100m N	AD943	FWQ	0
	2150m N	AD839	HWA	0
	2515m N	CP324	FWA	0
	2550m N	AD931	FWQ	0
	2740m N	CP363	FWA	0
	3000m N	AD947	HWA	0
810m (2650ft)	1190m S	CP387	FWA	0
	1020m S	AD899	HWA	1
	800m S	AD919	HWA	0
	800m S	AD879	HWA	0
	800m S	AD882	HWA	0
	880m S	AD895	HWA	0
	800m S	AD899	HWA	0
	800m S	AD910	HWA	180+
	500m N	AD870	HWA	1
	1260m N	AD928	HWA	0
	1715m N	AD948	FWA/HWA	0
885m (2900ft)	250m S	CP429	FWA	0
885m (2900ft)	100m N	CP452	FWA	1
	300m N	AD946	FWA/HWA	20
	482m N	CP445	FWA	17
950m (3120ft)	10m N	AD925	FWQ	0
	150m N	AD917	FWQ	0
960m (3150ft)	457m N	CP444	FWA	0
	255m S	CP443	FWA	0

Number 3 Shaft Underground Workings:

Level	Location	Borehole	Aquifer	Probable Number of E. Coli Bacilli per 100ml of Water
565m (1850ft)	Anticline Axis Fissure	BPN237	FWQ	0
	1250m W	ADN362	FWA/HWA	3
	720m W	BPN247	HWA	5
	500m W	Fissure	FWQ	20

FWQ = Footwall Aquifer
FWA = Footwall Aquifer
HWA = Hangingwall Aquifer

Table 5.21: Escherichia Coli Concentration in Konkola Mine Waters: July 1989

Location		Probable Number of E. Coli Bacilli Per 100ml of Water
Kafue River	- Pump Station	30
	- Weir	35
Lubengele	- Konkola Rd	25

Number 1 Shaft Underground Workings:

Level	Location	Borehole	Aquifer	Probable Number of E. Coli Bacilli Per 100ml of Water
810m (2650ft)	100m N	AD835	HWA	0
	500m N	AD865	HWA	3
	1260m N	AD928	HWA	0
	1400m N	AD933	HWA	0
	2220m N	CP470	FWA	0
	2000m N	AD956	HWA	0
	2000m N	AD957	HWA	0
	1915m N	CP438	FWA	0
	1875m N	CP436	FWA	0
	1800m N	CP430	FWA	0
	1800m N	AD949	HWA	0
	1150m S	CP386	FWA	0
	1190m S	CP387	FWA	0
	800m S	AD919	HWA	0
	800m S	AD906	HWA	0
	800m S	AD879	HWA	0
	1020m S	AD899	HWA	0
	880m S	AD895	HWA	0
670m (2200ft)	2100m N	AD850	FWQ	0
	2150m N	AD839	HWA	0
	3000m N	AD951	HWA	0
	3000m N	AD952	HWA	0
	3000m N	CP394	FWA	0
	2850m N	CP384	FWA	0
	2740m N	CP363	FWA	0
	2685m N	CP354	FWA	0
	2685m N	CP356	FWA	0
885m (2900ft)	250m S	CP429	FWA	0
	650m S	CP464	FWA	0
	700m N	CP468	FWA	0
960m (3150ft)	365m N	CP431	FWA	2
	457m N	CP444	FWA	2
950m (3120ft)	150m N	AD917	FWQ	0
	105m N	AD916	FWQ	0
	10m N	AD925	FWQ	0

Table 5.21: Escherichia Coli Concentration in Konkola Mine Waters: July 1989 (contd)

Number 3 Shaft Underground Workings:

Level	Location	Borehole	Aquifer	Probable Number of E. Coli Bacilli Per 100ml of Water
565m (1850ft)	500m W Fold Axis	Fissure	FWQ	0
	Fissure	BPN237	FWQ	0
	1550m W	Fissure	FWA	0
	1500m W	BPN253	HWA	0
	1500m W	BPN258	HWA	0
	1340m W	ADN362	HWA	0
	1340m W	ADN366	HWA	0
	1340m W	AD363	HWA	0
	700m W	Fissure	FWA	0
	725m W	BPN247	HWA	0
	725m W	Fissure	FWA	0
450m (1480ft)	1440m W	ADN286	HWA	0
	1120m W	AD206	HWA	0
	150m W	ADN320	HWA	0
	720m W	ADN220	HWA	0
488m (1600ft)	960m W	ADN352	HWA	0
	700m W	ADN358	HWA	0

Table 5.22: Residual Xanthate in Konkola Mine Waters: July 1988 & July 1989

Location				Residual Xanthate Concentration (mg/l)	
				July 1988	July 1989
Tailings Dam Area:					
Tailings effluent pipe from the concentrator				1.050	0.250
Lubengele Stream	- Opposite No. 3 Shaft			0.670	0.500
	- Konkola Road bridge			0.450	0.480
Tailings Drainage Canal	- Drainage Valve			0.600	0.350
Kafue River	- Pump Station			0.790	0.900
	- Tailings Drainage Canal			0.450	
	- Weir			0.140	
Number 1 Shaft Underground workings:					
Level	Location	Borehole	Aquifer		
670m (2200ft)	2150m N	AD839	HWA	0.025	0.025
	3000m N	AD947	HWA	0.025	
	2740m N	CP363	FWA	0.019	0.015
	3000m N	Fissure	FWA	0.050	
	2685m N	CP356	FWA	0.019	0.015
	2515m N	CP324	FWA	0.025	
	2310m N	CP293	FWA	0.025	
	2550m N	AD931	FWQ	0.038	0.100
	2100m N	AD943	FWQ	0.025	0.100
	1475m S	AD798	FWA/HWA	0.000	
	1400m S	CP164	FWA	0.000	
810m (2650ft)	1260m N	AD928	HWA	0.000	0.340
	500m N	AD870	HWA	0.000	0.010
	800m S	AD910	HWA	0.000	0.000
	1020m S	AD899	HWA	0.000	0.100
	1918m N	CP438	FWA	0.000	0.000
	1775m N	CP421	FWA	0.000	
	1715m N	CP424	FWA	0.000	
	1715m N	AD948	FWA/HWA	0.000	0.100
Number 3 Shaft Underground Workings:					
Level	Location	Borehole	Aquifer		
450m (1480ft)	3275m N	ADN265	HWA	0.000	
	1120m W	ADN206	HWA	0.10	0.150
	720m W	ADN220	HWA	0.050	0.050
	150m W	ADN320	HWA	0.000	0.010
488m (1600ft)	720m W	ADN358	HWA	0.050	0.080
565m (1850ft)	720m W	BPN247	HWA	0.000	0.010
	1340m W	ADN362	FWA/HWA	0.075	0.080
	1520m W	Fissure	FWA	0.050	
	500m W	Fissure	FWQ	0.025	
	Fold Axis Fissure	BPN237	FWQ	0.000	

FWQ = Footwall Quartzite Aquifer
FWA = Footwall Aquifer
HWA = Hangingwall Aquifer

MATHEMATICAL COMPUTATIONS FOR AQUIFERS
WATER MIXING PROPORTIONS

Quantification of Mixing Proportions

Using the average concentration values of the major dissolved ions in Konkola Mine waters, an estimate was made of the proportions of Hangingwall Aquifer water in boreholes drilled in the Footwall Aquifer. This research has clearly shown that the Footwall Aquifer water is a mixture of the Hangingwall Aquifer and Footwall Quartzite Aquifer waters.

If two waters A and B mix to produce C, then the proportion of B in C can be computed as follows:

$$Bx + A(1-x) = C \quad \text{(Mazor 1991)}$$

Using the average values for Mg, Ca, HCO_3 and TDS obtained from the July 1989 field studies, the computations are as follows:

TABLE 5.24a: AVERAGE CONCENTRATION OF DISSOLVED IONS IN MINE WATERS IN JULY 1989

Mine Level	Aquifer	Dissolved ion concentration in mg/l			
		Mg	Ca	HCO_3	TDS
565m (1850ft)	Hangingwall	24.25	37.0	125.25	262
North Limb Area		22.6	40.0	103.2	272
	Footwall Quartzite	8.0	29.5	60.0	170
610m (2200ft)	Hangingwall	33.5	47.0	120.0	302.5
North Area	Footwall	14.0	30.4	96.6	166.0
	Footwall Quartzite	7.0	20.5	91.5	125.0
810m (2650ft)	Hangingwall	41.7	69.7	148.0	360.0
	Footwall	15.5	32.3	75.0	171.7
	Footwall Quartzite	7.6	18.6	76.8	84.2

Calculations for the above Table

1. Proportion of Hangingwall Aquifer water in Footwall Aquifer water on 565m (1850ft) mine level, in July 1989.

   (a) Mg:

$$\begin{aligned} 24.25x + 8(1-x) &= 22.6 \\ 24.25x + 8 - 8x &= 22.6 \\ 15.75x &= 14.6 \\ x &= 0.93 \end{aligned}$$

(b) Ca:

$$
\begin{aligned}
37x + 29.5(1\text{-}x) &= 40 \\
37x + 29.5 - 29.5x &= 40 \\
7.5x &= 10.5 \\
x &= 1.4
\end{aligned}
$$

(c) HCO_3:

$$
\begin{aligned}
125.25x + 60(1\text{-}x) &= 103.2 \\
125.25x + 60 - 60x &= 103.2 \\
65.25x &= 43.2 \\
x &= 0.66
\end{aligned}
$$

(d) TDS:

$$
\begin{aligned}
262x + 170(1\text{-}x) &= 272 \\
262x + 170 - 170x &= 272 \\
92x &= 102 \\
x &= 1.1
\end{aligned}
$$

Average $= 1.025 \approx 1.0$

This figure implies that on this level practically all the Footwall Aquifer water came from the Hangingwall Aquifer water at that time. This is the area that lies directly below and adjacent to the Mine Tailings Dam and within the Luansobe and the Cross Anticline Axis fault fissure zones.

2. Proportion of Hangingwall Aquifer water in Footwall Aquifer water on 610m (2200ft) mine level north area, in July 1989.

(a) Mg:

$$
\begin{aligned}
33.5x + 7(1\text{-}x) &= 14 \\
33.5x + 7 - 7x &= 14 \\
26.5x &= 7 \\
x &= 0.26
\end{aligned}
$$

(b) Ca:

$$\begin{aligned} 47x + 20.5(1-x) &= 30.4 \\ 47x + 20.5 - 20.5x &= 30.4 \\ 26.5x &= 9.9 \\ x &= 0.37 \end{aligned}$$

(c) HCO_3:

$$\begin{aligned} 120x + 91.5(1-x) &= 96.6 \\ 120x + 91.5 - 91.5x &= 96.6 \\ 28.5x &= 5.1 \\ x &= 0.19 \end{aligned}$$

(d) TDS:

$$\begin{aligned} 302.5x + 125(1-x) &= 166 \\ 302.5x + 125 - 125x &= 166 \\ 177.5x &= 41 \\ x &= 0.23 \end{aligned}$$

Average = 0.275 $\approx$ 0.28

Therefore, about 28% of the Footwall Aquifer water in this area came from the Hangingwall Aquifer water at that time.

3. Proportion of Hangingwall Aquifer water in Footwall Aquifer water on 810m (2650ft) mine level north area, in July 1989.

(a) Mg:

$$\begin{aligned} 41.7x + 7.6(1-x) &= 15.5 \\ 41.7x + 7.6 - 7.6x &= 15.5 \\ 34.1x &= 7.9 \\ x &= 0.23 \end{aligned}$$

(b) Ca:

$$\begin{aligned} 69.7x + 18.6(1-x) &= 32.3 \\ 69.7x + 18.6 - 18.6x &= 32.3 \\ 51.1x &= 13.7 \\ x &= 0.26 \end{aligned}$$

(c) HCO_3:

$$\begin{aligned} 148x + 76.8(1-x) &= 75 \\ 148x + 76.8 - 76.8x &= 75 \\ 71.2x &= -1.8 \\ x &= -0.025 \end{aligned}$$

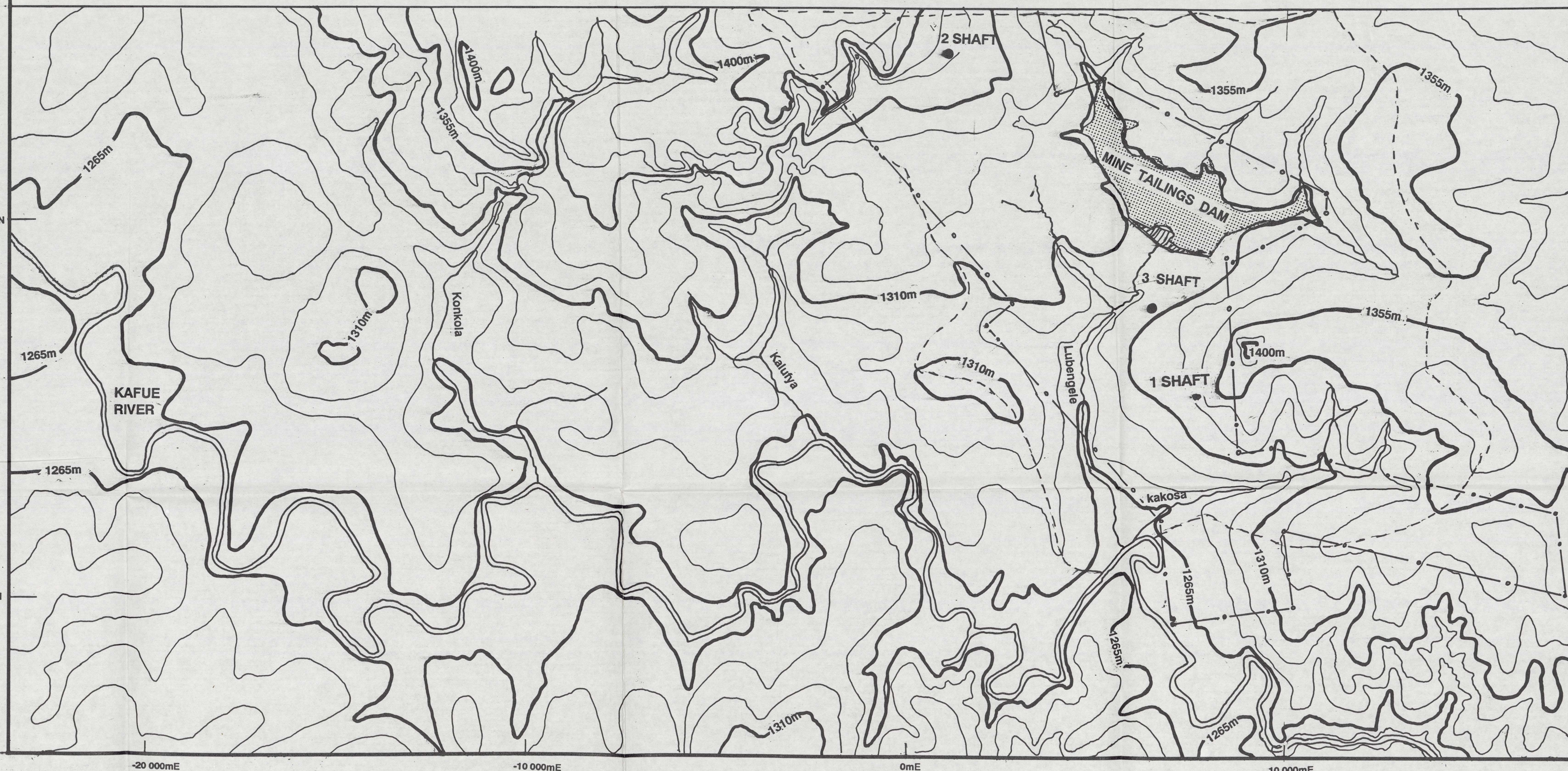
(d) TDS:

$$\begin{aligned} 360x + 84.2(1-x) &= 171.7 \\ 360x + 84.2 - 84.2x &= 171.7 \\ 275.8x &= 87.5 \\ x &= 0.3 \end{aligned}$$

Average = 0.2

Therefore, about 20% of the Footwall Aquifer water in this area came from the Hangingwall Aquifer water at that time.

Figure 3.2 : TOPOGRAPHIC MAP OF KONKOLA MINE AREA



SCALE
0 1 2 3 4 5 km

KONKOLA MINE (LUBENGELE) TOPOGRAPHIC CATCHMENT AREA BOUNDARY

MINING LICENCE AREA BOUNDARY

CONTOUR INTERVAL = 15metres

N

Figure 3.3: GEOLOGY MAP OF KONKOLA MINE AREA