Ore estimation and selection of ~~underground~~ mining methods

for

UNDERGROUND

some copper deposits.

by

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Abstract

This thesis analyses surface and underground drilling and chip sampling data collected from Khetri Copper Complex, India. It is suggested that systematic underground diamond drilling will provide a satisfactory basis for reserve estimation (using standard geostatical techniques) and mine planning. The current practice of chip sampling of mine workings could be reduced significantly. However, it still needs to be carried out as a check on underground drilling and to assist grade control.

Cutoff grades based on the mines' operating cost statistics and local price are established for three different stoping methods - sublevel, down-the-hole (DTH) and cut and fill. Optimum stope boundaries are located for each method using incremental analysis on kriged panels of 1m x 1m along the cross section of the orebody. The maximum operating margin is used to define the optimum position. The methodology provides a practical approach to both method selection and optimum stope design.

In the final part of the thesis stope layouts are modelled using computer graphics. This allows the investment in the stope development to be estimated for each of the extraction techniques. The economics of each method is then evaluated for a section of Khetri mine.

Although the research is based on two mines in India, the proposed methodology has general application to steeply dipping underground copper mines.

To my wife Shashi

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List of abbreviations

AM - Automated mapping.
ANFO - Ammonium nitrate fuel oil mixture.
APCOM - International symposium on the application of computers in operational research in the mineral industry.
(C) - The transition amplitude.
CAD - Computer-aided design.
(Co) - The nugget variance.
CS - Chip sampled assay values.
cu.m - Cubic metre
DCFROR - Discounted cashflow rate of return.
DK - Disjunctive kriging.
DTH - Down-the-hole.
GSI - Geological Survey of India.
HCL - Hindustan Copper Ltd., India.
HZL - Hindustan Zinc Ltd., India.
IBM - Indian Bureau of Mines.
IC - Imperial College of Science and Technology, London.
ICC - Indian Copper Complex.
ICCC - Imperial College Computer Centre.
IDW - Inverse square distance weighting.
IMM - The Institute of Mining and Metallurgy, London.
KCC - Khetri Copper Complex, India.
MMTCI - Metal Minerals Trading Corporation of India.
NPV - Net present value.
OMS - Output per manshift or Tonnes/manshift ratio.
RSM - Royal School of Mines, London.
SBH - Surface borehole.
UBH - Underground borehole .
WIP - Work in progress.

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Chapter 1

1. Introduction

The basic objectives in selecting a method to mine a particular deposit is to design an ore extraction system that will meet certain technical and economic criteria. This can be interpreted as aiming for maximum extraction with safe working conditions at low cost and maximum productivity.

In some circumstances the geological environment surrounding the deposit may be quite distinctive. This may dictate one particular method and an immediate exclusion of other methods. In such cases method selection is limited to adapting and refining the general method to suit that orebody. In other situations, the conditions may favour the application of several methods which must be evaluated and compared.

1.1 The interrelated factors in planning a mining method

There are certain interrelated factors that need consideration in planning a mining method:

In an underground mining situation, once the plans are agreed and development commences for ore extraction system, it is difficult and costly to change to an alternative method. In most cases, only minor changes can be made. For example it is virtually impossible to alter the vertical distance between levels once the shaft stations have been cut.

A project requires several years to reach its production stage and it is expected to produce ore for many years after that. While the basic principles of a mining method can be expected to remain the same, developments in the machinery and its utilization need an up-to-date knowledge of latest developments in mining techniques and a feeling for future trends. This can be taken care of while planning a mining method by building in flexibilities in the system.

An effective evaluation of mining methods depends upon the information available. Rarely is it possible to undertake more than a preliminary design when this information is based on boreholes drilled from surface. Final development plans and stope design is normally based on information derived from underground excavations.

Normally the boundaries of the mineralisation are not distinct. To define and map an orebody, it is necessary to establish a cutoff grade. Under typical conditions, gradually lowering the cutoff grade causes the estimated ore reserves to increase from a narrow high-grade vein to a massive low grade orebody. The relationship between production capacity, ore grade and available reserves, are factors that must be included in the selection of a mining method.

In selecting a mining method, the anticipated costs of mining exert a major influence. However, there are considerations other than simply finding the least costly procedure of excavating the rock. The characteristics and advantages of different mining methods must also be considered.

Mining techniques have become totally dependent upon machines of various types. Selecting a machine for a given type of work and matching its capacity to the required output have become two important tasks in mine planning and evaluation.

The factors discussed above, indicate that selection of a mining method needs a consideration of various interrelated factors and examination of various options and alternatives. The selection of underground mining method, to yield desired results, is thus a distinct challenge (problem) in the mining industry and warrants consistent research.

1.2 The use of computers in mine planning and design:

A mining method can be designed either manually or with the aid of a computer. The design process will include both conceptual aspects and routine processes. The manipulation and modification of large data sets, repetitive calculations, both simple and complex, numerous geological and engineering drawings, the tabulation of physical quantities, are some of the routine processes. Undertaking this part of the design process with the aid of computers would reduce labour, time and occurrence of arithmetical errors. Further, such an approach would permit spending more time on the conceptual aspects of method design which ultimately, should lead to improved designs.

1.3 Research objective:

The objective of this research has been to develop a methodology to select an appropriate stoping method for underground mining of a copper deposit. It involves the following:

* *Estimation of the mineral inventory from the basic geological information*

* *Formulating a basis for the cutoff grade decisions to predict mineable ore reserves*

* *Evaluating stope boundaries for different stoping methods, using cutoff grade and incremental analysis criteria*

* *Developing algorithms for designing different mining methods and their economic evaluation.*

A flow diagram of the proposed methodology is outlined in fig. 1.1.

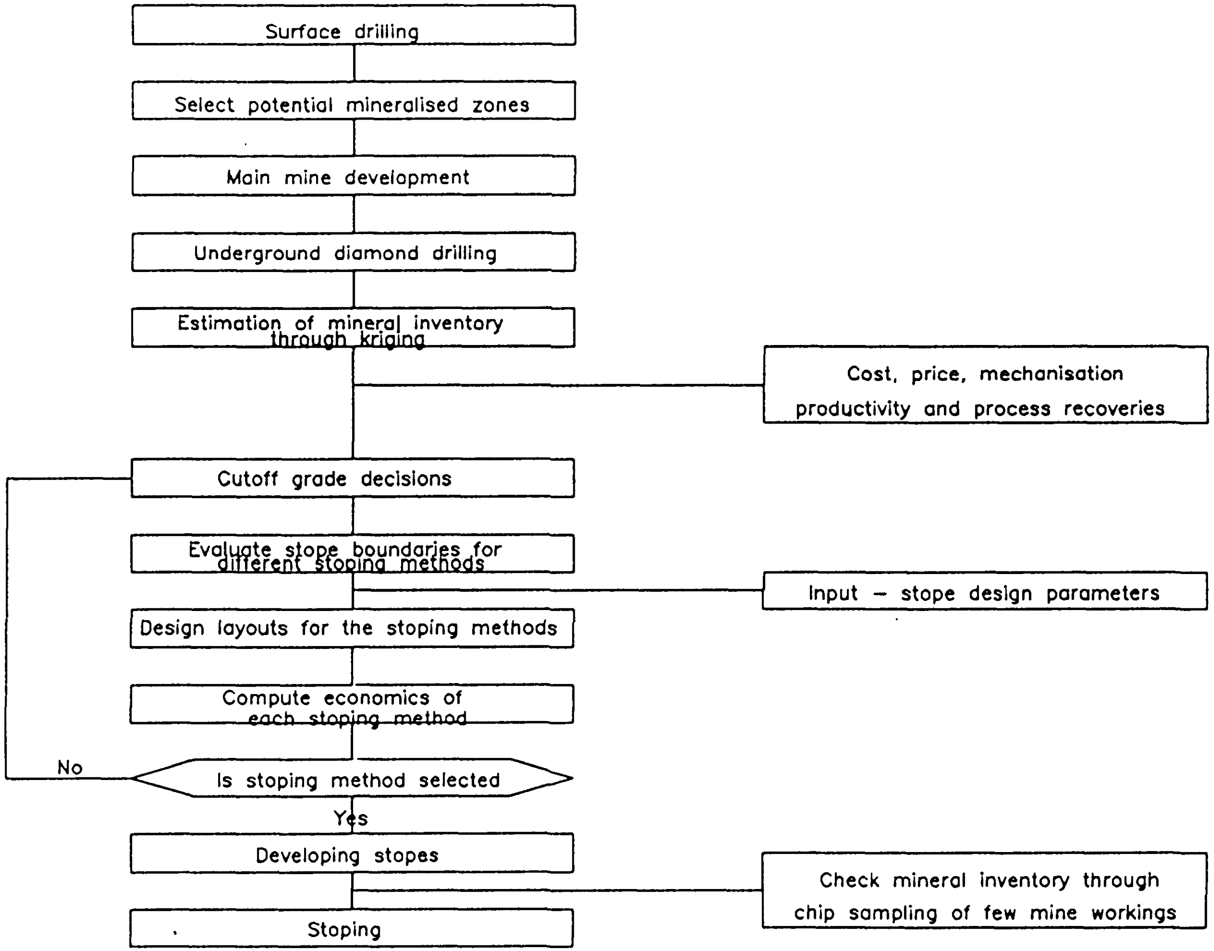


Fig. 1.1 Proposed methodology for underground mining of base metal deposits.

The research is based on the information obtained from two copper mines, Khetri and Kolihan, located in India. The author visited these mines in 1984 and 1986 and worked there from 1971 to 1979.

1.4 The layout of the thesis:

Chapter 2 contains a description of Khetri Copper Complex in general, and Khetri and Kolihan mines in particular. This includes the geology of the deposits, the mechanical properties of the rocks, operational details of the mines and a brief description of the various plants operating at KCC.

The approach to mineral exploration, ore reserve estimation and reserve classification at KCC mines is reviewed in chapter 3.

Chapter 4 briefly reviews the various mineral estimation techniques currently in use. The basis for selecting the geostatistical methods is outlined.

The global estimation of Kolihan mine reserves, based on the surface drilling information, using kriging, is worked out in chapter 5.

Chapter 6 describes the local estimation using underground diamond drilling information, through kriging, for a portion of the deposit between two levels, 60m apart, at Kolihan mine. In addition, the ore reserves estimated by kriging and the one estimated using conventional methods are compared.

A local estimation based on chip samples of a part of the Kolihan mine is described in chapter 7. These are compared with estimates using the drilling sampling information.

Chapter 8 examines important factors such as the stoping method, orebody thickness, degree of mechanisation, productivity and process recoveries to be considered in deciding a cutoff grade for underground mines. Cutoff grade theory is reviewed briefly.

A methodology developed to define the stope boundaries, based on the kriging of small 1m x 1m size panels and using incremental analysis, is described in chapter 9. The basic geological data obtained from Khetri mine, for a small portion of the deposit between two levels, has been used for this analysis.

The computer models developed to design sublevel, down-the-hole (DTH) and cut and fill stoping methods are presented in chapter 10. The input parameters essential for stope design purposes are described, and the basis of algorithm development for these methods is then outlined. A procedure to design ring drilling patterns is also described.

Chapter 11 describes the additional features that have been incorporated in the stope design models for the purpose of computing the economics of each of the designs. The economic evaluation, for a small section of the Khetri deposit, using different stoping methods is then outlined. A comparison of these results provides the basis to select a stoping method for the optimum exploitation of the deposit.

Chapter 12 summarises the observations made in the different sections. Finally, the recommendations for the future work related to this research are outlined.

Chapter 2

2. Khetri Copper Complex

2.1 Introduction

Copper metal in India is produced mainly by Hindustan Copper Limited (HCL), a government undertaking. HCL has two integrated copper production units, Indian Copper Complex (ICC) - the eastern sector and Khetri Copper Complex (KCC) - the western sector. KCC comprises an open pit, three underground mines, two concentrators, a smelter and electrolytic refinery, together with an acid-fertilizer plant. The photograph shown in fig. 2.2 is an aerial view of Khetri mine and plants.

The current mining practices followed at Khetri and Kolihan mines, have been used in the various models developed in this research and are described in this chapter.

The geological setting, mechanical properties of rocks and mining operations of these two mines are first described. A brief description of the central services, personnel employed and main features of the process plants of the complex then follows.

2.2 Location

Khetri Copper Complex is located in the Jhunjhunu district of Rajasthan. It is some 190km SW of Delhi and 180km north of Rajasthan state's capital Jaipur as shown in fig. 2.1. The Kolihan mine is about 7.5km away from the Khetri mine where all the processing plants are also located.

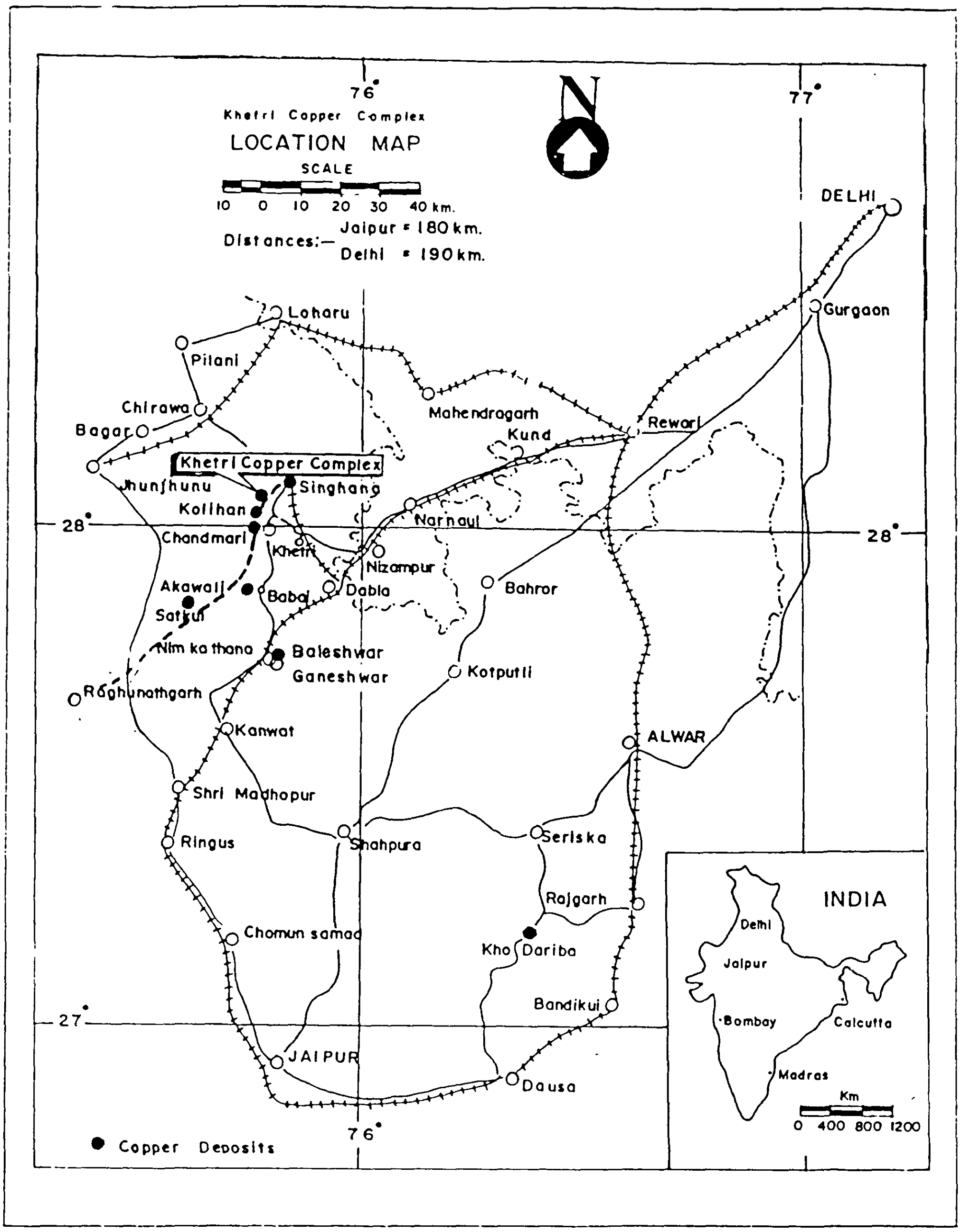


Fig. 2.1 Location map Khetri Copper Complex (Tiwari, K.K. et al. 1985).

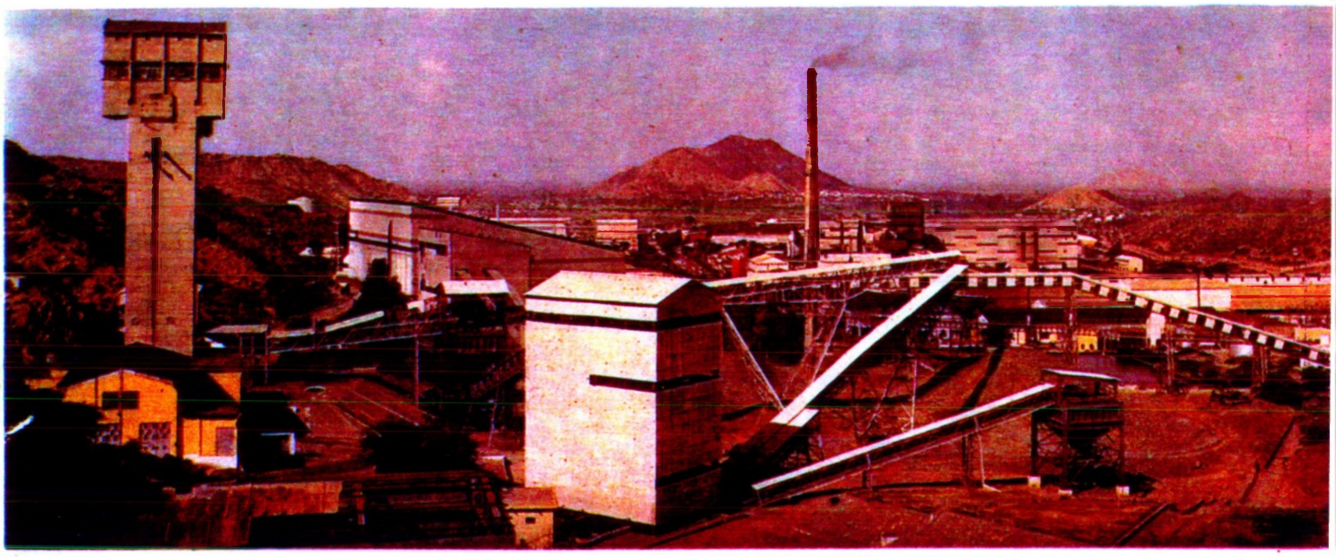


Fig. 2.2 An aerial view - Khetri Mine and plants, Khetri Copper Complex.
(source: Khetri Mine Planning Cell, 1986).

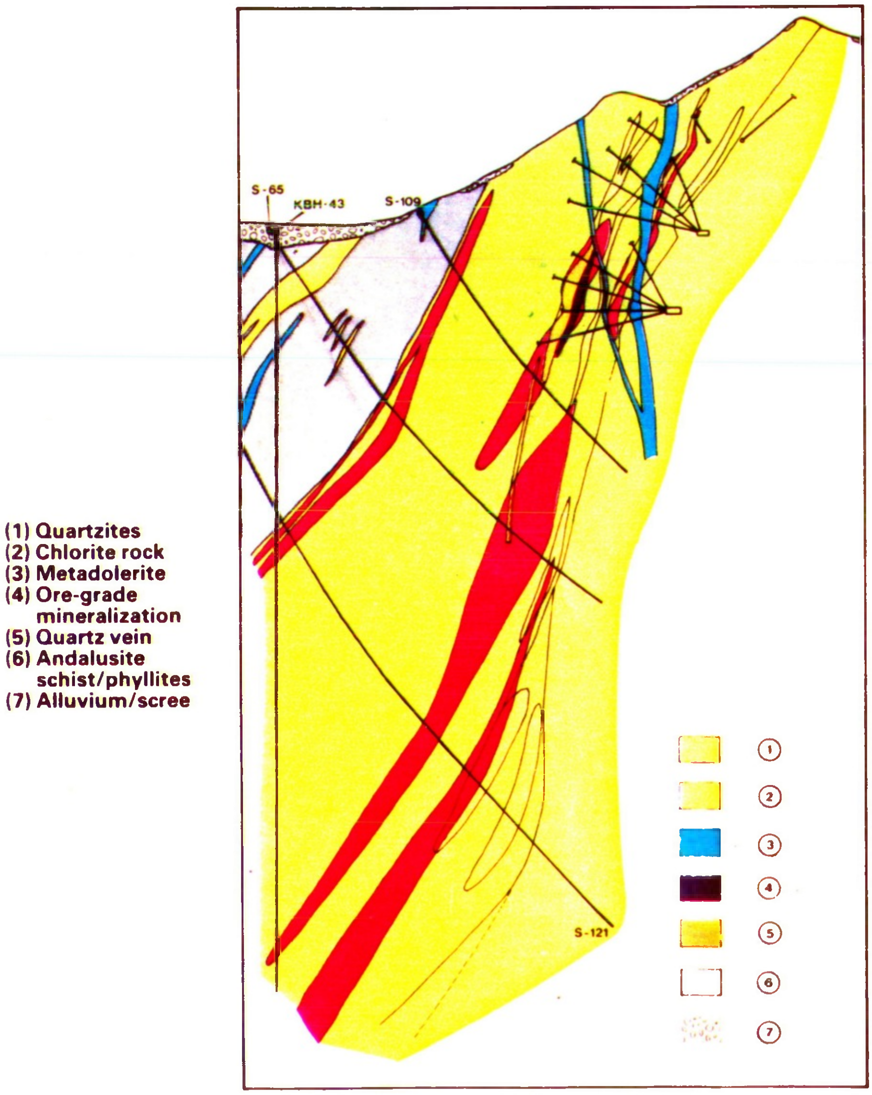


Fig. 2.3 Geological section through the Khetri mine showing ore lenses (based on 0.5% Cu cutoff).
(Geology Dept. Khetri Mine).

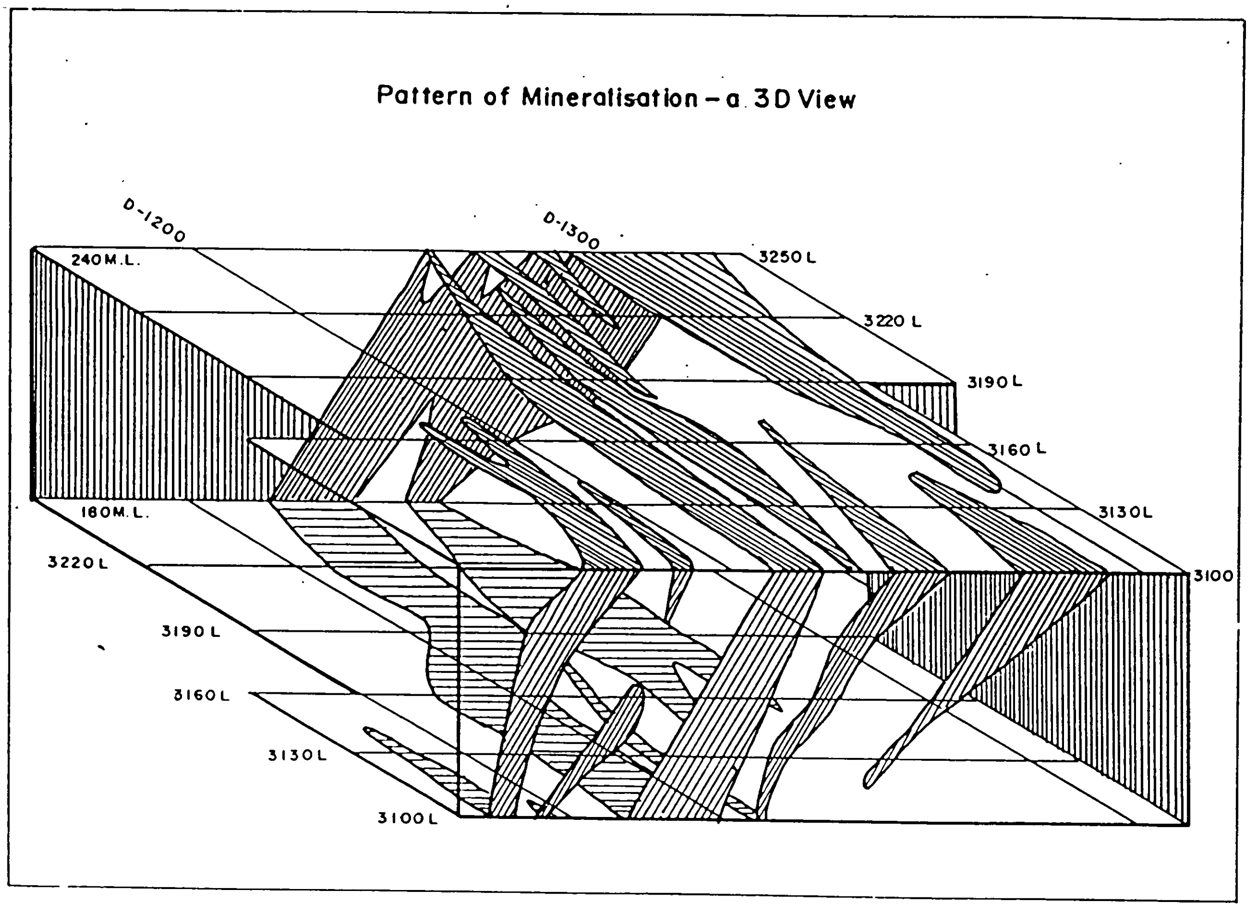


Fig. 2.4 A pattern of mineralisation, Khetri mine (Tiwari K.K. et al. 1985).

2.3 Project History

The deposits in this area were mined in ancient times some 2,000 years ago in the Mauryan period. The mines were also active in the Mughal period (1500 A.D.), but in 1872 the the exploration and mining activities in the area ceased.

Between 1953-1961 the Geological Survey of India and the Indian Bureau of Mines undertook the prospecting and exploratory mining in the area. The project was handed over to National Mineral Development Corporation in 1961 for further investigation. The decision to develop these mines was taken in 1962 and in 1967 the area was handed over to HCL, a new company established to produce copper in the country. The basic design of the complex was undertaken by a French company, Venot-Pic-Ensa. The mines are moderately mechanised and the plant design is based on modern technology.

Ore production from these mines commenced in 1971. The production of copper wirebars began in 1975.

2.4 Geology

The geology of the deposit has been published in detail by various authors (Bhatnagar et al., 1979, Tiwari et al. 1985, Mining Magazine, 1984).

The rocks in Khetri district belong to Delhi super-group of Precambrian age, which is sub-divided into Alwar and Ajabgarh groups. All the formations are metamorphosed to quartzites, schists and phyllites. Some intrusive dolerite dykes are present and veins of carbonates and quartz are common. The general strike of the formation is NNE-SSW with steep to gentle dips towards WNW. The strike extension at Khetri is 3.6km, but at Kolihan it is limited to 600m only.

The economic mineralisation which gives rise to the orebody is mainly localized in an amphibole-chlorite quartzite/schist. The mineralisation occurs in the form of veins, veinlets, stringers and disseminations (rarely massive).

At Kolihan mine three distinct lodes have been identified. Lode 1 - south lode, Lode 2 - north lode, Lode 3 - east or foot wall lode. Of these lode1 and lode3 are persistent at depth. Lode 2 is confined in the northern side of the deposit and does not extend below 364m level.

At Khetri mine there are two distinct lodes, the Madhan (or footwall) lode and the Kudan (or hanging wall) lode. The Madhan lode is fairly persistent and contains a number of ore lenses. The Kudan lode is narrower and has lower copper grades. In all some 50 lenses have been identified. There is a tendency for ore lenses to split along strike and depth. The underground drilling and development also revealed the echelon arrangement of ore lenses along strike and down dip. In fig. 2.3, a geological section through the Khetri mine showing ore lenses together with the country rocks is shown (based on 0.5% Cu cutoff). In fig 2.4, a typical pattern of mineralisation, occurring between 240m and 180m levels (Khetri mine), is presented. Comparatively, ore lenses at Khetri are thinner and scattered than those at Kolihan.

The felspathic quartzite rocks which form the footwall are highly jointed but are, nevertheless, quite competent. The hangingwall consists of phyllites. There is a shear zone which sometimes causes stability problems during stoping. There are also a number of faults and shear zones distributed throughout the deposit.

Chalcopyrite is the principal copper mineral. The other minerals associated with the ore are pyrite, pyrrhotite, magnetite and silica. Gold, silver and cobalt are also found in traces.

2.5 Mechanical properties of rocks

In table 2.1 and table 2.2, the results of the different mechanical tests obtained for the different rock samples from Khetri and Kolihan mines are given. A comparison of these results, with the rock classification described by Hans Hamrin (1982, p. 106) shows that the rocks of both mines possess a medium to high strength. Presence of minerals like quartz, garnet and magnesite show the hard and abrasive nature of mineralisation.

Sample No.	Rock type	Compressive strength MPa	Young modulus MPa	Poissons ratio -	Shear strength MPa	Tensile strength MPa
1	Amphibole quartzite	132	47970	0.17	33	8
2	Amphibole quartzite	130	38062	0.14	24	7
3	Metabasic rock	71	49932	0.23	43	13
4	Felspathic quartzite	140	23151	0.11	26	7

Table 2.1 Mechanical properties of rocks - Kolihan mine. (source: Rock Mechanics Section, KCC. kg/cm^2 converted to MPa . 1986)

Sample No.	Rock type	Compressive strength MPa	Young modulus MPa	Shear strength MPa	Tensile strength MPa
1	Garnet chlorite quartzite schist	151	36395	46	16
2	Garnet chlorite quartzite schist	175	46597	61	18
3	Amphibole quartzite schist	50	22563	25	11
4	Amphibole quartzite schist	63	25996	46	11

Table 2.2 Mechanical properties of rocks - Khetri mine. (source: Rock Mechanics Section, KCC, 1986. kg/cm^2 converted to MPa).

The hangingwall, consisting of solid precambrian rocks, will probably allow mining with open stoping (Hans Hamrin, 1982, p. 104). Based on the above description of the geological set up and mechanical properties of the rocks, both the country rocks and the orebody at KCC mines can be considered to be competent.

2.6 Ore reserves and mine planning

The exploration activities undertaken and ore estimation technique used at KCC mines have been described in chapter 3. Since 1974 the ore reserves at KCC mines have been estimated using a constant cutoff grade of 0.5% Cu. The stated ore reserves for Khetri and Kolihan are 39Mt at 0.94% Cu and

28Mt at 1.37% Cu respectively (Tiwari et al, 1985). Ore reserves at varying cutoff grades have not been estimated.

Mine planning cells (departments) prepare mine layouts, stope design and short term production planning for both mines. An account of the main activities undertaken at both mines since their inception is given in table 2.3. This includes the diamond drilling, mine development and the production statistics. It may be noted that the ore reserves at Khetri mine are 40% higher than that at Kolihan mine but the linear development undertaken at Khetri is three times than that of Kolihan. This is mainly due to the difference in the geometry of the orebodies at the two mines.

Activities	Kolihan	Khetri
Diamond drilling	67,524m	107,425*
Linear mine development	45.186km	193.78km*
Depletion of reserves	3,640,752x2.1%	4,364,504x0.85%
Mine production	3,384,237x2.03%	4,150,499x0.76%*

Table 2.3 An account of activities undertaken by KCC mines from starting as on 1-4-1983. * - As on 1-4-1984. % - grade % Cu. (source: Addendum reports and general information, Geology Dept. KCC mines, 1983-84).

Routine elements of mine design such as manipulation of large data sets, repetitive calculations, geological and engineering drawings and scheduling work are currently undertaken by hand or with the aid of calculators. The computers are yet to be used for mine planning, data storage and mineral inventory evaluation.

2.7 MINING

Khetri and Kolihan mines have been designed on a similar pattern. However, the workings at Khetri mine are more scattered than those at Kolihan. This is mainly because of the difference in the strike extension and orebody thickness prevailing at both mines. The main exploratory levels, which are subsequently used as tramming levels, are spaced 60m apart vertically. All the levels at both mines are denoted by their heights relative to mean sea level. Some of the levels are tracked and the rest are trackless. Fig. 2.5 represents a typical longitudinal section of the Khetri mine workings. The equipment commonly used at various levels in the mine are shown. Fig. 2.6 is an isometric view of Kolihan mine workings.

A flow diagram of the various activities undertaken at Kolihan mine is given in fig. 2.7. At Khetri mine also a similar pattern is followed.

2.71 Access

Access to the underground mine workings at Kolihan is through an adit, at 424m, and a decline commencing from surface at 428m. The 424m level is the first tramming level. An underground shaft or sub-vertical shaft of 5.9m dia. has also been sunk at this level to hoist the production from the levels below. The other tramming levels have been positioned at 364m, 306m, 246m, 184m, 120m, 60m and 0m (measured above the mean sea level) horizons. The decline was driven to obtain production from the levels below 424m before the completion of the underground shaft. It terminates at 306m level.

Khetri mine, similarly, has adits, inclines, an inclined shaft and two vertical shafts. Access to the uppermost level of Khetri mine, at 421m, is by adits. The ore at this level has already been worked out. The next level, 350m, is served by number of inclines and an inclined shaft. 300m, 240m, 180m, 120m, 60m and 0m horizons are the other tramming levels at Khetri mine. Two

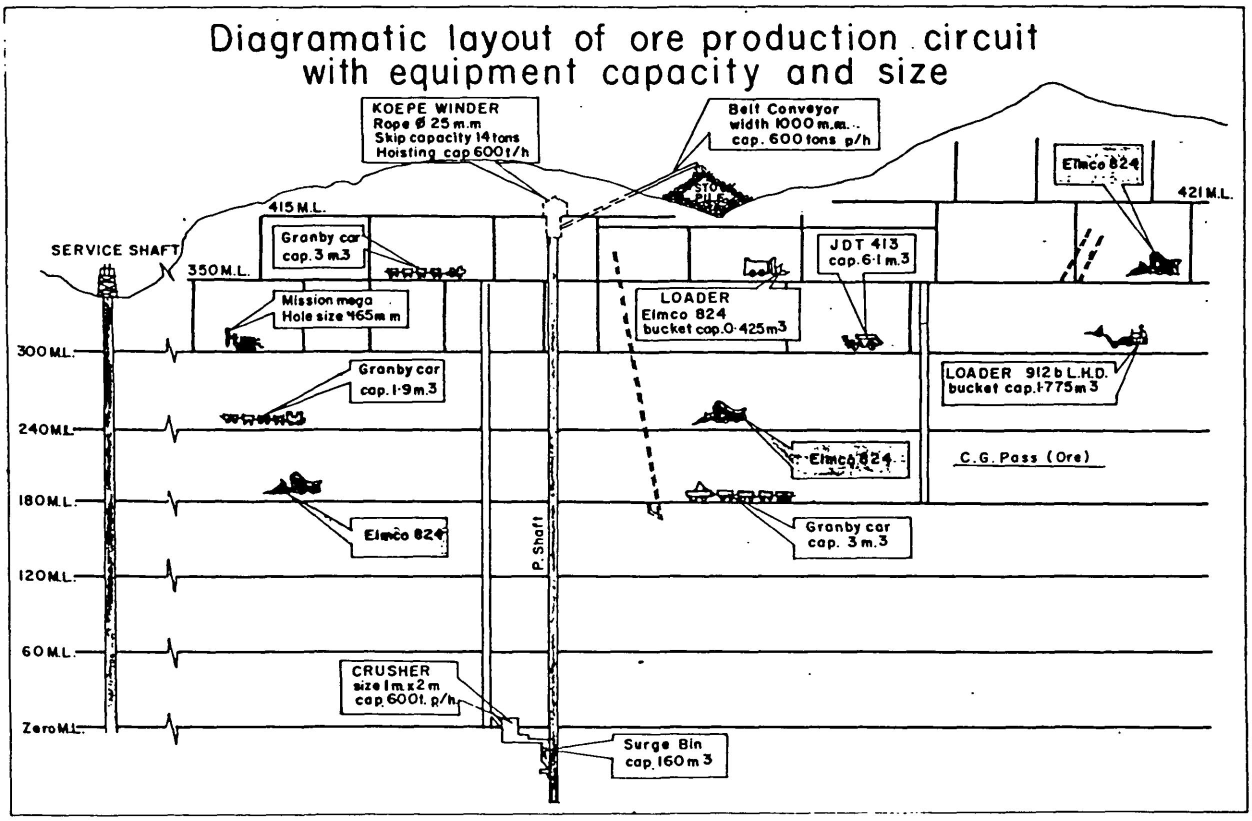


Fig. 2.5 Diagrammatic layout of ore production circuit with equipment capacity and size, Khetri mine (Singh U.B. et al. 1985).

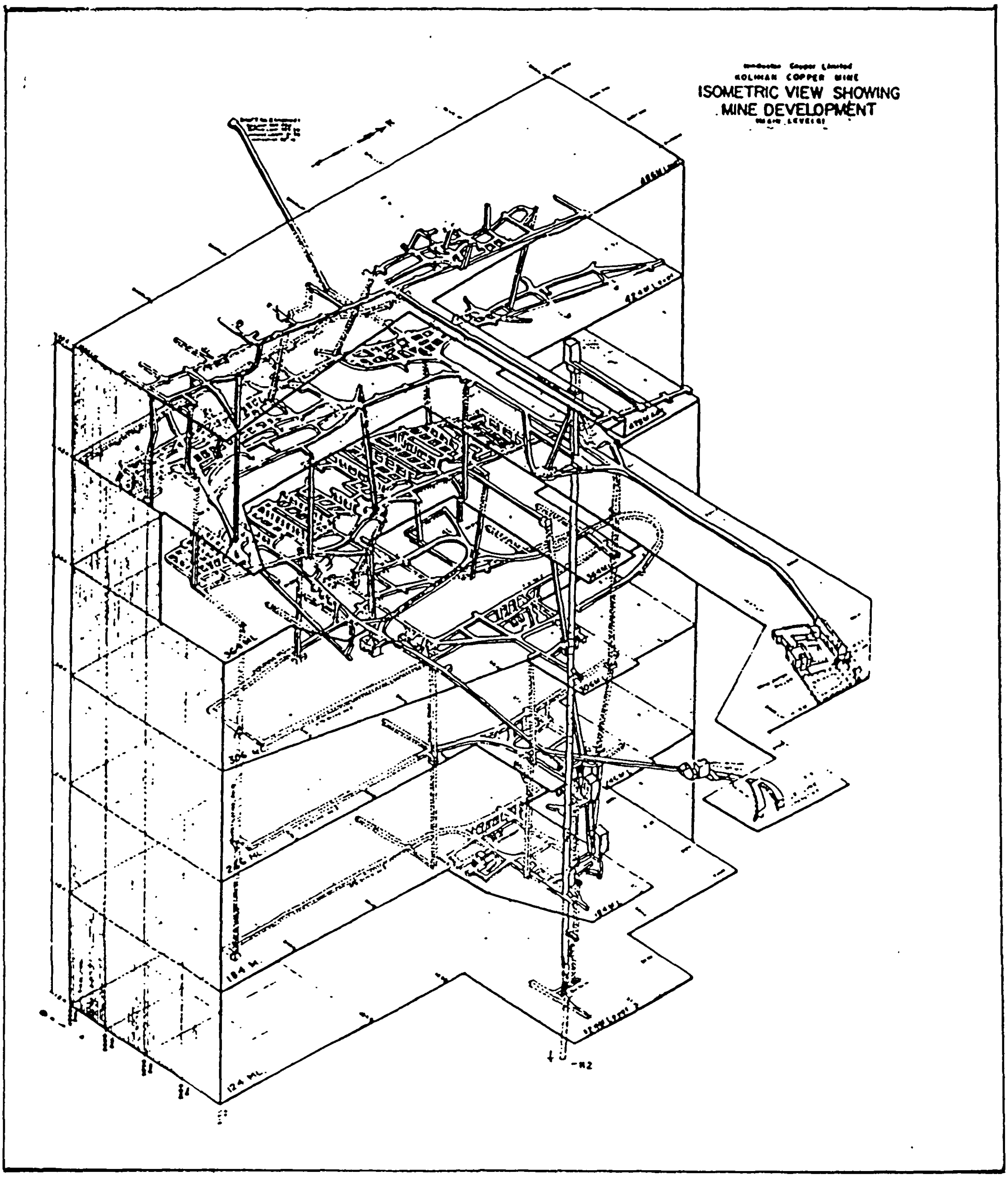


Fig. 2.6 An isometric view of Kolihan mine workings (Mine Planning Cell, Kolihan).

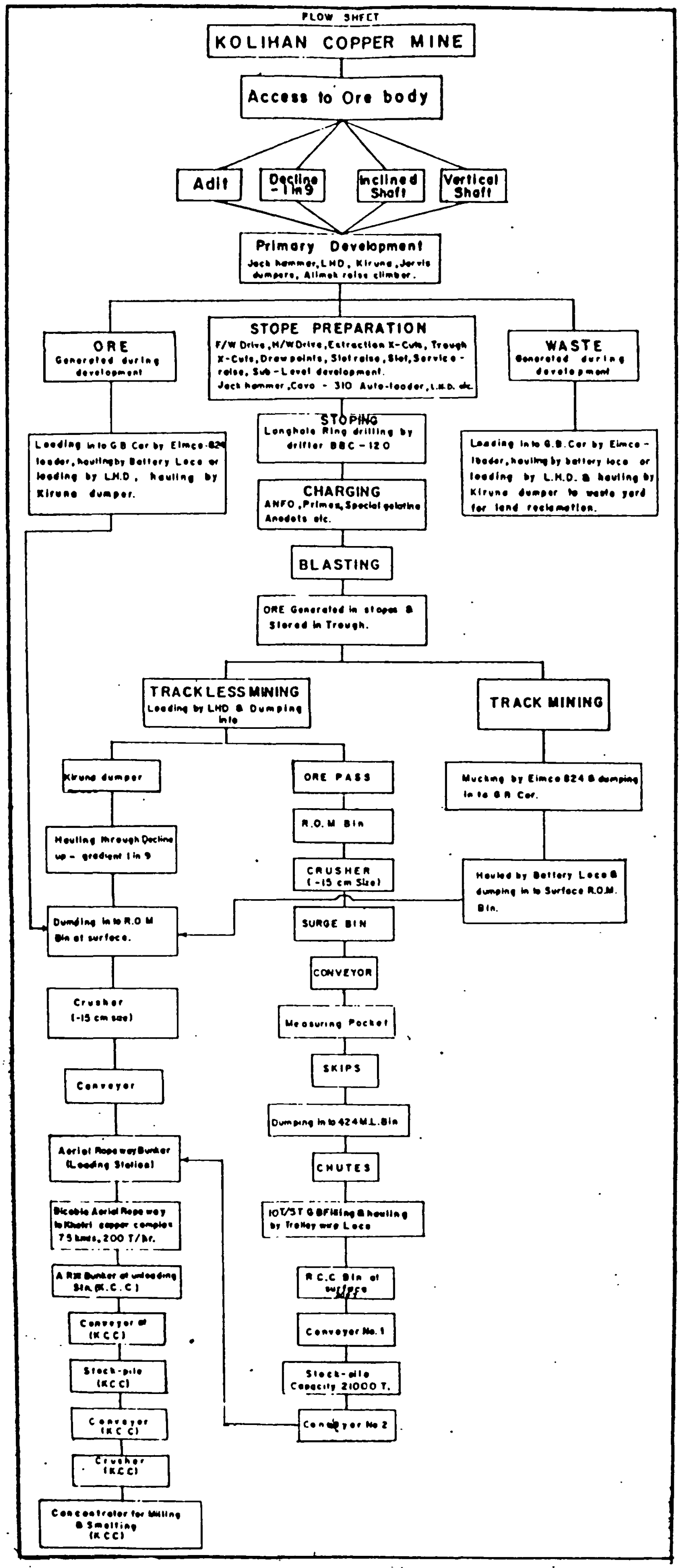


Fig. 2.7 Flow sheet - Mining activities at Kolihan mine (Mine Planning Cell, Kolihan).

vertical shafts, a service shaft and production shaft, have been sunk from the surface to 0m level in order to serve these levels. The technical details of shafts at Khetri and Kolihan mines are summarised in table 2.4.

	Khetri		Kolihan
Technical details	Production shaft	Service shaft	production/service
Collar elevation	390	370	424
Depth (m)	475.5	388.4	270.1
Cross-section	5.5m dia.	6.1m x 4.93m	5.9m dia.
Use	ore hoist./vent.	Waste hoist/vent.	ore/waste hoisting vent. and services
Levels served	-	300m-0m	306m-184m
Loading levels	0m	180m, 0m	246, 184
Skips	2x14t	-	2x6t
Cages	-	2x2-deck	2x1-deck
Guides	8 ropes	rails	rails
Shaft lining	concrete	concrete	300mm concrete
Hoist type	Koepe winder	double-drum winder	Koepe (skips), double-drum
Hoist kw	2,870	1,600	650(skip), 400(cage)
Headgear construction	concrete	steel	underground

Table 2.4 The technical details of the shafts at Khetri and Kolihan mine. (source: Mine Planning Cell, KCC, 1986).

2.72 Mine development

The common development workings driven at the mines, with their specific purposes and dimensions are described, in table 2.5. Normally 37-63 holes of 33-40mm dia., using burn cut pattern, are drilled. Generally jack hammer drills with pusher legs are used for development drilling purposes. However, drilling jumbos have been introduced recently.

Development Type	Usual dimensions (width x height)
Single track haulage	3m x 3m
Double track haulage	5.5m x 3m
Trackless haulage level	5m x 3m
Sublevels (drill drives/x-cuts)	3m x 3m
Extraction cross-cuts/draw points	4m x 3m
Services raises	2.5m x 2.5m - 3m x 3m
Slot raises	2m x 2m
Decline	5.5m x 3.25m

Table 2.5 Types of development workings with their usual dimensions at Khetri and Kolihan mines. (Mine planning cell, KCC, 1984).

Special gelatine (80%) and half second delay detonators are used to blast development headings with 1.5 to 3.0m long rounds. A pull of 80% of drilling length is usually achieved.

On tracked levels rocker shovels load the rock into Granby cars which are hauled by electric trolly-wire or battery locomotives. On trackless levels the loading is undertaken by LHD - low profile dump truck combinations.

Completion of a round of 1.5 to 3m in any of the development headings is a 2-3 shift operation.

As the strata is usually competent no support, except occasional rock bolting, is needed at most places in the mines. Timber props or steel sets are employed for roadways support wherever needed.

Earlier the raises were developed using Alimak and conventional methods. Then conventional methods were replaced by the longhole raising technique. Raises up to 30m length are driven using this technique. Raises longer than 30m are driven using DTH drills.

2.73 Stoping methods

Sublevel open stoping, using blasthole drills, is the usual method at the KCC mines. Depending upon the orebody geometry and the type of drill deployed, the following three variants of sublevel stoping are practised:

Longitudinal sublevel stopes

Transverse sublevel stopes and

DTH stopes.

The designs of longitudinal and transverse sublevel stopes are based on the application of the blasthole drills capable of drilling holes of 50-60mm dia. and lengths below 30m in any direction.

Longitudinal sublevel stopes are used where the orebody is 5-30m thick. The stope length ranges from 40 to 90m (measured along the strike direction) and the stope height is 60m. First a service raise between two adjacent tramming levels is driven. The drill drives (sublevels) are developed at 18-22m vertical spacing, depending on the orebody thickness and the drill deployed. These drives usually follow the footwall contact of the orebody at each of the drilling horizons. However, an additional drill drive at the hanging wall contact of the orebody is also driven if the orebody thickness exceeds 20m. A slot is developed by enlarging the slot raise, using parallel hole blasting, to the width of the stope. The slot is positioned in the widest section within the stope. The winning of ore from the stope commences from the slot position along the strike direction. In fig. 2.8, a typical sublevel stope is presented.

The transverse sublevel variant is used for the orebodies usually thicker than 30m. The interval between two consecutive sub-horizons and number of sub-horizons in a stope remain the same as that of a longitudinal sublevel stope. Stope length is 30m. The slot is positioned at the extreme hangingwall contact of the orebody in the stope. At each of the horizons the drilling cross-cuts, a footwall drive and a slot drive are driven as shown in fig. 2.9.

A 10m rib pillar between the two adjacent stopes, a 5m crown pillar to support the immediate upper level and sill pillar of 10m at the extraction level are included in both stoping variants described above.

The blasthole-rings are drilled in a vertical plane keeping toe spacing in the range of 1.2m to 2.2m (depending upon the blasthole length) and the ring burden between 1.5 and 1.8m.

Ore in a stope is fragmented using ring drilled blastholes and ANFO explosive. The ANFO explosive is initiated using 20mg PETN boosters and antistatic detonators, 'anodets'. First the stope rings are blasted, followed by the rib pillars, sill pillar and crown pillar.

DTH stoping

Designing the stopes based on the use of blasthole drills capable of drilling 165-200m dia. holes up to 150m is very recent. Imported Mission Mega drills or indigenous WDS-400 drill rigs are used for the stope drilling.

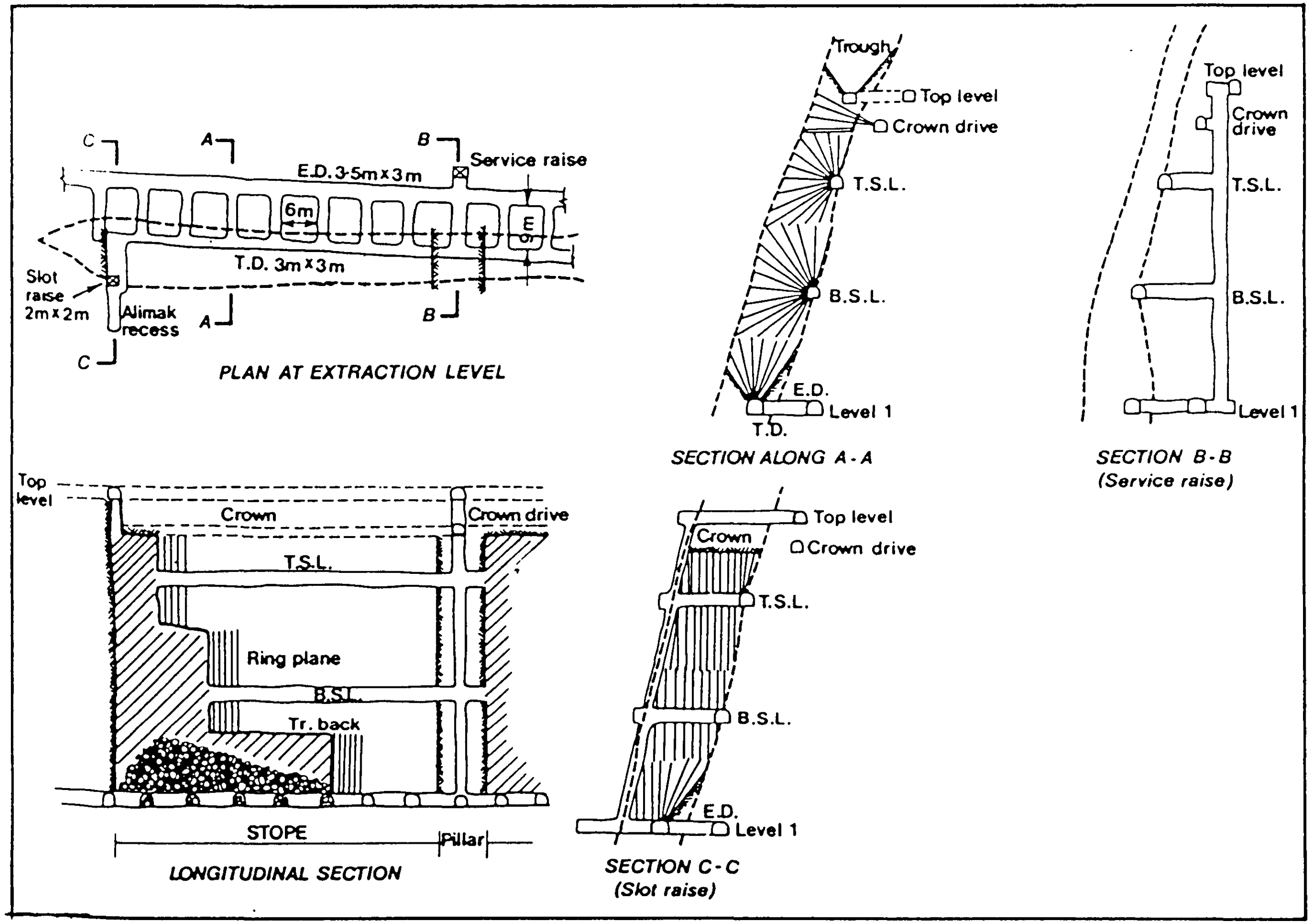


Fig. 2.8 Plan and sections of a typical sublevel stope Kherti and Kolihan mines (Mine Planning Cell, KCC).

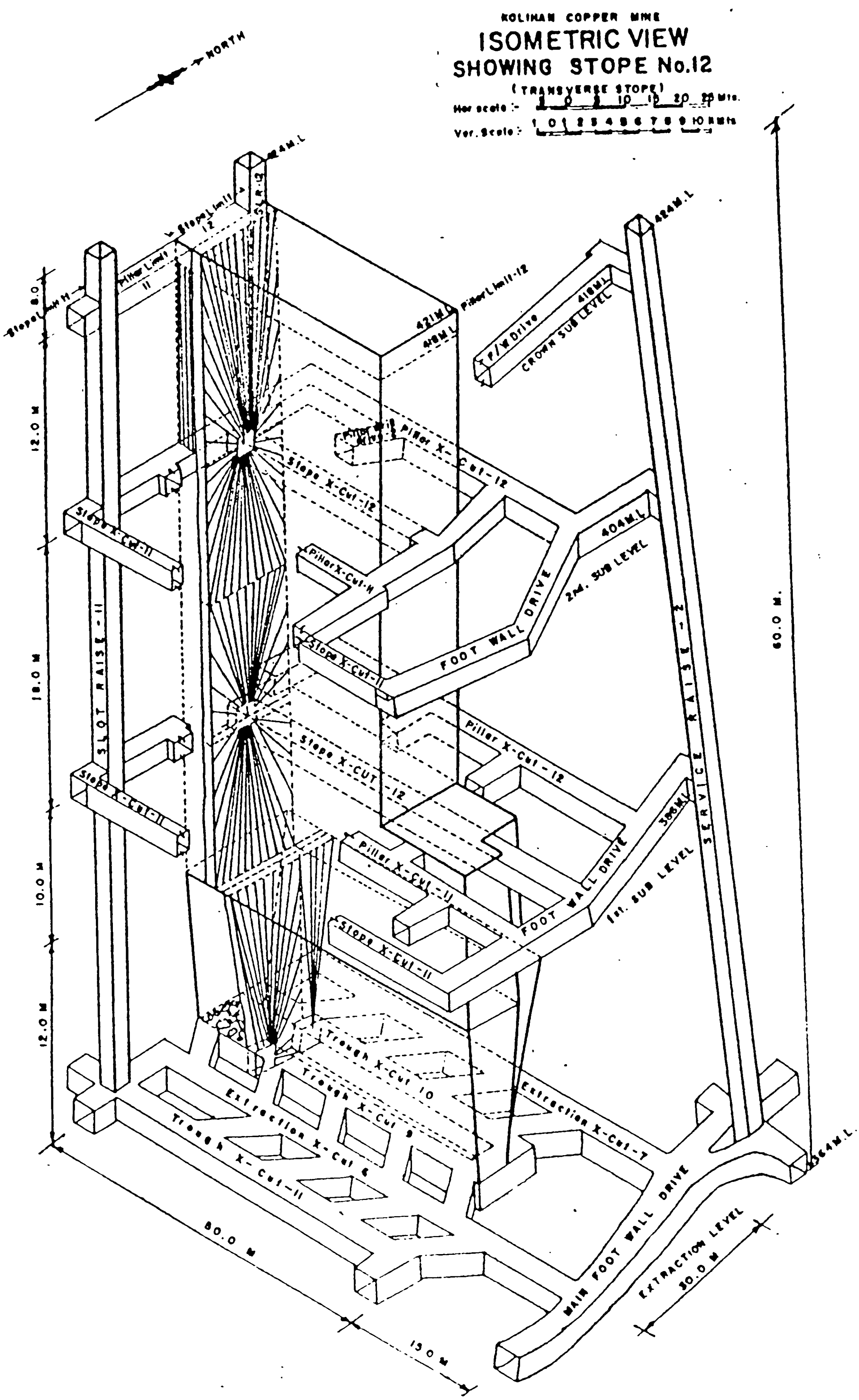


Fig. 2.9 An isometric view - Transverse sublevel stoping (Mine Planning Cell., Kolihan mine).

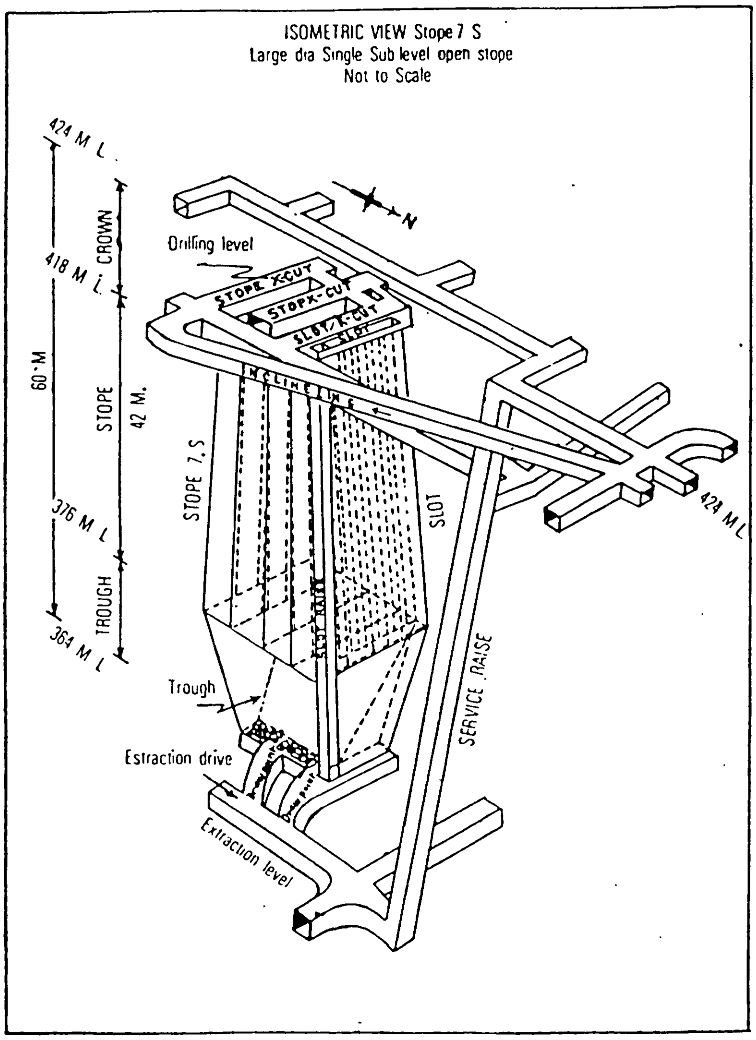


Fig. 2.10 An isometric view - DTH stoping (Report - Introduction of new technologies KCC, 1983).

Only one sublevel 9-10m below the immediate upper level is planned in this stope layout. The sublevel horizon is developed by driving a number of drilling cross-cuts and creating a 3m wide slot. Trials are on the way to determine the hole spacing, burden and other stope design parameters. Burden and spacing in the range of 3-6m are used at present. On completion of stope drilling a few rows of holes are blasted together. Slurry and ANFO explosives are used. An isometric view of a typical DTH stope is shown in fig. 2.10.

At the tramming level for each of the stopes (for all the three variants) the trough/troughs, draw points and extraction drives or x-cuts are driven to receive the blasted muck from the stope. Tracked or trackless loading and transportation equipment are used to transfer muck from the stope to the nearby orepass.

Basu, et al. (1985) reported that the DTH stoping has proved to be an improvement over the other two variants. This method is likely to be progressively introduced throughout both mines.

2.8 Ore handling and transportation

Both mines have central orepass systems. The ore is brought to the orepass from the stopes and development headings. In the Khetri mine the orepass extends from the 350m level to 0m level. A primary jaw crusher at -18m level reduces the ore to -150mm size before it passes, via a bunker and measuring pockets, to the 14t skips for hoisting.

The orepass at the Kolihan mine is from 364m level to 246m level. The double toggle jaw crusher installed at 229m level crushes the ore to -150mm size. The crushed ore is then fed into a 400 tonnes capacity underground surge bin constructed between 200 and 229m levels. The ore from surge bin is fed via a conveyor to the measuring pockets and finally to the skips. The loaded skips are hoisted and discharged into a 250 tonne bin located at 440m

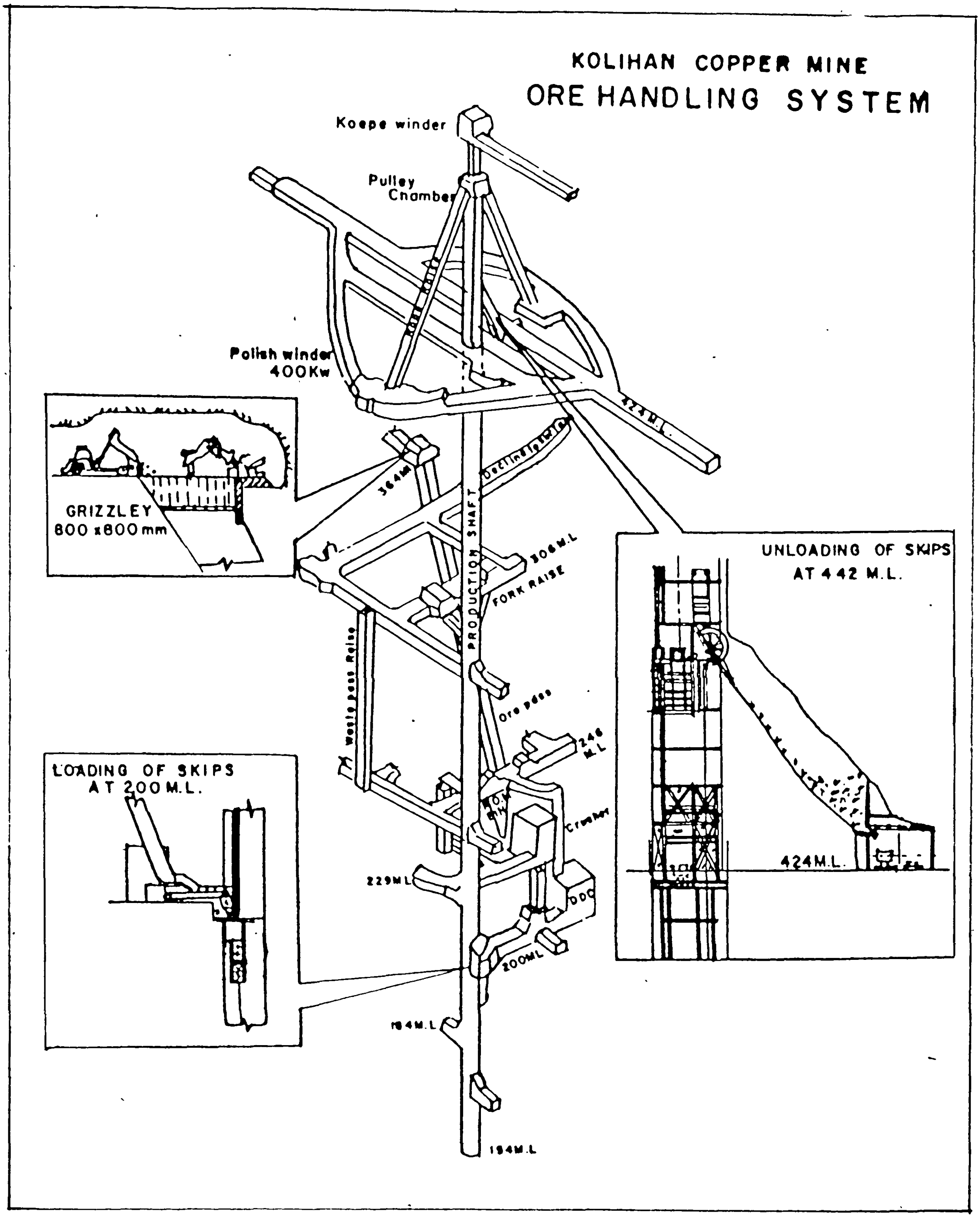


Fig. 2.11 Ore handling system at Kolihan mine (Mine Planning Cell, Kolihan).

level. The ore from this bin is loaded into a train of 5/10t granby cars which are then hauled by a 16-tonne trolly wire locomotive for discharge into the surface bin of 300t capacity. Ore from this bin is conveyed to the surface stockpile. The ore is fed, via another conveyor, to the aerial ropeway system for despatch to KCC. The 7.5km long ropeway has a capacity of 200tph. A typical ore handling layout at Kolihan mine is shown in fig. 2.11.

2.9 Equipment in use

The details of various types of equipment used at the mines are shown in table 2.6. The KCC mines were the first to use such equipment in India. 'The mines at Khetri are most modern and mining techniques and methods developed are innovative and of high standard' (Vasudeva, 1981).

Equipment and its make	size/capacity	Population	
		Kolihan	Khetri
LHD, Eimco 912b	1.26cu.m	13	10
Cavo loader, Eimco Elecon	1cu.m	5	10
Eimco 824	-	1	21
Trucks, Jarvis Clark	13t	7	5
Trucks, Kiruna	20t	7	-
Loco, Goodman trolly wire	115kw	3	-
Loco, MAMC (battery)	8t	4	14
Loco, ASEA (battery)	9t	-	4
Drifters, Atlas copco	-	9	10
Mission mega	-	1	1
WDS-400 drilling rig	-	1	1
Jack hammer	-	54	103

Table 2.6 The main equipment in use at Kolihan and Khetri mines. (source:

2.10 Mine services

The various data specified below to describe the mine and central services are based on the information gathered by the author during his visit to the mines in 1984 and 1986.

Water inflow to Khetri mine is 150,000 cu.m/month. The sumps are provided at 0m, 180m and 300m levels. There are 10 pumps in all, ranging from 130 to 135kw to deal with the average flow of water. A total of nine pumps, ranging from 22kw to 187kw, handles an average flow of 158,400 cu.m/month at Kolihan mine.

Ventilation

Main ventilation at Khetri mine is provided by 175kw Sirrocco axial flow fan at the production shaft giving an air flow of 7,800 cu.m/min at 25mm water gauge. At Kolihan there is a 175kw axial flow fan serving the decline section of the mine. This provides a flow of 4,250 cu.m/min. Two smaller fans serve the shaft section, giving a further 2,250 cu.m/min. Booster fans are used at both the mines for ventilating the development headings in conjunction with 600mm rigid ducting.

Compressed air

At the Khetri mine 17 compressors of 30-130 cu.m/min capacity meet the demand of 650 cu.m/min of compressed air at 690 KPa. The Kolihan compressor house has seven compressors of 21 to 226 cu.m/min capacity. Total usage of compressed air at Kolihan is about 400 cu.m/min at 690 KPa.

2.11 Production and productivity

The Kolihan and Khetri mines have been designed to produce a daily production of 3,000 and 5,000t of ore respectively. It is envisaged that the Khetri mine is likely to achieve its designed production of 1.5Mt of ore (per annum) at 0.75% Cu by 1990-91 and Kolihan that of 0.9Mt at 1.15% Cu by 1992-93 (Mine planning cell KCC, 1986).

In table 2.7, the production and productivity statistics of Kolihan mine for the past years have been shown. A steady rise both in production and productivity is indicated. However, it can be seen that the 511,346 tonnes of ore produced in 1983-84 is far short of the 900,000tpa target.

year	Mine production		Manshift ratio			
	tonnes	grade % Cu	ore	tramming	u/g	overall
1974-75	260,770	1.80				
1975-76	271,980	2.06				
1976-77	407,516	1.58				
1977-78	385,646	1.40				
1978-79	409,062	2.01				
1979-80	354,044	2.74				
1980-81	362,569	2.74	7.48	30.34	-	0.99
1981-82	337,591	2.14	7.50	27.88	-	0.92
1982-83	386,559	2.08	8.42	32.02	-	1.10
1983-84	511,346	2.16	11.60	40.26	-	1.49
1984-85	-		12.18	40.50	3.14	1.57

Table 2.7 Production and Productivity - Kolihan mine. (after Gupta, B.L., 1985).

2.12 Mining cost

The breakdown of mining cost of ore production is given in table 2.8. A considerable amount is spent in acquiring the imported plant and equipment (Jain, K.C. personal communication, 1986). In 1981 Vasudeva (1981) stated that the copper price in 1981 was more or less at the same level as it was in 1977, whereas the costs of all inputs such as wages, power, raw materials, lubricants, stores and spares, etc. have all increased to nearly double since then. There has been considerable increase in all the resource inputs since 1981. The rise in copper price has not matched the increases (Jain, 1986 - personal communication).

Cost breakup	Khetri cost/t, in Rs	%	Kolihan cost/t, in Rs	%
Variable cost	70.3	26.8	53.8	29.1
Direct fixed cost (salary and wages)	78.3	29.8	66.4	35.9
Indirect fixed cost (depreciation, interest on cash credit, general and social overheads)	48.4	18.6	33.2	18.0
Interest on govt. loans, head office expenses)	64.9	24.8	31.3	17.0
Total	261.9	100.	184.7	100.

Table 2.8 Mining cost/t of ore production and its distribution amongst various heads (allocations) for Khetri and Kolihan mines. (cost reports KCC, 1985-86).

It may be noted that overall mining cost at Khetri is 45% higher than that of Kolihan. This is mainly due the geometry of the orebody at Kolihan which has resulted in low variable costs, fixed costs and overheads compared to those at Khetri.

It may be noted that the copper price is decided by the Metal Minerals Corporation of India (MMTCI). The copper price during the period 1985-86 was Rs 44,500/t equivalent to 3100 U.S. Dollars. The current price of copper in the international market is 1400 U.S. Dollars/t. Thus, there is a considerable difference in the Indian copper price compared with the one in the international market.

2.13 Concentrator and Process plants

The run-of-mine ore from Kolihan of -150mm size is conveyed by aerial ropeway to KCC where it is blended with the ore from Khetri mine in a stockpile of 30,000t capacity. The ore from this stockpile is fed to a 530tph crusher. The crushed ore is fed to the grinding circuit of 12,000t/day capacity. Conventional froth flotation practice is followed for ore concentration.

The head grade of ore supplied to the concentrator averages around 1.5%Cu. The recovery is around 90% and the concentrate grade is just over 16%Cu, dry basis.

The Khetri smelter has a design capacity to produce 31,000t/y of copper wirebars (mining magazine, 1984). The blended concentrates from the Khetri, Malanjkhand and some imported containing some 16-22% Cu and 10-20% moisture, are fed to the Outokumpu flash furnace. The slag tapped from the furnace contains some 1.5% Cu and is recycled to the concentrator. The mate obtained from the furnace contains about 45% Cu and this is further treated into two converters and cast into 220kg anodes containing 99.4% Cu.

The anodes are refined in a tank house to yield cathodes of 99.95%Cu purity which are finally sent to wirebar casting plant.

The gasses from the sulphur are used to generate sulphuric acid and super phosphate fertilizer by-products.

2.14 Central services

Electric power is supplied by the Rajasthan state's electricity board and the maximum demand on this utility is 32 MVA. In addition the complex has its own diesel generator sets with a total capacity of 26 MW.

There are number of workshops undertaking maintenance and repair work. The central store holds about 50,000 items, the annual turnover being about 13,000 items.

2.15 Personnel

The Khetri Complex employs some 8,600 persons. The mines and process plants operate on a continuous three shift basis plus a general day shift. In the mines, 40% of the total labour strength is allocated for direct jobs (operations). The average absenteeism at both mines is about 23% (Industrial Engineering Report, KCC, 1986). the distribution of manpower amongst various units of the complex is given in table 2.9.

Unit	Officials	Total	%
Mining	249	4,377	51
Concentrator	28	373	4
Smelter and Refinery	78	989	12
Maintenance			
(Electrical and Mechanical)	78	703	8
Services			
(Civil and Hospital)			
Administration	289	2,185	25
Total	702	8,627	100

Table 2.9 Personnel employed at KCC. (Mine planning cell KCC, 1984).

2.16 The Rajpura Dariba Mine

The Rajpura Dariba mine belongs to Hindustan Zinc Limited, India. The mine has got Pb-Zn-Cu deposits. The flat back cut and fill stoping method is used to mine the deposit. The data related to stope design, costs and performance were collected from this mine by the author during his visits in 1984 and 1986.

2.17 Summary

The deposits at Khetri and Kolihan mines have a low grade copper content. The rocks of the deposit are hard and abrasive. Strata are quite competent. The deposit at Khetri mine is thin and scattered compared to that at Kolihan which is thick and concentrated. This difference in orebody geometry has considerable influence on mining costs.

Ore reserves are estimated at a fixed cutoff grade of 0.5% Cu since 1974 whereas there has been a constant increase in the costs of input materials and wages. Conventional methods of ore estimation, without the assistance of computers, are used.

The normal mining method is mechanised sublevel open stoping. However, DTH stoping is being introduced in some of the sections of the mines.

There is a steady growth to achieve the rated production and higher productivities from the mines, but the Kolihan mine was operating at 51% of the rated capacity in 1983-84.

Besides production of copper, the complex also produces sulphuric acid and supper phosphate fertilizer as by-products.

2.18 Discussion

The mines and plants have been designed to use modern equipment and latest techniques. The price of copper (Indian) has not kept pace with the increase in cost of resources. Most of the ore reserves have yet to be mined out. It is obvious that following the present trend, the project may not achieve its desired goals in the long run.

In this type of operation a dynamic decision making process is warranted. This is not feasible without the use of computers. Therefore one of the aims of this research is to analyse basic geological information and to develop various stope designs using main frame computer and its graphical aids.

Chapter 3

3. Exploration activities and ore reserves estimation - Khetri mines

3.1 Introduction

The objective of this chapter is to overview the exploration and ore reserve estimation activities undertaken at KCC. These provide a basis for the comparison of the existing ore reserves and the one estimated through the technique used in this research project.

Firstly, details of preliminary exploration and core logging procedure are described. The details of exploratory mine workings, underground drilling and chip sampling follow.

Ore estimation method is described briefly and finally, the reserve classification and grade control practices are outlined.

3.2 Brief background:

The ancient exploration and exploitation activities in the Khetri district, as mentioned in chapter 2, were discontinued in 1872. The prospecting and exploratory mining undertaken, by the Geological Survey of India (GSI) and the Indian Bureau of Mines (IBM) during the period 1953-61, included surface drilling, aditting and shaft sinking and underground drilling. The initial reserve estimates of the area were assessed by the IBM and were based on a limited, widely spaced data set. The following section describes the exploratory work undertaken by HCL since 1967.

3.3 Surface exploration

The preliminary exploration included a topographical survey, geological mapping and the location of the large number of old workings and slag heaps. The presence of old workings proved a useful guide in identifying potential mineral resources in the lease hold area and planning the surface drilling pattern (Tiwari, et al. 1981). The potential areas were then diamond drilled to prove the depth and strike continuity of potential ore zones.

At Kolihan the surface holes were drilled at intervals of 25m along the longer axis of the orebody. But the spacing across this axis varied according to the planned depth of hole. The direction of drilling was kept either due East or West and the length of holes were in the range of 250-550m.

At Khetri mine the surface holes were spaced at 60m intervals along strike. However, the 'echelon' pattern of orebodies called for some sections to be drilled at 30m spacing. Beyond 180m depth the drilling from surface proved to be very costly (due to slow rate of drilling) and less useful as the boreholes were found to deviate considerably (Bhatnagar, et al. 1979).

The diamond drilling work undertaken by various agencies until 31-4-1984 is summarised in table 3.1.

Agency	Diamond Drilling Surface	U/g	Total metres	Number of Holes Drilled Surface	U/g	Total
GSI	8,348	3,970	12,318	19	47	66
DAE	674*	620*	1,294	5	9	14
MECL	1,450	-	1,450	3	-	3
HCL	7,235	48,832	56,064	18	527	545
Total	17,704	53,422	71,126	45	583	628

Table 3.1 An account of diamond drilling work undertaken at Kolihan mine.
* - Drilled exclusively for radio active minerals in hangingwall of the orebody. DAE - Atomic Mineral Division, MECL - Mineral Exploration Corporation Ltd. (source: addendum report Kolihan mine, 1984-85).

A total of 53,422m of underground diamond drilling undertaken by HCL resulted in some 17 million tonnes of reserves including all categories of reserves, described later (addendum report Kolihan, 1984-85). Thus the 'reserve' tonnage of underground diamond drilling amounted to 338 tonnes pre metre.

The surface drilling data is indicative only and could not therefore be relied upon for detailed mine layout and stope planning purposes. Underground drilling is therefore carried out to confirm the mineralisation indicated by the surface drilling.

3.4 Logging of Drillhole Data:

Table 3.2 describes a typical core logging pro-forma (record form) used at Kolihan mine to record the core details for both surface as well as underground drilling. A similar pro-forma is used at Khetri mine.

KOLIHAN GEOLOGY DEPARTMENT

- CORE LOGGING DETAILS

Bore Hole No. Date Logged by Box No. Page No.

Latitude Departure Bearing Reduced level Hole depth

RUN From	RUN To	RUN Length	Core length	Core recovery	Size of core	Colour	Texture/ Grain size	Mineralogical composition	Rock type & structural features	Mineralisation pattern	Percentage of sulphide & oxide

Table 3.2 A typical core log pro-forma used at Kolihan mine. (Geology Dept. Kolihan mine).

Initially the lithology and physical properties of the rocks are described as a result of visual inspection by the mine geologists. The structural features, that are present in the core are noted. Such features include faults, folds, dykes, joints, bedding planes, etc. present in the core recovered. A note is also made of any infilling present. The rock coding pattern used at Kolihan mine is given in table 3.3.

Code number	Rock type
1	Quartz vein
3	Meta basic rock
4	Hybrid rock
6	Argillite
7	Quartzite
8	Andalusite phyllite/schist
9	Carbonaceous phyllite
10	Garnet chlorite/biotite schist
13	Amphibole quartzite
15	Amphibolite
16	Crystalline lime stone
17	Magnetite rock+amphibole

Table 3.3 Rock coding pattern used at Kolihan mine. (Geology Dept. Kolihan mine).

Prior to 1985 drill core samples were taken all the length of sulphide mineralisation. Since then the core has been split into 1m lengths (Tiwari, et al. 1985). The mineralogical, lithological and structural features of each 1m length are recorded. The core is split using pneumatic core splitters. Samples are prepared using jaw and roller crushers and pulverisers for chemical analysis.

3.5 Exploratory mining

The decision to develop these mines was taken on the basis of the initial estimates given by the IBM, and the results of surface drilling. Information derived from the surface boreholes, exploratory workings and some additional adits and inclined shafts (described in chapter 2) formed the basis for deciding to undertake underground exploration. This included sinking shafts and developing exploration drives in the footwall of the mineralisation at 60m vertical intervals. In most cases these drives have become haulage ways.

Positioning the footwall drive: The mine planning requires the footwall drive to follow the footwall contact of the mineralisation within 10 to 15m. Placing the drive further away into the footwall, as a precaution against the fluctuations in ore geometry, results in an increase in development and longer distance for hauling the ore from the stopes (fig. 3.1). It should be noted that placing the drive within 10m to 15m from the footwall does not help the drilling because such a lead (parting) is not ideal for exploratory drilling, as the angles of intersection of some drill holes to the orebody become acute (Bhatnagar, et al. 1979).

3.6 The underground diamond core drilling

Bhatnagar, et al. (1979) reported that "The area (Lat. 3200-2900) between 350-300m levels (Khetri mine) was first interpreted on the basis of surface drilling. The same area when explored by underground diamond drilling showed a marked change in the reserves and grade of individual lenses. Surface drilling is a costly and time consuming affair and the information gathered has only restricted use for initial mine planning only".

At Khetri mine the underground drilling 'fans' are spaced at 15-30m interval. Generally up holes are drilled, but whenever necessary a few down-holes are also drilled. At Kolihan, the 'fan' spacing is 25m and generally down-holes are drilled. Horizontal holes are preferred due to less drilling required to obtain maximum information. Using this information the extent of lateral development at a working level can be planned.

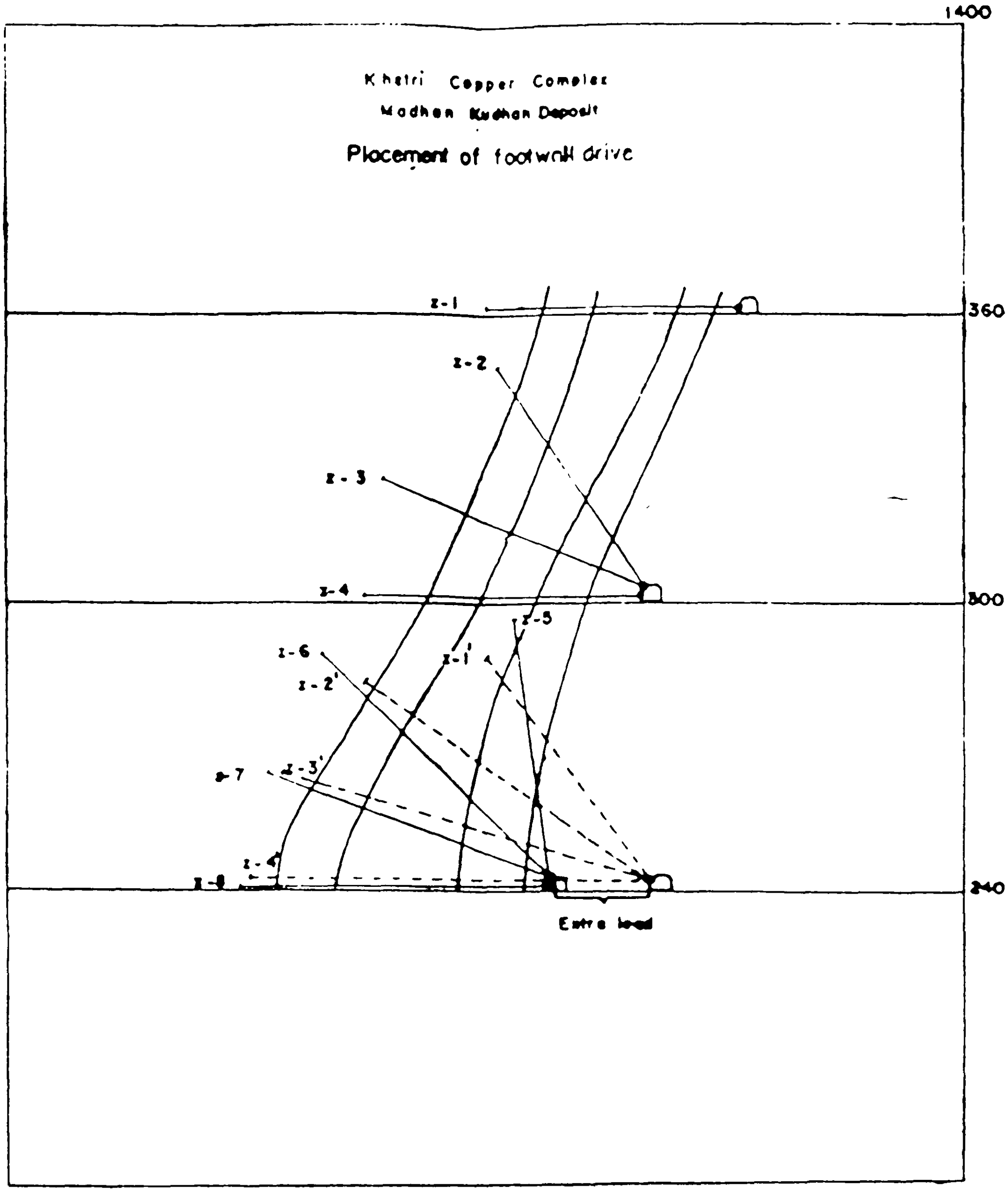


Fig. 3.1 Positioning footwall drive, Khetri mine (Bhatnagar, et al. 1979).

3.7 Ore reserve estimation:

The assay values obtained by surface as well as underground drilling programs are used to formulate a zone which contains the values at or above the cutoff grade of 0.5%Cu with a minimum width of 3m.

The assay zones as well as the lithology and other geological features of the deposit are plotted on cross sectional maps, prepared at underground drilling positions, all along the deposit.

The ore reserves are estimated on longitudinal sections. Reserves blocks are demarcated taking into account the area of influence for any particular category of ore reserves. "The geological interpolation for 15-30m at Khetri (and 12.5m at Kolihan) is only a broad guide to the boundaries, and any development along them may bring unexpected grade variations" (Bhatnagar, et al. 1979). In order to define the boundary of ore zone in stopes and development workings, a close chip sampling is also undertaken.

It should be noted that following this practice a grade-tonnage relationship for the deposit is not established.

3.8 Chip sampling

Followed by the underground diamond drilling program, the decision to develop stope workings is taken. The stope workings are sampled through chip sampling to prepare assay plans and sections to delineate the ore zones and ore geometry.

In this method the wall of a cross-cut is divided into 1m wide samples. Within each sample, 25cm spaced grids on the entire wall are marked and chips are cut from these grids. Similarly, a sample is cut from the face of a drive at 25cm spaced grid centres as shown in fig. 3.2 (Tiwari, et al. 1985).

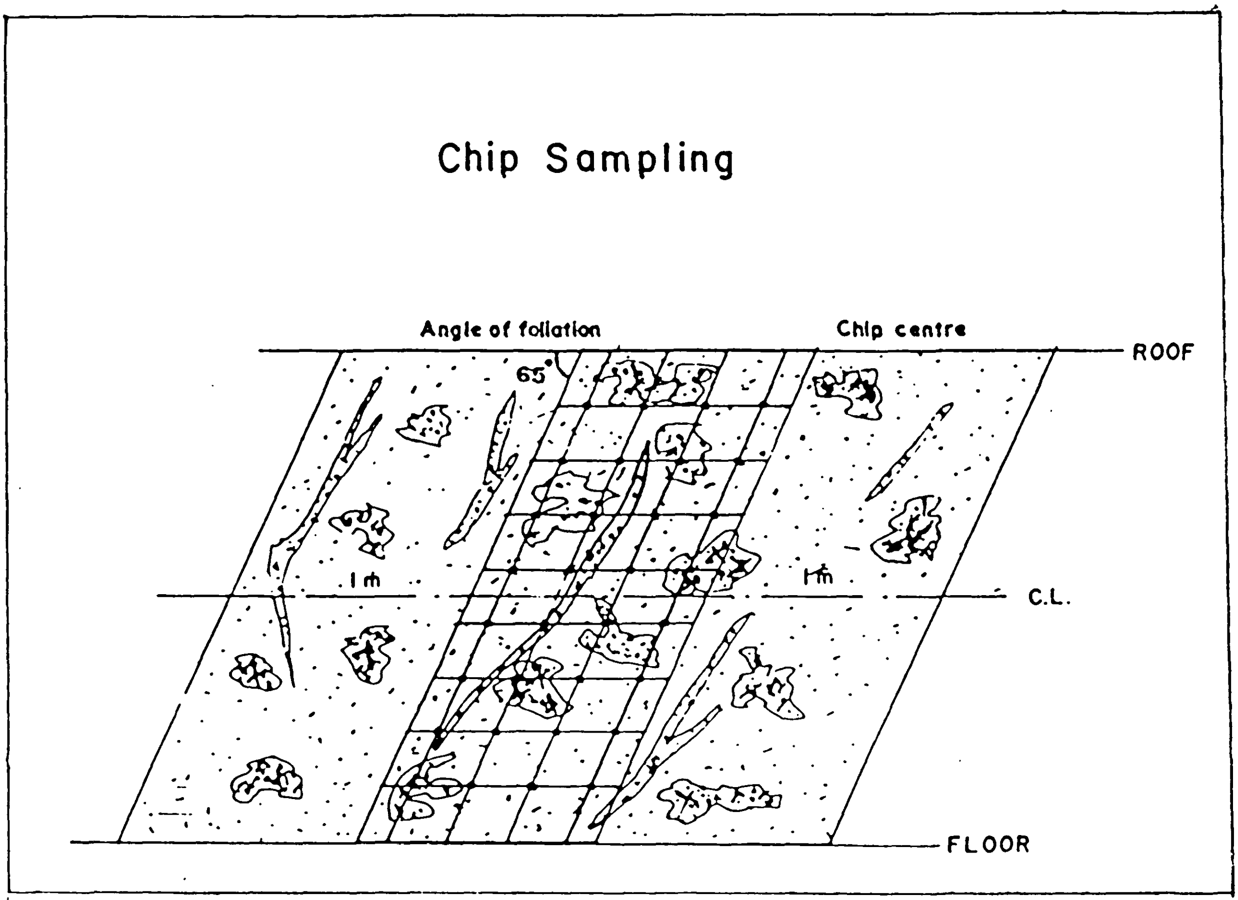


Fig. 3.2 A typical chip samplinng pattern, Khetri mines (Tiwari, et al. 1985).

This supplementary sampling information provides a check to the orebody profiles projected on the basis of the underground drilling.

3.9 The ore reserves classification

The ore reserves are classified as outlined below:

Drill inferred reserves: The reserves are estimated outside the periphery of drill indicated category (defined below), projected on the assumption of continuity.

Drill indicated reserves: These reserves are estimated by taking a 30m square around a surface drill hole intersection on longitudinal section at Khetri mine. The corresponding distance considered at Kolihan mine is 12.5m, around a surface/underground drill hole intersection.

Partly blocked reserves: Estimated on the basis of underground drilling. Stope preparation in progress.

Fully blocked reserves: The stopes ready for production.

The reserves as classified above are diagramatically presented along a longitudinal section (fig. 3.3). The progression of reserves from 'drill' to 'block' category is often accompanied by many changes in configuration with regard to quality (grade) and quantity (tonnage) (Bhatnagar et al. 1979). This is mainly due to amount of information used to estimate different categories of reserves.

On the basis of this reserve classification, the status of the reserves at Khetri and Kolihan mines, as on 1-4-1983, is summarised in table 3.4. It can be seen that at Kolihan mine 44% of the total reserves are ready for stope design purposes, but the corresponding percentage at Khetri mine is 20. This is due to the difference in the geometry of the deposits at the two mines, as

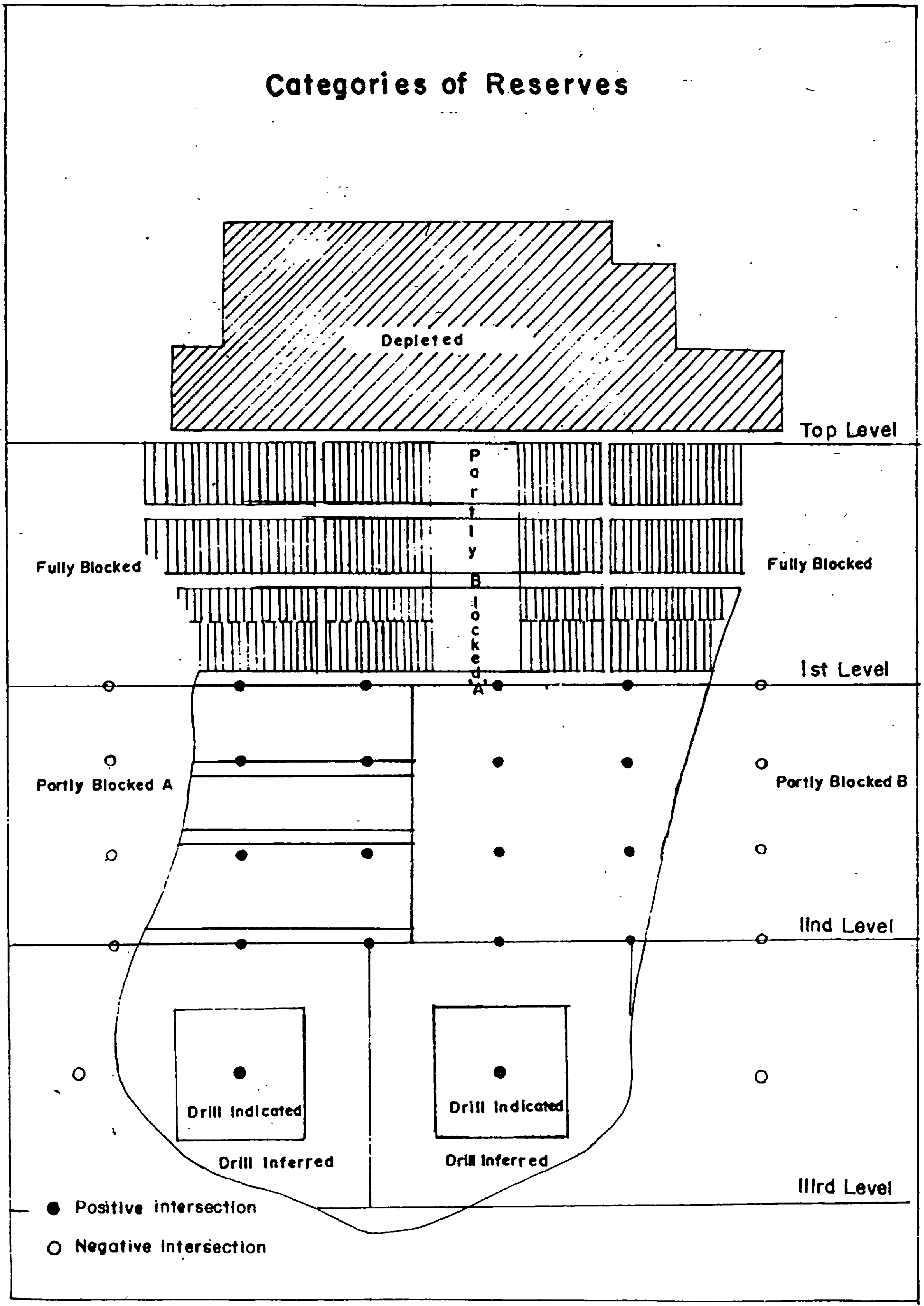


Fig. 3.3 Diagrammatic presentation - Ore reserves classification at Khetri mines (Tiwari, et al. 1985).

described in chapter 2.

Mine	Estimated reserves *	Depleted reserves *	Net reserves *	Category of Reserves: Fully blocked *	Partly blocked *	Drill indicated *
Khetri	43.4x0.92	4.5x0.85	38.9x0.93	1.2x.80	6.7x.97	31x.92
Kolihan	32.8x1.47	3.6x2.1	29.1x1.4	3.1x2.5	9.8x1.4	16x1.1
Combined	76.2x1.15	8.1x1.4	68x1.14			

Table 3.4 Status of reserves as on 1-4-1983, KCC mines.
(* - Reserves X grade, Reserves in million tonnes and Grade in % Cu, source: addendum reports Kolihan and Khetri mines, 1983)

3.10 Grade Control:

Verma (1981) stated that "The grade of ore fluctuates from 0.3 to 0.4% Cu to as high as 6.0% Cu". Regular grab sampling at the draw points, loaded mine cars, grizzlies and run of mine conveyors is carried out for the purpose of grade control and reconciliation.

Mill feed production schedules are made on the basis of the blasting schedule of various stopes. The calculation of grades for all the stopes is made on the designed ring patterns.

The longitudinal sublevel stopes provide a better mixing of the low and high grade ore stringers as the rings are blasted across the axis of mineralisation. However, transverse sublevel stopes sometimes pose serious problems with high variation of stope grades. These arise out of the fact that the rings are blasted along the axis of mineralisation by which there is hardly any scope for inherent mixing of blasted ore (Tiwari, et al. 1985).

Tonnage from development headings amounts to 20% of the total production. In order to segregate ore and waste from these headings strict control is exercised, but this is not always possible as some of the development headings are partly in ore as well as waste rock.

Based on the recommendations of the Task Force II, set up by HCL in 1979, a specific gravity of 3.0t/cu.m. is used (Soni, 1981). Associated moisture with the ore is also accounted for.

3.11 Summary:

The preliminary exploration of Khetri deposit was carried out by the surface drilling. There is a continuous on-going programme of core drilling from underground footwall drives to prove additional reserves.

The information gathered by surface drilling was used for initial mine planning purposes only. The underground drilling provides the basis for development, stope design and ore reserve calculations.

The conventional longitudinal section technique of ore estimation is used. Reserves are estimated at 0.5% Cu grade and ore zones below 3m widths are not included. The interpolation on this basis is only a broad guide to the ore boundaries and any development along them may bring unexpected grade variations. Chip sampling of most of the development workings is therefore an essential step in the exploration sequence at these mines.

The ore reserves are classified taking into consideration the sampling density and also the parameters of exploitation.

3.12 Discussion:

The ore estimation technique used at KCC mines does not consider mineralisation characteristics but an area of influence around a point sample. This is specified on the basis of experience, which in turn needs a close sampling of the mine workings and strict measures on grade control.

In chapter 2, the mining cost at the Kolihan mine was found to be less than that at the Khetri mine. This was mainly due to difference in the geometry of their orebodies. The question arises: why should the possibility of lowering or raising the cutoff grades at Khetri and Kolihan mines, from the present one, not be explored? In the absence of a grade-tonnage relationship this would be difficult.

Thus, there is a need to review the ore estimation technique and establish the grade-tonnge relationship for the deposits. Another mineral inventory estimation technique for analysing the basic sampling information obtained from these mines will be investigated in the following chapter.

Chapter 4

Evaluation of mineral inventory

4.1 Introduction

In chapter 3 a need to select a mineral estimation technique was pointed out. It is the intent of this chapter to briefly review the mineral inventory evaluation techniques available and select one which will be used to evaluate the mineral inventory. The considerations to be made while modelling a deposit and the logical steps that should be followed to evaluate it, are outlined. A distinction is made between mineral inventory and ore reserves.

4.2 Literature review - Mineral inventory estimation techniques.

The basic geological data derived from boreholes, exploratory test pits, drivages, and mine development, includes information on parameters such as mineral thickness, grade, depth below surface, structural attitude of beds, rock types and lithology of the beds over-lying and under-lying the deposit. This data is usually supplemented by analytical information obtained from chemical, physical and structural investigations. The relatively small number of sample points in the mineralised zone, and sparsity of sample volume makes it necessary for the available information to be interpolated between sample points and extrapolated into area beyond the limits of sample points. Such interpolation and extrapolation are usually undertaken by making suitable assumptions about the parameter's continuity in the mineralised zone and building a suitable mathematical model.

The mathematical model provides the values of the chosen parameters at the nodes of the superimposed grids of any desired dimensions. This may be a grid cell equal to size of a mining block or any other criterion. The network of values for any parameter, can then be used for mine planning, designing, and controlling functions. Additionally, this data can also be used to calculate the mineral inventory and grade in total and/or for any desired block.

The in-situ mineral inventory or reserve which can be mined at profit is termed as ore. Converting the mineral inventory into ore reserve thus introduces the process of mine design. Mineral computations are made at all stages of the mine life. Mineral reserves computed during the the exploration phase of the mine life may not require such precision as needed during production stage.

4.3 Mineral inventory modelling techniques

Popoff (1966) described following three principles used to calculate mineral inventories by the conventional methods.
-the rule of gradual change
-the rule of nearest point or equal influence and
-rule of generalisation.

Under the first two principles fall the conventional geometrical methods such as:
-triangle or triangular prism method
-method of polygons
-cross sectional method
-angular bisection
-rectangular blocks and
-inverse square distance weighting (IDW) method can be explained.

The rule of generalisation is really not a rule at all, and is usually arbitrarily applied as a matter of judgement reflecting past experience. The conventional methods mentioned above are the functions of geometry and distance between samples which simplify the calculations of volume and grade. They are not function of the mineralisation characteristics which they propose to measure (Barnes 1980, p 54).

4.31 Classical statistics

Classical statistics and tests have been used in mineral reserve evaluation for many years. However, these methods assume that samples taken from an unknown population are randomly selected and are independent of each other. In addition, they assume that data has either a normal (Gaussian) or a log-normal distribution (David, 1977). In the context of a mineralised zone, this implies that the position from which any sample was taken is not important. Sample assays taken from holes drilled in close proximity to one another, within a mineralised zone, obviously should not be random or independent. Closely spaced samples should demonstrate some correlation, in other words reflect some degree of continuity in the mineralisation. If this is not the case either there is no continuous mineralisation or the samples have been taken over an excessively large spacing.

Despite of the limitations of classical statistics in mineral reserve estimation, much can be gained from studies of sample distribution in terms of providing estimates of ore tonnage, grade and metal content (Barnes 1980, p). Traditional statistical decision theory has played an important role in orebody modelling and ore evaluation methods (Hazen, 1967). Excellent descriptions of statistical methods, approaches, and calculations with respect to mineral reserve estimation have been given by Sichel (1952, 1973), Krige (1962, 1978), Hazen (1967) and Koch and Link (1970, 1971).

4.32 Geostatistics

The geostatistical grade estimation method is known as "kriging". It was named after the pioneer of the technique in South African gold mines, D.G. Krige, in early 1950, by Matheron, to designate a best linear unbiased estimator for assigning values to mining blocks using geostatistical techniques. Like IDW method, kriging also involves assigning grades to a uniformly-spaced grid of points as a weighted average of nearby sample grades. However, in kriging, weighting factors are not assigned as an arbitrary function of distance. Instead a mathematical model of sample grade variation within the orebody is first determined by a variogram study of the sample themselves. Next, the variogram model and the sample locations, including distance separating the samples themselves, are used to setup a series of linear equations which are solved to determine the optimum weighting factors.

Raymond (1982, p20) described kriging as an improvement over IDW method in several respects:

1. The most important advantage is that, with kriging, weights are not chosen arbitrary (as explained above).

2. With kriging, overweighting to clusters of samples is overcome by considering not only the distances from samples to the point to be estimated, but also the distance separating the samples themselves.

3. By selecting only the nearest samples, optimum estimates are provided by kriging even in the areas of widely sample spacing.

4. With Kriging, expected estimation error may also be calculated for each point estimate or for the estimate of larger blocks of material. This calculation provides a method of checking variogram and an appreciation of the reliability of the estimates. Although determination of estimation error is often quoted as the major advantage of kriging . it is in fact of secondary interest in production estimates. The major requirement is in the best grade estimate from the available information.

The merits of kriging over other methods with regard to improved estimation have been discussed by Royle (1979, p. 101) and Rendu (1979, p. 207). Full description of theory has been dealt with by O' Leary (1980), David (1977), Barnes (1980), Journel et al (1978), Royle (1975) and Clark (1979).

4.33 Non-linear estimation techniques in geostatistics

There are several non-linear estimation techniques available in geostatistics. Most common amongst them is log-normal kriging. When the grade distribution of the samples is log-normal, the use of log-normal kriging is often recommended (e.g; Rendu, 1978). Log-normal kriging involves variogram calculations and kriging with logarithms of grade (or grade modified by a constant). Raymond (1982) states 'sometimes the logarithmic variogram is quite variable from one area to another. By assuming one variogram for the total area, an additional error is introduced in converting from logarithmic to arithmetic grades'. A naive application of the log-normal approach will inevitably lead to erroneous results as the method is highly distribution dependent (Kwa, 1984).

Instead of log-normal kriging, cross-validation techniques can be used to circumvent any biasedness in the estimation (O' Leary, 1986, personal communication).

Common practice currently used for the grade and tonnage estimation of a mineral deposit is to estimate the grade of a panel (or block) and sum up the tonnage of the panels whose estimated average grade is higher than a cutoff grade. If the recoverable tonnage and grade is made on these panels, a bias is made, over-estimation of tonnage for cutoffs below the mean, which leads to the result of so called "Disappearing tonnage problem". To solve this problem Matheron (1975) provided a more promising non-linear technique known as Disjunctive kriging (DK) (Young, 1982). The objective of this technique is to estimate the density function of the grade distribution of selective mining units, within a panel, from the grades of the nearby samples. Using this density function a complete grade-tonnage curve can be calculated for the panel. This is an area where future research could be undertaken to evaluate a mineral deposit taking into account the density function at varying grades within it.

The limitations of the method has been expressed by Young (1982) - "the positiveness of the estimated conditional density function may not be guaranteed for any location and DK estimates may face the problems of yielding odd recoveries and figures such as negative recoveries and decreasing ore grade with higher cutoff grades."

Verly (1983) states that a more promising non-linear technique is the multivariate gaussian kriging approach currently being developed at Stanford university. This method is very attractive in the sense that it is more mathematically robust and is fairly easy to implement. However, there are some problems: namely the 'despiking' of zero values and of the considerable computer run time needed (Kwa, 1984).

In addition to the non-linear techniques, there exist some non-parametric methods. These are indicator kriging (Journel, 1983) and its variant, probability kriging (Sullivan, 1983). Both methods are easy to implement but suffer from order relation problem (Kwa, 1984). Sullivan (1983) and Davis (1983) showed how this problem can be solved using quadratic programming and an approximation technique has been suggested by Journel (1983). According to Sullivan, the relative merits of probability kriging over indicator kriging are firstly, that it provides a more accurate estimate (lower estimation variance) and that of the smoothing problems that affect indicator kriging are eliminated. However, disadvantages are that more variograms must be calculated and modeled for each value of cutoff target.

4.34 Application of geostatistics to evaluate copper deposits

Since its inception the use of geostatistics in mineral inventory evaluation is very considerable and the application of geostatistics has grown rapidly in recent years (David, 1977). Matheron mainly dealt with vein type of deposits. Serra and O' Leary (1973) evaluated some of the iron ore deposits and Clark (1979) applied this theory to Cornish tin deposits.

Raymond (1975-79) initially used this technique for evaluating very erratic low grade copper deposit of Newmont Similkameen mine in Canada. In 1979-80 Raymond applied geostatistics for Mount Isa's copper ore bodies in Australia. He reported that results by log-normal kriging were inferior to ordinary kriging. Kriging gave a reasonably consistent ore boundaries from section to section, which allowed confident design of large stope even in erratic areas.

A study by Journel and Segovia (1974) provides a model for copper deposits of El Teniente mine in Chile. The mine is ranked as largest copper mine of the world.

Huijbergts and Segovia (1973) modelled Extico copper deposit in Chile.

Hinde et al. (1984) used isotropic semi-variogram for modelling south and north zones of Malanjkhand open-pit copper mine, India. The semi-variogram for both zones have a range of 45m, the sill in the south zone is considerably higher than that for the north, reflecting higher variability of grade in the south.

Biswas et al. (1985) made geostatistical studies for Mosaboni copper mine, India and concluded that an optimum prospecting drilling pattern can be obtained using a fraction of range obtained through semi-variogram studies.

Thus, a brief review of the available mineral estimation techniques indicates that the geostatistical approach (kriging) is an improvement over the conventional methods in several respects, and has proved to be a reliable estimation procedure for several deposits all over the world. Hence, this technique has been used to evaluate the mineral inventory in this study.

4.4 Important considerations for evaluation of the mineral inventory

4.41 Homogeneity and mode of origin

King, et al. (1982) presented a useful diagram (fig. 4.1) and table 4.1 to present

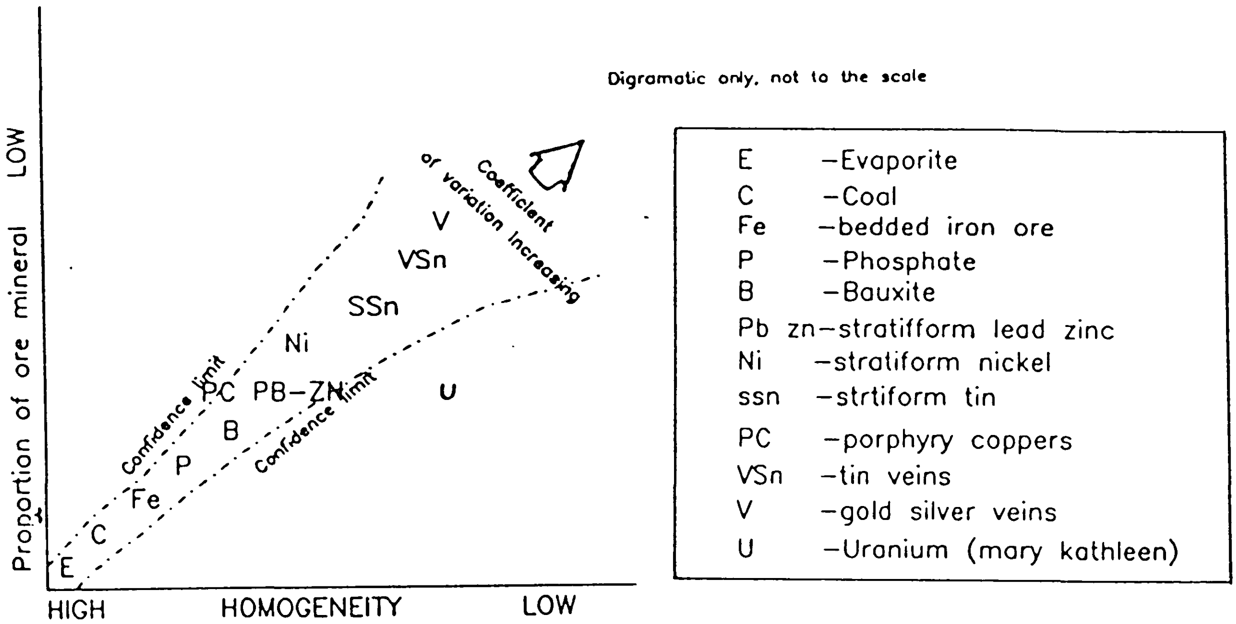


Fig. 4.1 Relative magnitude of confidence limits for key element evaluation (after Caras, 1984)

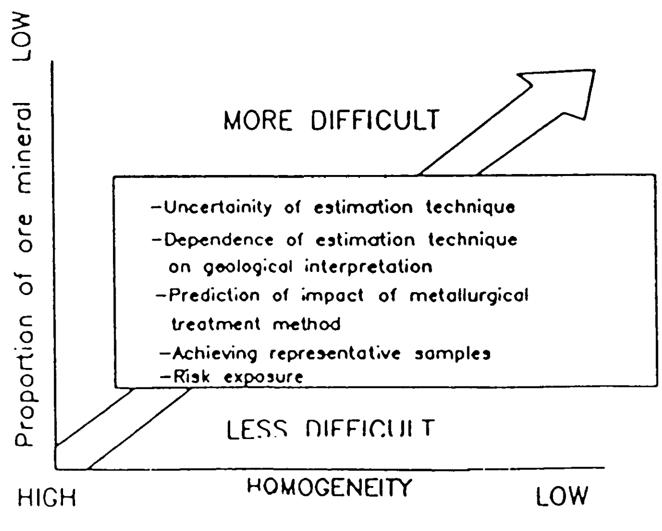


Fig. 4.2 General trend of increasing difficulty in deriving ore reserves estimates (after Caras, 1984)

proportion of ore in a mineral inventory as a function of homogeneity. It can be seen that how the magnitude of the confidence limits decreases with an increase in the coefficient of variation. In general it can be stated that the more homogeneous a deposit, the less difficult it is to evaluate. A gold deposit can be considered to be more difficult to evaluate than a coal deposit. A copper deposit can not be considered easy to evaluate.

Metal or mineral	Percentage of element	Percentage of mineral
Diamond	1/50 ppm	same
Gold (in veins)	5 to 10 ppm	same
Tin	0.3% to 1%	0.5% to 1.5%
Copper	0.5% to 3%	1.5% to 10%
Nickel	1.0% to 4%	5.0% to 20%
Lead plus zinc	10% to 20%	15% to 30%
Iron (high grade)	60% to 65%	85% to 93%

Table 4.1 Proportion of valuable content in mineral deposit. (After King, et al. 1982).

The degree of difficulty associated with evaluating mineral inventory can be represented by a diagram (fig. 4.2) given by Caras et al. (1983). From fig. 4.2 it is evident that moving from concordant (sedimentary and biological, indigenous) deposit to more discordant (migratory, exotic) deposit the difficulty associated with evaluating orebodies increases.

Copper deposits often occur as massive and/or disseminated bodies. The metal content varies significantly in sympathy with lithology. The metal tends, however, to occur in swarms of thin veins and this combined with low average grade makes porphyry coppers difficult to evaluate to the precision demanded by economics (King, 1982).

Referring fig. 4.1 and table 4.1 it is possible to assess without mathematics whether the mineral to be evaluated involves low or high risk in estimating the grade.

4.42 Geological and mineralogical boundaries

Mineral deposits rarely are completely homogeneous and often consist of subsets separated by geological and mineralogical boundaries. Sometimes different mineral populations are difficult to distinguish and may be individually lumped together - often without serious consequences, but more often with detrimental effects upon analytical results (Barnes 1980, p. 62). Before applying any mineral estimation technique it is, therefore, important to identify different populations.

The geological boundaries are usually defined by a change in lithology and/or stratigraphy, or the structure. For example wallrock, fault and fold boundaries. The geological boundaries represent sharp environmental changes in terms of mineral zone, usually ore-waste or mineral non-mineral. For example, in porphyry coppers there is commonly an association between lithology and higher and lower metal content, making interpretation of lithological outlines an essential feature in the process of mineral estimation.

4.5 Computation of the mineral inventory

The geostatistical analysis has been undertaken using 'MINPAK' library of

computer programs developed by O'Leary and Forbes (1983). This library contains a series of Fortran subroutines which undertake the following:
-statistical analysis
-structural analysis
-the estimation of grade - kriging and
-miscellaneous

4.6 Logical steps followed:

Having installed the basic data set, on the computer at Imperial College Computer Centre (ICCC), the following procedure was adhered to in order to calculate the mineral inventory:

4.61 Graphical presentation of data:

Using the common plotting library and some of the graphics software available at (ICCC), the geographical locations of the boreholes and the sampling points has been presented graphically. The same routines were used to represent the results of an analysis afterwards.

4.62 Statistical analysis and cumulative probability distribution:

Following the graphical presentation of input data, a statistical analysis has been carried over to assess:

-general statistics of the deposit
-grade distribution and
-presence of one or more populations within the data set.

The cumulative probability distribution is a fairly sensitive measure of the continuity of local mineralisation. A smooth curve with a similar shape throughout suggests homogeneous mineralisation. An abrupt change in the curve suggests a geological discontinuity limiting mineralisation.

Based on the probability distribution and geological reasoning the data set, if needed, is further split to represent different geological regimes or populations.

4.63 Structural analysis - the semi-variogram:

Experimental semi-variograms in different directions are computed to identify the continuity of the mineralisation in these directions. Any anisotropy, if it exists, is noted. In this study it has been observed that enough sample pairs were not available in the directions other than those of drilling. Experimental semi-variograms in these directions showed 'noise'. An average experimental semi-variogram, using an angle of regularization of 180 degrees, in most of the cases, was therefore evolved.

Several intrinsic schemes or mathematical models have been proposed over the years to represent a probability function that describes the behaviour of various experimental semi-variograms. These are: the Linear model, the DeWijsian model and Spherical model (Clark, 1982).

David (1973, p. 103) stated that the spherical model occurs naturally in all the deposits where grades become independent of each other once a given distance, or range, is reached. Spherical models are usually applicable in most of the sedimentary deposits and porphyry copper deposits. In this study the spherical model has been used to fit the experimental semi-variogram.

4.64 Trend surface analysis:

Sometimes semi-variogram exhibits trend or drift in data set. In order to deal with this problem, any trend in the data set should be removed. This can be achieved by fitting a simple surface to the sample data by trend surface analysis. The grade at any given point can be considered to be a function of drift component and residual component. Deducting the drift component from the data set the residual component can be obtained. The residual values can then be processed to compute the semi-variogram.

4.65 Checking the variogram model:

The technique referred as point kriging cross validation is used to test the validity of semi-variogram parameters. In this process a point value is estimated, not the block value (explained later). The kriging method remains the same except that each sample value is taken in turn, removed from the data set and its value kriged from the nearest surrounding sample values. In order for the model parameters to be valid, two criteria should be met:

a close correlation between the mean of the kriged estimates and mean of the actual value.
a close correlation between the average kriging variance and the average variance of estimation.

4.66 Block kriging

Block dimensions:

In underground mines, particularly where there is large variation in grade, blocks of small size are essential. In general, it can be stated that the more the precision of ore boundaries warranted the smaller should be the size of the block. For example, short term planning require a good knowledge of grade in relatively small blocks (Diehl and David, 1982). Since selectivity is an important point of the study (designing the stopes), a small block of 1m size is required (Deraisme, et al. 1984).

If a block or panel is relatively large, i.e. encompassing several samples, one can have more confidence in predicting its overall grade than for a small block size in which no samples have been taken (Barnes, 1980, p. 49). This is an important aspect which should be taken into account when planning the underground diamond drilling fans. Close sampling is essential for stope design purposes. Availability of data usually decides the smallness of the block (Royle, 1977).

In order to model the Kolihan deposit two different block sizes have been used:
Global estimation - 30m x 30m x 30m
Local estimation - 10m x 10m x 10m
However, for stope designing purposes the panels of 1m x 1m across the Khetri deposit have been used.

4.67 Kriging procedure:

The kriging parameters as outlined below have been used.

Number of samples required to krige a block: In this study minimum 4 (O'Leary personal communication) and maximum 15 samples were used to krige a block.

Radius of sample search: this parameter should be specified, preferably within the range obtained for model variogram.

Model variogram: for each of the mutually perpendicular directions, the nugget variance (Co), the transition amplitude (C) and range (A) should be specified. If an anisotropy exists, the ratio of anisotropy should also be specified.

Once the input parameters are specified the following procedure is adopted to krige each block in order:
1. Calculation of the average variability of samples contained within the dimensions of the block - the extension variance.
2. Location of the nearest 4-15 sample values to the block centre within the range prescribed.
3. If sufficient samples are not located (within the search range), the procedure is repeated for the next block.
4. Establishment of the kriging matrices-

a) set up the S-matrix which considers the expected variability between each of the nearest surrounding values and themselves.
b) set up the T-matrix which considers the expected variability between each of the nearest surrounding values and the block centre.
5. Calculate the kriged estimate and its kriging variance.
6. Locate the next block and repeat the same procedure till the last block is kriged.

4.7 Graphical presentation of the kriged results:

The kriging results have been presented in the form of:
-plans and sections showing the mineralised zones (chapters 5 and 6).
-grade tonnage curves to indicate tonnage and grade contained within a volume of ground for a series of given cutoff grades(chapters 5, 6 and 7).

4.8 Conclusions

There are a number of methods available to evaluate a mineral inventory from the basic geological data.

Selection of a method depends upon the geological considerations; exploration and/or sampling method; availability; reliability and volume of data; specific purpose of estimation and requirement of accuracy.

The geostatistical technique (kriging) appears to be an improvement over conventional methods in several respects. Based on this review kriging has been selected for the global and local estimations of mineral inventories of some of the sections of the mines.

Chapter 5

5. Global estimation - Kolihan mine

5.1 Introduction

This chapter describes the global estimation of Kolihan deposit using kriging. The estimation is based on the surface borehole information obtained from the mine. The purpose is to develop grade-tonnage relationships for the three lodes that have been identified.

5.2 Input data

5.21 Boreholes:

The collar details of 18 surface boreholes received from Kolihan mine are given in table 5.1. Table 5.2 is abstracted from a typical borehole record. Fig. 5.1 represents the geographical location of surface boreholes. On the basis of the information described in table 5.1 and fig. 5.1, it can be seen that the surface drilling is confined within a spread of 575m along the N-S and 550m along the E-W directions. The surface holes have been put at a regular interval of about 25m along N-S direction, but along E-W direction the interval is not fixed. The inclination of the holes is in the range of 44-71 degrees with horizontal. The holes were drilled from hilly terrain with their inclined lengths in the range of 176-580m.

The sampling interval is not uniform, it varies from hole to hole and within a hole

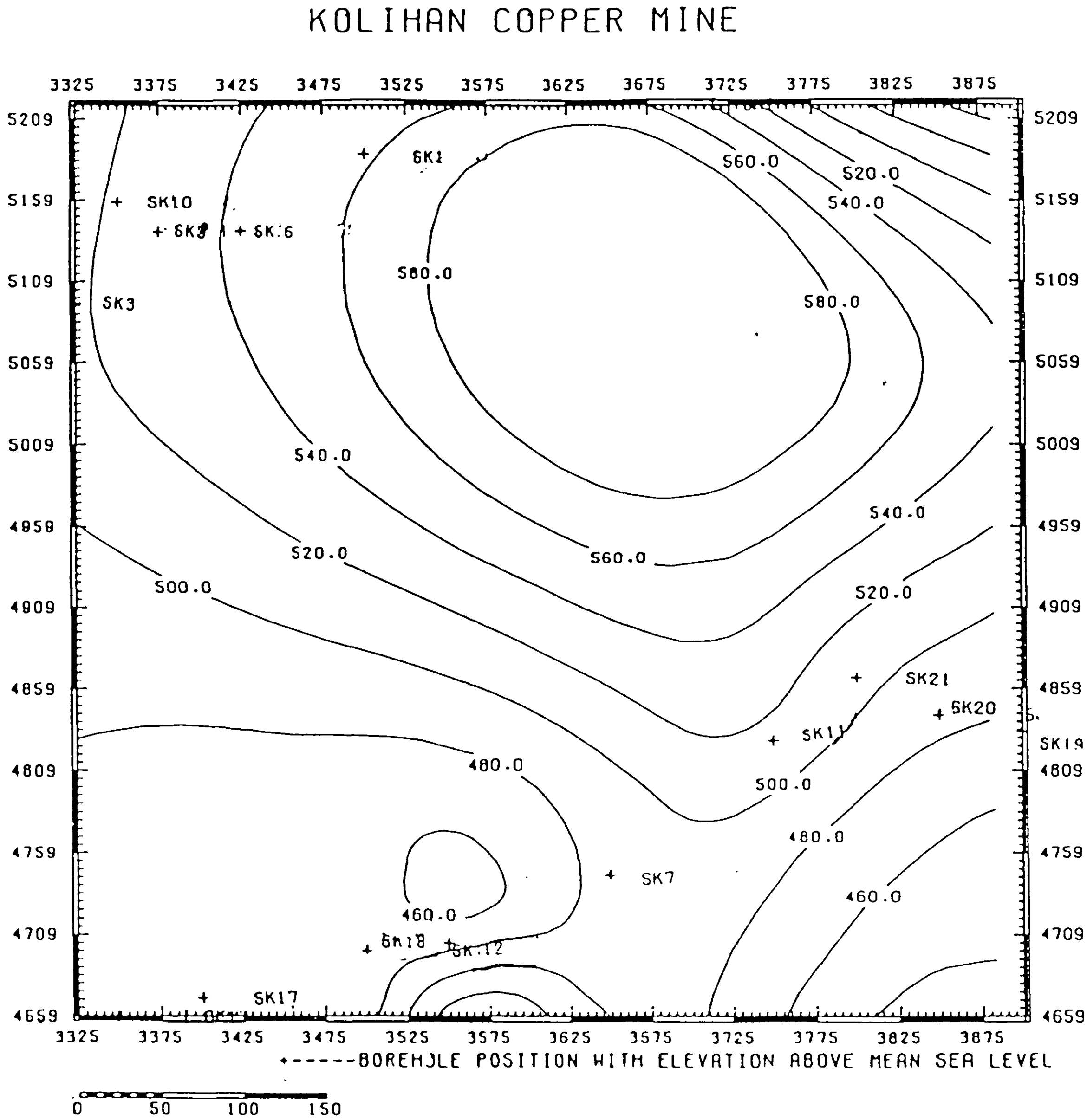


FIG. 5.1 GEOGRAPHICAL LOCATION SURFACE BOREHOLES.

itself. In most of the cases the recording of assay values is not consistent.

Borehole Nos.	Latitude (m)	Departure (m)	Azimuth *D/Deg.	Inclination Deg.	Surface R.L. (m)	Length (m)
sk3	3325.0	5095.8	270.0	68.00	517.3	274.0
sk13	3348.9	4659.4	090.0	44.25	460.8	551.1
sk10	3350.1	5158.2	270.0	68.00	522.1	543.2
sk5	3375.0	5140.3	269.8	58.25	530.3	326.8
sk17	3400.0	4670.8	090.0	38.50	460.9	501.8
sk6	3425.0	5140.7	272.7	65.16	542.7	481.3
sk18	3499.9	4699.5	090.0	38.00	469.9	501.5
sk1	3500.2	5187.7	271.8	46.00	557.2	274.8
sk4	3500.2	5188.6	271.8	71.75	557.1	483.3
sk12	3549.9	4704.0	090.0	59.50	470.2	579.8
sk8	3549.9	4704.5	090.0	68.50	426.1	176.6
sk7	3649.6	4745.7	090.0	45.16	484.7	528.0
sk16	3649.6	4745.4	090.0	54.50	484.7	502.4
sk2	3675.0	5218.0	270.0	51.58	559.5	365.1
sk11	3749.8	4827.6	090.0	60.00	514.0	472.1
sk21	3800.0	4866.2	090.0	62.00	507.3	470.8
sk20	3850.0	4843.2	090.0	56.00	485.7	492.0
sk19	3899.7	4833.1	090.0	56.00	478.9	487.2
Total						8011

Table 5.1 Collar details of surface boreholes Kolihan mine. * D - the direction of drilling.

From meters	To meters	Core length meters	Core recovered meters	% recovery	Assay %Cu
479.10	481.10	2.0	1.07	53	0.30
481.10	483.10	2.0	1.08	54	1.37
483.10	487.10	4.0	0.40	10	1.18
487.10	489.10	2.0	1.58	79	0.64
489.10	491.10	2.0	1.58	79	0.64
491.10	493.10	2.0	1.36	68	0.83
------BREAK--------					
499.00	500.22	1.22	0.77	63	3.63
500.22	501.45	1.23	0.78	63	1.02
501.45	504.05	2.60	0.32	12	0.97

Table 5.2 Assay values and core recovery surface borehole-12.

It can be seen in table 5.2 that the core recovery for surface borehole 12 varies between 10% to 79%. Overall the core recovery varies between 10% - 100%, but for more than 70% of the total samples recorded, the core recovery is greater than 50% of a run i.e. the sampling interval. However, for the purpose of this analysis all the assay values have been considered.

In addition, the deviation of each hole was recorded at 30m intervals. This information is compiled in table 5.3 together with the number of samples in each hole and the range of sampling interval. It can be seen that the deviation in some of the holes is considerable, particularly in the longer holes.

Borehole nos.	Sample nos.	Sampling interval	Collar angle	Deviated angle*	Deviation +/-
sk1	136	0.20-1.04	46.00	46.80	0.80
sk2	161	0.30-1.45	51.58	51.25	0.33
sk3	67	0.30-0.87	68.00	67.00	1.00
sk4	91	0.35-1.44	71.75	80.75	9.00
sk5	125	0.36-1.10	58.25	57.50	0.75
sk6	403	0.24-1.00	65.16	67.75	2.59
sk7	129	0.27-1.11	45.16	25.00	20.16
sk8	124	0.35-1.00	68.50	27.00	41.50
sk10	230	0.20-1.84	68.00	77.00	09.00
sk11	48	1.40-2.00	60.00	45.70	14.30
sk12	9	1.22-4.00	59.50	38.50	22.00
sk13	28	0.70-2.00	44.25	32.25	12.00
sk16	49	2.00	54.50	48.50	6.00
sk17	37	0.80-2.00	38.50	62.00	23.50
sk18	13	0.90-3.00	38.00	60.00	22.00
sk19	41	1.15-2.55	56.00	57.50	1.50
sk20	55	0.95-2.1	56.00	50.00	6.00
sk21	67	0.35-2.00	62.00	41.00	21.00
Total	1872				

Table 5.3 Details of sampling interval and deviation of boreholes. * The angle where the hole terminates.

5.22 Geological information:

The geological description and maps representing the cross sections through the deposit describe three distinct mineralized zones. These zones have been designated as lode1, lode2 and lode3, as described in chapter 2. Lode1 occurs in Chlorite-Biotite-Schist. Lode3 persists in Amphibole-Quartzite. Lode2 also occurs in host rocks as lode1 but it is confined to the northern side of the deposit at the hanging wall contact. It is also limited to the upper horizons of the deposit.

5.3 Sampling density:

The total diamond drilling core amounts to 8,011m , and 1872 samples were recovered. Considering mineralisation within (on an average) 550m along strike direction, 400m along depth and 100m across strike direction, the volume of ground evaluated averages 2,746 cu.m./meter drilled or 11,752 cu.m./sample.

5.4 The analysis:

The geological information of the deposit indicates that the sampling data should be separated into three populations. Lode1, lode2 and lode3 contain 1164, 195 and 453 samples respectively. Applying correction to the deviated boreholes, the position of each of the assay values at their geometrical centres were then computed. The spatial positions, with regard to Northing, Easting and elevation (measured above mean sea level), could then be calculated from the 'corrected' sample positions.

It can be seen from table 5.3, that the samples were not taken at equal intervals. A compositional change is necessary to consider the data with their equal 'support'. The sampling data can be composited at a regular interval using either rectangular integration or linear interpolation techniques (Davis, 1973). In this study, since the data recording was not consistent, see table 5.2, the compositing was not feasible. The data with their original assay values at their respective geometrical centres have been considered.

5.41 Statistical analysis:

Statistical software 'MINITAB' available at ICCC as well as the 'MINPAK' library were used for the statistical analysis. The descriptive statistics and frequency distribution for the raw data and log-transformed values for all the lodes were computed. The results are described in tables 5.4-5.6. The histogram plots of these lodes are shown in fig. 5.2.

Parameters	Raw data		Log-transformed values	
General Statistics:				
Number of data values	1164		1164	
Range of values	0.01	15.10	-4.60	2.71
Mean value	1.28		-0.42	
Mode value	0.56		-0.57	
Standard deviation	1.84		1.17	
Variance	3.41		1.37	
Coeff.of variation	1.58		-2.36	
Range for 95% C.I.	1.17	1.39	-0.49	-0.35
Frequency Distribution:				
Degree of skewness	-3.42		0.008	
Degree of kurtosis	17.81		3.07	
Chi square (7df)	1217		37.80	

Table 5.4 General statistics and frequency distribution for Lode1.

Parameters	Raw data		Log-transformed values	
General Statistics:				
Number of data values	195		195	
Range of values	0.02	19.54	-3.91	2.97
Mean value	2.04		-0.58	
Mode value	0.56		-0.57	
Standard deviation	3.21		1.76	
Variance	10.33		3.11	
Coeff.of variation	1.58		-2.36	
Range for 95% C.I.	1.59	3.22	-0.83	-0.33
Frequency Distribution:				
Degree of skewness	-2.09		0.05	
Degree of kurtosis	07.69		2.01	
Chi square (7df)	330		22.69	

Table 5.5 General statistics and frequency distribution for Lode2.

Parameters	Raw data		Log-transformed values	
General Statistics:				
Number of data values	453		453	
Range of values	0.02	3.46	-3.91	1.24
Mean value	0.57		-0.95	
Mode value	0.57		0.39	
Standard deviation	0.57		0.92	
Variance	0.33		0.84	
Coeff.of variation	1.00		-0.96	
Range for 95% C.I.	0.52	0.62	-1.04	-0.87
Frequency Distribution:				
Degree of skewness	-2.28		0.07	
Degree of kurtosis	9.14		2.82	
Chi square (7df)	364.24		5.47	

Table 5.6 General statistics and frequency distribution for Lode3.

A program GIRAF (Geological interactive frequency distribution analysis and comparison based on probability scale cumulative curves) developed by Earle (1977), was used to obtain the cumulative frequency plots. The cumulative frequency plots, for both raw data and log-transformed values obtained for all the lodes, are shown in fig. 5.3.

On the basis of the statistical analysis and cumulative probability plots (tables 5.4 - 5.6 and figs. 5.2 - 5.3), the following inferences are drawn:

The descriptive statistics and statistical tests indicate the grade distributions in all the three lodes are neither normal nor log-normal, but their tendencies are to exhibit a nearly log-normal habit. The raw data of all the lodes produced negative skewed histograms as shown in fig. 5.2.

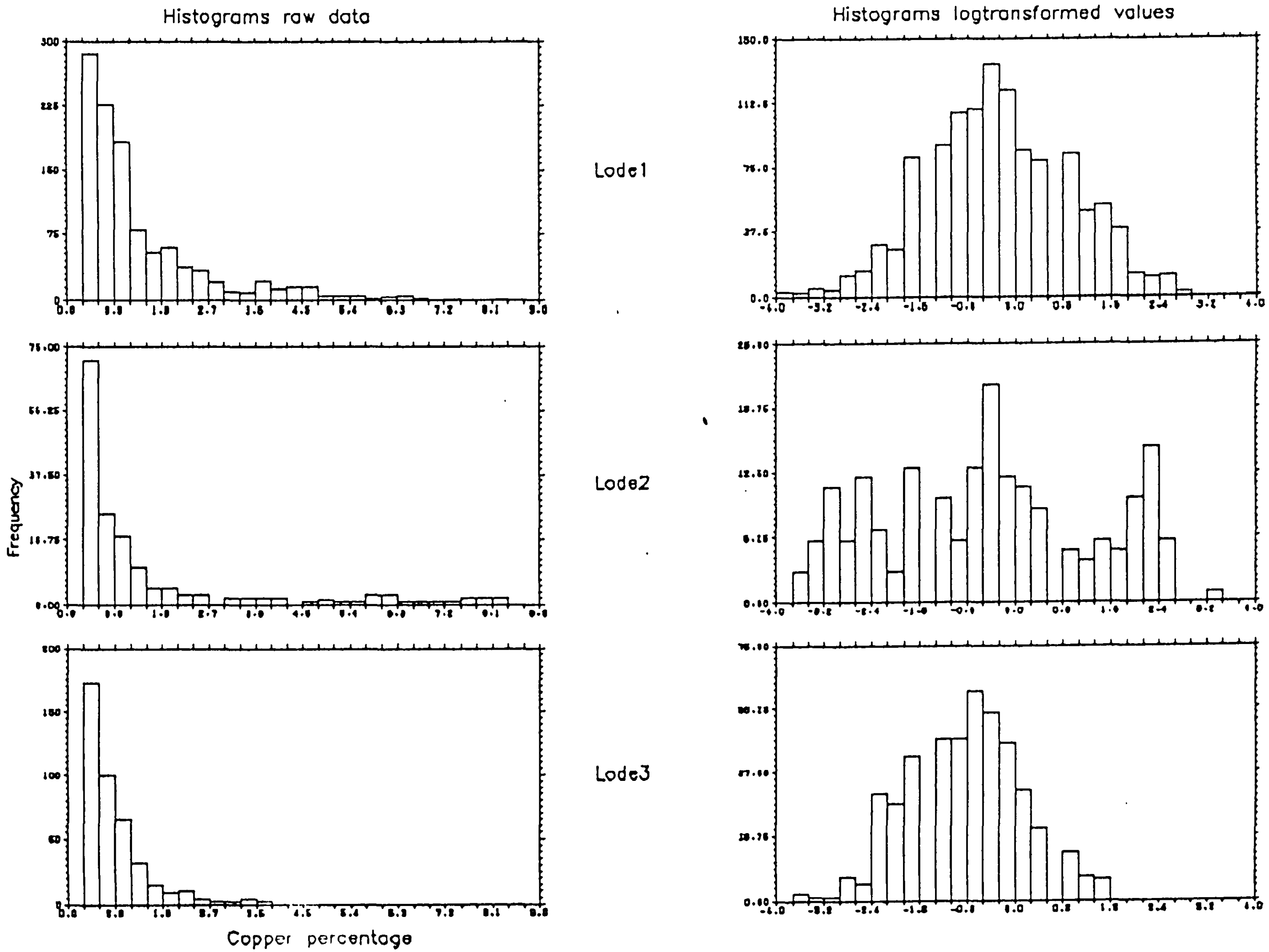


Fig. 5.2 Histogram plots of grade distribution for raw data and log-transformed values - lode1, lode2 and lode3.

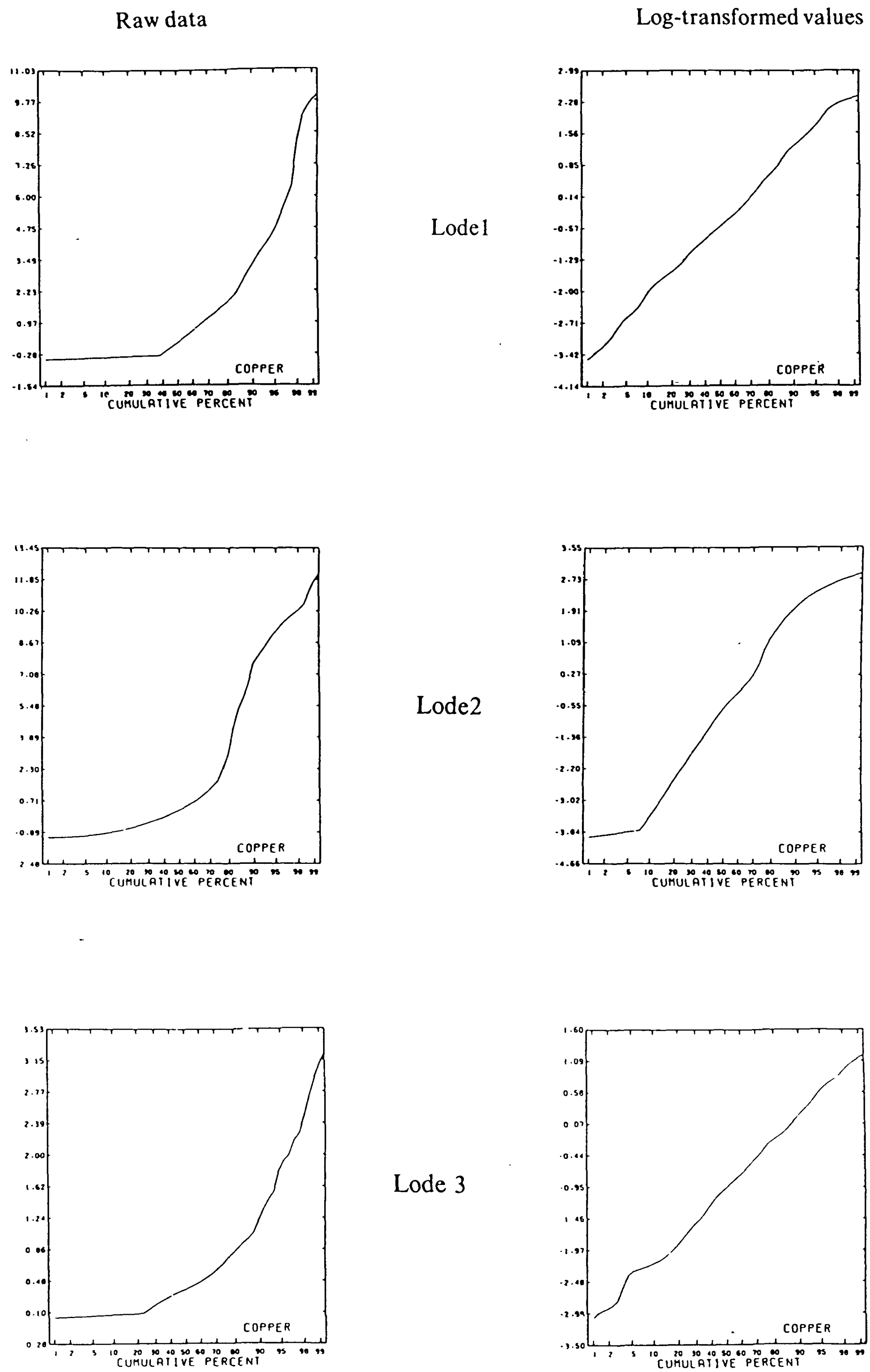


Fig. 5.3 Cumulative probability plots of grade distribution for raw data and log-transformed values - lode1, lode2, and lode3.

Based upon geological information and number of core samples recovered lode1 is the largest mineralised zone. About 50% the population of this lode has grade values below 0.5% Cu and more than 30% of its population accounts for the grade values above 1.0% Cu.

Lode2 is the smallest. About 60% of the total population of this lode indicated the grade values to be above 0.5% Cu. About 20% of the population constitutes a high grade above 2% Cu.

Lode3 is the second largest. It is a low grade zone. Only one third of its population indicated the grade distribution above 0.5% Cu. The grade variability in this lode is low compared to the other two.

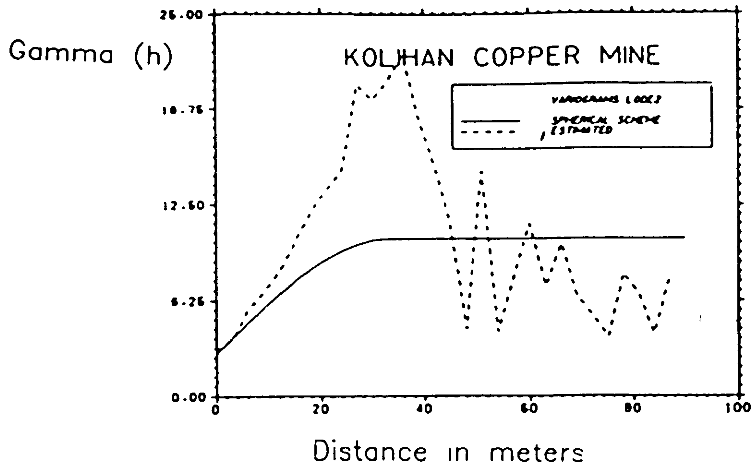
5.5 Geostatistical analysis:

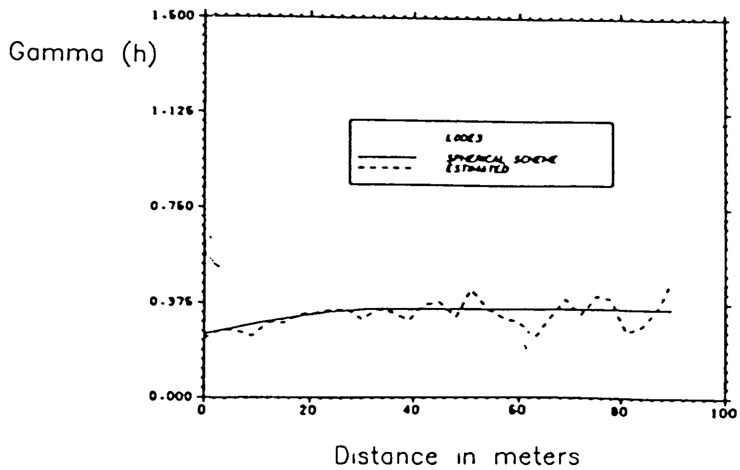
5.51 Semivariograms:

The experimental semivariograms, in the directions N-S, E-W, NE-SW and SE-NW for all the three lodes, were first calculated. The semivariograms in the direction East-West showed a good continuity but in the other directions they were noisy. This was mainly due to a smaller number of sample pairs encountered in these directions. The geological description of the deposit indicated a continuous mineralisation along strike direction N-S. Taking into consideration all these factors, the isotropic semivariograms for all the lodes, were computed. In fig. 5.4 the semivariogram plots of the different lodes, together with their fitted models, are shown.

5.52 Trend surface analysis

The grade at any location (point) is a function of a drift component and the residual component. When the data set is freed from the respective drift, the residual values at each point can be obtained. In order to examine the





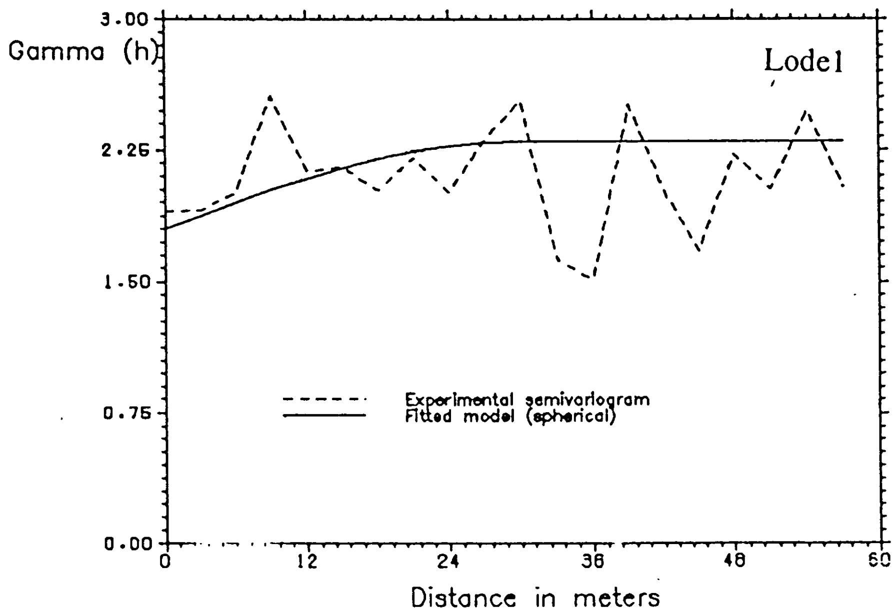


Fig. 5.4 Experimental semivariograms with fitted models - lode1, lode2 and lode3,

presence of any trend within the data set, trend surface analysis, from second to fourth degree transformation (Davis, 1973), for each of the lodes was carried out. The semivariograms, computed for the residual values obtained in each of the lodes, did not show any structure. It was therefore concluded that there was no trend present in these data sets.

5.53 Point kriging cross validation:

A spherical model was fitted to the isotropic semivariogram. This model was then cross validated through point kriging. The parameters of the fitted model were varied and adjusted until a ratio of the estimated variance to the kriging variance was equal to 1 or fairly close to 1. In addition, the correlation coefficient for the actual grade values compared with the estimated ones, was computed.

A criterion of correlation coefficient to be greater than 0.5 and ratio of estimated to kriging variance 1 +/- 0.03 (O'Leary, 1985, personal communication) was used to select the model semivariogram to be used for the purpose of block kriging.

The parameters of the model semivariograms viz. nugget variance (the random component) Co, transition amplitude (the spatial component) C and range of influence (a) for the three lodes are summarised in table 5.7. In addition, the results of cross validation have been shown.

Parameters		Lode1	Lode2	Lode3
Nugget variance Co		1.80	2.75	0.25
	C	0.50	7.50	0.10
Range	A	36m	33m	35m
	AA	0.0	0.0	0.0
	CC	0.0	0.0	0.0
Search radius (m)		30	30	30
Min. max.Points		4 15	4 15	4 15
RESULTS:				
Points kriged		1164	189	439
Ratio of kriging variance to:				
estimated variance		0.99	1.00	1.03
Mean of points used for kriging		1.28	2.10	0.56
Mean of kriged estimate		1.29	2.11	0.56
Mean estimated variance		1.98	3.50	0.308
Correlation coefficient of				
estimated to kriged grade		0.70	0.81	0.57

Table 5.7 Input parameters and results of point kriging for lode1, lode2 and lode3.

From the study of experimental semivariogram calculated for the three lodes, the following observation could be made:

The random components Co are high and for lode2 it is the highest. A high value of this parameter indicates inaccurate sampling, analytical errors and erratic mineralisation (O'Leary and Forbes, 1983, p.16).

The variability in grade distribution in lode3 is the least. The range of influence is found to be 30-36m for the all the three lodes.

5.54 Mineral inventory computation - Block Kriging:

After selecting the model semivariograms for the three lodes, the next step was to estimate the grade and variance of the blocks through kriging.

Royle (1977) stated "Ideally, the size of blocks would be that of a future stope and even more ideally the blocks would be stope themselves-----------------, the availability of data usually decides the smallness of block". Taking sampling density of the surface boreholes, stope dimensions which are 30m (along strike direction), 60m (along depth) and the thickness of orebody into consideration, a block size of 30m x 30m x 30m was selected for the purpose of kriging.

The details of other input parameters and results of block kriging for the three lodes are shown in table 5.8.

Parameters	Lode1	Lode2	Lode3
Nugget value Co	1.80	2.75	0.25
C	0.50	7.50	0.10
A	36m	33m	35m
AA	0.0	0.0	0.0
CC	0.0	0.0	0.0
Search radius	30m	30m	30m
Min. max. Points	4 15	4 15	4 15
RESULTS:			
STATISTICS:			
Blocks considered	1170	570	1170
Blocks kriged	204	23	84
Mean of kriged estimate	1.08	0.26	0.61
Mean with 95% geostat. conf. limits:	0.85-1.31	1.35-1.87	.49-.73
Mean estimated variance	2.70	15.04	0.31
Standard error of estimation	0.1151	0.8086	0.0591

Table 5.8 Input data and results of block kriging for lode1, lode2 and lode3.

5.55 Grade-tonnage estimation:

The block kriging results were used to compute the grade-tonnage relationship for the mineral inventory. Using a specific gravity of 3.0 tonnes/cu.m. a 30m cube block contains 81,000 tonnes of mineral. In fig 5.5, the grade-tonnage curves for the individual lodes and deposit as whole are shown. The results are summarised below:

Lode1 was found to contain 16.5 million tonnes of ore at a zero cutoff with

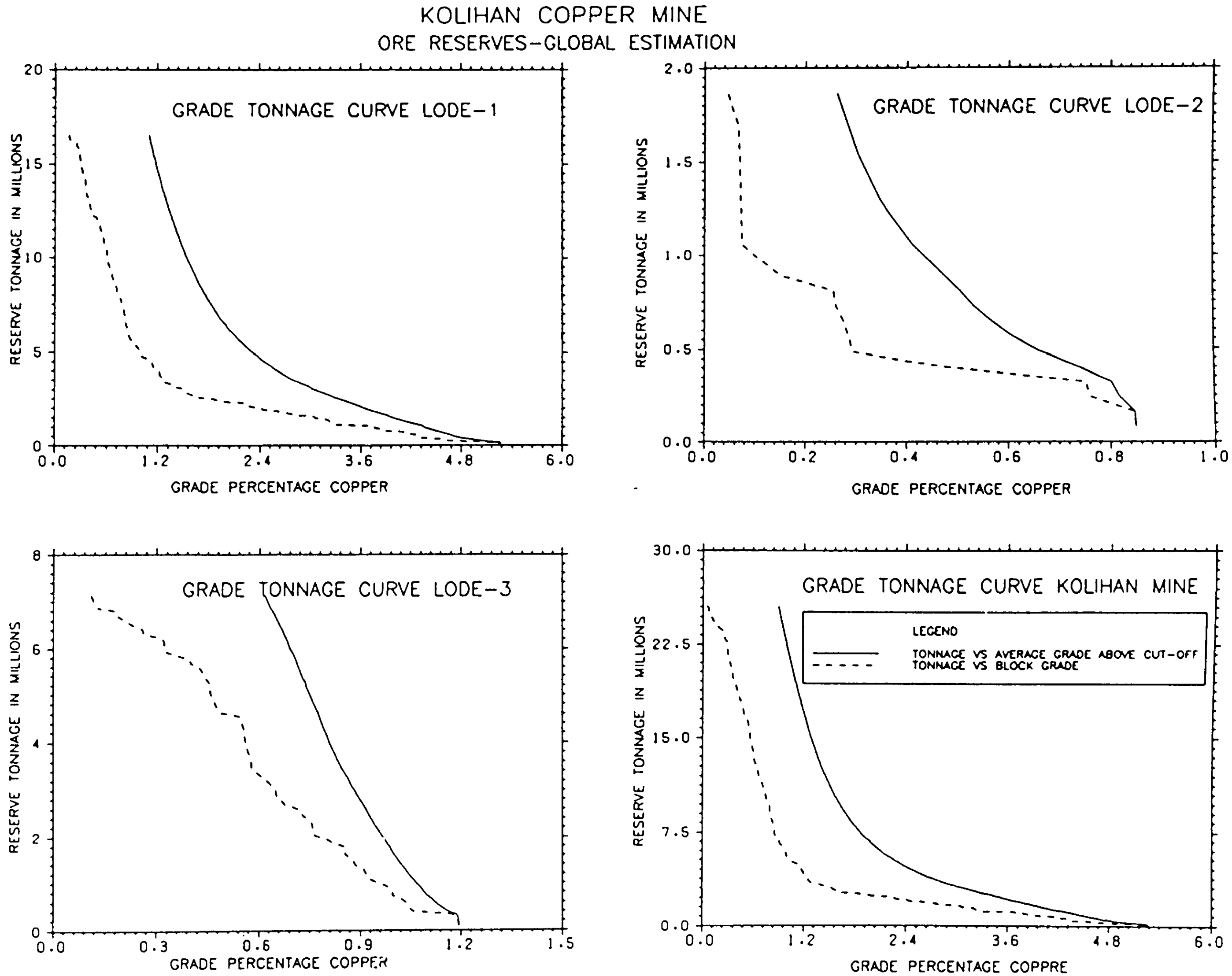


Fig. 5.5 Grade-tonnage curves for individual lodes and Kolihan deposit as a whole, through global estimation, using surface borehole information only.

an average grade of 1.008% Cu.

Lode2 is the smallest lode with a low overall grade of 0.261% Cu and 1.863 million tonnes of reserves at zero cutoff.

Lode3 was estimated to contain 7.12 million tonnes at zero cutoff with an average grade of 0.615% Cu.

By combining the mineral inventory of all the lodes, the overall reserves are estimated to be 16.6 million tonnes at 0.50% cutoff, with an average grade of 1.21% Cu. as against 28 million tonnes at 0.5% cutoff and 1.37% average grade estimated at the mine including all categories of the reserves (fully blocked, partly blocked, drill indicated and drill inferred) already described in chapter 2.

5.6 Graphical presentation of kriging results:

In figs 5.6 and 5.7 the kriged blocks obtained on two consecutive horizons, 30m apart, are shown. The contour lines represent the grade values of the kriged blocks.

The profile of mineralisation obtained through kriging along one of the cross sections (lat. 3410) is shown in fig. 5.8. The digitiser was used to transfer the coordinates of interfaces of different rock types from the mine's geological sections (maps) to the graphical data base. In fig. 5.8 the orebody plans at 246m and 306m levels along with interface of different rock types are shown.

5.7 Discussion:

The mineral inventories, for the individual lodes have shown a high variance and hence greater estimation errors (table 5.8). In addition, there is considerable difference in the estimates by kriging and the one computed at the mine using conventional methods. This comparison can be considered to be

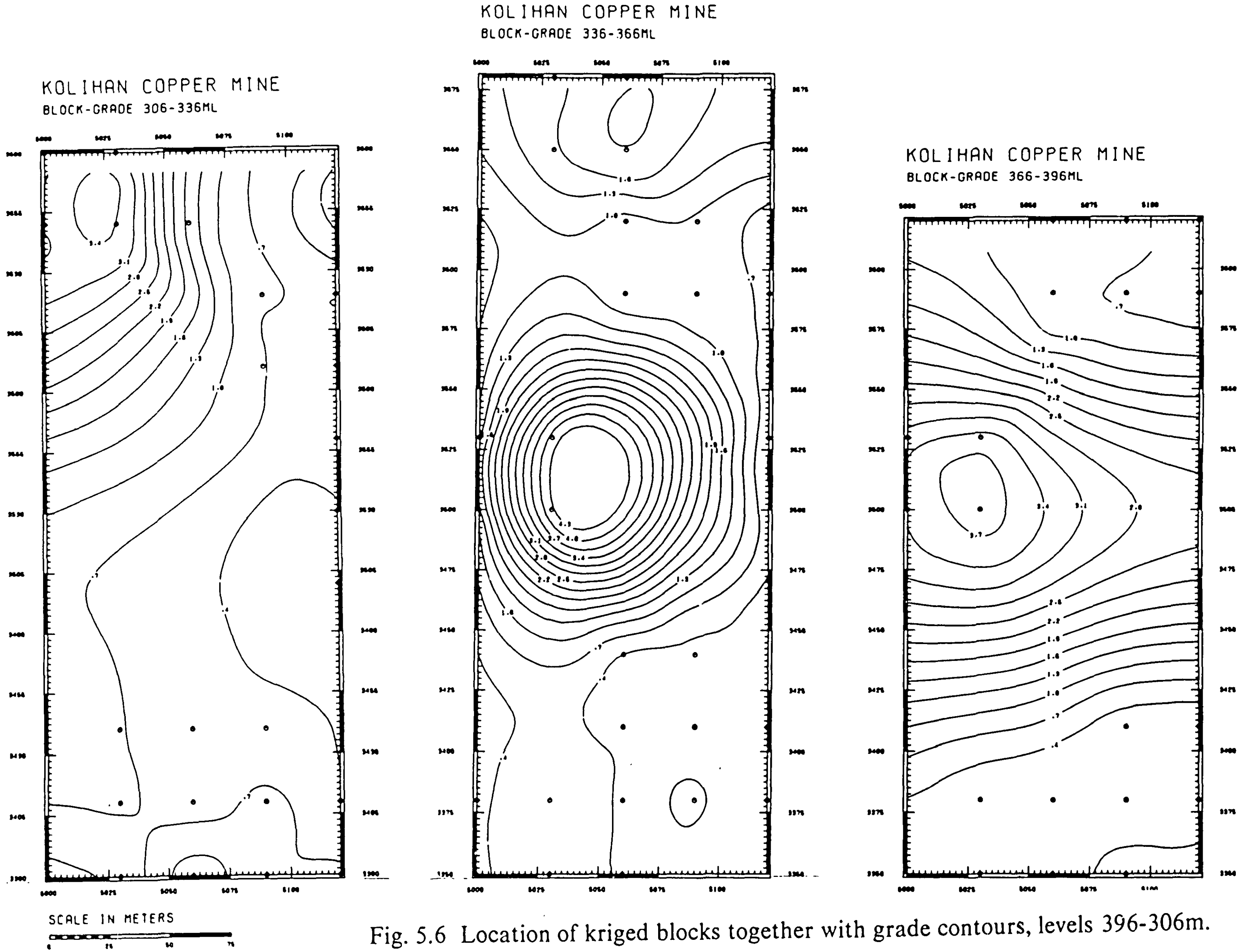


Fig. 5.6 Location of kriged blocks together with grade contours, levels 396-306m.

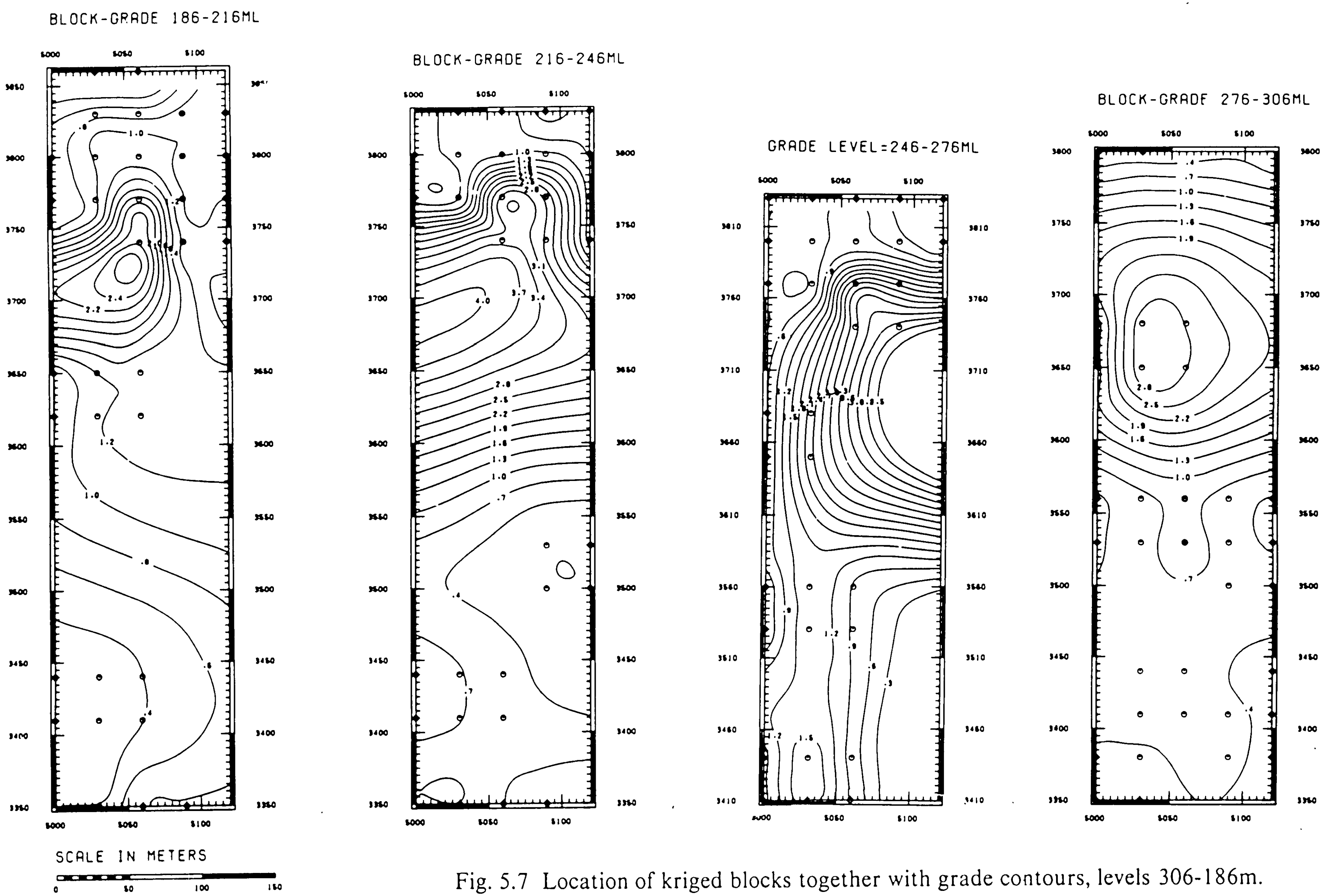


Fig. 5.7 Location of kriged blocks together with grade contours, levels 306-186m.

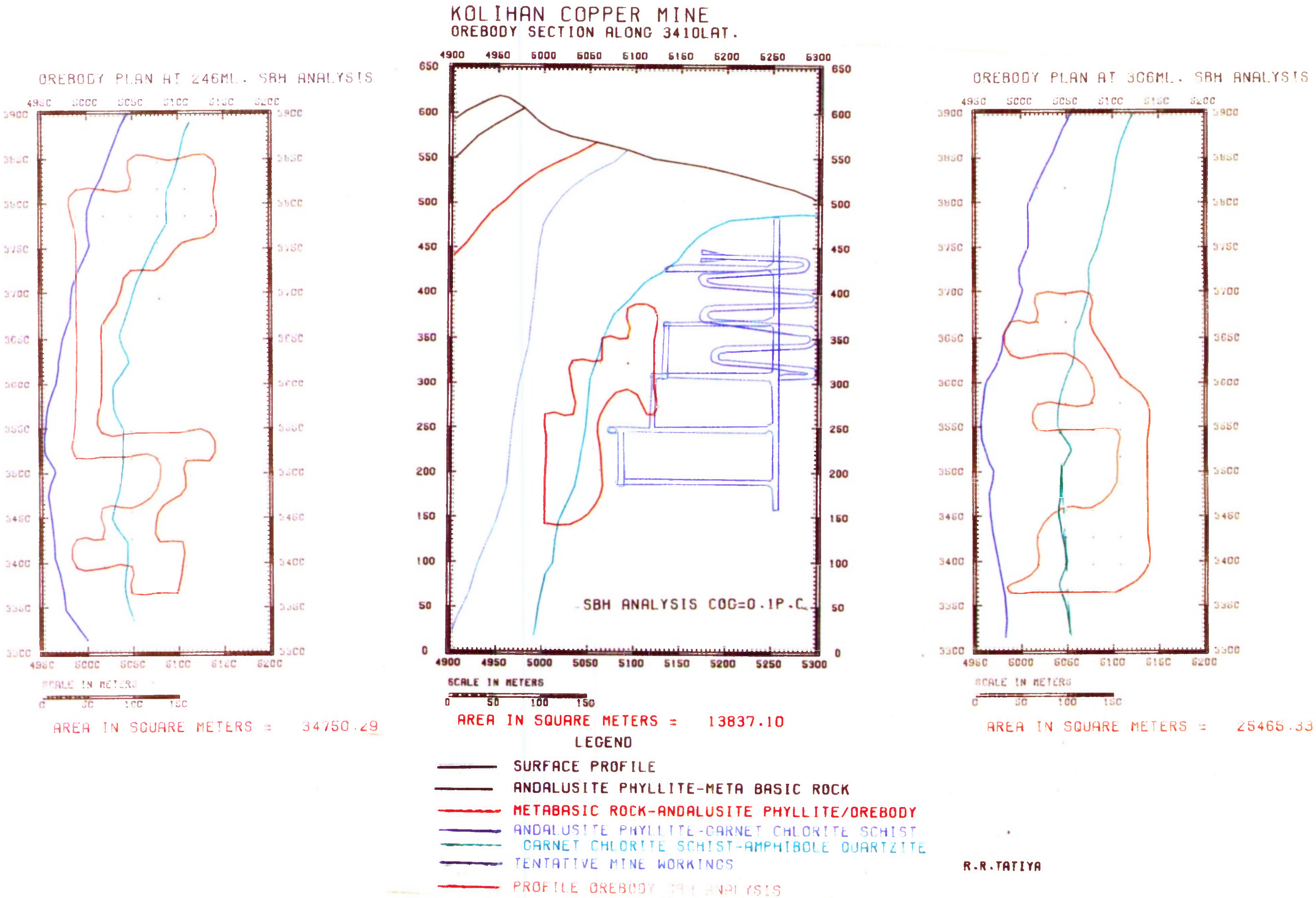


Fig. 5.8 Orebody plans and section (0.1% Cu) together with lithological boundaries, global estimation.

of less importance due to the fact that the estimation at the mine is based on all kinds of sampling information. Is likely to be changed as more information, such as current sampling, if added (as described in chapter 3).

In the semivariogram studies the presence of a nugget variance (random component) of high magnitude has been observed. This may be considered to be mainly due to a low density, irregularly spaced and inconsistently recorded sampling data. In a situation like this the semivariograms can not be considered to truly represent the spatial variability of the grade within the deposit considered. Hence, this estimation is not very reliable and precise, and suggests that more information with systematic sampling should be included for the estimation to be more precise and reliable.

A similar observation has been made by Mendelshon (1980): 'In kriging the prime requirement is that semivariograms of the variable of interest can be obtained to represent their spatial variation, which is, seemingly, not often possible for global estimates. Geostatistics has important applications, particularly to second stage estimates where evaluation of blocks becomes necessary, but does not seem to be most suitable for global ore reserve estimates'. At low data density present methods can introduce bias and other problems (Watson, 1977).

In a situation like this the statistical methods may be considered to provide better estimates, particularly of grade. Again quoting Mendelshon (1980) 'It has been shown in the various case studies that statistical methods give results (particularly for grade estimates) that are as good as or better than the geometric methods, and where data are few or sparse they are virtually mandatory'.

5.8 Conclusion:

The statistical studies indicated the grade distribution to exhibit a nearly log-normal habit.

A high nugget variance was observed for all the mineralised zones (lodes) analysed. In addition, higher standard errors of estimation have been found. Based on these observations, the geostatistical estimation can not be considered reliable. It suggests that more information with systematic sampling should be included for the estimation to be more precise and reliable.

In a situation like this, the conventional statistical methods may be considered to provide better estimates, particularly of grade.

Chapter 6

6. Local estimation - Level 4 area, Kolihan mine

6.1 Introduction

This chapter describes the local estimation by kriging for the deposit between two working levels at Kolihan mine. Data for this estimation was obtained by underground diamond drilling.

The mineral inventory computed in this analysis and the one computed by global estimation, described in chapter 5, have been compared. A comparison between ore reserves estimated through kriging and the one estimated at the mine using conventional method has also been made.

6.2 The basic information:

The underground diamond core drilling information between two working levels - level 3 (306m R.L.) and level 4 (246m R.L.) of Kolihan mine was used in this analysis. This information records the sample locations and values for a series of diamond drilling 'fans' drilled across the deposit in this area.

The collar details of each hole in a fan include the information with regard to : hole no., Easting, Northing, collar elevation, drilling direction, hole inclination, and hole length. The information with regard to the core length recorded at each of its run (the sampling interval) together with its assay value then follows.

6.21 Geological information:

Lode1 and lode3, described in chapter 5, extend to this part of the deposit. The host rocks for the two lodes remain the same.

Fig. 6.1 represents a typical drilling pattern ('fan') along a cross section of the deposit.

6.3 Observations:

A voluminous data set consisting of details of fans, drilled from 29 locations all along the strike extension of the mineralisation, were considered for the purpose of this analysis. This data set was transferred to ICCC computer data base. Observations based upon this data are outlined below:

In all, there are 120 holes drilled either due East or due West at inclinations ranging from 0 to 76 degrees to the horizontal. The interval between consecutive fans, measured along strike direction, is 25m. Usually 4-6 holes were drilled in a fan. The inclined length of these holes is in the range of 66-155m.

The sampling interval is not uniform and it varies from hole to hole and within a hole itself. It ranges between 0.45m to 3.0m measured along the hole's inclination. In most of the holes there are few interruptions in the core recording.

The core recovery is in the range of 40 - 100%, but for more than 70% of the total samples recorded, the core recovery is greater than 50%.

Deviation occurred in most of the holes and the holes were surveyed for each 30m run.

Sampling density: Drilling is confined within a distance of 525m along strike of

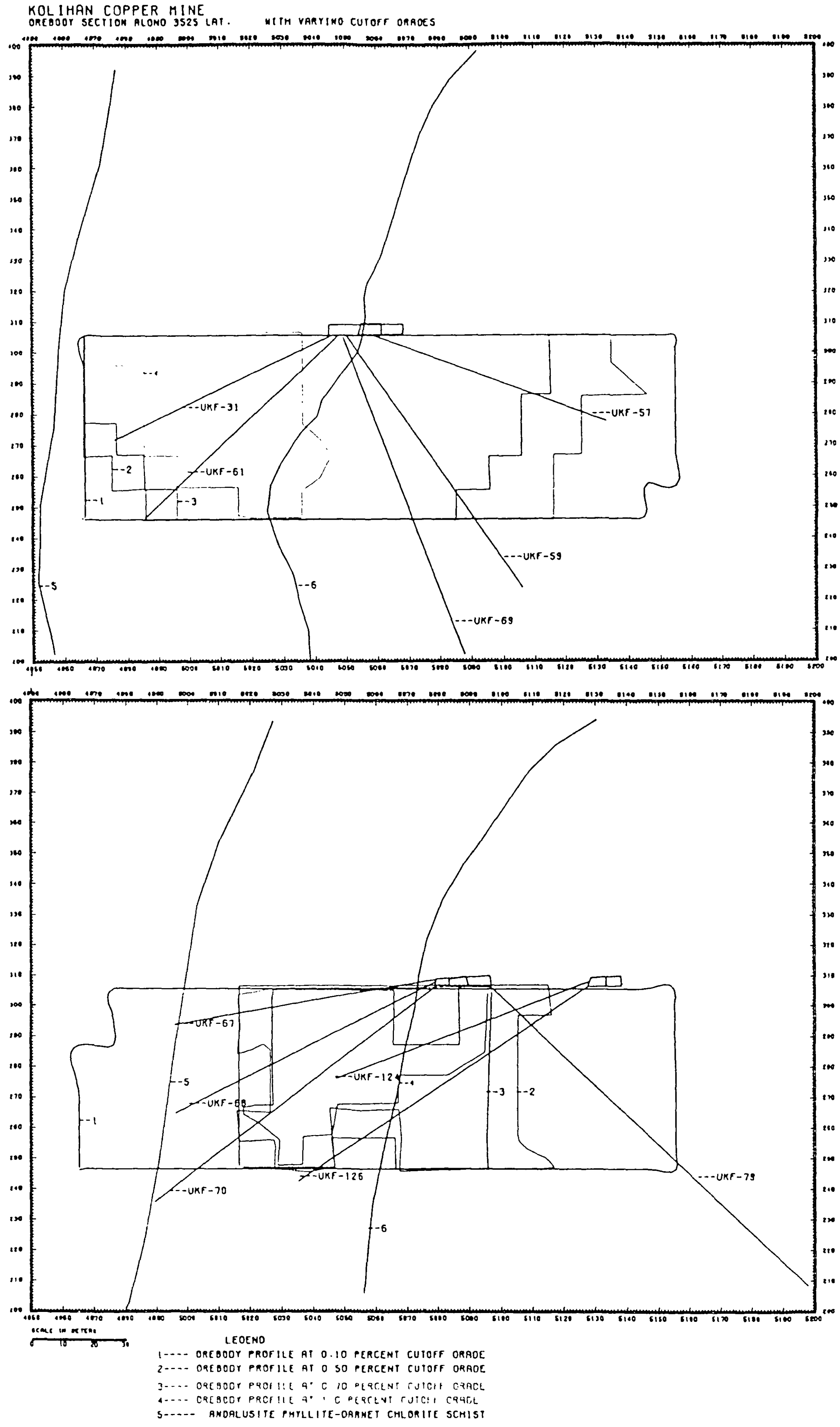


Fig. 6.1 A 'fan' drilling pattern, underground diamond drilling at Kolihan mine, along 3525 latitude together with lithological boundaries.

the deposit, 150m across it and 60m from top to bottom. In total 12,000m were drilled and account for 4538 samples recovered. Hence, the volume of ground covered per metre of drilling amounts to 350 cu.m or 925 cu.m/sample. The corresponding volume of ground covered by surface drilling has been 2,746 cu.m/m drilled or 11,752/sample, as described in chapter 5.

6.4 The analysis:

Cross- sectional maps of the deposit, prepared at an interval of 25m, were used to separate lode1 and lode3 depending upon the host rocks to which they belonged. A correction was applied to the deviated positions of the sample points using borehole survey information. The spatial coordinates of each sampling point together with its associated assay values were then computed.

6.41 Statistical analysis:

The descriptive statistics and frequency distribution for the raw data as well as log-transformed values for both lodes were computed. The results are described in tables 6.1 and 6.2.

Parameters	Raw data	Log-transformed values
General Statistics:		
Number of data values	2226	2226
Range of values	0.10 9.99	-2.30 2.30
Mean value	1.13	-0.50
Standard deviation	1.47	1.14
Variance	2.16	1.30
Frequency Distribution:		
Degree of skewness	-2.78	0.13
Degree of kurtosis	12.59	2.23
Chi square (7df)	2368	45.17

Table 6.1 General statistics and frequency distribution for Lode1.

Parameters	Raw data	Log-transformed values
General Statistics:		
Number of data values	2312	2312
Range of values	0.06 9.45	-3.81 2.24
Mean value	0.67	-0.73
Standard deviation	0.62	0.84
Variance	0.39	0.71
Frequency Distribution:		
Degree of skewness	-4.06	0.20
Degree of kurtosis	37.64	2.58
Chi square (7df)	696.20	174.60

Table 6.2 General statistics and frequency distribution for Lode3.

As described above, lode1 and lode3 belong to different mineralisation regimes. Lode3 geographically persists in footwall of the deposit. In continuation to this mineralised zone laterally, lode1 is encountered. Lode1 is richer in copper than lode2.

The semivariogram studies, described later, indicated that the spatial variability for lode3 is less than lode1. The sampling assay values at the extreme footwall are of very low grade. Thus, it can be considered that there is a gradual change in grade values when moving from the footwall side of the deposit towards its hangingwall side. To quote Barnes (1980, p. 64) in dealing with a situation like this in order to apply any extension technique, "If the mineralogical changes are gradual over a considerable distance, and gradational zone has been sampled, the gradual change principle employed by most mathematical functions will adequately represent the transition zone. Put another way, gradation and zonal mineralogical changes often are adequately accounted for by the mathematics of the extension function without imposing definite boundary limits".

Lode1 and lode3 were merged to obtain a composite semivariogram to be used for the estimation of overall mineral inventory for the deposit under study. The general statistics and frequency distribution of the grade is given in table 6.3. The histogram plots for raw data and log-transformed values of lode1, lode3 and overall deposit within this area, are shown in fig. 6.2.

Parameters	Raw data	Log-transformed values
General Statistics:		
Number of data values	4538	4538
Range of values	0.06 9.99	-2.81 2.30
Mean value	0.90	-0.62
Standard deviation	1.14	1.00
Variance	1.31	1.01
Frequency Distribution:		
Degree of skewness	-3.63	-0.13
Degree of kurtosis	20.81	2.60
Chi square(7df)	4009	146.3

Table 6.3 General statistics and frequency distribution for overall assay values.

The cumulative frequency plots for both the raw data and log-transformed values, obtained for each lode as well as jointly, are shown in fig. 6.3.

On the basis of statistical analysis and cumulative probability plots the following inferences can be drawn:

The descriptive statistics and statistical tests indicate the grade distributions of

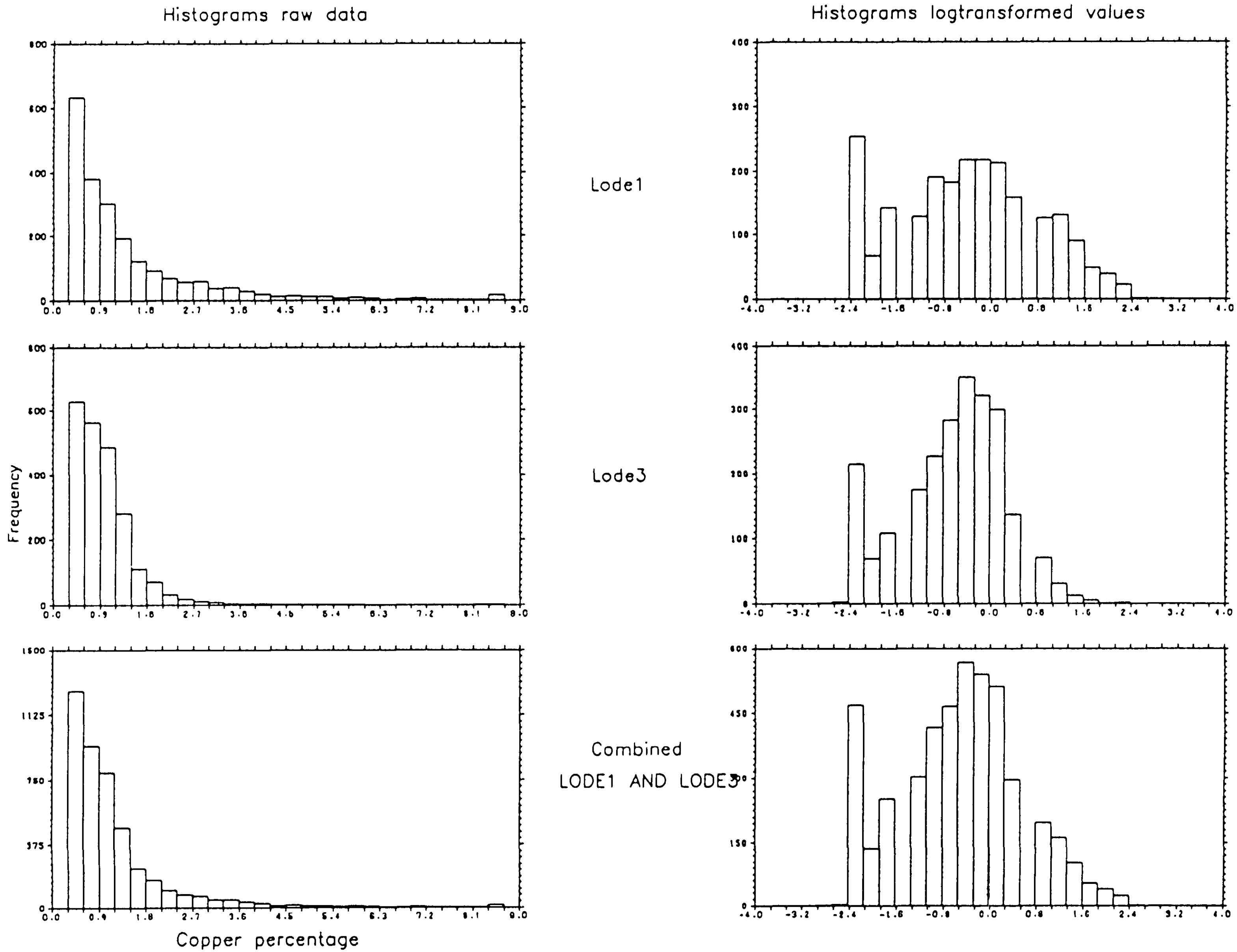


Fig. 6.2 Histogram plots of grade distribution for raw data and logtransformed values lode1, lode3 and combined lode1 and lode3

Raw data　　　　Log-transformed values

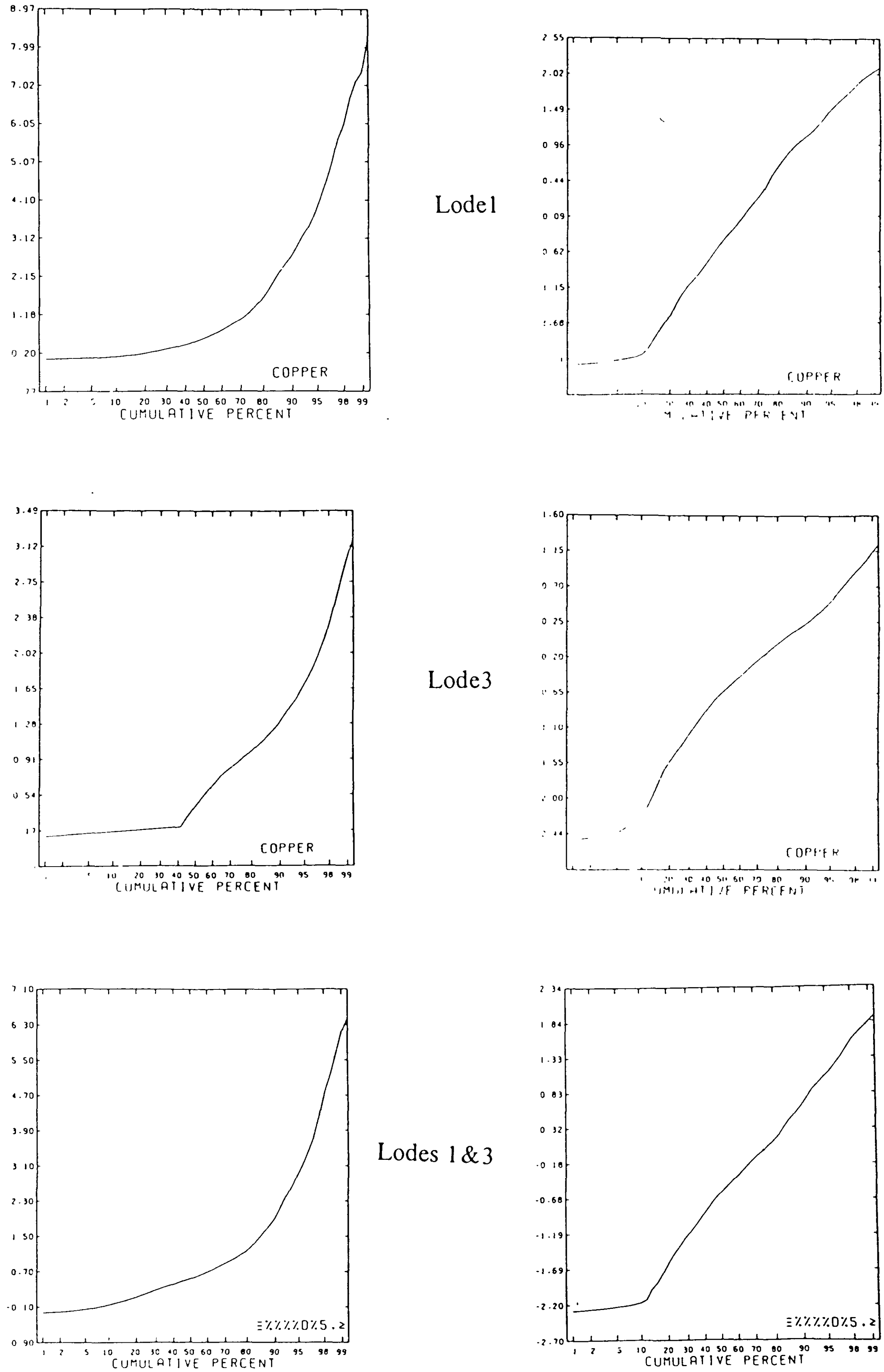


Fig. 6.3 Cumulative probability plots of grade distribution for raw data and log-transformed values - lode1, lode2, and combined lodes 1 and 3.

both lodes, and when they are combined are neither normal nor log-normal. However their tendencies are to exhibit a nearly log-normal habit. The raw data of all the cases produced negatively skewed histograms, shown in fig. 6.2.

6.42 Geostatistical analysis:

6.421 Semivariograms:

The experimental semivariograms in the directions N-S, E-W, NE-SW and SE-NW for each lode and combined lodes, were calculated. The semivariograms in the East direction showed good continuity. But in other directions they were noisy. Less number of sample pairs were encountered in the directions other than N-S. Considering all these factors, and again assuming mineralisation to be continuous along the strike direction N-S, the isotropic semivariograms were computed. The semivariogram plots along with the fitted models (spherical) are shown in fig. 6.4.

6.422 Point kriging cross validation:

Models selected to fit the experimental semivariograms were cross validated through point kriging. The parameters of the model semivariograms are summarised in table 6.4. In addition, the results of cross validation are also given.

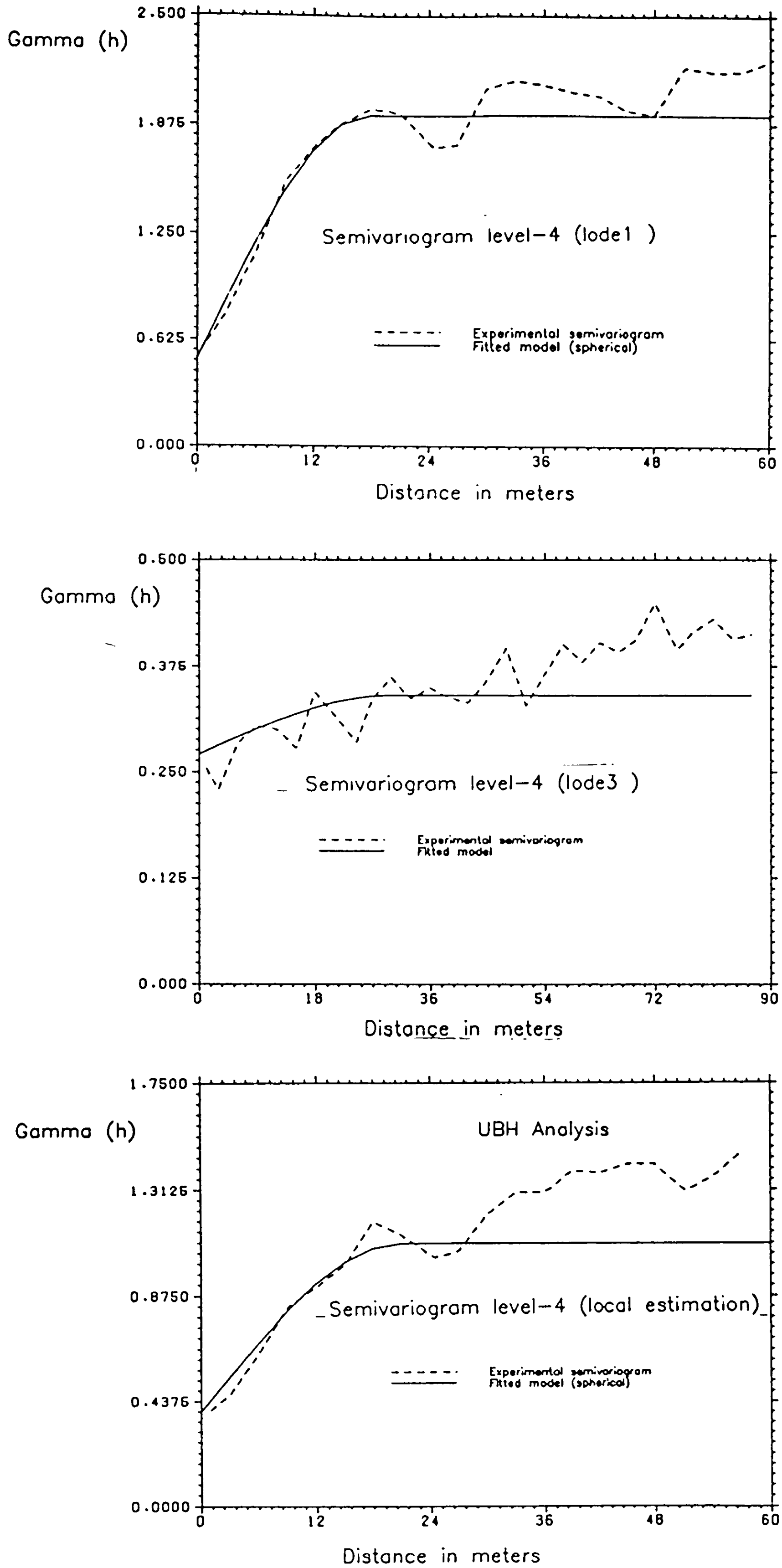


Fig. 6.4 Experimental semivariograms with fitted models - lode1, lode2 and combined lodes 1 and 3.

Parameters		Lode-1	Lode-3	Lodes 1&3
Nugget variance Co		0.50	0.26	0.39
	C	1.43	0.08	0.70
Range	A	18m	30m	21m
	AA	0.0	0.0	0.0
	CC	0.0	0.0	0.0
Search radius (m)		30	30	30
Min. max.Points		4 15	4 15	4 15
RESULTS:				
Points kriged		2226	2312	4538
Ratio of kriging variance to:				
estimated variance		1.00	1.00	1.00
Mean of points used for kriging		1.13	0.67	0.90
Mean of kriged estimate		1.14	0.67	0.90
Mean estimated variance		1.98	0.30	0.63
Correlation coefficient of				
estimated to kriged grade		0.75	0.47	0.72

Table 6.4 Input parameters and results of point kriging for lode1, lode3 and lodes 1&3.

From the study of experimental semivariograms, the following observations could be made:

The nugget variance, Co, is high in each case studied, but of less magnitude than the one estimated in the semivariogram studies made on the basis of surface borehole information.

The variability in grade distribution in lode3 is low.

The range of influence is the same for the all the three cases studied.

6.423 Mineral inventory computation - Block Kriging:

The model semivariograms were then used to estimate the grade and variance of the blocks to be kriged.

Block size: Taking sampling density of these underground boreholes into consideration, a block size of 10m x 10m x 10m was selected for the purpose of kriging. The deposit was then divided into six horizons or benches each 10m apart. The blocks of each of these horizons were then kriged. In table 6.5, the details of input parameters and kriging results for lode1, lode3,individually as well combined, are given.

Parameters		Lode1	Lode3	Lodes 1&3
Nugget value	Co	0.50	0.26	0.39
	C	1.43	0.08	0.70
	A	18	30	21
	AA	0.0	0.0	0.0
	CC	0.0	0.0	0.0
Search radius		30m	30m	30m
Min. max. Points		4 15	4 15	4 15
RESULTS:				
STATISTICS:				
Blocks kriged		4128	4280	6231
Average no. of blocks kriged per horizon		684	713	1028
Mean of kriged estimate		0.958	0.658	0.721
Mean with 95% geostatistical limits:		0.84-1.06	0.61-0.70	.65-.78
Mean estimated variance		2.01	0.37	1.16
Standard error of estimation		0.0543	0.0228	0.0335

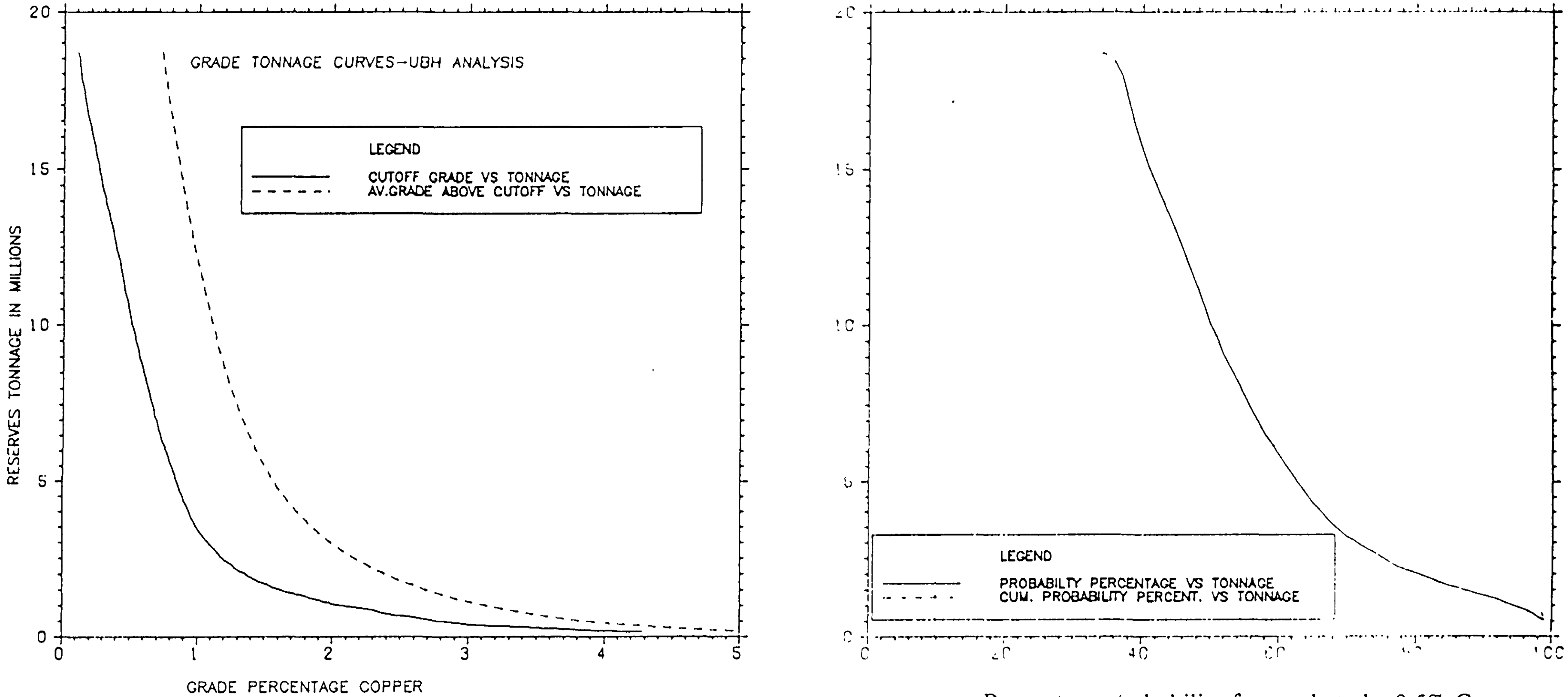
Table 6.5 Input data and results of block kriging for lode1, lode3 and lodes 1&3.

6.424 Grade-tonnage estimation:

The block kriging results were used to compute the grade-tonnage curves. Using a specific gravity of 3.0 tonnes/cubic metre a block will contain 3,000 tonnes of mineral. In fig 6.5, the grade-tonnage curves for the deposit of the area under consideration are shown. The reserves at varying percentage probability, tested at 0.5% Cu grade, are also presented in fig. 6.5.

- Chapter 6 -

KOLIHAN COPPER MINE



Percentage probability for grade to be 0.5% Cu.

Fig. 6.5 Grade-tonnage curves, level 4 area - Kolihan mine.

The overall mineralisation for this area are estimated to be 18.693 million tonnes at zero cutoff, with an average grade of 0.734% Cu.

6.5 Graphical presentation of kriging results:

Computer and its graphical aids were used to plot the kriging results. The software TSC (Tile, Solid and Conicon, Graham and Raby 1987) available at ICCC, with certain modifications, was used to obtain graphical output of the mineral inventory. An important feature of this graphical output is that first a profile of an orebody using plotting routines is derived. Use of Tiles's subroutine 'AREAS' was made to compute the areas of the irregular profiles of the orebodies.

Fig. 6.6, in plan, shows how the areas decreased in size and increased in irregularity when raising the cutoff grade. The 0.1%Cu cut-off line outlines the areas of mineralisation. At 1.0% the 'orebody' is beginning to look unminable and certainly more costly to mine. Table 6.6 summarises the parameters for 0.1, 0.5, 0.7 and 1.0%Cu cutoff grades.

Cutoff grade % Cu	Area in plan in sq. m.	Av.Thickness in m.	Strike extension in m.
0.1	90,500	150	600
0.5	40,268	70	550
0.7	28,735	55	500
1.0	9,345	35	250

Table 6.6 Change in profile of orebody (in plan) with varying cutoff grades.

In fig.. 6.7 · ·, the orebody profiles, in plan, with varying cutoff grades at 246m and 306m levels, have been superimposed. In addition, the host rocks interfaces are shown.

Decrease in size and increase in irregularity by raising the cutoff grade have also been shown in the cross sectional and longitudinal sections shown in figs 6.8 and 6.9.

The observation made above was presented at the Institute of Mining and Metallurgy, London, when a discussion on the paper 'Cutoff grades - some further reflections' by H.K.Taylor (1986) was held. The purpose of this presentation was to demonstrate the fact that increasing a cutoff grade not only reduces the volume of material above the cutoff but also is likely to increase the ore generation costs.

6.6 A comparison with global estimation:

The mineral reserves obtained through surface borehole analysis described in chapter 5 for the level 4 area amounts to be 6.39 million tonnes as against 18.69 million tonnes obtained through the local estimation. In addition, from table 6.7, it can be seen that the large estimation errors are associated with the mineral inventory computed for lode1 and lode3 in their global estimation.

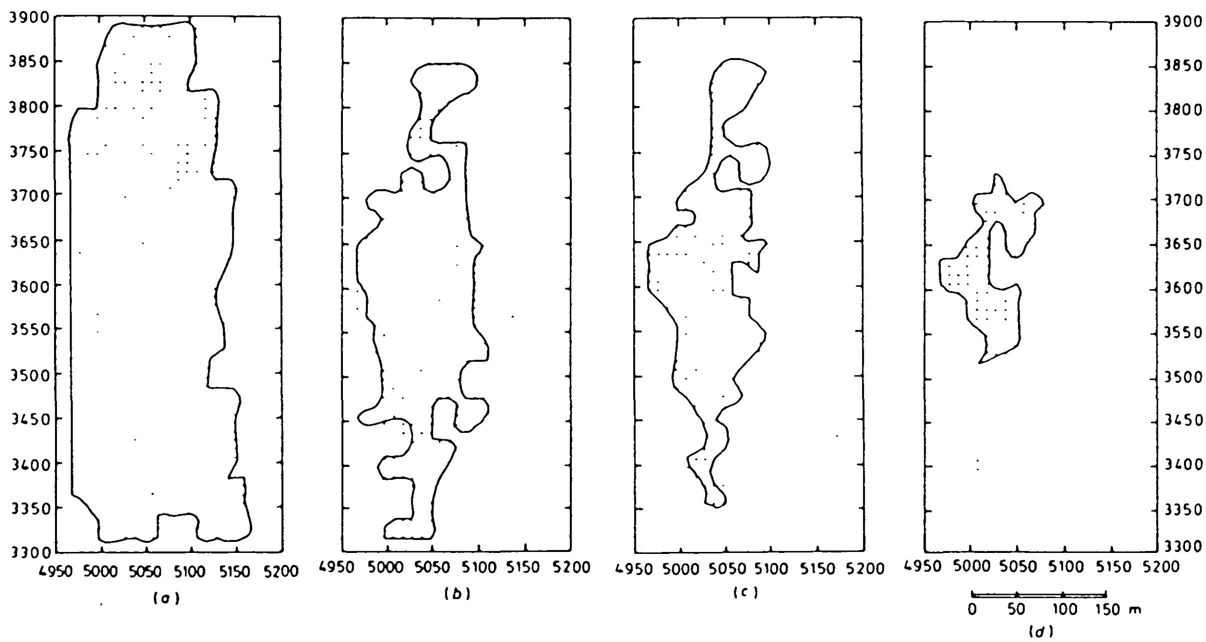
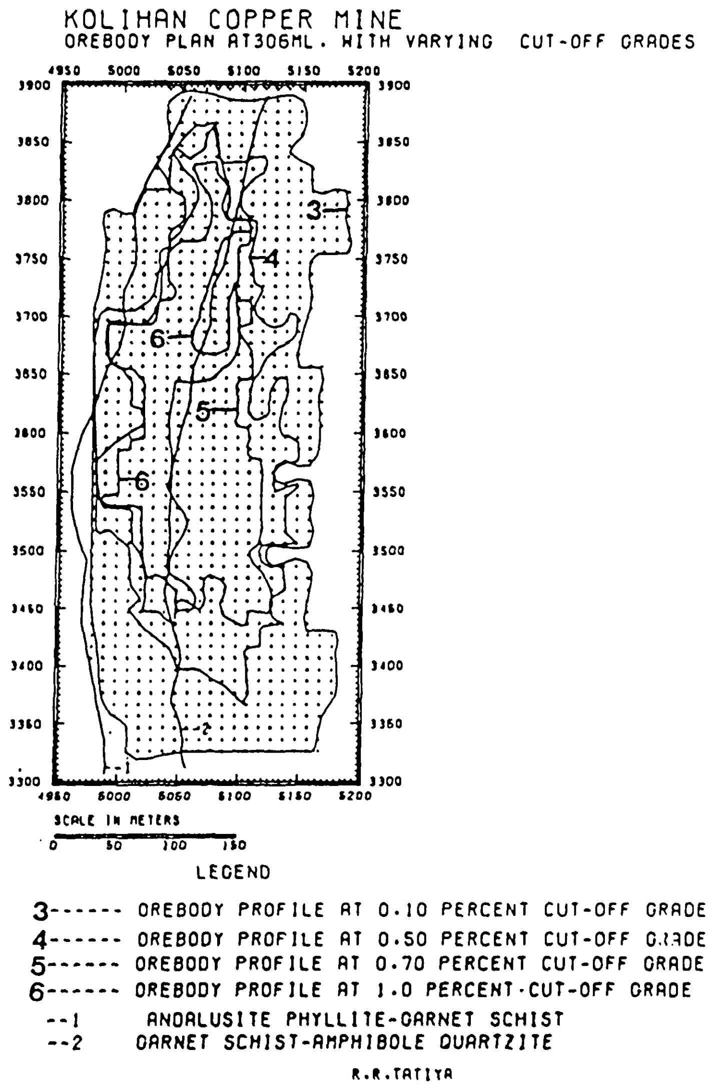


Fig. 6.6 Orebody plan with varying cutoff grade, 246m level, Kolihan mine (Tativa R. R. 1986. contribution - 'Cutoff grades - some future reflections'. H.K. Taylor, 1986, Trans. Instn. of Min. Metall. 95, A210-211)



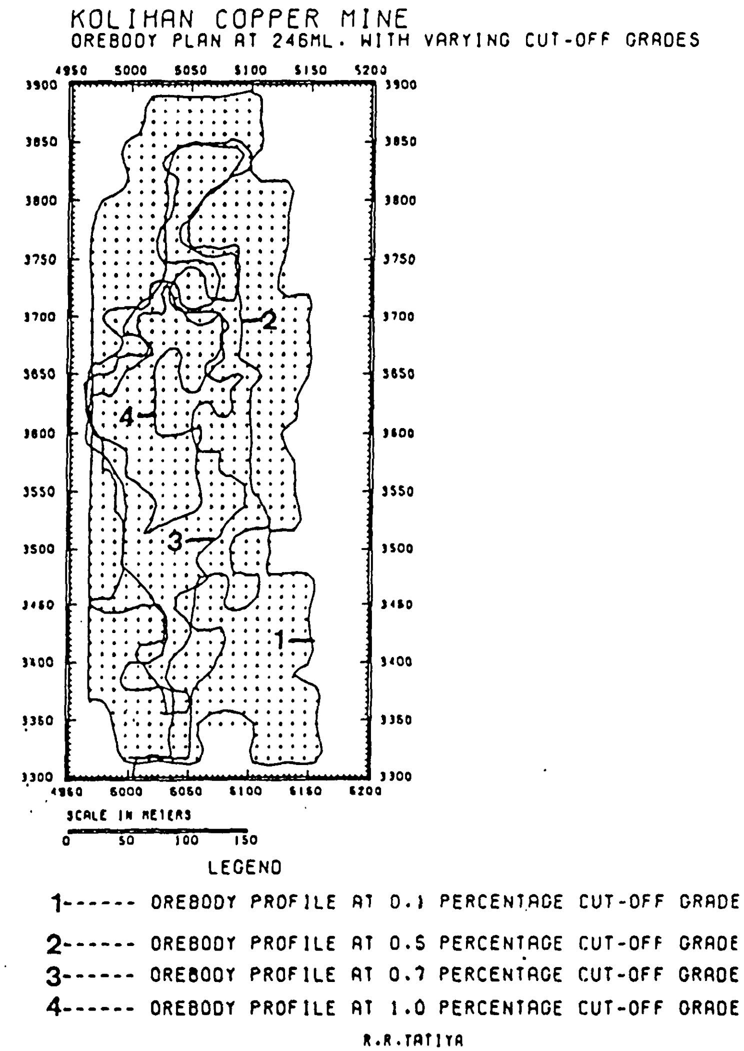


Fig. 6.7 Orebody plans at varying cutoff grades (superimposed) at 246 and 306m levels together with lithological boundaries.

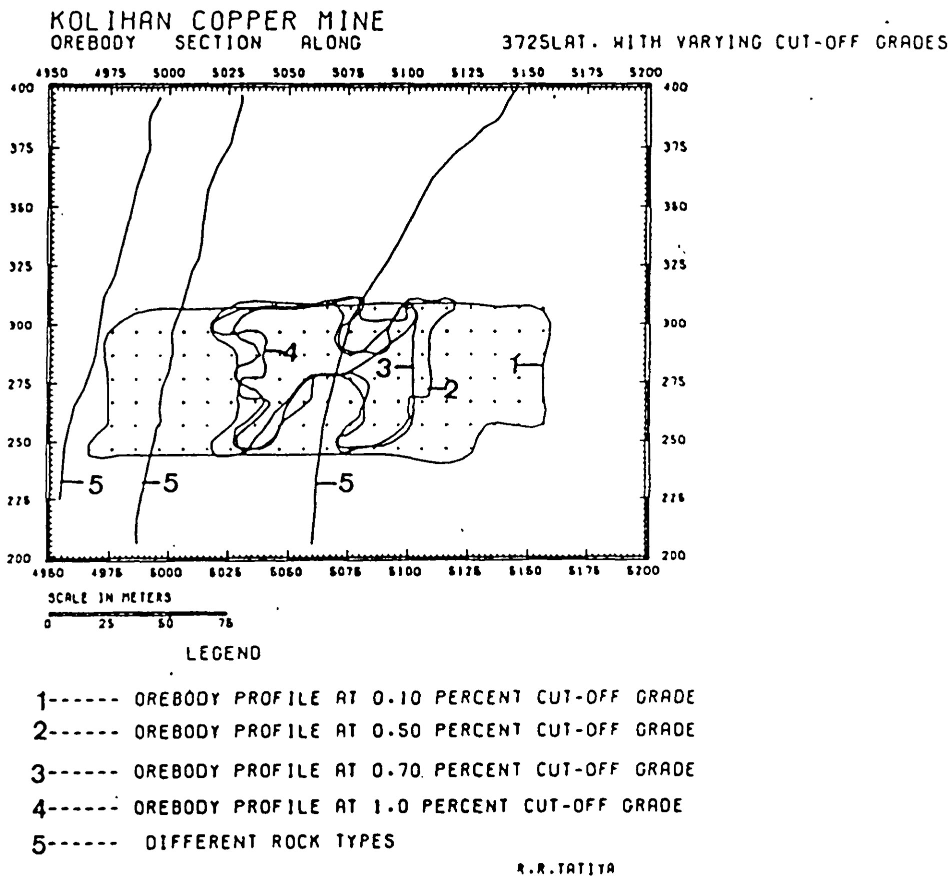


Fig. 6.8 Orebody sections at varying cutoff grades along 3725 lat. (superimposed).

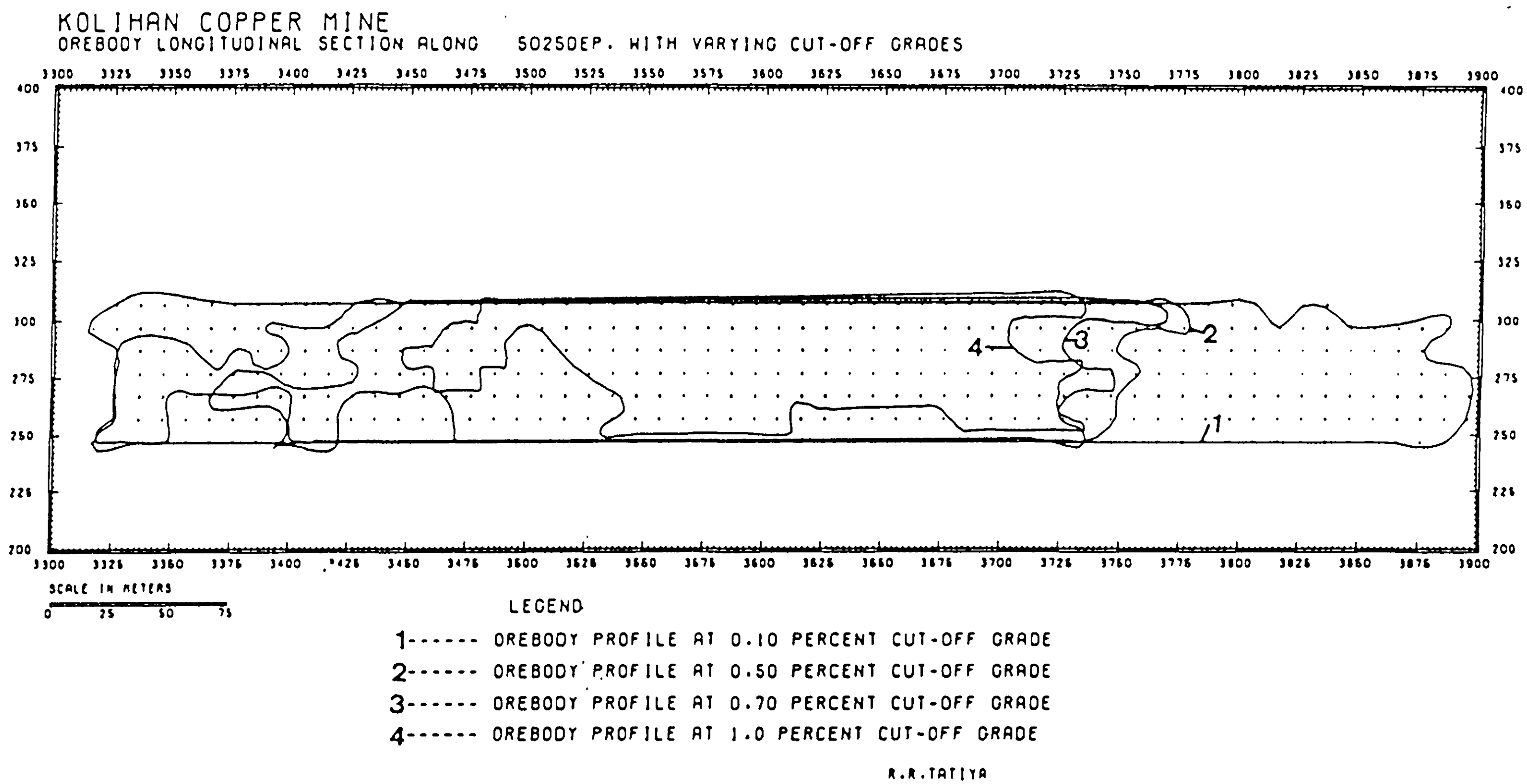


Fig. 6.9 Orebody longitudinal sections at varying cutoff grades (superimposed) along 5025 departure.

Lodes/parameters	Global estimation	Local estimation
Lode1		
Standard error of estimation	0.1151	0.0543
Mean (95% C.I)	0.85-1.31	0.84-1.06
Lode3		
Standard error of estimation	0.0591	0.0228
Mean (95% C.I.)	0.49-0.73	0.612-0.703

Table 6.7 A comparison of results obtained by global estimation and local estimation.

Thus, the global estimation through kriging can not be considered reliable with the amount of information used. Conversely, low estimation errors when analysing the lodes separately as well as jointly were obtained in the local estimation through kriging.

6.7 A comparison with conventional method:

A comparison of the ore reserves estimated by the conventional cross-sectional method used at the Kolihan mine and the one obtained through this analysis is made in table 6.8. The reserves estimated at the mine (Report - status of reserves as on 1-4-1983, Kolihan mine) are at 0.5% cutoff grade and includes "partly block B", "drill indicated" and "drill inferred" categories of the reserves.

The ore reserves estimated through kriging show an increase in quantity and decrease in overall grade at 0.5% cutoff when compared with the one obtained through the conventional method. However, the kriging estimates show an overall increase of 16.8% in metal content.

Reserve details	Conventional method	Kriging
Cutoff grade, % Cu	0.5	0.5
Ore reserves (in million tonnes)	6.89	10.248
Average grade, in %Cu	1.40	1.009
Metal content, in tonnes	96472	112718

Table 6.8 A comparison of ore reserves obtained through the conventional method and the kriging. (source: addendum report Kolihan mine, 1983-84).

The following reasons could explain the difference in the reserves obtained by kriging and the conventional method.

The conventional methods used at the mine are based upon the geometric relationships among samples but not the function of mineralisation characteristics which they intend (propose) to measure.

The reserve estimates shown at the mine are likely to under go a further change as the parameters of exploitation are taken into account (described in chapter 3).

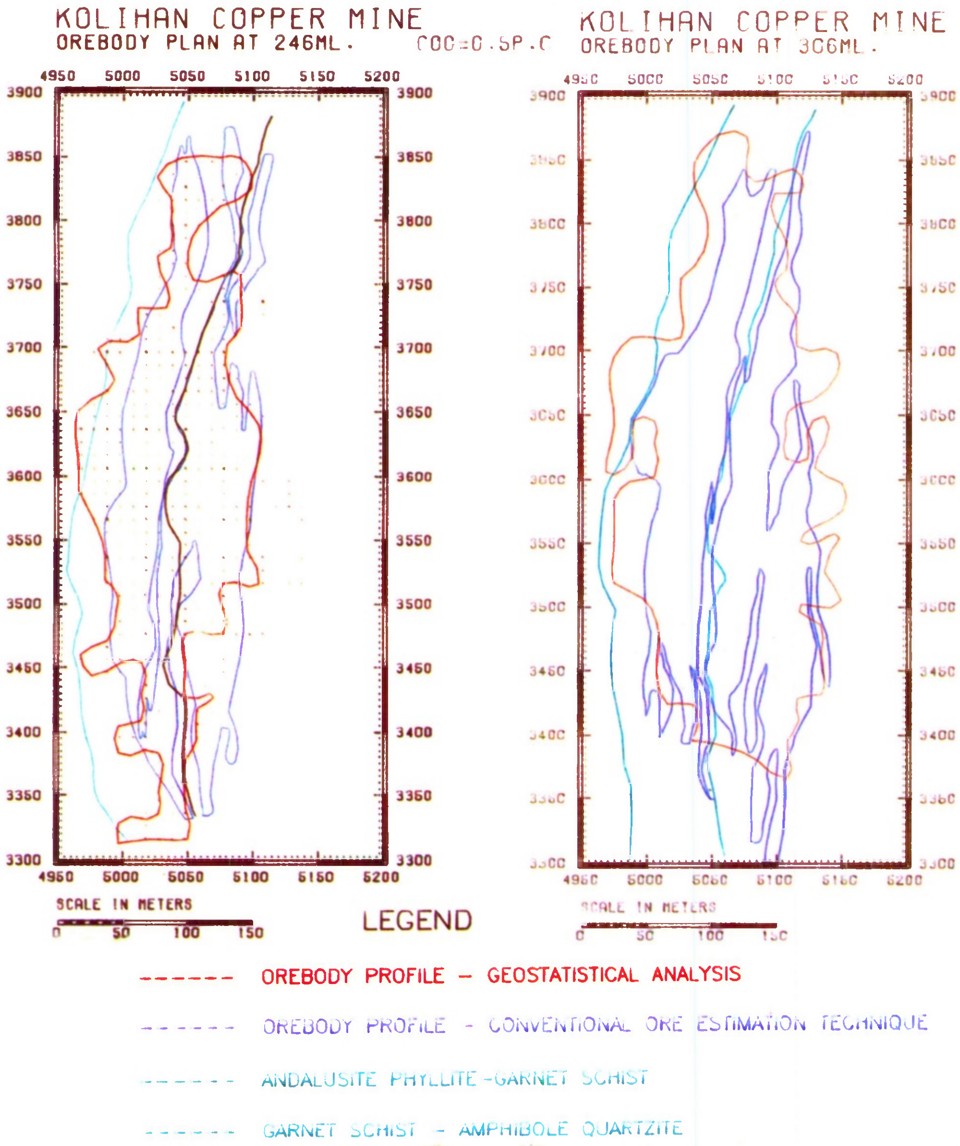


Fig. 6.10 Comparison orebody profiles obtained through kriging with those obtained through conventional method (0.5% Cu cutoff), at 246 and 306m levels.

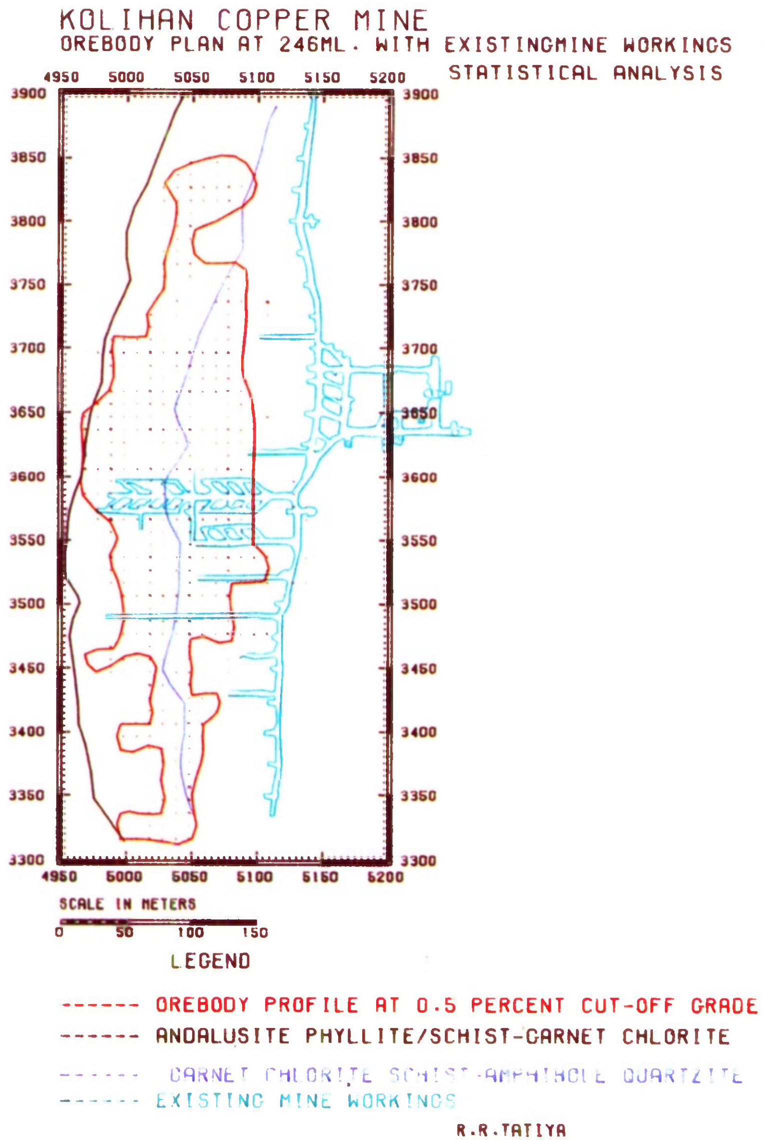


Fig. 6.11 Orebody plan obtained through kriging at 0.5% Cu cutoff together with existing mine workings at 306m level, Kolihan mine.

The estimation by kriging would have been further improved if the basic geological data were more systematically recorded.

In figs. 6.10 and 6.11, the orebody profiles (in plan) at 0.5% cutoff grade for the 246m level and 306m level horizons are shown. The orebody profiles obtained by conventional method have been superimposed. It is apparent that the orebody profiles are more continuous and regular through kriging which can allow stopes to be designed more confidently. However, these profiles are not radically different and show a good correlation.

6.8 Conclusions:

From the foregoing analysis, it is apparent that geostatistical methods can be used to estimate the values of the mineralisation from underground core drilling to evaluate the orebodies such as at Kolihan mine. However, the global estimation of the deposit, using kriging, made on the low density sampling could not provide reliable results.

The high nugget variance observed in the semivariograms indicates a need to devise systematic sampling procedures. The improvement in sampling procedures currently implemented, as described in chapter 3, may provide improved estimates.

The grade-tonnage relationship, plans, sections and other plots produced provide a quick, simple and effective ways of evaluating an orebody at its early stage of development.

The decrease in size and increase in irregularity when raising the cutoff grade is to be expected.

The results indicate that in this part of the mine there could be an increase in the reserve tonnage but with significant drop in the estimated grade.

Chapter 7

7. Local estimation - Chip sampled assay values

7.1 Introduction

This chapter describes local estimation of a closely sampled area measuring 100m x 100m, at 306m level of Kolihan mine. Estimation is based on the data derived through chip sampling.

The results from the chip sampling analysis have been compared with those obtained by the surface and underground drilling analyses, described in chapters 5 and 6.

7.2 Input data:

The assay plans of two working levels - level 3 and level 4 mine were obtained. The position of stope workings, at their extraction levels and the chip sampled points with their assay values, have been shown on these plans (assay plan 306m level, Geology Dept., Kolihan mine).

Use of a digitiser was made to determine the spatial positions of the sampling points, from the assay plan of 306m level (between 3400-3500 latitudes and 4980-5980 departures), and the coordinates of each sampling point together with its assay value were fed into the computer. In addition, the stope workings, shown in the assay plan, were also digitised. The digitised assay plan is shown in fig. 7.1.

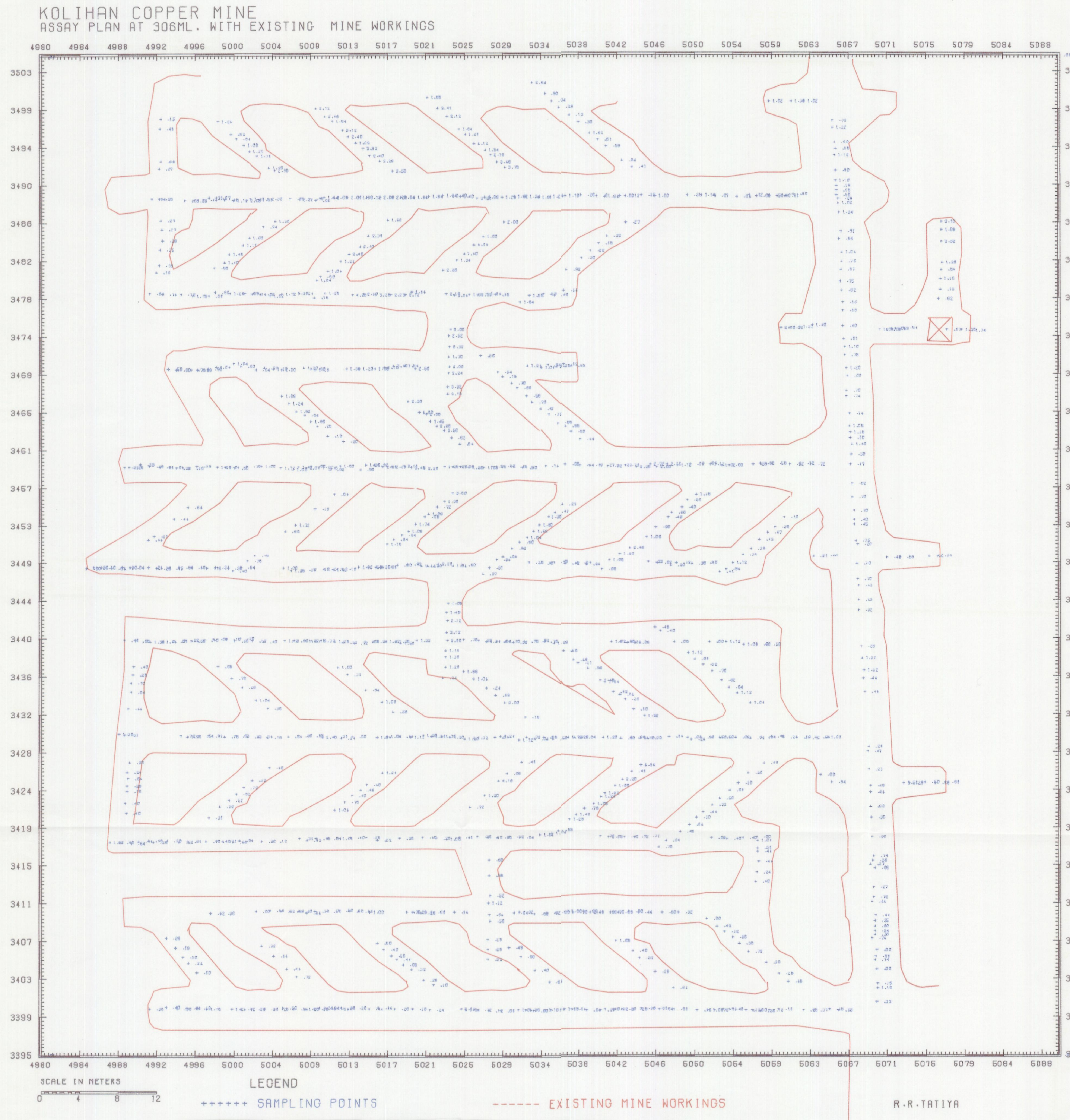


Fig. 7.1 Assay plan showing positions of chip sampled points together with mine workings, 306m level - Kolihan mine.

From fig. 7.1 it can be seen that the chip samples were taken all along the workings at an interval of 1m. In all, 905 aggregated samples were encountered within the area under study. Thus the sampling density in the area is quite high. The area covered per sample amounts to 11 sq.m.

The chip samples were taken in a mineralised zone at or above the mine cutoff grade of 0.5% Cu.

7.3 The analysis:

7.31 Statistical analysis

The descriptive statistics and frequency distribution for the raw data and log-transformed values were then computed. The results are described in tables 7.1 and 7.2. For the purposes of comparison the results of statistical analyses, obtained from SBH (surface borehole) analysis (described in chapter 5) and UBH (underground borehole) analysis (described in chapter 6), are also shown.

Parameters	CS analysis	UBH analysis	SBH analysis
General Statistics:			
Number of data values	905	4538	1812
Range of values	0.10 7.40	0.06 9.99	0.01 19.54
Mean value	0.98	0.90	1.19
Standard deviation	0.80	1.14	1.89
Variance	0.65	1.31	3.56
Frequency Distribution:			
Degree of skewness	-2.48	-3.63	-3.68
Degree of kurtosis	12.86	20.81	20.32
Chi square (7df)	365.41	4009	2349

Table 7.1 General statistics and frequency distribution of assay values obtained through surface drilling, underground drilling and chip sampling.

Parameters	CS analysis	UBH analysis	SBH analysis
General Statistics:			
Number of data values	905	4538	1812
Range of values	-2.30 2.00	-2.81 2.30	-4.60 2.97
Mean value	-0.28	-0.62	-0.57
Standard deviation	0.72	1.00	1.21
Variance	0.53	1.01	1.48
Frequency Distribution:			
Degree of skewness	-0.024	-0.13	-0.79
Degree of kurtosis	2.96	2.60	3.04
Chi square (7df)	24.46	146.3	37.18

Table 7.2 General statistics and frequency distribution of log-transformed assay values obtained through surface and underground drillings, and chip sampling.

In figs. 7.2 and 7.3, the histogram and cumulative frequency plots for both the raw data and log-transformed values are shown.

On the basis of statistical analyses and cumulative probability plots for this three fold analysis the following inferences can be drawn:

The grade distribution of the chip sampled assay values is neither normal nor log-normal. However, its tendency continues to exhibit a nearly log-normal habit as shown during UBH and SBH analyses. Also, both raw data and log-transformed values continue to produce negative skewed histograms. On the basis of these observations it can be concluded that the grade distribution for Kolihan deposit exhibits a nearly log-normal habit.

The high density sampling (the chip sampled assay values) showed a low variance compared to the one with low density sampling (surface boreholes), as shown in table 7.5.

7.32 Geostatistical analysis:

7.321 Semivariograms:

The experimental semivariograms computed for all the principal directions N-S, E-W, NE-SW and SE-NW are shown in fig. 7.4. These semivariograms showed good continuity of mineralisation, in the principal directions, within a range of 30m. Table 7.3, describes an account of sampling pairs encountered in each of the directional semivariograms computed with a lag length of 3m. In addition, the sampling pairs that were obtained while computing the directional semivariograms during SBH analysis with a lag length of 5m and UBH analysis with a lag length of 3m are also shown.

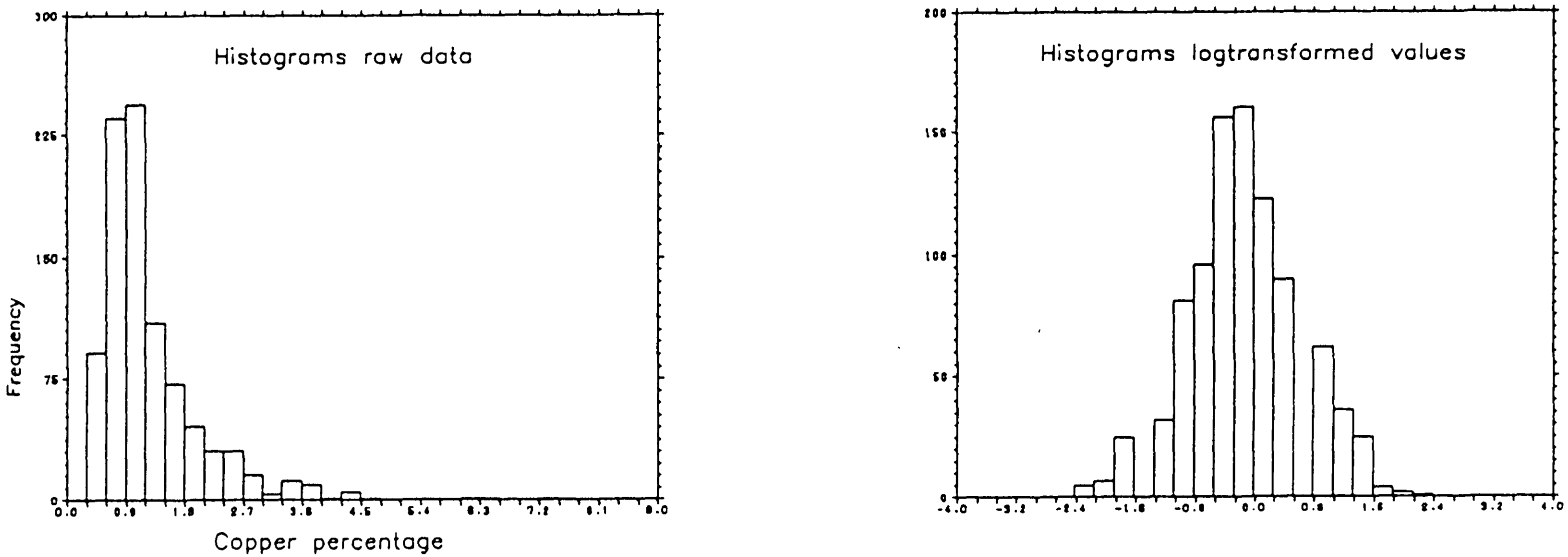
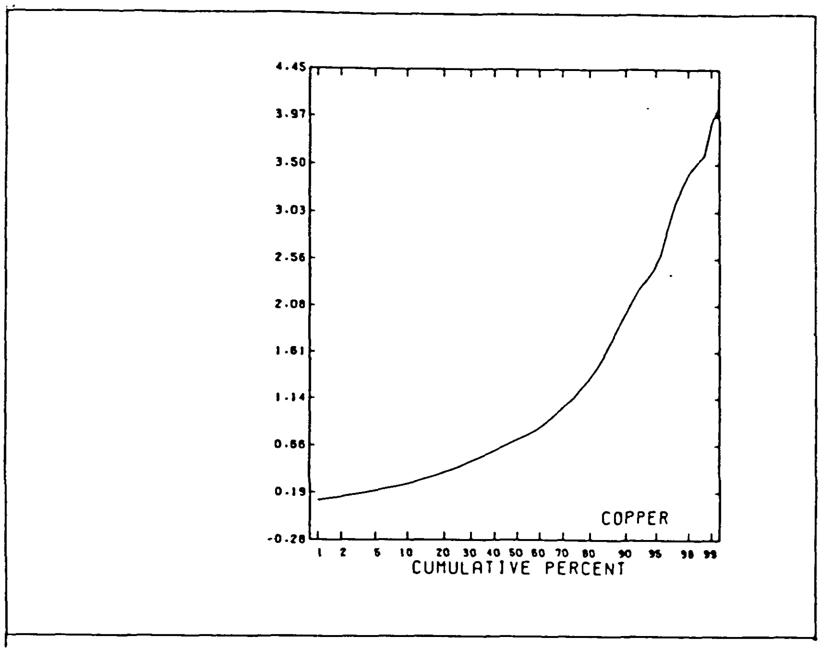


Fig. 7.2 Histogram plots of grade distribution for chip sampled assay values level – 4 area



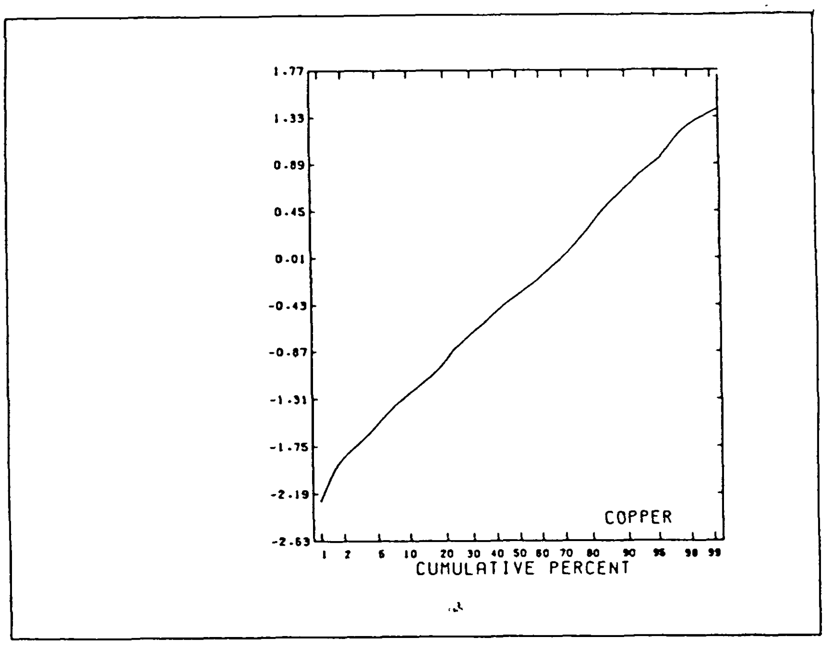
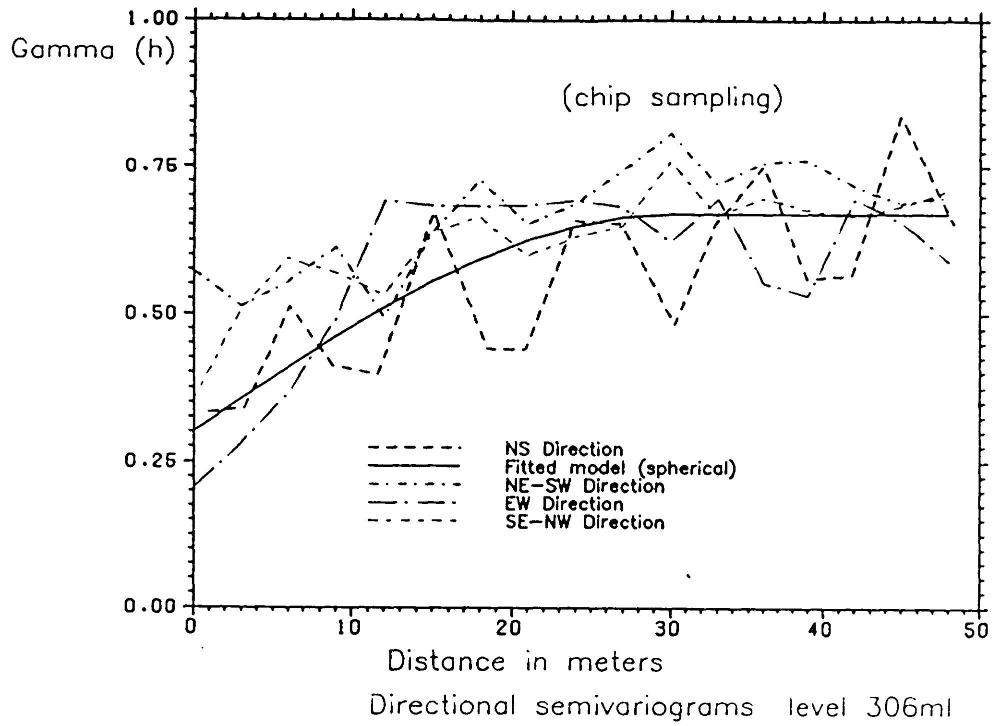


Fig. 7.3 Cumulative probability plots of grade distribution for raw data and log-transformed values.



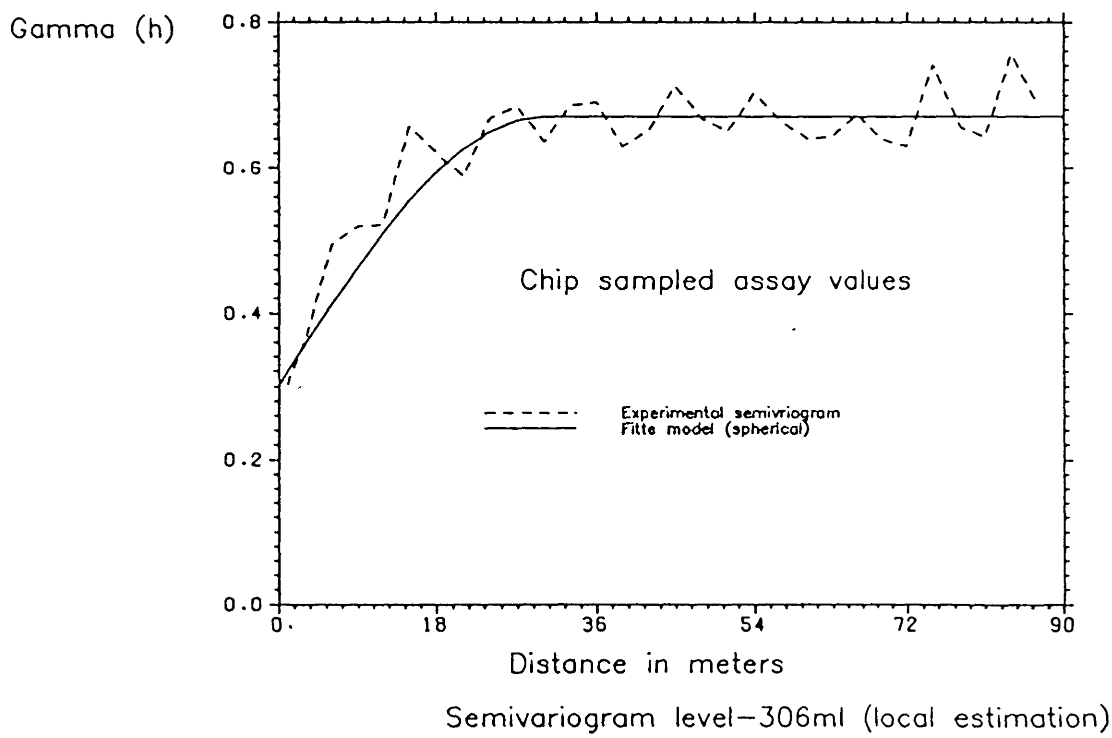


Fig. 7.4 Isotropic and directional semivariograms with fitted models.

	SHH analysis				UHH analysis				CS analysis			
Lag No.	NS	NE-SW	EW	SE-NW	NS	NE-SW	EW	SE-NW	NS	NE-SW	EW	SE-NW
1	335	40	2917	72	10	27	2400	18	117	97	355	94
2	619	46	5535	48	33	60	10195	62	579	362	1093	417
3	403	60	5136	7	63	136	11167	72	1136	937	1312	1652
4	144	218	4890	9	51	186	12858	54	1750	1606	1755	2352
5	122	166	4620	19	51	213	14189	58	3030	2372	2506	2803
6	102	66	4198	45	51	165	15924	60	2052	2753	2711	2864
7	45	67	4089	86	38	114	17564	84	3422	2857	3799	3499
8	13	96	4032	95	52	96	18453	94	3906	3366	3215	3753
9	96	12	4202	77	197	185	19245	157	3751	3713	3684	4095
10	192	4	3925	62	13537	5393	19800	4658	3090	3942	3953	3819

Table 7.3 Number of sampling pairs encountered while computing the directional semivariograms.

An isotropic semivariogram was then computed for the purpose of selecting the model semivariogram. The model selected, shown in fig. 7.4, was cross validated through point kriging. The details of model semivariogram parameters together with the results of cross validation are summarised in table 7.4.

Parameters		Chip sampled assay values
Nugget variance Co		0.30
	C	0.37
Range	A	30m
	AA	0.0
	CC	0.0
Search radius (m)		30
Min. max.Points		4 15
RESULTS:		
Points kriged		905
Ratio of kriging variance to:		
estimated variance		1.00
Mean of points used for kriging		1.13
Mean of kriged estimate		1.14
Mean estimated variance		1.98
correlation coefficient of		
estimated to kriged grade		0.658

Table 7.4 Input parameters and results of point kriging cross validation - chip sampled analysis.

In table 7.5, a comparison of models selected for mineral inventory computation purposes for CS, UBH and SBH analyses is made.

Semivariogram Parameters	SBH analysis (lodel)	UBH analysis	CS analysis
Co	1.80	0.39	0.30
C	0.50	0.70	0.37
A	36m	21m	30m

Table 7.5 A comparison of models selected for mineral inventory computation purposes for CS, UBH and SBH analyses.

On the basis of the semivariograms study the following inferences can be drawn:

Directional semivariograms obtained in CS analysis have shown the continuity of mineralisation in all the directions within a range of 30m. This confirms continuation of mineralisation, which was assumed on the basis of the geological description of the deposit during global and local estimations described in earlier chapters (along the strike direction of the deposit (N-S) and other directions as well).

A decrease in the magnitude of nugget variance, Co, has been observed with an increase in sampling density, as shown in table 7.5. However, its presence indicates that sampling and analytical errors within the data set prevail, and also the grade distribution in the deposit is erratic.

The grade variation, within the area considered for CS analysis, is low.

7.4 Mineral inventory computation - Panel Kriging:

Using the model semivariogram the panels within the area under study were kriged. A panel size of 10m x 10m was selected. In table 7.6, the details of the input parameters and kriging results are shown.

Parameters	Chip sampled assay values
Panels kriged	121
Mean of kriged estimate	0.958
Mean with 95% geostatistical limits:	0.84-1.06
Mean estimated variance	2.01
Standard error of estimation	0.0543

Table 7.6 The results of panel kriging - chip sampled analysis.

The mineral reserves, within the area under study, have been presented by way of grade-tonnage curves, shown in fig. 7.5.

The mineral reserves obtained through UBH analysis, for the same area, are also presented by grade-tonnage curves shown in fig 7.5.

A comparison of mineral reserves estimated in CS (chip sampling) and UBH analyses shows that the mineral quantity within the area is the same. However, there is an increase in overall grade of about 20-25% and also that of the individual blocks by CS analysis . This is due to the fact that the samples driven for CS analysis were relatively from a high grade mineralised zone (an area to be mined at 0.5% cutoff). The kriging results of the two analyses thus correlate well.

A comparison of kriging results for global estimation of the whole deposit and local estimations for level 4 area, shown in table 7.7, indicates a lower estimation error associated with the local estimations using underground diamond drilling and chip sampling data.

KOLIHAN COPPER MINE

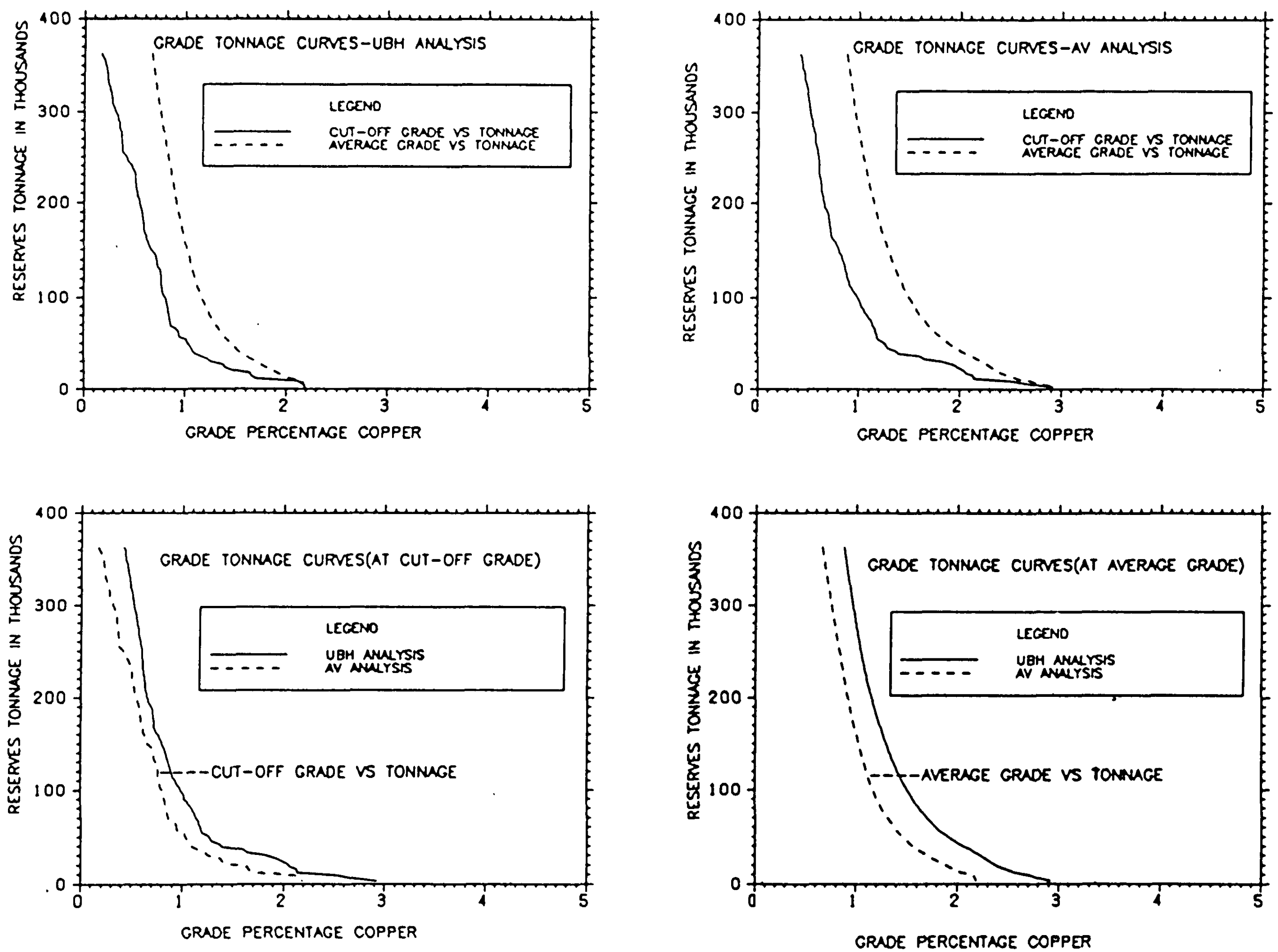


Fig. 7.5 Comparison: grade-tonnage curves, estimated through kriging, using underground drilling and chip sampling informations.

Kriging results	SBH analysis (lodel) Global Estimation	UBH analysis (lodes 1 3) Local Estimation	CS analysis (partly lodes 1 3) Local Estimation
Blocks/panel size	30m x 30m x 30m	10m x 10m x 10m	10m x 10m
Mean (95% C.I.)	0.85-1.31	0.65-0.78	0.749-1.009
Standard error of estimation	0.1151	0.0335	0.065

Table 7.7 A comparison of results obtained by global estimation of the whole deposit and local estimations for level 4 area.

7.5 Conclusions:

On the basis of the mineral inventories computed through kriging, the following conclusions could be drawn:

No matter how close or sparse the sampling was, the statistical analyses, in all the cases, indicated a near log-normal grade distribution as far as Kolihan mine is concerned. Thus, the nature of the grade distribution of a deposit can be assessed by undertaking sampling of a few locations within a deposit.

As expected, the local estimation using underground drilling and chip sampling data provided more accurate estimates of the mineral inventory than the global estimation using surface drilling.

High density sampling has shown low variability of grade in this part of the deposit.

The mineral inventory evolved for an area between level 3 and level 4 correlates well with the one estimated using chip sampling information. It also correlates well with the one estimated by the conventional method used at the mine (described in chapter 6).

It could be possible to assess the mineral inventory satisfactorily through kriging using only the underground drilling information, provided a systematic sampling procedure is adopted. Estimation using chip sampling , could be used as a check only. In this manner the chip sampling of most of the mine workings, as is the practice currently followed at KCC mines, could be avoided. This would mean a considerable saving in sampling costs and time. The aim would be to develop a semivariogram for the deposit, which in general, could be used to estimate the reserves by kriging.

Chapter 8

8. Cutoff grade analysis

8.1 Introduction

In the preceding chapters the geological aspects, which mainly concern the estimation of mineralisation in a deposit, were dealt with. Only the mineralisation that can be exploited commercially to yield profit is classified as ore. To determine and map an orebody, it is necessary to establish a cutoff grade that represents the lowest grade at which the mineralised rock qualifies as ore.

In this chapter an attempt is made to examine some factors, such as stoping methods, orebody thickness, mechanisation, productivity and process recovery, when deciding cutoff in an underground mine.

8.2 Literature review

Evaluation of the Kolihan deposit indicated that the grade distribution is not uniform. There are areas of high mineral concentration and low mineral concentration within it. Areas with an intermix of high and low grade are also encountered. In a situation like this some kind of selectivity is needed for the purpose of mining. A cutoff grade is the criterion which is employed to define geographically and quantitatively the potential ore limits. Taylor (1972) defined cutoff grade as "any grade that for any specific reasons, is used to separate two courses of action, e.g. to mine or to dump". Where grade of the mineralised material is less than cutoff grade it is classified as waste and where it is equal to or above cutoff grade it is classified as ore.

The reasons for continuing interest in cutoff grades are obvious. Too high a grade can reduce the mineral recovered and possibly the life of the deposit. Too low a cutoff would reduce the average grade (and hence profit) below an acceptable level. In project evaluation it is important to determine a cutoff grade which is normally set to achieve the financial objectives for the project. This is discussed in chapter 11.

Studies on cutoff grade theory may fall into two basic categories. The fixed cutoff grade concept assumes a static cutoff for the life of the mine, while the variable cutoff grade concept assumes a dynamic cutoff maximizing the mine net present value. Callaway (1958), Vickers (1961), Erickson (1968), Hall, Bellium and Lewis (1969), Plewman (1970), Douglass (1971), and Nilsson (1982), adopt a fixed cutoff grade approach, whereas Henning (1963), Lane (1964,1979), Johnson (1969), Backwell (1971), Taylor (1972,1985), Roman (1973), Elborned and Dowd (1975), Wells (1978) and Redenno (1979), favoured a variable cutoff grade concept.

Lane (1964) outlined three distinct stages in a mining operation: ore generation (mining), concentration (milling) and refining. He demonstrated that in establishing cutoff grades, consideration of costs, capacities, waste:ore ratios and average grade of different increments of ore of the orebody as well as the present values of annual cash flows are essential.

For each stage, there is a grade at which the cost of extracting the recoverable metal equals the revenue from the metal. This is commonly known as breakeven grade.

If the capacity of an operation is limited by one stage only, the breakeven grade for that stage will be the optimum cutoff grade. Where, an operation is constrained by more than one stage, the optimum cutoff grade may not necessarily be a breakeven grade. In such a case the "balancing" cutoff grade for each pair of stages needs to be considered as well.

For calculation of breakeven grade assuming that output of the mining operation is constrained only by its capacity to handle ore, Lane (1979) gave the following formulae to be used depending upon the policy of a company.

Constant price and cost:
Simple formula
$(s-r)y\,g = c$ .... (1)

Maximum total profit
$(s-r)y\,g = c + f/C$ .... (2)

Maximum present value
$(s-r)y\,g = c + (f + dxv)/C$ .... (3)

varying prices and costs
Maximum present value
$(s-r)y\,g = c + (f + dxv + \partial v)/C$ .... (4)

where:
s price of metal/mineral
r smelting and marketing costs
y recovery of mineral
c marginal (variable) processing costs
f annual fixed cost
d discount rate
C capacity:units of ore p.a.
g cutoff grade
v present value
∂v the decline in present value over a year.

For cutoff grade calculations, in on-going underground mining situations, usually the marginal costs only are considered (Lane, 1985 personal communication).

Dowis (1982) mentioned that only variable costs are used in cutoff grade analysis because the inclusion of other operating costs (fixed,direct etc) reduces the ore reserves by an amount that would pay for these other costs. The total costs should be used in the optimization analysis with regard to net present value and fixing production rates. In chapter 9 the operating margin (revenues less operating costs) is used to compare differing mining methods and conditions to evaluate stope boundaries.

Taylor (1985, p. A214) emphasized the importance of quantities (whether of ore or valuable content) in cutoff grade investigation. He mentioned: "these quantities influence and determine the unit costs, breakevens and all other economic matters".

Kelsey (1979) mentioned: "the formulae for the calculation of break even grade assume that the variable costs are constant across a range of cutoff grades. In underground mines in particular this need not be the case". Nilsson (1982) stressed the need to study mining methods as one of the most important factors to decide cutoff grades.

Based on the observations made above the simple formula 8.2(1) has been used for the calculations of the cutoff grades in this study. In this case the marginal costs are equated with variable costs.

As will be seen in chapter 11 the cutoff grade derived on this basis may result an average grade that does not provide sufficient revenues to cover all the fixed costs of the operation. In this case it would be necessary to consider formula 8.2(2) to generate a high cutoff grade. Alternatively an intermediate value might be selected.

8.3 Procedure for establishing a cutoff grade:

The study is based on the basic data obtained from KCC. The cutoff grade analysis has been carried out following the steps outlined:

-outlining the orebody profile
-preparation of stope layout (mining plan) and quantifying the work involved
-formulating equipment performance norms
-unit cost calculations
-input metal price and process recoveries
-cutoff grade calculation

8.31 Outlining the orebody profile

Stope boundaries are normally defined by incremental analysis, described in chapter 9. However, in examining the effect of stope width on cutoff grade hypothetical orebody thicknesses 5m, 10m, etc. have been considered.

Using a stope design model (chapter 10), the stope layout (mining plan) of a mining method is then prepared. This presents the workings at the extraction level, working level/levels and the cross sections along the main raises. See figs. 8.2 - 8.5. Based on these designs, the activities involved to mine out the stope are assessed. This includes the stope development work and stoping tonnage.

8.32 Formulating equipment performance norms

The next step is the selection and assessment of equipment to be deployed in the stoping activities described above. This has been designated as the 'degree of mechnisation'.

Using standard sizes of stope workings to accommodate the equipment to be used, the quantities of development and stoping works are computed.

The term productivity has different connotation in different disciplines. A common basis for expressing equipment efficiency in mining is the calculation of rock produced by operator/operators during a shift. This is expressed as the tonnes/manshift ratio or output/manshift (OMS). Different equipment yields different output per unit time depending upon its size, working conditions and training and experience of the operator. The basis for evaluating OMS for equipment should be investigations in the form of work and time studies with an allowance for local working conditions.

The equipment performance norms, established by the Industrial Engineering Department of KCC to determine performance index of the mine and to provide basis for their incentive scheme, were obtained for use in this study. The performance norms of various types of equipment are shown in appendix 1. For example a LHD (1 cu. yd. capacity) should give an output of 200t/shift/man when operating in stopes, and 160t/shift/man in development headings.

8.33 Unit cost calculations

The equipment cost per unit time includes the cost of power/energy, wages, spares, consumable items and maintenance. Equipment costs were obtained from the mines using the standard pro formas such as the one shown in appendix 2.

Using the OMS and cost/shift for each piece of equipment, the unit operational costs for the operation performed by each piece of equipment are calculated.

The unit operational costs for three stoping methods with different levels of mechanisation are reported in table 8.1.

Operation	Mining Method				
	Sublevel	DTH	Cut and Fill		
	Unit cost in Rs			units	Remarks
Extraction level development					
Drilling	123.0	65.0	123.0	cu.m	pusherleg drills
Blasting	76.0	76.0	76.0	cu.m	gelatine explosive
Mucking	32.0	26.0	26.0	cu.m	Rocker shovel
Transport.	39.0	18.0	15.0	cu.m	locomotive haulage
Sublevel development					
Drilling	123.0	65.0		cu.m	pusher leg drills
Blasting	76.0	76.0		cu.m	gelatine powder
Mucking	26.0	26.0		cu.m	auto loader (CAVO)
Transport.	15.0	18.0		cu.m	chute loading
Stoping					
Drilling	71.5	165.0	34.4	m	blast hole drills
Blasting	5.8	5.8	5.8	t	using ANFO
Mucking	8.4	6.9	6.9	t	rocker shovel
Transport.	10.5	7.1	4.9	t	loco haulage
Hoisting	5.2	5.2	5.2	t	Koepe winder
Miscellaneous				t	10% of above costs

Table 8.1 Unit costs for sublevel stoping (with deg.1 mechanisation), DTH stoping (with deg. 3 mechanisation) and cut and fill stoping at 100% productivity. (derived from Cost and Industrial Engineering reports, KCC, 1986).

8.34 Input metal price and process recoveries

The copper price, the variable processes' costs, concentrator through wirebar plant and their associated recoveries: concentrator - 91.0%, smelter - 93.4%, refinery - 99.90% and wirebar - 99.95% were used for the basic calculation of the cutoff grades. The metal price was assumed to be constant for the duration of a stoping operation.

Having established the cost and revenues, the cutoff grade can be calculated for the different stoping methods and factors.

8.4 Cutoff grade calculations for various factors

8.41 Methods of stoping

Cutoff grade analysis was undertaken for the following stoping methods:

-Sublevel stoping
-Down-the-hole(DTH) stoping
-Flat back cut and fill stoping with posts and pillars.

The cutoff grades for the above methods were computed at the varying productivity levels. The results are shown in table 8.2 and fig. 8.1. An example of calculation is given in chapter 11.

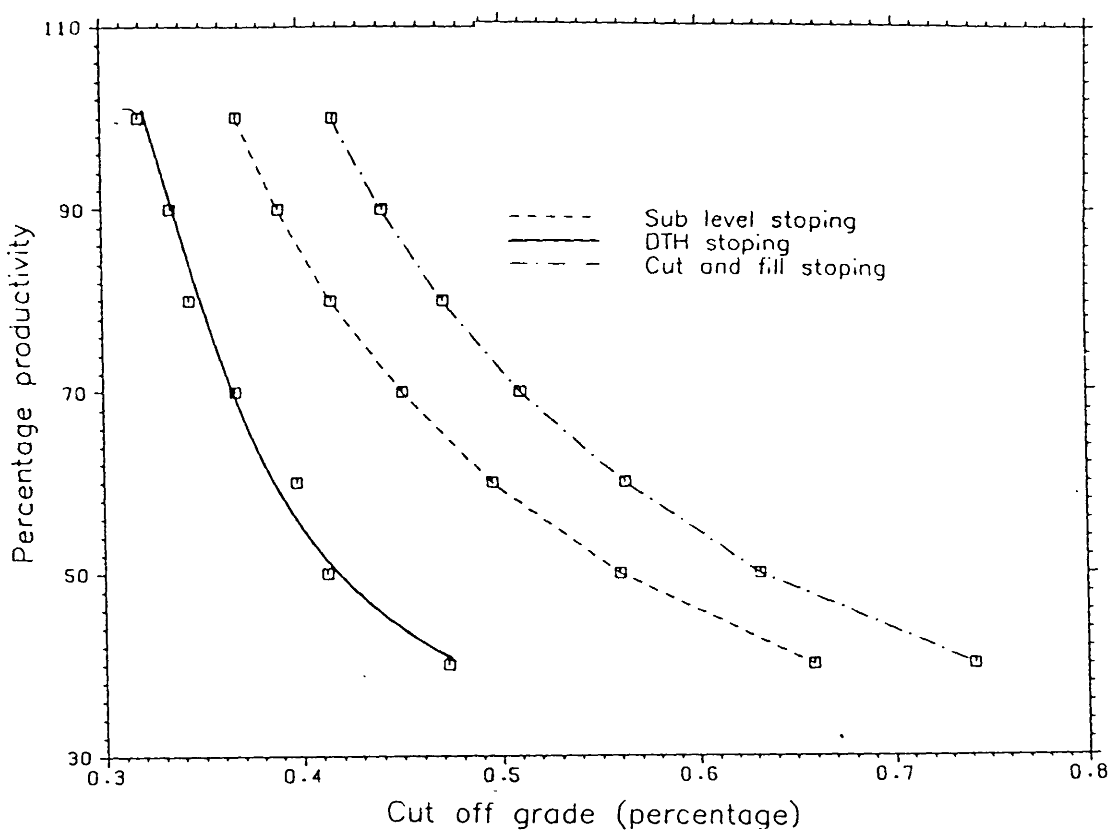


Fig. 8.1 Influence of stoping methods on cutoff grades.

Productivity %	Sublevel stoping %Cu (base case)	DTH stoping %Cu	Cut and Fill stoping %Cu
100	0.369	0.319	0.418
90	0.390	0.335	0.442
80	0.416	0.442	0.472
70	0.451	0.367	0.511
60	0.496	0.398	0.563
50	0.560	0.413	0.632
40	0.659	0.473	0.742

Table 8.2 Change in cutoff grades at different levels of productivity for different stoping methods.

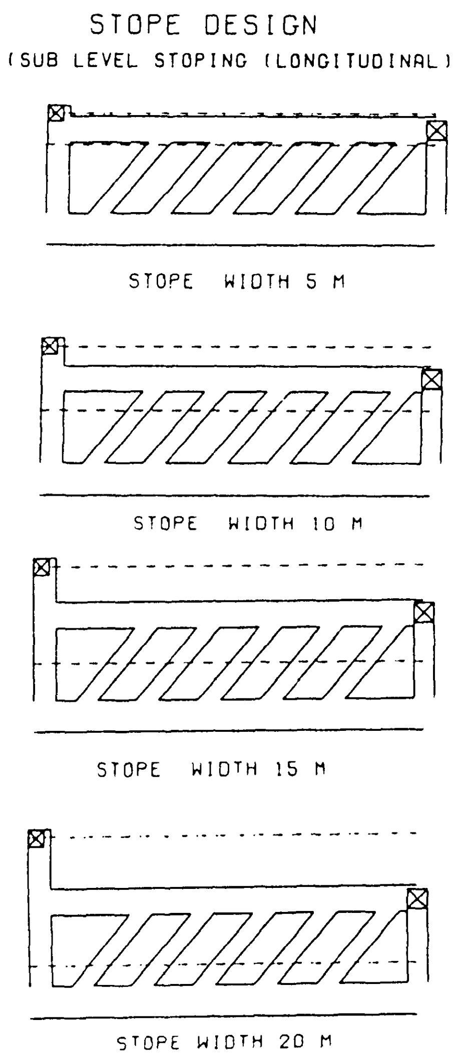
A comparison of these results indicates that cut and fill stoping will require the highest cutoff grade and DTH stoping the lowest.

8.42 Orebody thickness

The total mining cost and unit mining costs are also influenced by stope width. O'Hara (1980) concluded that total labour costs for Canadian mines tend to vary 0.5 exponent of width and the supply costs tend to vary with the 0.8 exponent. In fact, stope width affects all the aspects of mining costs as well as direct stoping costs. This is due to the fact that as the stope width increases, the operations, such as mine development, ore transportation and mine services, are more concentrated and efficient.

The author analysed the development plans for Khetri and Kolihan mines, having stated ore reserves of 40 and 28 millions respectively. It was found that the total development work, with the existing stoping methods, for Khetri mine amounts to 110 km whereas that for Kolihan is 56 km. Partly, this difference can be accounted for by the difference in their reserves, but the rest is due to orebody thickness. The orebody at Khetri is thin and scattered but the one at Kolihan is thick and continuous (as described in chapter 2).

Stope designs (plans and sections) shown in figs. 8.2, 8.3, 8.4 and 8.5, for sublevel stoping with thickness ranging from 5-60 m, were generated, using the stope design models described in chapter 10. The resulting cutoff grades for each of the thicknesses considered are shown in table 8.3. A plot of thickness versus cutoff grade in fig. 8.6 illustrates that cutoff grade is almost constant once a certain minimum thickness of the orebody is attained.



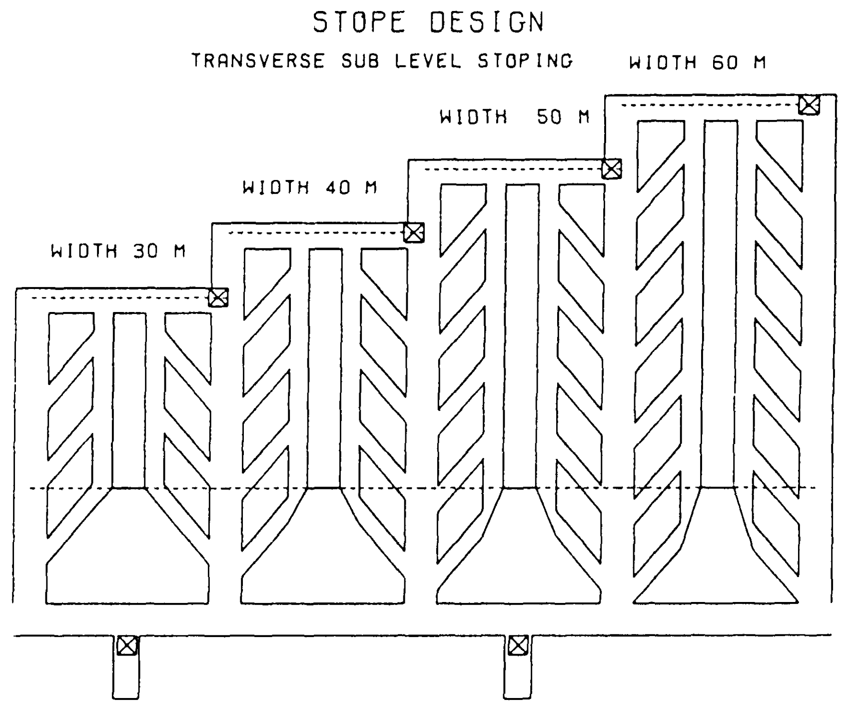
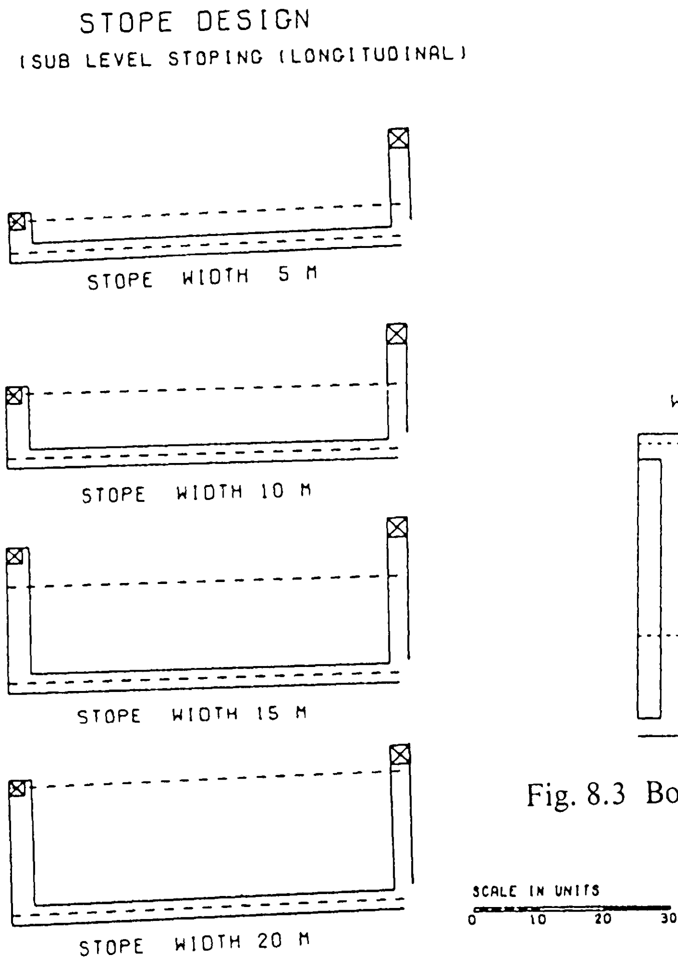


Fig. 8.2 Extraction layout with varying orebody thicknesses - sublevel stoping.

SCALE IN UNITS 0 10 20 30

-------- OREBODY PROFILE



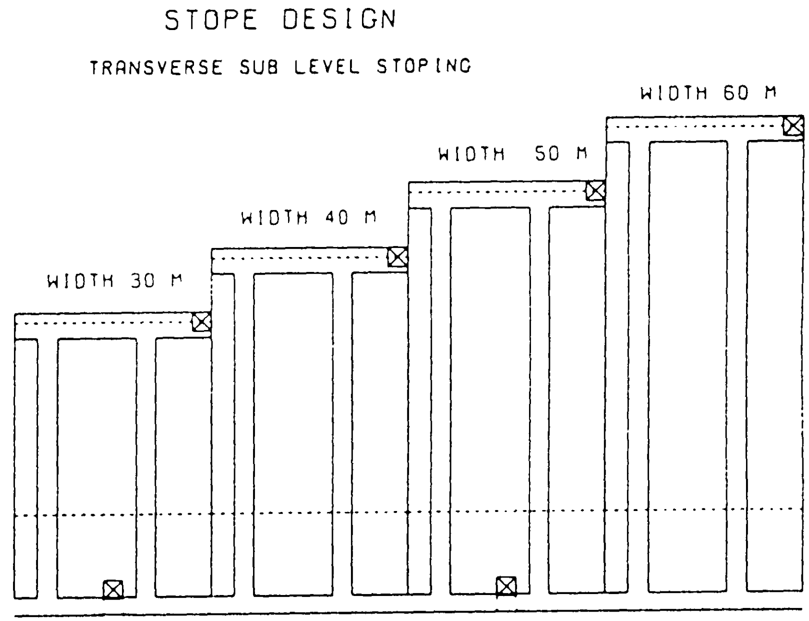


Fig. 8.3 Bottom sublevel layout with varying orebody thickness - Sublevel stoping.

-------- OREBODY PROFILE

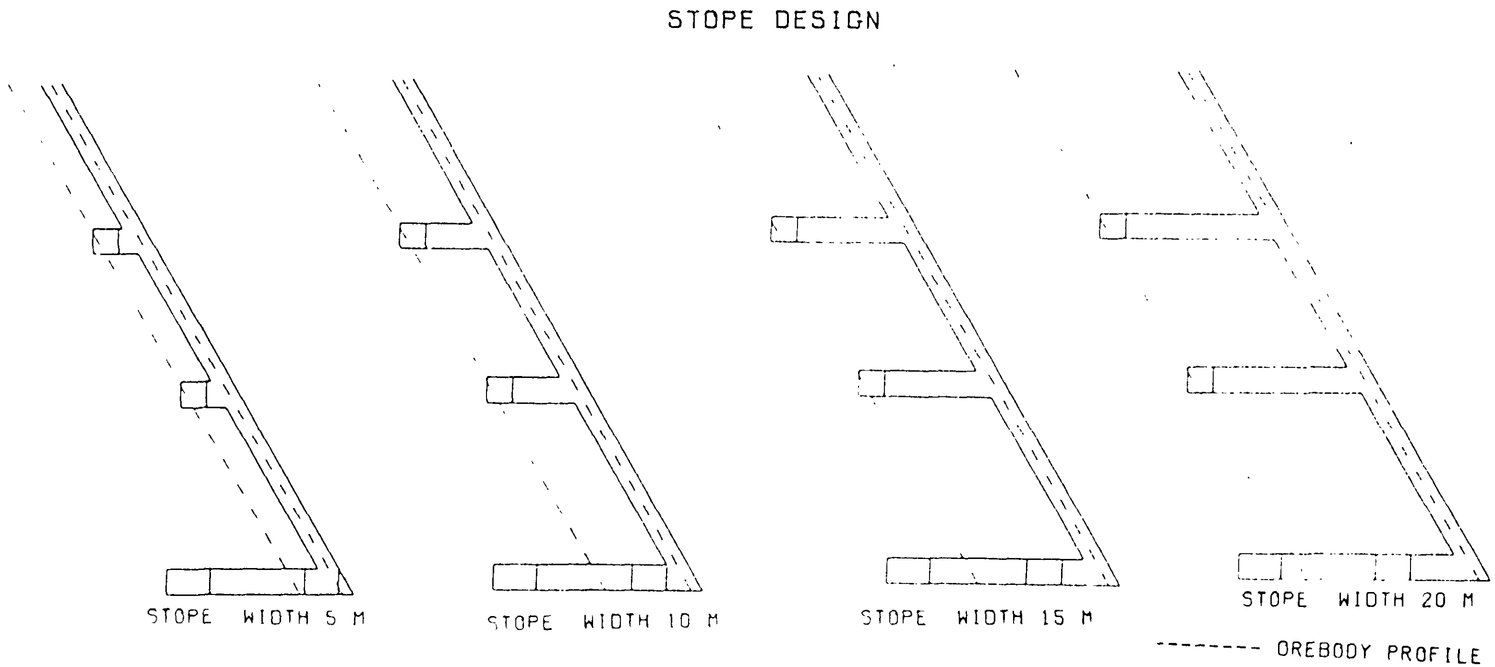


Fig. 8.4 Cross-sections along slot raises with varying orebody thicknesses
- Sublevel stoping.

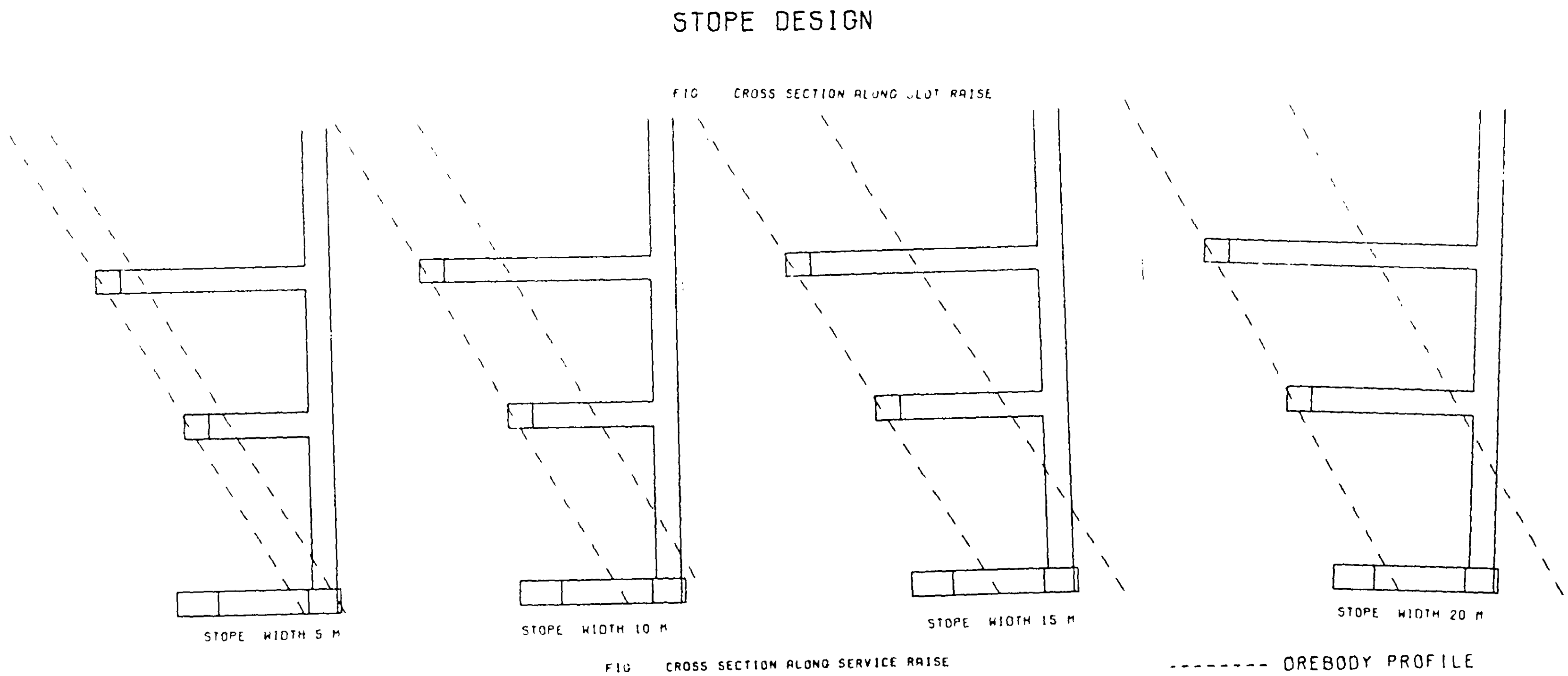


Fig. 8.5 Cross-sections along service raises with varying orebody thicknesses
Sublevel stoping.

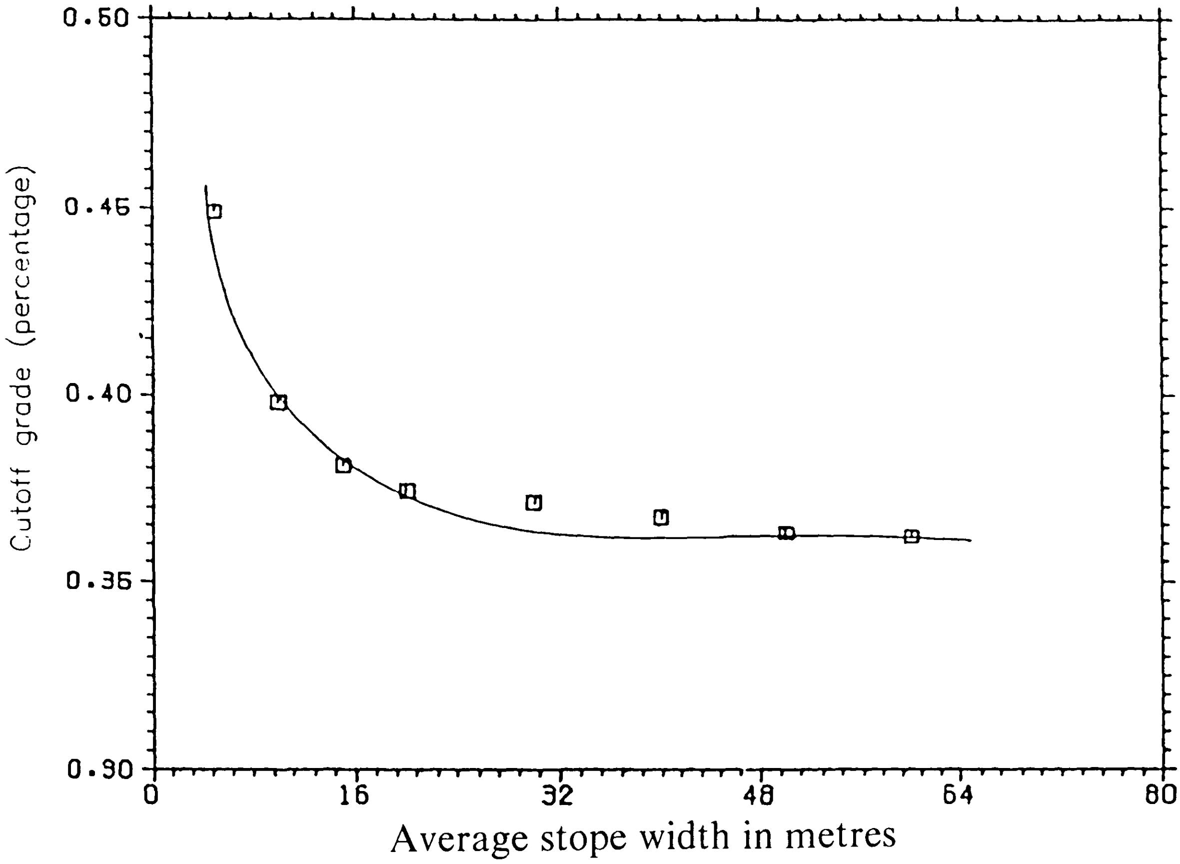


Fig. 8.6 Influence of orebody thickness on cutoff grades - Sublevel stoping

Average Thickness in metres	Cutoff grade %Cu	Remarks
5	0.449	productivity 100%
10	0.398	longitudinal sublevel stoping
15	0.381	
20	0.374	
30	0.371	Transverse sublevel stoping
40	0.367	
50	0.363	
60	0.362	

Table 8.3 Change in cutoff grade with thickness, sublevel stoping.

8.43 Degree of mechanisation

Mechanisation means performing the underground operations using machines. The capacity of a machine is usually related to its size. Therefore, it is advantageous to select the largest units possible taking into account the aspects of flexibility, excavation and access size.

In order to examine the influence of varying degrees of mechanisation on cutoff grades, an equipment grouping was first formulated to designate different levels of mechanisation.

Use of higher bucket capacity LHDs, multi boom jumbos, large capacity dumpers and trucks in large underground mines is not uncommon, but the types of the equipment that are available at KCC mines, have been selected and grouped in the following manner:

Use of conventional pusherleg drills, rocker shovels, loco haulage and blasthole drills of 50-60 mm dia. forms degree-1 mechanisation.

Degree-2 mechanisation means uses of jumbos, trackless equipment such as LHDs (1 cu. yd. capacity), low profile dumpers and small capacity trucks. Stope drilling is by the same drills as in degree-1 mechanisation.

Degree-3 mechanisation has the same set of machines as degree-2 mechanisation except that drilling (for stoping) is by down-the-hole drills capable of drilling holes of 150-200 mm dia. of +40 m length.

The cutoff grades, computed at varying degrees of mechanisation, are shown in table 8.4. The cutoff grade plots with varying degrees of mechanisation for sublevel stoping are shown in fig. 8.7. It can be seen that even with same mining method, there can be an appreciable difference in cutoff grades while adopting different degrees of mechanisation.

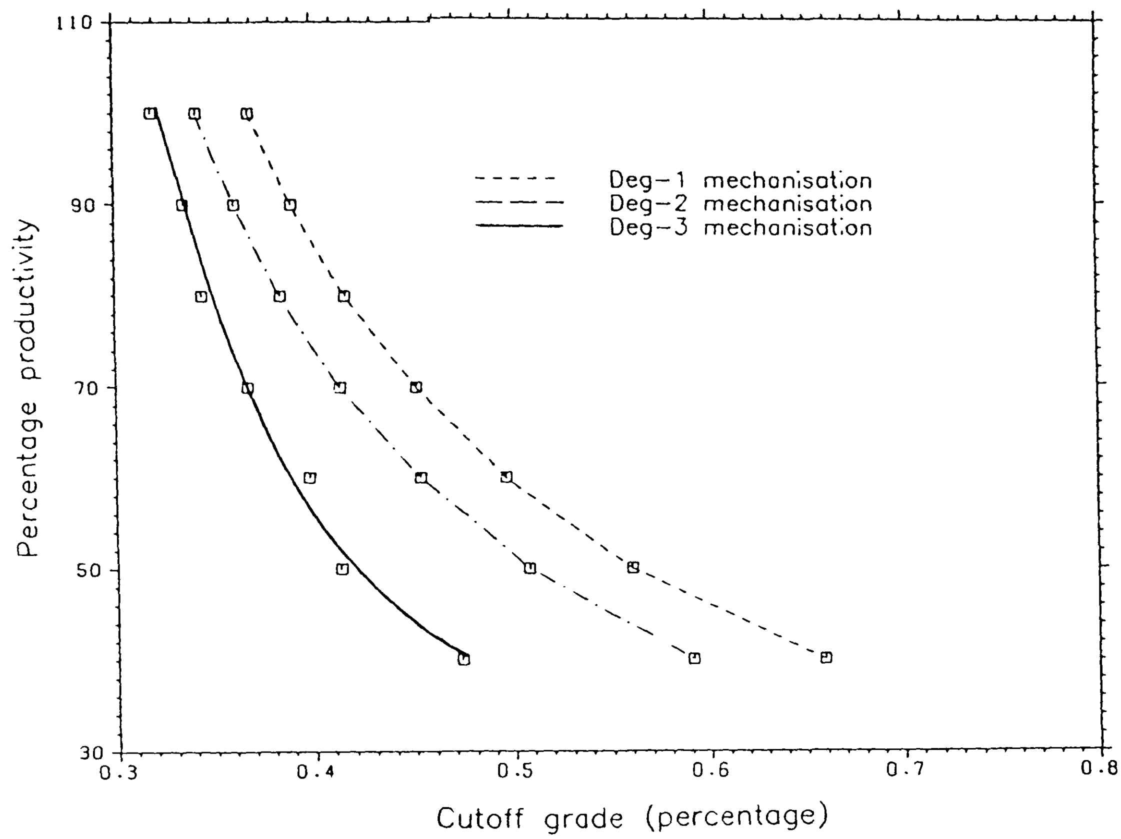


Fig. 8.7 Influence of mechanisation on cutoff grades.

Productivity	Degree of mechanisation			
%	Deg.-1 (base case)	Deg.-2	Deg.-3	Remarks
100	0.369	0.342	0.319	
90	0.390	0.361	0.335	
80	0.416	0.384	0.344	
70	0.451	0.413	0.367	
60	0.496	0.453	0.398	
50	0.560	0.508	0.413	
40	0.659	0.591	0.473	

Table 8.4 Change in cutoff grade with change of mechanisation (sublevel stoping)

8.44 Productivity

As described above, the measure of productivity in mines is the OMS. It is hardly possible to operate any mine at full efficiency, but maximum utilization of equipment may lead to a higher efficiency. However faulty layout, adverse working conditions, mismatching and abuse of equipment, poor fragmentation or lack of trained personnel, often leads to low productivity.

The plots shown in figs. 8.1 and 8.7, and tables 8.2 and 8.4, clearly indicate how working efficiency (productivity) influences the cutoff grade calculations. Productivity certainly has a predominant effect on cutoff grades in any situation.

From table 8.4 and fig. 8.6 it can be seen that at 100% productivity the difference in cutoff grade between degree-1 and degree-3 mechanisation is 15%, but at 40% the same difference rises to 38%. This indicates that productive mechanisation is related closely to higher utilization of machines i.e. operating machines with fewer and shorter interruptions. It is sensible to calculate cutoff grade taking an appropriate level of productivity into consideration to achieve the desired results.

8.45 Process recovery factor

Taylor (1985) considered that recovery of content must be high enough to offset the risks and costs of discovery. A series of cutoff grade calculations were undertaken by allowing a variation in the existing process recoveries of the various plants of the mining complex.

The variation in cutoff grade with the process recovery factors has been shown in the spider diagram in fig. 8.8 This illustrates that there is a sharp decline in cutoff grade with the increase of process recovery percentage.

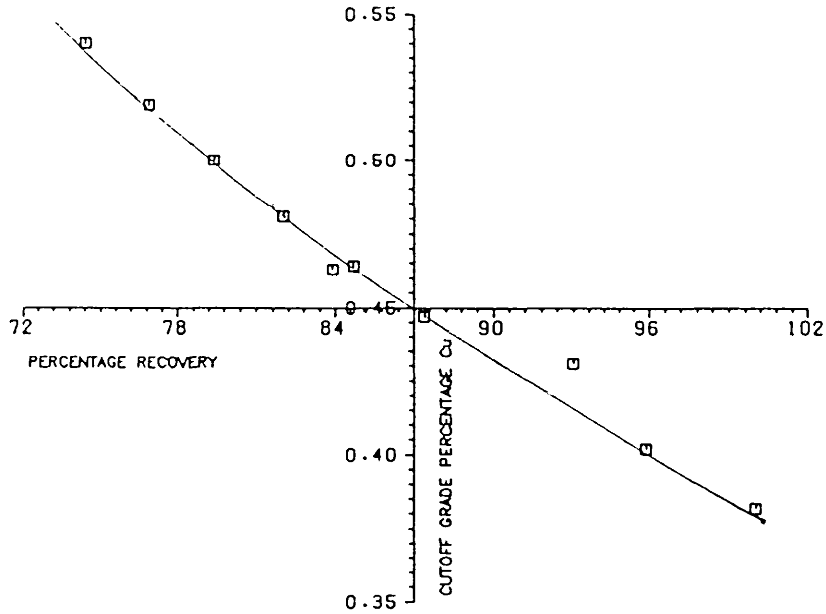


Fig. 8.8 Influence of overall process recoveries on cutoff grades

Overall Recovery %	Cutoff grade %Cu	Remark
100.00	0.382	
95.86	0.402	
93.00	0.431	
87.40	0.447	
84.70	0.464	
83.90	0.463	present operating level
82.00	0.481	
79.40	0.500	
76.90	0.519	
74.40	0.540	
72.00	0.561	

Table 8.5 Change in cutoff grade with change of overall process recovery percentage.

8.46 Depth of the deposit

A specific study to examine the effect of increasing depth of an orebody on cutoff grades was not made. However, mining of deeper levels is more costly than mining at shallow depths, because more energy will be needed for ventilation, water pumping and rock hoisting. The productivity in the mine will be decreased because of longer transportation distances for the men, material and the rock. Change in rock conditions necessitate a change in the mining method used. This is an area where further research in the future can be undertaken.

Increasing depth may therefore need a higher cutoff grade.

8.5 Conclusions

As a result of this study made for the mining complex, under Indian conditions, the following observations for the underground mines in general could be made.

In underground mines, even for the same deposit, the cutoff dependent costs vary significantly according to stoping methods, orebody thickness, degree of mechanisation and working efficiencies of the men and machines.

The sublevel stoping and its variants can be operated at a lower cutoff grade than cut and fill and its variants.

Thinner orebodies adversely affect the cutoff grades but once a certain minimum thickness is attained, the cutoff grades become almost stable.

The cutoff grade for the same stoping method, in the same mine, can vary if operated using different sets of machines. Labour intensive methods need higher cutoff grades.

An allowance for working efficiency, that is the productivity, should be made when calculating cutoff grades.

There is a sharp decline in cutoff grades with an increase of process recoveries.

Appendix 1

Equipment	Output/shift	units	Remarks
1. Drilling			
Development:			
Pusherleg drills (face size 3m x 3m)	0.36	m	linear adv./manshift
Jumbo drills	40.0	m	drilling
Stoping:			
Blasthole drills	30.0	m	
Wagon drills	46.0	m	Rajpura Dariba mines
DTH drills	7.3	m	crew strength 4
2. Blasting			
Blastholes	112.0	m	for stoping
DTH	22.0	m	for stoping
3. Mucking			
Development:			
Rocker shovel	84.0	t	
Auto loader	84.0	t	
LHD	160.0	t	lead within 60m
Stoping:			
Rocker shovel	105.0	t	
Auto loader	104.0	t	
LHD	200.0	t	
4. Transportation			
Development:			
Loco	84.0	t	
JDT	160.0	t	
stoping:			
Loco	105.0	t	
Loco	225.0	t	chute loading
JDT	200.0	t	lead ÷400m
JDT	240.0	t	chute loading

Table 8.6 Performance norms of various equipment at 100% productivity. (source: Industrial Engineering Report, KCC, 1986).

Appendix 2

Equipment owning and operating costs and its performance:

1.Name of mine:
2.Name of equipment,its make,model and motive power:
3.Price of equipment(FOB factory):
4.Excise duty,freight,insurance etc:
5.Errection cost(if any):
6.Delivery cost:
(Sum of items no.3-5)
7.Salvage value(if any):
8.Depreciation value:
(Item no.6-Item no.7)
9.Depreciation period:years
shifts
hours

Fixed cost(per shift):

X-Depreciation cost/shift:
(Depreciation value/depreciation period in shifts)

Y-Variable cost/shift:
Fuel/power/c.air:
Lubricants:
Spares:
Tyre and tubes:
Repair cost:
(specify the wages of maintenance crew allocated for this equipment)
Miscellaneous variable cost,if any:
Z-Operator cost:
(specify the wages of the operator)

Total owning and operating cost/shift(A):
(Sum of the variable and fixed costs)

Table 8.7 Standard pro forma for cost calculation.

Chapter 9

9. Modelling stope boundaries using incremental analysis

9.1 Introduction

In chapter 6, it was demonstrated how irregularities in the profile of the orebody increase with a rise in cutoff grades. Some mining methods, such as cut and fill can follow an irregular profile closely. However, methods such as sublevel and DTH stoping require even stope walls to allow the broken material to gravitate to loading points. With irregular ore boundaries having even stope walls could mean the inclusion of certain sub-grade mineralised material and sacrificing some high grade mineralisation.

Barnes (1982, pp. 69-72) described the use of cones (coning concept) to establish an ultimate open pit design. Each cone is expanded until an economic optimum criterion is met. An approach to determine the ore reserve and plant size in the context of open pit mines, using incremental financial analysis was, demonstrated by Halls, et al. (1969). No published literature has been found in which underground stope boundaries were defined using this technique. However, Tatiya and Allen (1987) will be presenting a procedure in a conference in Sept. 1987.

This chapter outlines a method for defining and evaluating stoping boundaries for alternative underground mining methods. Fifteen geological sections lying between two levels of Khetri mine were kriged in 1m x 1m panels. This resolution allowed stope boundaries to be defined using a grade cutoff criterion. Three stoping methods - Sublevel, Down-the-hole (DTH) and flat back cut and fill - were modelled. Optimum stoping boundaries, partly constrained by the technical aspects of the mining method, were evolved using an incremental analysis similar to that found in open pit optimisation. Costs, revenues and recoveries were based on those prevailing at the mine.

9.2 Orebody modelling

9.21 The basic geological data

The basic geological data obtained from Khetri mine was used for the investigation described in this chapter.

A small section of the deposit was chosen, measuring 400m along strike and 60m vertically (between two main haulage levels). Geological studies indicated the mineralisation along strike to be continuous. The sampling data were obtained from diamond drillholes, drilled in 'fans', across the mineralisation at 30m intervals along the strike.
In all, there are 136 holes drilled either due East or West at inclinations in the range 0 to 82 degrees from the horizontal. The inclined length of these holes was in the range of 42-171m and the number of samples recovered were 3643. The average sampling density for the section under study is 20 sq.m/sample.

9.22 The structural analysis - semi-variogram:

Figure 9.1 shows an isotropic semi-variogram for the part of the deposit analysed. Directional semi-variograms were also calculated. Due to insufficient sampling pairs those in the direction of the strike (N-S) were found to be 'noisy'. A spherical model, with a nugget variance (Co) $0.325(\%)^2$, continuity (C) $0.335(\%)^2$, sill variance (Co + C) $0.660(\%)^2$ and a range of 18m was 'fitted' to the isotropic experimental semivariogram. The model was then cross validated through point kriging. This gave a correlation coefficient (actual vs estimated) of 0.62. The model then formed the basis for subsequent kriging of the sections across the deposit (E-W direction).

9.22 Estimation mineral inventory

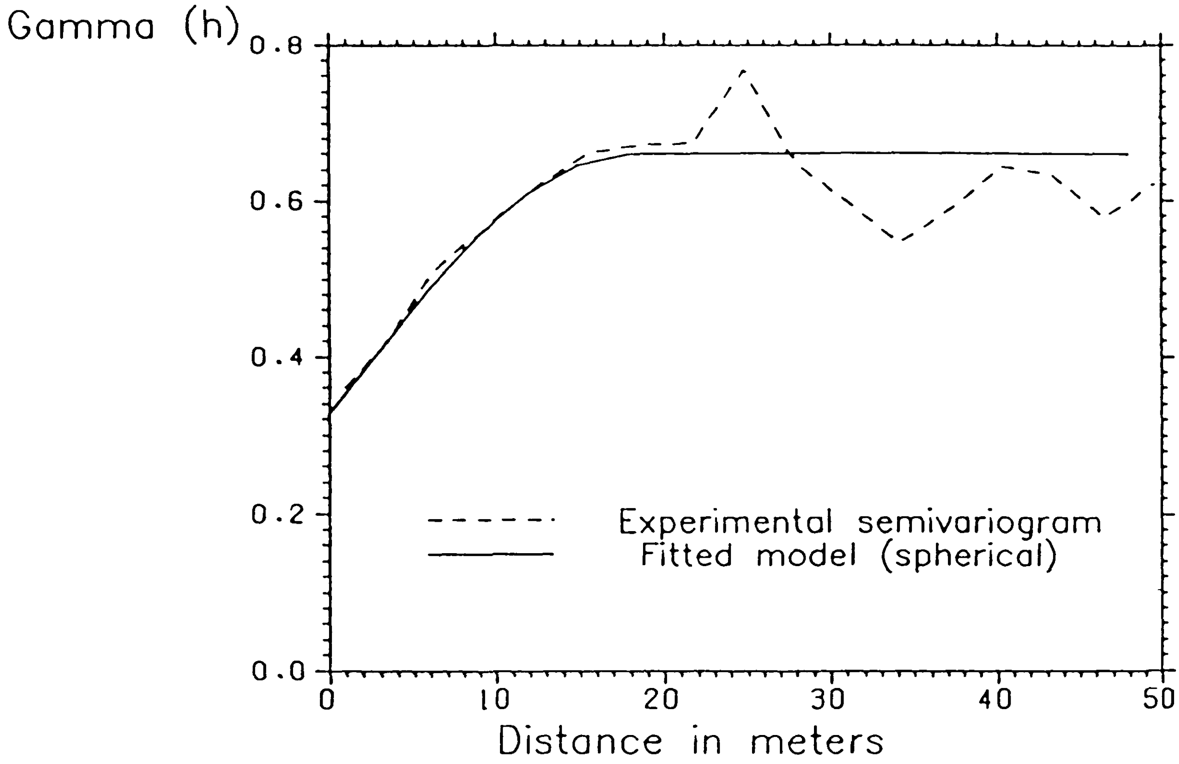


Fig. 9.1 Experimental semivariogram with fitted model.

As the section under consideration has been sampled at a fairly close interval and selectivity is an important criterion for stope design purposes, small size blocks were considered.

A matrix of panels with 1m x 1m centres was created for each of the 15 sections. A grade value for each panel was estimated by kriging. There were 4,000 to 10,000 panels (squares) indicating some mineralisation in each of the 15 sections kriged. Fig. 9.2 shows a display of kriged values of panels along one section. The shape and value of the mineralisation was assumed to be projected 15m north and south of each section. Thus, using a tonnage factor of 3.0 tons per cubic metre, each block represents 90 tons of material. Kriging this section gave an overall mean of 0.428% Cu and standard error of estimation 0.0108

9.3 Techno-economic parameters

9.31 Methods studied

As mentioned above three mining methods - sublevel, DTH and cut and fill stoping were studied. The design features of these methods have been described in chapter 10.

9.32 Economical aspects

The cutoff grade has been calculated using the criteria described in chapter 8, by considering the mechanisation, productivity and process recovery factors.

Mining costs for both methods in use were computed for a range of productivity levels. A 70% productivity level was used in this exercise. Data for cut and fill stoping was obtained from another mine having similar working conditions and where this method is in use.

In this study the operating margin has been taken to establish the optimum

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Legend

-Blocks marked 1 to 9 are the blocks of grade .1 to .9% Cu respectively.

-Blocks marked R are the blocks of grade 1% or more

-Blocks marked 0 are the blocks of grade >0.0< 0.1%Cu

Fig. 9.2 Mineralization along a section as obtained by kriging economic criterion . This follows the approach suggested by Dowis (1982). The following have been included to calculate the operating margin for a stoping unit or an 'envelope' (to be defined later) to estimate revenue and costs:

-Price of metal
-Revenues from the byproducts
-Ore recovery while mining
-Metallurgical recoveries
-Total variable costs(mining through wirebar casting)
-Royalty, excise duty and selling costs of finished goods.
-(Primary development costs are considered to be 'sunk')

The operating margin is partly the cash flow which is used to calculate the financial criterion (see chapter 11).

9.33 The characteristics - mining methods:

The three mining methods are characterised by geometrical constraints as follows:

For sublevel and DTH stopes the wall slopes should be greater than 45 degrees from the horizontal. The draw point level is imposed.
The straightening or smoothing of stope walls for long hole stopes is often required for the free flow of the blasted muck from the stopes. It may also improve blasting efficiencies. This is a standard stope designing practice in the mines where these stoping methods are adopted (King, 1982, p.12), (Hans Hamrin, 1982, p. 107). However, this aspect may require the inclusion of some sub-grade material as well as exclusion material above a cutoff grade.

For the cut and fill method the stope height is determined by the distance between levels (60m). The stope should have a minimum width of 4m, to allow access for drills and mucking machines.

The mining costs for DTH, sublevel and cut and fill stoping methods have been estimated at 63, 87 and 103 Rupees per tonne respectively. The resulting lower cutoff grade will result in DTH stoping including more mineralised material than, say, the cut and fill method.

As indicated above, the stope design geometry of sublevel and DTH stopes require their walls to be flat and having a minimum slope to allow smooth flow of the blasted muck. This may require the sill level to be located in subgrade material. However, this is not the case with cut and fill stopes where the operations can follow the grade boundaries more closely. In all cases it is not easy to exclude internal waste or subgrade material.

9.4 Selection of stope boundaries

9.41 The envelope (stope boundaries):

The principle adopted is to consider a number of incrementally different mining excavation layouts. In the underground situation this is interpreted as a number of 'envelopes' that could form possible stope boundaries. An envelope is defined as the mineralised zone bounded by the upper and lower development levels (60m apart), the interfaces with adjacent stopes (here 30m apart) and the two sidewalls.

9.42 Defining sidewall (long hole stopes):

The procedure for defining the sidewalls for the different methods follows:
1. The mineral inventory of a section to be studied is established by kriging. See fig. 9.2.
2. The cutoff grade of the mining method under consideration is then calculated.
3. The next step establishes the slopes of the stope sidewalls for the two long hole methods. Initially those blocks which are at and above cutoff grade are displayed. See fig. 9.3. From this display, the wall angles for both walls are then selected manually so that the angles chosen should include maximum number of blocks. The wall angles are incremented by 3 to 5 degrees and the frequency distribution of the values and grade-tonnage relationship of the mineral inventory contained within each of the envelopes are established. The economics of each of the envelopes is then calculated (table 9.1). That which yields the maximum operating margin establishes the wall angles of the stope.

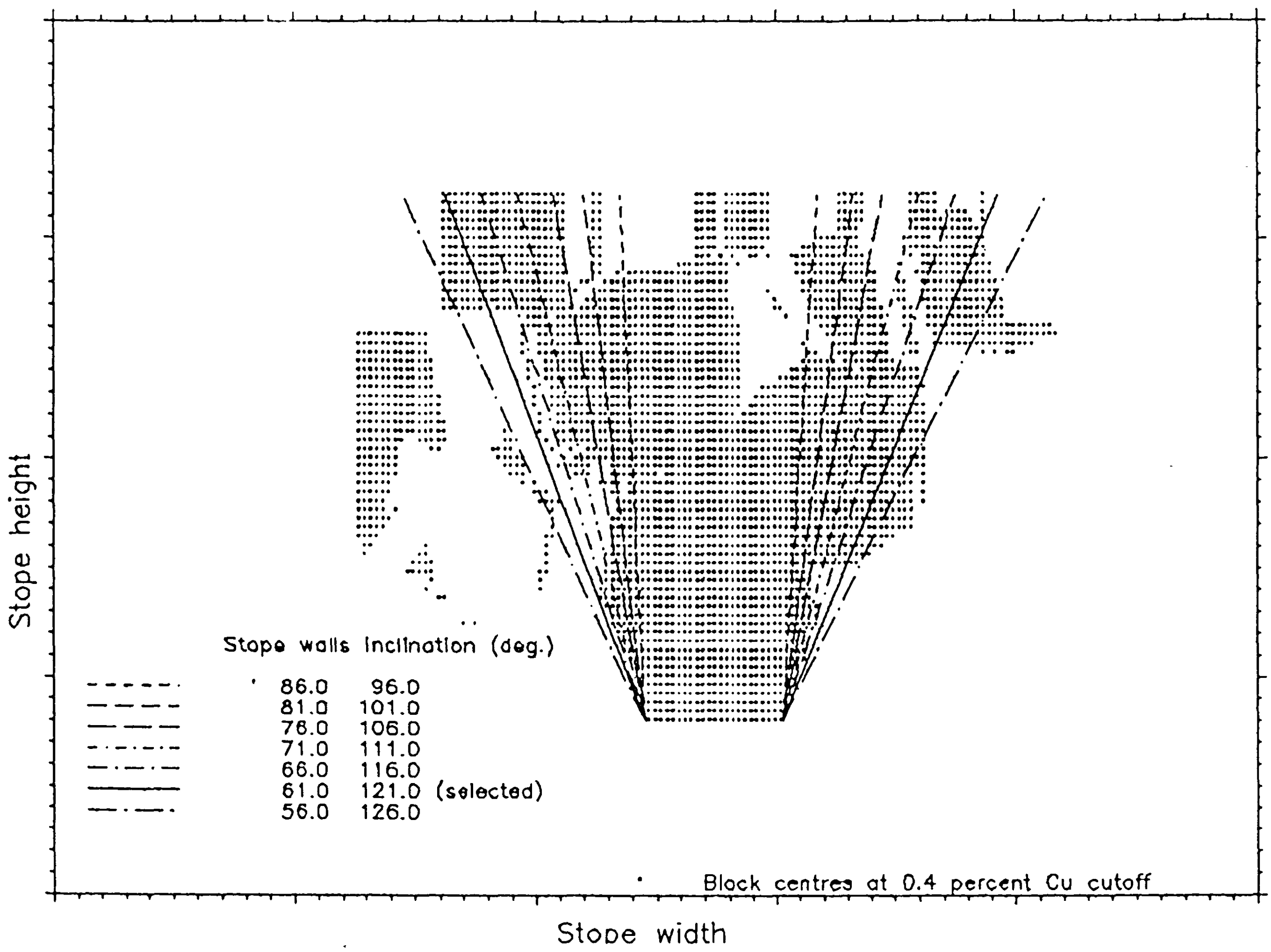

Fig. 9.3 Selection of stope walls' inclination - sublevel stoping.

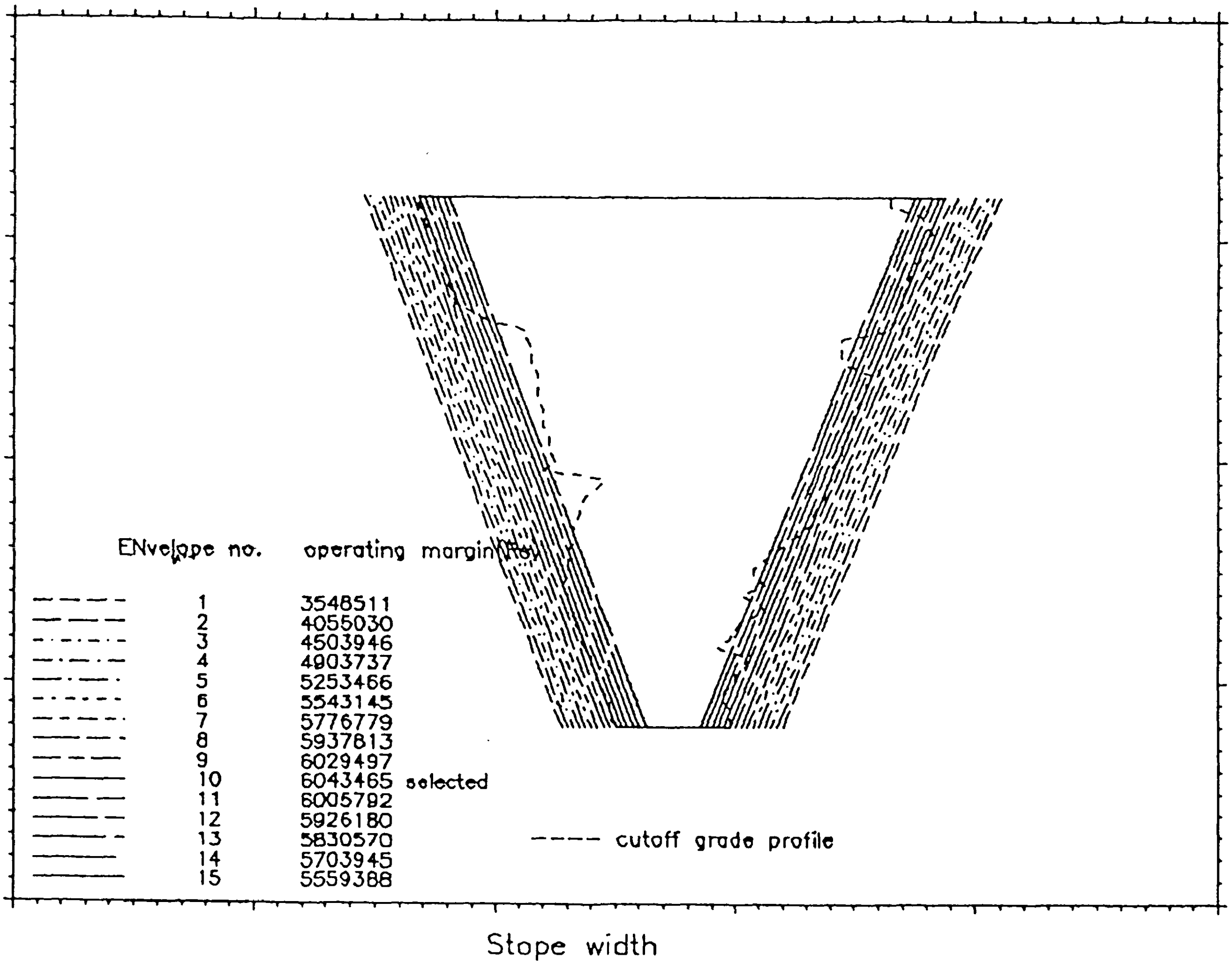

Fig. 9.4 Selection of stope boundaries by increment/decrement in stope size, sublevel stoping.

envelope no.	slope 1	slope 2	average grade	ore tons.	metal tons.	total revenues(Rs)	total costs(Rs)	operating margin(Rs)	remark
1	86	96	.577	141213.6	684.6	29059190.1	24093423.3	4965766.8	
2	81	101	.566	166953.6	794.0	33704942.7	28347102.6	5357840.1	
3	76	106	.554	193010.4	897.8	38113111.1	32590658.0	5522453.1	
4	71	111	.545	221522.4	1014.1	43050430.3	37258180.9	5792249.4	
5	66	116	.536	250826.4	1128.3	47894409.8	42006503.8	5887905.9	
6	61	121	.527	283298.4	1253.0	53190886.2	47253072.6	5937813.6	selected
7	56	126	.505	319017.6	1353.1	57438293.6	52689678.3	4748615.2	

Table 9.1 Techno-economics for stope walls slope - Sublevel stoping

9.43 The ultimate stope profile

The ultimate stope profile is then derived by moving the stope walls horizontally by parallel increments of 1m. Table 9.2 and figure 9.4 show how the maximum operating margin enables the optimum stope profile to be chosen for a sublevel stope. In table 9.3 the details of frequency distribution of grade values within the envelope chosen is given. Table 9.4 describes the grade-tonnage distribution of the mineral inventory contained within it for sublevel stoping.

envelope no.	positions wall1	positions wall2	average grade	ore tons.	metal tons.	total revenues(Rs)	total costs(Rs)	operating margin(Rs)	remark
1	1241	1278	.488	350935.2	1438.4	61059201.3	57510689.9	3548511.4	
2	1242	1277	.494	341272.8	1416.8	60144777.3	56089746.7	4055030.7	
3	1243	1276	.500	331610.4	1393.6	59157257.0	54653310.1	4503946.9	
4	1244	1275	.506	321948.0	1368.8	58107397.4	53203660.3	4903737.1	
5	1245	1274	.512	312285.6	1342.6	56994012.5	51740545.9	5253466.6	
6	1246	1273	.517	302623.2	1314.6	55804425.6	50261279.9	5543145.7	
7	1247	1272	.522	292960.8	1284.9	54543718.7	48766939.6	5776779.2	
8	1248	1271	.527	283298.4	1253.0	53190886.2	47253072.6	5937813.6	
9	1249	1270	.531	273636.0	1219.1	51750050.2	45720552.6	6029497.5	
10	1250	1269	.534	263973.6	1182.8	50210594.8	44167129.6	6043465.1	selected
11	1251	1268	.536	254311.2	1145.0	48605609.6	42599817.1	6005792.5	
12	1252	1267	.538	244648.8	1105.9	46947404.4	41021224.3	5926180.1	
13	1253	1266	.540	234986.4	1066.4	45268899.4	39438328.8	5830570.7	
14	1254	1265	.542	225324.0	1025.9	43551037.0	37847091.2	5703945.9	
15	1255	1264	.544	215661.6	984.9	41810418.5	36251030.2	5559388.3	

Table 9.2 Techno-economics for stope profile - Sublevel stoping

CLASS INTERVAL	FREQUENCY	PERCENTAGE	CUMUL FREQ	CUMULATIVE %
.00000 - .10000	0	.0000	0	.0000
.10000 - .20000	23	.6901	23	.6901
.20000 - .30000	172	5.1605	195	5.8506
.30000 - .40000	497	14.9115	692	20.7621
.40000 - .50000	828	24.8425	1520	45.6046
.50000 - .60000	823	24.6925	2343	70.2970
.60000 - .70000	393	11.7912	2736	82.0882
.70000 - .80000	293	8.7909	3029	90.8791
.80000 - .90000	240	7.2007	3269	98.0798
.90000 - 1.00000	50	1.5002	3319	99.5800
1.00000 - 1.10000	13	.3900	3332	99.9700
1.10000 - 1.20000	0	.0000	3332	99.9700
1.20000 - 1.30000	0	.0000	3332	99.9700
1.30000 - 1.40000	1	.0300	3333	100.0000

Table 9.3 Frequency distribution of grade values within the envelope, (sublevel

stoping).

CUT OFF GRADE	TONNES OF COPPER	TONNES OF MINERAL/ORE	AVERAGE GRADE ABOVE CUTOFF
.0000	1600.6	299970.0	.53358
.1000	1600.6	299970.0	.53358
.2000	1596.8	297900.0	.53602
.3000	1556.5	282420.0	.55114
.4000	1397.3	237690.0	.58786
.5000	1061.2	163170.0	.65038
.6000	660.3	89100.0	.74112
.7000	433.5	53730.0	.80688
.8000	235.7	27360.0	.86156
.9000	56.0	5760.0	.97136
1.0000	13.4	1260.0	1.06679
1.1000	1.2	90.0	1.35800
1.2000	1.2	90.0	1.35800
1.3000	1.2	90.0	1.35800

Table 9.4 Grade-tonnage computation of the mineral inventory within the envelope.

The difference in tonnage of two adjacent envelopes forms one increment. Should the strip have a grade below cutoff it should not normally be included. See table 9.5.

envelope nos.	strip no.	incremental tonnage	ore grade	metal tons.	operating margin(Rs)	remarks
1-2	1	9662.4	.265	21.5	-506519.3	excluded
2-3	2	9662.4	.287	23.3	-448916.3	excluded
3-4	3	9662.4	.305	24.7	-399790.1	excluded
4-5	4	9662.4	.323	26.2	-349729.5	excluded
5-6	5	9662.4	.345	28.0	-289679.1	excluded
6-7	6	9662.4	.366	29.7	-233633.5	excluded
7-8	7	9662.4	.393	31.9	-161034.5	excluded
8-9	8	9662.4	.418	33.9	-91683.9	excluded
9-10	9	9662.4	.447	36.3	-13967.6	excluded
10-11	10	9662.4	.466	37.8	37672.7	included
11-11	11	9662.4	.481	39.1	79612.3	included
12-13	12	9662.4	.487	39.5	95609.5	included
13-14	13	9662.4	.499	40.5	126624.8	included
14-15	14	9662.4	.505	41.0	144557.6	included

Table 9.5 Techno-economics for stope's incremental tonnage - Sublevel stoping

Following this methodology, stope profiles are obtained for sublevel and DTH stoping for sections along the orebody as shown in figures 9.5 & 9.6.

method	mining cost Rs/ton	cutoff grade	average grade	ore tons	metal tons	revenues in ,000Rs	costs in ,000Rs	operating margin in ,000Rs	remarks
sublevel	87	0.452	0.534	263973	1183	50210	44167	6043	
DTH	63	0.367	0.522	292960	1285	54543	41735	12807	
cut & fill	103	0.509	0.547	251697	1156	49068	46393	2675	including internal waste
cut & fill	103	0.509	0.577	141213	0685	29060	26352	2706	excluding sub grade material

Table 9.6 Techno-economical summary of stope profile for different methods

9.44 Stope boundaries - cut and fill stoping:

```
  1203      1213      1223      1233      1243      1253      1263      1273      1283      1293      1303
    .         .         .         .         .         .         .         .         .         .         .
301                                                                                                              301
300                  555555555555565754333344333333433333322444544568766632111323333644463222133444443223323342  300
299                  55555555555556575433334433333343333332244454456876663211132433364446322213344444322332 34  299
298                   5555555555556575433334433333333333333224445445687666321113243336444632221334444434654334  298
297                   5555555555556575433334433333333333333224445445687666321113243336444632222334544444546533  297
296                    555555555556575433334433333333333333224445445687666321113243336444322222345444454446 5  296
295                     55555555556575433334433333333333333224445345687666321113278787445333222244444455444 5  295
294                     5555555555657543333443333333333333322444534568745422222666464 46R932332334444455554 4  294
293                      555555555657543333443333333333333322444343433453422298766664555782323333535455544 4  293
292                      55555555565754333344333333333333366665544434223332222866666455578933334333555554 4  292
291                       5555555565754333344334356676766767644454322223322229666666455576893334334555554 4  291
290                       555555556575433443344566767788887777645432222233222287666645557893334433455554  290
289                        55555556574444334444678778878888778765443322224333228766664555789333443345555  289
288                        5544445554443334555678778878888778765453323223433333666666455788333443344555  288
287                         444555544464344555567878878888878765454333222444333R866664555789333443345 5  287
286                          3334544344444455555678778888888877876554333222344333396666455578933344334 55  286
285                          334444444444444555567877887888877876544433322233333338666644578833344334 5  285
284                           344344444444444555567878878888778765454332222222333336556445578933343 33  284
283                           344344444444444555567877888888877876454332222222433327544455 78R3334433  283
282                            443444444444445555678778878888778765543222222222244323855445 78R333443  282
281                            34434444444444455567787887888877876564322222234444455 44R9876678R33333  281
280                             443444444444445555678778788888777756432222344444555554443444449 8888  280
279                             4434444444444455556787788788887787664322244444555554444455445445 98  279
278                              44344444444444555678778878888778765432244445555544444554444444 549  278
277                              443444444444455556787787888887875543355555555544444554444444555 9  277
276                               43444444444445555678778888888778554355555555444445544444445555 54  276
275                                4344444444445555678778878788877854455555544444554444444555555 5  275
274                                434444444444555567878878788876445555544444554444444555555 56  274
273                                 34444444444555567877888788876445554444455444444455666556 66  273
272                                 3444444444455555678778878788765554444455444444445566656666 6  272
271                                  3343344444445555678788787786564444455444444455665566666 65  271
270   Legend                         3434444544455556787787877666444455444444455665566666 655  270
269                                   3444433344454567877886665676455444444455665566666 6555  269
268  -Blocks marked 1 to 9 are the    34444333444444444477777776698444444445566556666665 5555  268
267   blocks of grade 0.1 to 0.9%Cu    43333444434446556555555546544444455665566666655555 5  267
266   respectively.                     333333443456555555555544755444555655666666555555 55  266
265                                     33333334555655555555666666776455555666666555555 554  265
264  -Blocks marked R are the blocks     3333334556655555555666666775665566666655555555 44  264
263   of grade 1%Cu or more.             333333455565555555566666677856566666655555554 4444  263
262                                       3333455565555555566666677886665565555555544 4444  262
261                                       33334555655555555666666678867556555555554444 444  261
260                                        333555655555555566666778867555556655444444 44  260
259                                        33455565555555566666677876755554455544444 44  259
258                                         3455565555555566666677877555555445434444 44  258
257                                          4555655555555666666787675555554454433333  257
256                                          5556555555556666667787675555555444443333  256
255                                           55655555556666666778765555555544444333  255
254                                           556555555566666667786755555555544444 2  254
253                                            565555555666666678767555555554444443  253
252                                            45544554566666778767555555544444 44  252
251                                             455545555666 67R9R7665544554444444  251
250                                             455444555666679RR966655544444444 4  250
249                                              5445555666777R8996765554444444 4  249
248                                               55555666777R889999665444444 44  248
247                                               555556665798888999665444444 4  247
246                                                5556666998888778R965544444 4  246
245                                                465666898878777799955444 44  245
244                                                 578997888787 7788R9555444  244
243                                                 48997778887777789R955443  243
242                                                  58777788877777789955 44  242
241                                                  5877778887777778 9R9544  241
240                                                   877778887777777789954  240
239                                                                                                              239
    .         .         .         .         .         .         .         .         .         .         .         .
  1203      1213      1223      1233      1243      1253      1263      1273      1283      1293      1303
```

Fig. 9.5 Stope boundaries with mineral inventory - Sub level stoping (walls straightened)

```
      1203      1213      1223      1233      1243      1253      1263      1273      1283      1293      1303      1313
301                                                                                                                         301
300   4675555555555555565754333344333333433333322444544568766632111323333644463222133444443223323422 21                     300
299   46755555555555555657543333443333334333333224445445687666321113243336444632221334444432233234222                       299
298   6755555555555555657543333443333333333333224445445687666321113243336444632221334444434654334222                        298
297   6755555555555555657543333443333333333333224445445687666321113243336444632222334544444546 53432                        297
296   755555555555555657543333443333333333333224445445687666321113243336444322222234544445444 65432                         296
295   5555555555555657543333443333333333333224445345687666321113278787445333222244444455444 5553                            295
294   555555555555565754333344333333333333322444534568745422222666464 46R9323323344444555544565                             294
293   55555555555565754333344333333333333322444343433453422298766664555782323333535455544 4556                              293
292   555555555555657543333443333333333336666554443422333222866666455578933334333555554 4444                                292
291   5555555555565754333344333435667676676764445432222332222966666645576893334334555554 4455                               291
290   455555555555657543344334456676778888777764543222223322228766664555789333443345555 4444                                290
289   54555555555657444433444678778878888778765443322224333228766664555789333443345555444                                   289
288   5555544445554443334555678778878888778765453323223433333666666455578833344334455 5544                                  288
287   3344445555444643445555678788788888787654543333222444333R8666645557893333443345 5554                                   287
286   3333334544344444455555678778888888877876554333222344333396666455578933344334 55554                                    286
285   3333344444444444455556787788788887787654443332223333333866664457883334433455 55                                       285
284   333344344444444445555678788788887787654543322222223333365564455789333433345 5                                         284
283   33334434444444444555567877888888877876454332222222243332754445578R3334433455                                          283
282   3334434444444444555567877887888877876554322222222224432385544578R333443333                                            282
281   33334434444444444555677878878888778765643222222344444 5544R9876678R33333333                                           281
280   333443444444444455556787787888887777564322223444445555544434444498888333                                              280
279   33344344444444445555678778878888778766432224444455555444445544544598823                                               279
278   333443444444444455567877887888877876543224444555554444455444444 4549843                                               278
277   3334434444444444555567877878888878755433555555554444455444444445559893                                                277
276   33443444444444455556787788888887785543555555544444554444444455555 4444                                                276
275   33443444444444455556787788787887785445555554444455444444445555555554                                                  275
274   3344344444444444555567878878788876445555544444554444444455555556555                                                   274
273   344344444444444555567877888788876445554444455444444445566655666555                                                    273
272   33434444444444455556787788787887655544444554444444455666566666655                                                     272
271   3443343344444445555678788787786564444455444444445566556666665555                                                      271
270   33334344445444555567877878776664444554444444556655666666 55555                                                        270
269   334344443334445456787788666567645544444444556655666666555555                                                          269
268   3333444433344444444477777766984444444445566556666665555555 5                                                          268
267   333433334444344465565555555465444444556655666666555555555                                                             267
266   34333333344345655555555555447554445556556666666555555555 44                                                           266
265   43333333334555655555555666666776455555666666655555555 4444                                                            265
264   333333333455665555555566666667756655666666655555555 44444                                                             264
263   33333333455565555555566666667785656666666555555544444444                                                              263
262   3333333455565555555566666667788666556555555554444444 44                                                               262
261   3333333455565555555566666667886755655555555444444444 4                                                                261
260   33333355565555555556666667788675555566554444444444 4                                                                  260
259   33333455565555555566666667787675555445554444444444                                                                    259
258   Legend   333345556555555556666666778775555554454344444444 4                                                           258
257   33345556555555556666666787675555554454433333333                                                                       257
256   -Blocks marked 1 to 9 are the   3345556555555556666667787675555555444443333333                                        256
255   blocks of grade 0.1 to .9%Cu    345556555555566666667787655555555544444333333                                         255
254   respectively.                   34555655555556666666778675555555555444442223                                          254
253   34556555555556666666787675555555554444443322                                                                          253
252   -Blocks marked R, T and S are the   33445544554566666677876755555554444444 4332                                       252
251   blocks with 1%Cu or more, 2%Cu or   345455545555666667R9R766554455444444 4322                                         251
250   more and 3%Cu or more respectively.  3344554445556666 79RR96665554444444 44332                                        250
249   3355445555666777R8996765554444444333                                                                                  249
248   345555555666777R8899996654444444 4332                                                                                 248
247   3345555556665798888999665444444 4433                                                                                  247
246   3345556666998888778R9655444444333                                                                                     246
245   33346566668988787777999554444 4333                                                                                    245
244   3335789978887877788R9555444333                                                                                        244
243   3334899777888777 7789R955443333                                                                                       243
242   33458777788877777789955 44432                                                                                         242
241   33458777788877777789R9544332                                                                                          241
240   3458777788877777778995 4433                                                                                           240
239                                                                                                                         239
      1203      1213      1223      1233      1243      1253      1263      1273      1283      1293      1303      1313
```

Fig. 9.6 Stope boundaries with mineral inventory - DTH stoping (walls straightened)

```
  1203      1213      1223      1233      1243      1253      1263      1273      1283      1293      1303      1313
   .         .         .         .         .         .         .         .         .         .         .         .
301                                                                                                                    301
300                6755555555555556575                         5  5687666           6   6                              300
299                6755555555555556575                         5  5687666           6   6                              299
298                6755555555555556575                         5  5687666           6   6              6               298
297                6755555555555556575                         5  5687666           6   6                65             297
296                6755555555555556575                         5  5687666           6            5    5   65             296
295                6755555555555556575                         5  5687666       78787  5             55   55             295
294                6755555555555556575                         5  5687 5    666 6  6R9               55   565            294
293                6755555555555556575                               5     9866666  5568      5 5 555     56             293
292                6755555555555556575                  666655              866666   5789       55555     65            292
291                67 5555555555556575           566767667675   5           9666666  57689      55555   5  6            291
290                56    55555555556575         667677788877776 5             876666  55689       5555    5 65           290
289                555555   555555657         678778877888778765              866666  55789       5555    5 56           289
288                    555555    555        55678778878888778765 5             666666  5788        5555    56            288
287                            5 55    6    555678788788888878765 5            R86666  55789       5555    55            287
286                              5          5556787788888887787655              96666  55789       5555    5 5           286
285                                          55678778878888778765               86666   5788       55555          66285
284 55 5555                                  55567878878888778765                6556   55789       5555     5555577284
283 55 5555                                  5556787788888887787 6                 6    5578R       5555566666677778 283
282 5555555                                  5556787788788887787655                 855   578R                         282
281 5555555                                   556778788788887787656             55   R9876678R                         281
280 5555555                                   5556787787888888777756           55555          98888                    280
279                                           5556787788788888778766            5555    55  5  5988                    279
278                                            55678778878888778765          55555     55       5 98                   278
277     Legend                                 55567877878788878755        55555      5        55989                   277
276                                            55567877888778877855      55555      5         555                      276
275     -Blocks marked 5 to 9 are the blocks    556787788787887785     55555      5         5555555                    275
274      of grade .5 to .9%Cu respectively.     55567878878788876   55555      5         5555565 5                     274
273                                             55567877888787876   555      5       55555556665                       273
272    -Blocks marked R are the blocks of       55567877887877876555     5       555555666666555                       272
271     grade 1%Cu or more.                      55567878878778656      55      555555666666555                        271
270                                              5556787787877666     5       55555566666655555                        270
269                                              5 567877886665676  5       5555556666665555                           269
268                                                    77777776698       555556666665555                               268
267                                                 55555555555 65       55556666665555                                267
266                                               565555555555   755   55556666665555                                  266
265                                             55565555555666666676 5555666666555                                     265
264                                             556655555556666667756655666666555                                      264
263                                            55565555555666666778565666666555                                        263
262                                            5556555555566666667788666 56555                                         262
261                                            555655555556666667886755655555                                          261
260                                           5556555555556666677876755555566                                          260
259                                           555655555556666667787675555   55                                         259
258                                           5556555555566666676877555555                                             258
257                                           5556555555566666678767555555  5                                          257
256                                          55555555555566666677876755555555                                          256
255                                          5555555555556666667687655555555                                           255
254                                          55565555555666666768675555555555                                          254
253                                            565555555666666678767555555555                                          253
252                                             55   55 5666667787675555555                                            252
251                                            5      5 56666799R76655  55                                             251
250                                              55    555666679RR96665                                                250
249                                             5     5556667779899676555                                              249
248                                                  5555666777R888999665                                              248
247                                                 5 5555665798888999665                                              247
246                                                  5556666988888778R9655                                             246
245                                                   65666898878777799955                                             245
244                                                   57899788877777788R9555                                           244
243                                                    7997778887777679R955                                            243
242                                                    58777788877777789955                                            242
241                                                    58777788877777789R95                                            241
240                                                    58777788877777777899                                            240
   .         .         .         .         .         .         .         .         .         .         .         .
  1203      1213      1223      1233      1243      1253      1263      1273      1283      1293      1303      1313
```

Fig. 9.7 Stope profile Cut and Fill stoping (excluding blocks below cutoff grade)

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    1203      1213      1223      1233      1243      1253      1263      1273      1283      1293      1303      1313
      .         .         .         .         .         .         .         .         .         .         .         .
301                                                                                                                       301
300                6755555555555555657543333443333333433333332244454456876663211132333364446                             300
299                6755555555555555657543333443333333433333332244454456876663211132433364446                             299
298                67555555555555556575433334433333333333333322444544568766632111324333644463222133444444346             298
297                675555555555555565754333344333333333333333224445445687666321113243336444632222334544444 5465          297
296                675555555555555565754333344333333333333333224445445687666321113243336444322222234544445 44465         296
295                6755555555555555657543333443333333333333332244453456876663211132787874453332222444444 5544455         295
294                67555555555555556575433334433333333333333322444534568745422222666464 46R93233233444444555544565       294
293                675555555555555565754333344333333333333333224443434334534222987666645557823233335354555444556         293
292                6755555555555555657543333443333333333333366665544434223332222866666455578933334333555554444465        292
291                6745555555555555657543333443343566767667676444543222233222296666666455768933343345555544445546        291
290                5654455555555555657543344334456676778888777764543222223322228766664555789333443345555 44445465        290
289                5555554555555555657444433444678778878888778765443322224333228766664555789333443345555 44445456        289
288                    5555554444555444333455567877887888877876545332322343333366666645557883334433445555 4444456        288
287                            555544464344555567878878888878765454333222444333R8666645557893334433455555 4444455         287
286                              54434444445555678778888888778765543332223443333966664555789333443345555 44444545        286
285                                          5567877887888877876544433322233333333866664457883334433455555 44444444444665 285
284 5555555444442332233333333333443444444444455556787887888877876545433222222233333655644557893334333455554444444 55555776 284
283 5555555444442332233333333333443444444444455556787788888887787645433222222224333275444557 8R33344334555556666 6677778   283
282 55555554444423332333333333334434444444444555567877887888877876554322222222224432385544578R                            282
281 55555554444442332233333333333443444444444455567787887888877876564322222234444445544R9876678R                          281
280 555555544444423322333333333334434444444444555567877878888877775643222234444455555444344444 98888                      280
279                                           5556787788788887787664322244444555554444455445445988                        279
278                                            556787788788887787654322444455555444445544444445498                        278
277                                            55567877878888878755433555555555444445544444444555989                      277
276                                            5556787788888888778554355555555444445544444444 55555                       276
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274                                             55567878878788887644555555444445544444444 555555556555                    274
273                                             5556787788878888764455544444455444444445 5666556665                       273
272                                             5556787788787887655544444554444444 556665666666555                        272
271                                              55567878878778656444445544444444 556655666666555                         271
270                                              5556787787877666444455444444445566556666 6655555                         270
269                                              54567877886665676455444444445566556666665555                             269
268                                                    77777776698444444444556655666666 5555                              268
267                                                 65565555555546544444455665566666 65555                                267
266      Legend                                    56555555555554475544455565566666665555                                 266
265                                              5556555555556666667764555556666666555                                    265
264     -Blocks marked 1 to 9 are the            556655555555666666775665566666 6555                                      264
263      blocks of grade 0.1 to 0.9%Cu          5556555555566666667785656666 66555                                        263
262      respectively.                          555655555555666666778866655 6555                                          262
261                                             55565555555566666678867556 5555                                           261
260     -Blocks marked R are the blocks        5556555555556666667788675555566                                            260
259     of grade 1%Cu or more.                 555655555555666666778767555544555                                          259
258                                            55565555555566666677877555555                                              258
257                                            5556555555556666667876755555 5445                                          257
256                                           55565555555556666677876755555 55                                            256
255                                           555655555555666666778765555555 5                                            255
254                                           55565555555566666677867555555555                                            254
253                                             5655555555666666787675555555 5                                            253
252                                              5544554566666778767555555 5                                              252
251                                             545554555566667R9R766554455                                               251
250                                               5544455566667 9RR96665                                                  250
249                                              554455556667 77R899676555                                                249
248                                                 5555666777R889999665                                                  248
247                                                555556665798888999665                                                  247
246                                                 5556666998888778R9655                                                 246
245                                                  65666898878777799955                                                 245
244                                                  57899788878777788R9555                                               244
243                                                   8997778887777789R955                                                243
242                                                   5877778887777778 9955                                               242
241                                                   5877778887777778 9R95                                               241
240                                                   5877778887777777899                                                 240
239                                                                                                                       239
      .         .         .         .         .         .         .         .         .         .         .         .
    1203      1213      1223      1233      1243      1253      1263      1273      1283      1293      1303      1313
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Fig. 9.8 Stope boundaries with mineral inventory - Cut and Fill stoping (including internal waste)

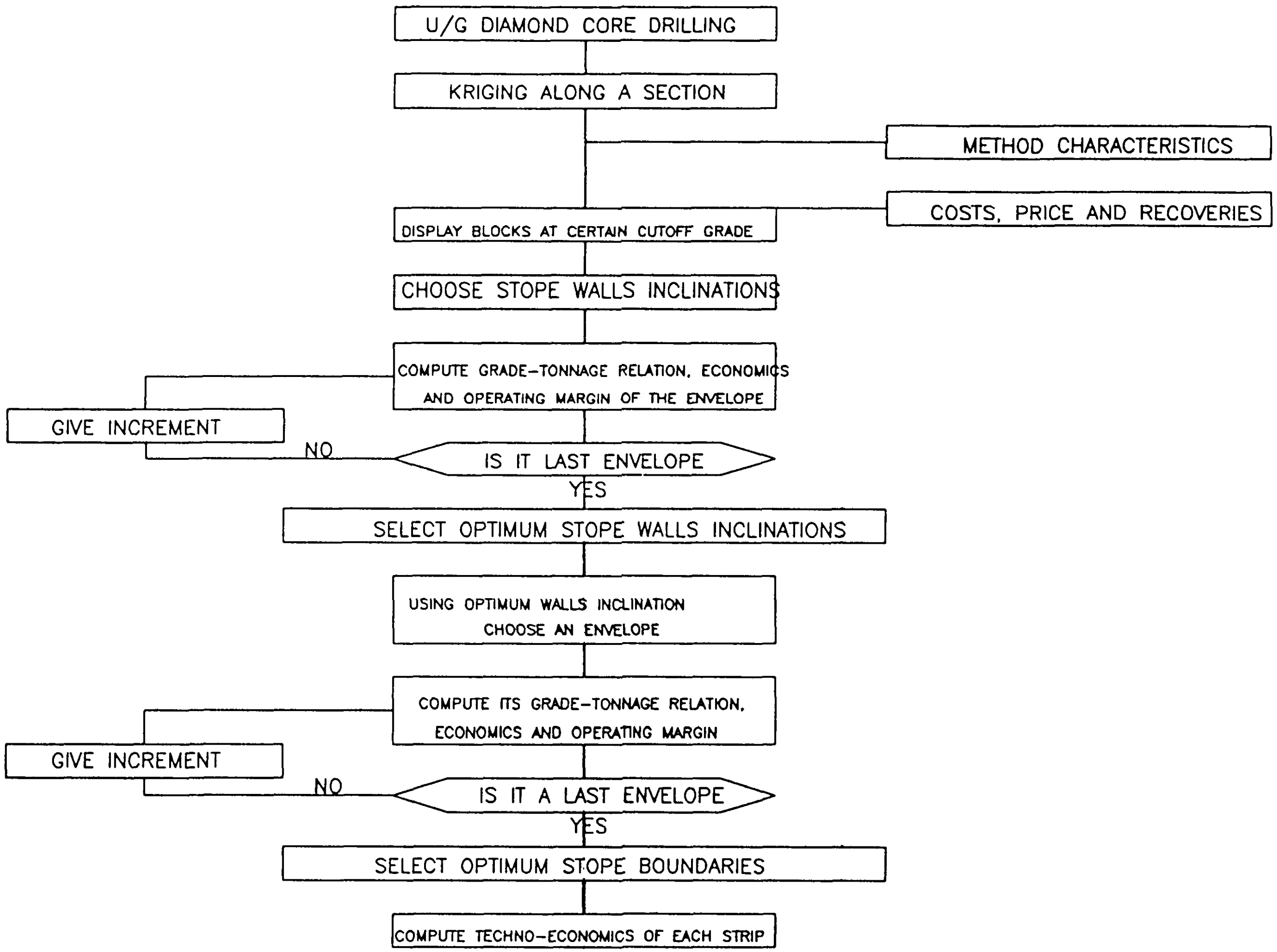


Fig. 9.9 FLOW DIAGRAM FOR THE SELECTION OF OPTIMUM STOPE BOUNDARIES

9.44 Stope boundaries - Cut and fill stoping:

For cut and fill stoping better selectivity can be achieved. Figure 9.7 shows the display of mineral blocks which have grade values above the cut and fill cutoff grade. The sidewalls have been located by moving block by block in from the extremities of the mineralisation until a block at (or above) the cutoff grade is encountered. It can be seen that there are areas where subgrade material would be taken if the outer profile were adopted (see figure 9.8). Cases where the inclusion of a large number of blocks of subgrade material is caused by a few above cutoff grade blocks need special attention. One method would be to relocate the extremity.
The main features of the methodology developed have been illustrated in a flow diagram shown in fig. 9.9.

9.5 Observations

Table 9.6 summarises the results obtained with the different methods for the same mineralised section. The cut and fill method that excludes all sub-grade material is a bit unrealistic. It can be seen that the DTH long hole method would give the best operating margin.

Following the procedure outlined above, the economics for the other sections can be derived. The overall operating margin is then computed and used to calculate the cash flow and profitability. If the profitability does not meet corporate targets higher cutoff grades could be tested.

9.6 Conclusions

The methodology enables stoping boundaries to be defined and provides a basis to compare the economics of different mining methods.

The two dimensional approach was justified in this case by the assumed continuity of the mineralisation along strike direction and by the fairly high density of sampling in the cross sections.

Future work in this area could be the investigation for more elegant ways to demarcate the optimum stoping boundaries. This includes designing an algorithm which eliminates specifying wall angles manually.

A three dimensional approach, where it is not possible to assume continuity of mineralisation in third direction, would be a suitable area for further research.

Chapter 10

10. Computer Assisted Stope Design

10.1 Introduction

This chapter describes the use of computing facilities in order to design sublevel, DTH and cut and fill stopes.

10.2 Use of computers in mine planning - literature review

In the context of underground metal mining one or more of the following activities can be undertaken with the assistance of suitable computing facilities:

1. assimilation and display of base data
2. detailed modelling of the size, shape, quantity and quality of the mineral deposit
3. determination of feasible and efficient mineral exploitation limits.
4. calculation of a schedule for the deposit exploitation
5. design and display of exploitation sequence
6. calculation of resources consumed and production generated during the exploitation
7. economic evaluation of the deposit
8. ancillary activities such as - surveying, environmental control, blasthole pattern design etc.

In the nineteen symposiums held on the Application of Computers in Operational

Research in the Mineral Industry (APCOM) (Ramani, 1976. Johnson and Barnes, 1982. Weiss, 1980. O'Neil, 1979), it can be seen that considerable work has been undertaken in some of the above areas, notably numbers 1, 2, 7 and 8. There are some companies and academic institutions offering software in these areas, and for open pit designs. Some simulation programs for Room and Pillar mining are also available (Haycocks, et al. 1984). However, the remaining areas are executed using subjective, non-repeatable manual methods. There is hardly any published literature available for computerised stope design purposes.

As described by Kim (1986) Computer-Aided Design (CAD) and Automated Mapping (AM) are becoming very popular in the engineering world. These terms relate to application of computers, specially computer graphic technology to mapping and design. CAD refers primarily to the design of components and structures in two or three dimensions. However, it can be applied to two or three dimensional modelling of mine workings, orebodies and geological structures.

AM/CAD systems use computers to store, manipulate, produce and update drawings and maps in two and three dimensions. Ideally the calculating capabilities of the computers are also used in conjunction with design and drawing generation, offering functions not available with conventional manual mapping and design processes.

Ferguson (1985) states "Whilst the computer will be increasingly used in the development of the conceptual activity, it is in the field of detailed mine planning that existing techniques and processes can have the most immediate impact. Historically the routine processes have been labour intensive, time consuming and somewhat prone to errors in the handling of large data sets and repetitive calculations. Performing this part of the design process with the aid of computers should therefore be undertaken to reduce overall duration of project work and the occurrence of arithmetical errors. Further, such an approach would permit the engineer to spend more time on the intellectual element of mine planning and should ultimately lead to an improved design".

Again Clews (1985) describing a heuristic planning system justifies the the use of computers. He wrote "At the moment, long time periods by manual design of a mine precludes the consideration of more than one or two alternatives for deposit exploitation. This often tends to a non-optimal design being used for a mining operation. It may also result into lower operating efficiencies, high production costs and wasteful exploitation of a natural resource. A more systematic design process would greatly reduce these problems".

Although recent advances in computer graphics have provided the hardware necessary for developing and evaluating mine plans rapidly; the software that will allow the mining engineer to make use of the new technology is slow in developing (Chatterjee, et al. 1986).

10.3 Objective

The ultimate objective of mine design is to determine the development and utilization of mining resources, in financial terms, for one or more specific alternatives. However, part of the process is the production of mining plans/designs, to outline of the amount of work and costs involved as well as predict the revenues therefrom.

Defining stope boundaries, using a computer, was described in chapter 9. Using the stope boundaries so defined, or even the hypothetical one, algorithms have been prepared to design the layouts of the following stoping methods:

1. Sublevel stoping (longitudinal)
2. Sublevel stoping (transverse)
3. DTH stoping
4. Flat back cut and fill stoping with posts and pillars.

An additional feature that has been incorporated with these algorithms, is the computation of the economics of a stope. This aspect is dealt with in the following chapter.

10.4 Stope design

Program development:

The algorithms have been written in Fortran 77 and the graphics library subroutines available at Imperial College Computer Centre (ICCC). The graphics programs have been written in two distinct phases:

1- Creating and saving on permanent files the arrays of (x,y) values/co-ordinates of the different workings that constitute a stope layout.

2- Using these files as input to the graphic routines that produce stope layouts in the form of plans and sections.

This method of developing a program saves time and resources and can greatly ease debugging.

A Tektronix terminal 4014 available at Imperial College (IC) was used for the production of interactive graphical output. This terminal can display output at over 200 lines per second. Thus, once a graphics program has generated a plot file, the picture can be displayed almost instantaneously on the screen. A Benson 1645 Drum Plotter was used for hard copy production on A4 - A0 size papers and Calcomp 1670 Microfilm plotter for the production of 35 mm microfilms.

10.42 Common features - stope design algorithms

The common features that have been incorporated in an algorithm to design a stope are:

1. The provision of stope workings in order to provide:
   - -access to man, material, equipment and air current into the stope
   - -arrangement for the rock fragmentation
   - -initial free face for the rock fragmentation

-extraction of the blasted muck and
-connection to the main levels.

2. The dimensions, position and orientation of stope workings:

The dimensions of the stope are selected according to the strata conditions and the equipment to be deployed to perform the basic operations of rock fragmentation and mucking. The orebody profile and mining sequence decide their orientation. Sometimes equipment movement also dictate this aspect.

A subroutine can be incorporated to compute the dimensions of the various stope workings on the basis of the criteria discussed above. They can alternatively be specified on the basis of the existing practices and experience.

Since these designs are based on the working conditions at the KCC mines, the dimensions of the stope workings and their orientation follow the current practice together with experience of the author. The shape of all stope workings was assumed to be rectangular.

3. Stope dimensions:

Stope dimensions can be controlled by strata conditions and size of the equipment deployed to mine the ore. Therefore, stope dimensions should be based on the existing practices that are known to be safe, such as, those at the KCC mines. Alternatively geotechnical investigations are necessary to establish the limits of stoping excavations. This aspect of stope design has not been examined in this research project. It is an area where further research could be carried out in the future.

10.5 Input data:

The input data for a stope design comprise two parameters:
1. Model parameters
2. Design parameters

The model parameters describe the deposit in terms of its position at four corner points, at a specified horizon (level), and its hangingwall and footwall dips. If orebody configuration at the four corners of a stope differs then the position of these corners at the immediate upper horizon (the tramming level) is defined.

The design parameters include the design specifications of a stope, the stope dimensions, equipment dimensions, dimensions of the various stope workings and their orientation constitute the design parameters.

Stope reserves and grade are model parameters, whereas operational costs, process recoveries and product price are design parameters. These parameters have been included in the stope design models and are discussed in the next chapter.

The model parameters remain constant but the design parameters can be varied from run to run. The advantage of separation of model and design parameters is that changes to design parameters can be made easily and cheaply.

10.6 Algorithms -Stope designs

The listing of the programs are not included in this thesis. However, they will be handed over to the Department Library for future reference. The flow diagram and main features incorporated with each of the designs will be described.

10.61 Algorithm - Longitudinal sublevel stopes.

The main features of the algorithms for both longitudinal and transverse sublevel stopes have been illustrated in the flow diagram shown in fig. 10.1. The essential features of the algorithm, for designing the longitudinal sublevel stopes, are summarised below:

This algorithm is essentially for the orebodies having average thickness in the range of 5 - 20m, but the stope length and height can be varied. Stope lengths up to 90m and heights up to 120m are not uncommon under suitable conditions.

Access to the stope is made through a service raise connecting the two main haulage levels. Provision is made for three drill drives, commonly known as bottom sublevel, top sublevel and crown sublevel. The blastholes are drilled radially in the form of rings. The algorithm for designing such rings is described later. A slot raise, to be ultimately converted into a slot, is positioned at the wider end of the orebody of the stope. The blasted muck is collected at the extraction level, which is comprised of a trough, a number of draw points and an extraction drive. The length and orientation of the draw points vary depending on the dimensions of the equipment in use for stope mucking.

The algorithm can be used for the wider orebodies, up to 30m or more, with certain modifications. In this case, provisions for the double drill drives at each of the drilling horizons and double troughs and extraction drives at the extraction level, are necessary.

The input parameters (consisting of model and design parameters), shown in table 10.1, were used. The model parameters used are hypothetical. In fig. 10.2, the stope layout, at the extraction level, considering uniform thickness of the orebody, is shown. The stope layout at sub-horizons, the cross-sections along service raise and slot raise are also shown.
In fig. 10.3, the stope layout with varying thicknesses of orebody, is shown.

Input parameters	Dimensions	Remarks
Design parameters:		
stope height	60m	
stope length	60m	
stope workings:		
Trough drive width	4.0m	
Extraction drive width	5.0m	
Draw point width	4.0m	
Sublevel width	3.0m	
Slot x-cut width	4.0m	
Height of workings	3.0m	
Slot raise x-section	2.0m x 2.0m	
Service raise x-section	2.5m x 2.5m	
Equipment length	6.0m	
Turning angle	40.0	angle in degrees
Draw angle of muck	55.0	
Position of sublevels:		
Bottom sublevel	22.0m	height above reference
Top sublevel	40.0m	level
Crown sublevel	52.0m	
Upper level	60.0m	
Model parameters		considering 8 corner points at two levels
Orebody starting point-1	(3565,1260)	at lower horizon
Orebody starting point-2	(3565,1275)	
Orebody ending point-1	(3625,1260)	
Orebody ending point-2	(3625,1275)	
Orebody starting point-1	(3565,1225)	at upper horizon
Orebody starting point-2	(3565,1240)	
Orebody ending point-1	(3625,1225)	
Orebody ending point-2	(3625,1240)	
Reduced level of main level	240m	

Table 10.1 Input parameters - Longitudinal sublevel stoping (hypothetical orebody).

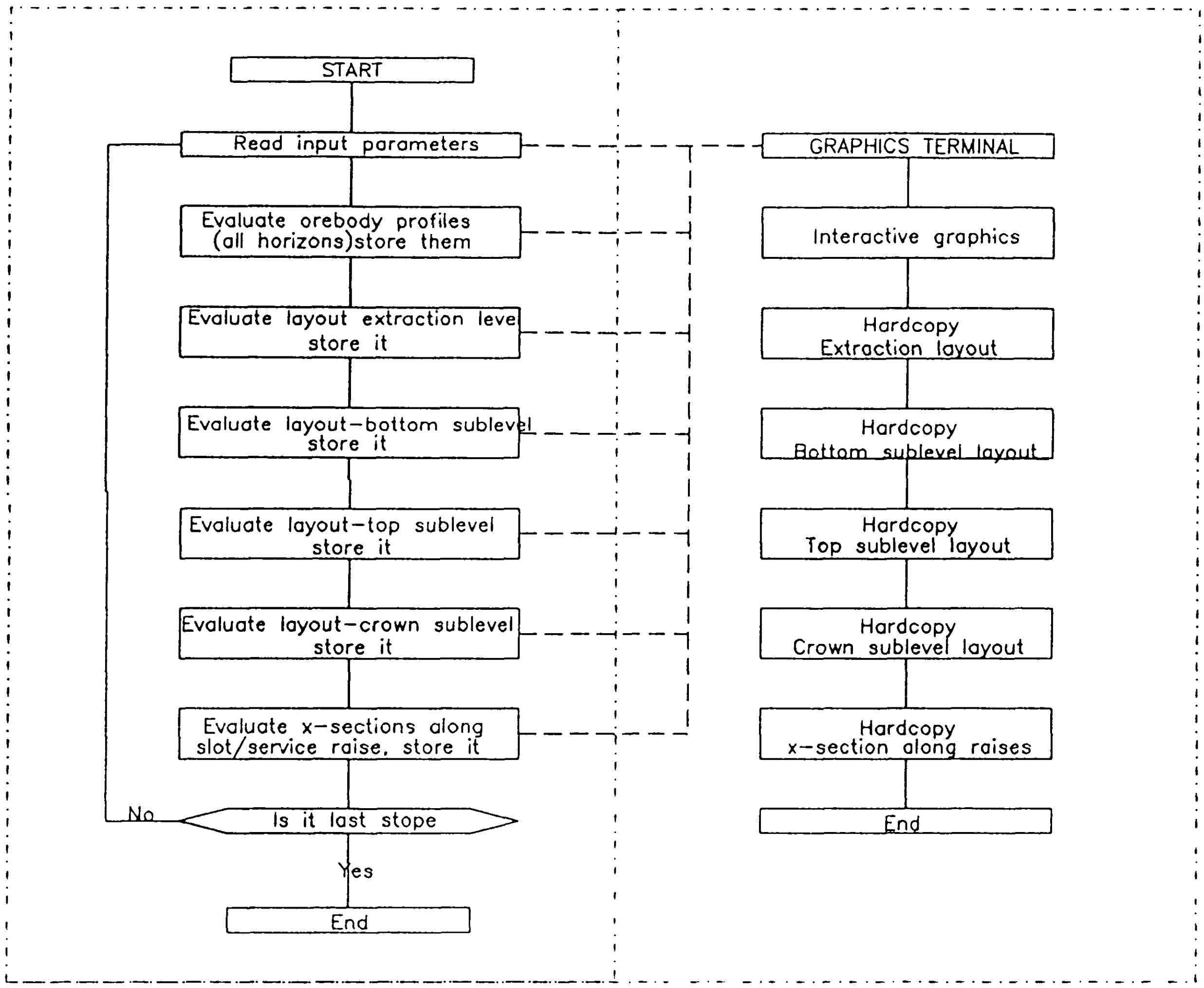


Fig. 10.1 Flow diagram stope design - Sublevel stoping.

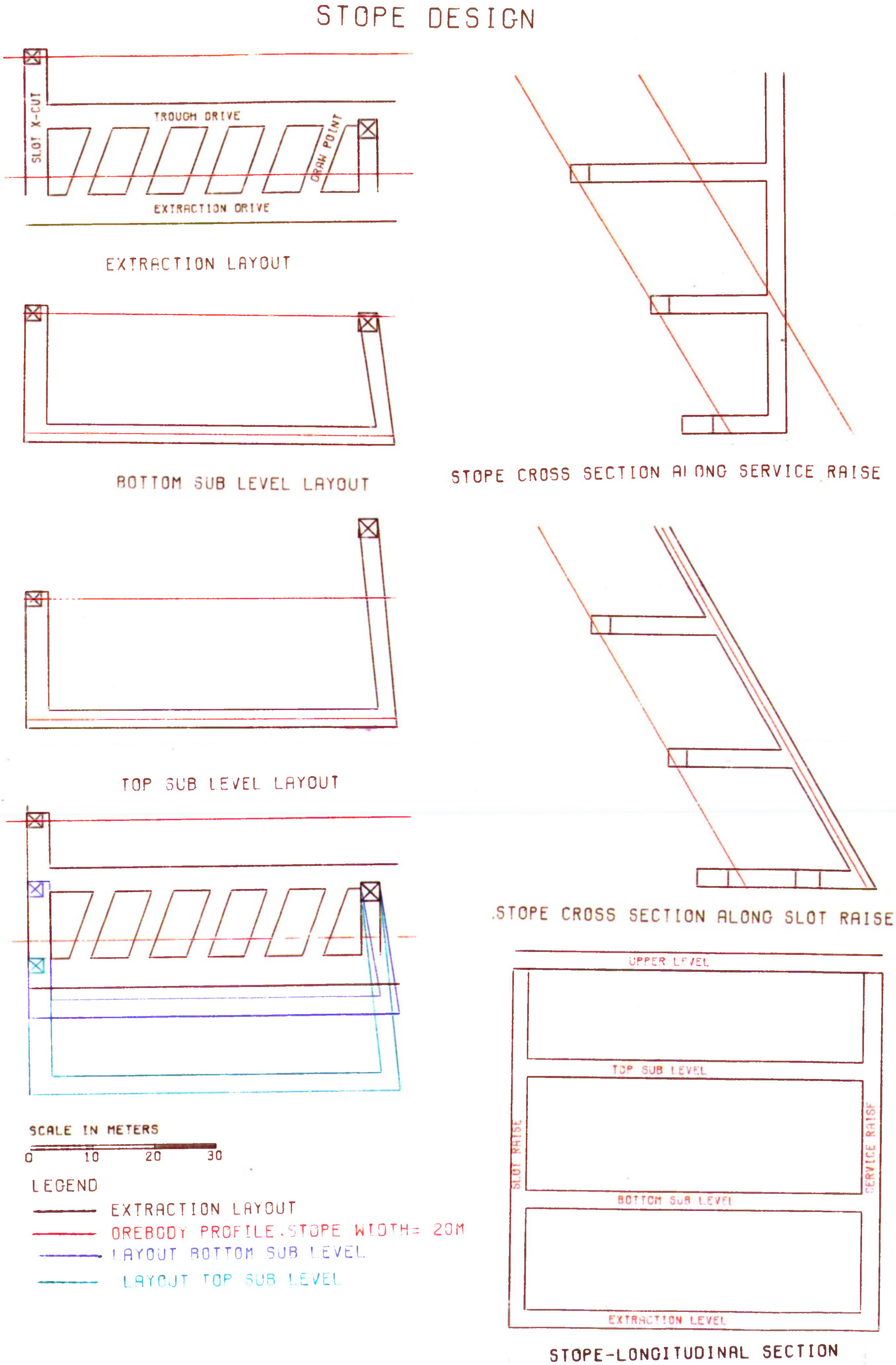


Fig. 10.2 Stope layout - Sublevel stoping (longitudinal) with uniform orebody thickness.

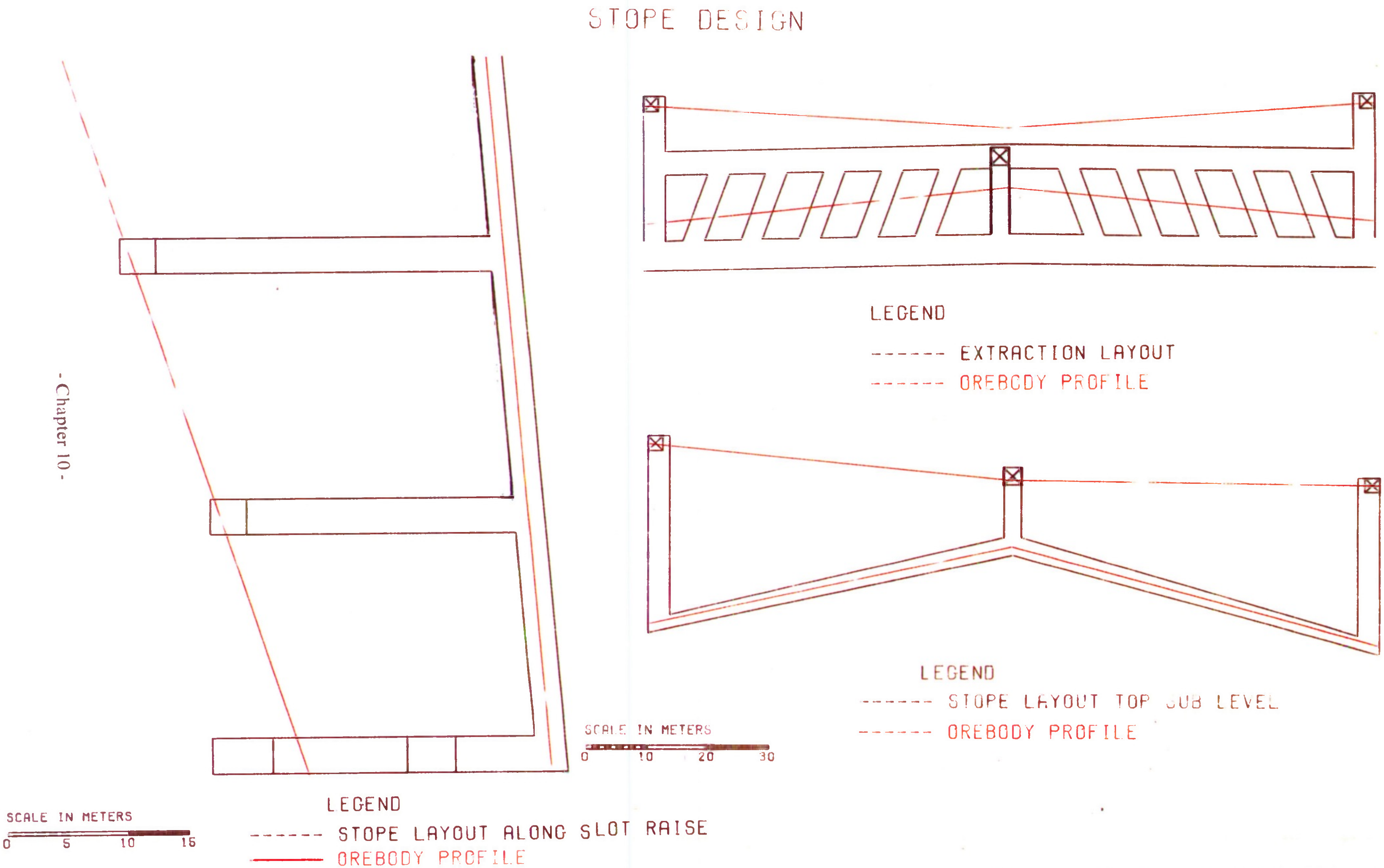


Fig. 10.3 Stope layout - Sublevel stoping (longitudinal) with varying orebody thickness.

10.62 Algorithm - Transverse sublevel stoping

The algorithm to design this method has the same features, with regard to the stope height and equipment deployment, as that of longitudinal sublevel stoping. The dimensions and position of the various drilling horizons and size of stope workings also remain the same. The following are additional features incorporated to design a transverse sublevel stope:

The width of orebody should exceed 20m. The stope length is normally taken to be 30m but it can be more under suitable conditions.

At the extraction level, double troughs and double extraction cross-cuts, are essential. Drilling cross-cuts at the various horizons should also be double.

The slot follows the extreme hangingwall of the orebody. Stoping operation commences from the extreme hangingwall towards the footwall.

The extraction cross-cuts at the haulage level and the drilling cross-cuts at the various horizons, should be connected to a common footwall drive at each of the horizons. The footwall drives should be positioned at a minimum distance of 10m from the extreme footwall (orebody) contact at each of the horizons.

The footwall drive at the main haulage level is usually driven, during the exploration phase of the level in the country rock. To represent the position of this drive in the stope design at the extraction level, use of a digitiser can be made, or its position can be assessed by the conventional methods.

In order to design a footwall drive serving a number of stopes, the following criteria were followed:

The drive should be as straight as possible or have turns which can be easily negotiated by the haulage system used. The configuration of the footwall drive, shown in the designs, is computed using the following procedure:

First, a linear regression is performed on the extreme footwall (orebody) points, for a battery of stopes in line (considering points at the stope's extremities along its length). The footwall drive is positioned at a minimum distance of 10-15m away from the regressed points.

The other method, that can be used to design this drive, is by the use of the 'despline' curve fitting routines (NAG library, ICCC) to the points considered to frame it (see fig. 10.12). The alternative which leads to a minimum quantity of overall development work should be preferred.

Design details (output) for the three drill drives and extraction layout are computed. The algorithm is capable of providing design for a number of stopes in a continuous orebody. The stope designs using the input data listed in table 10.2, are shown in figs. 10.4 to 10.7. The stope boundaries of six continuous stopes, evaluated in the previous chapter, have been used. In fig. 10.4, the extraction layout is shown. The provision of a service raise, common to two consecutive stopes has been made in this design. Stope layouts at the bottom sublevel, top sublevel and crown sublevel horizons have been shown in figs. 10.5 to 10.7, respectively.

Input parameters	Dimensions	Remarks
Design parameters:		
stope height	60m	
stope length	30m	
stope workings:		
Trough x-cut width	4.0m	
Extraction x-cut width	5.0m	
Draw point width	4.0m	
Sublevel x-cut width	3.0m	
Slot drive width	4.0m	
Footwall drive width	5.0m	
Height of workings	3.0m	
Slot raise x-section	2.5m	
Service raise x-section	2.5m	
Equipment length	6.0m	
Turning angle	40.0	angle in degrees
Draw angle of muck	55.0	
Position of sublevels:		
Bottom sublevel	22.0m	height above reference
Top sublevel	40.0m	level
Crown sublevel	52.0m	
Upper level	60.0m	
Model parameters		considering orebody profile at lower horizon
Orebody starting point-1	(3565,1256)	
Orebody starting point-2	(3565,1300)	
Orebody ending point-1	(3595,1256)	
Orebody ending point-2	(3595,1300)	
Dip at starting point	73.0	in degrees
Dip at ending point	66.0	in degrees
Reduced level of main level	240.0m	

Table 10.2 Input parameters - Transverse sublevel stoping

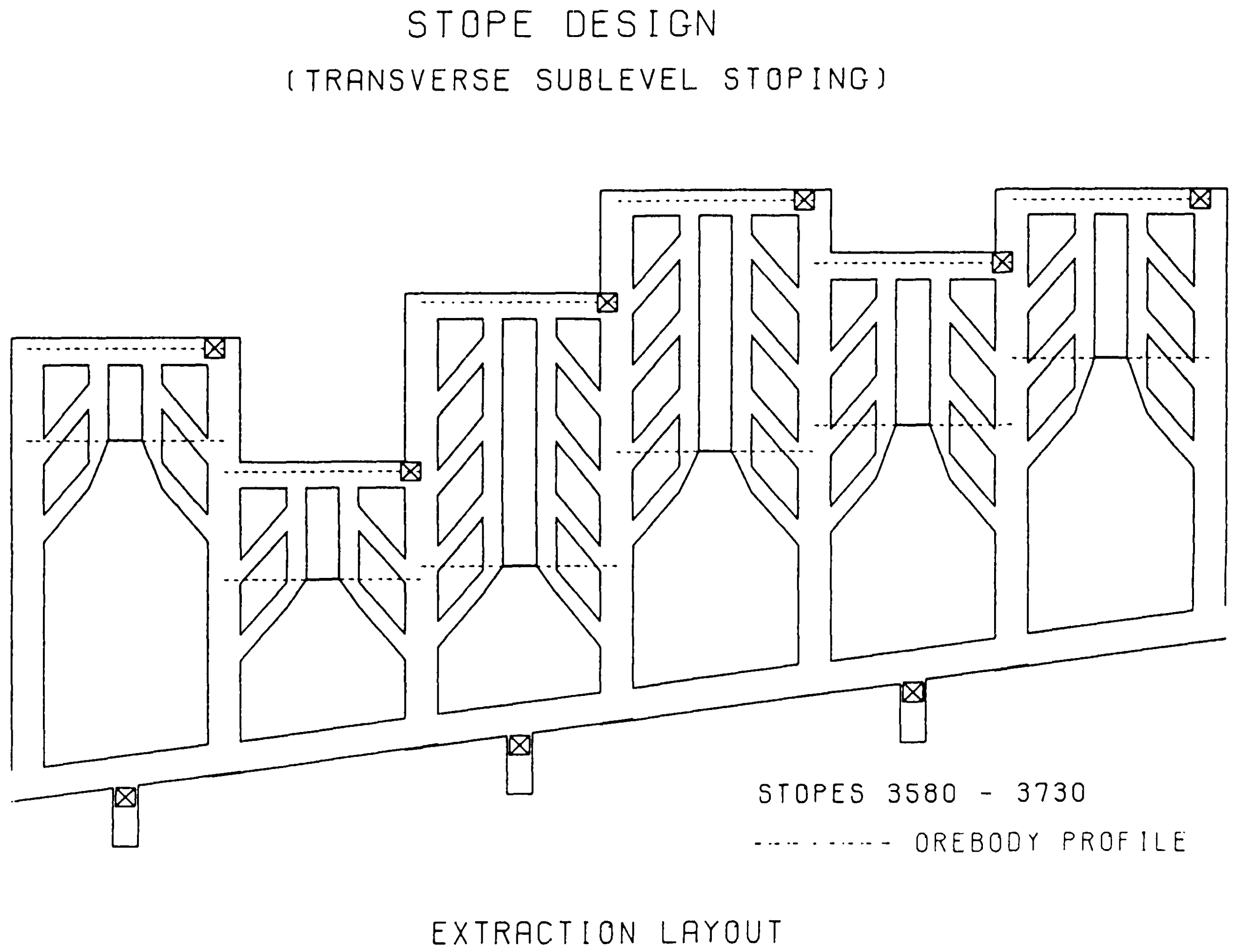
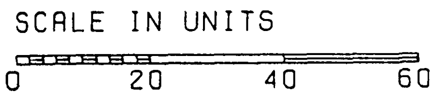


Fig. 10.4 Extraction layout for six continuous stopes - Transverse sublevel stoping.



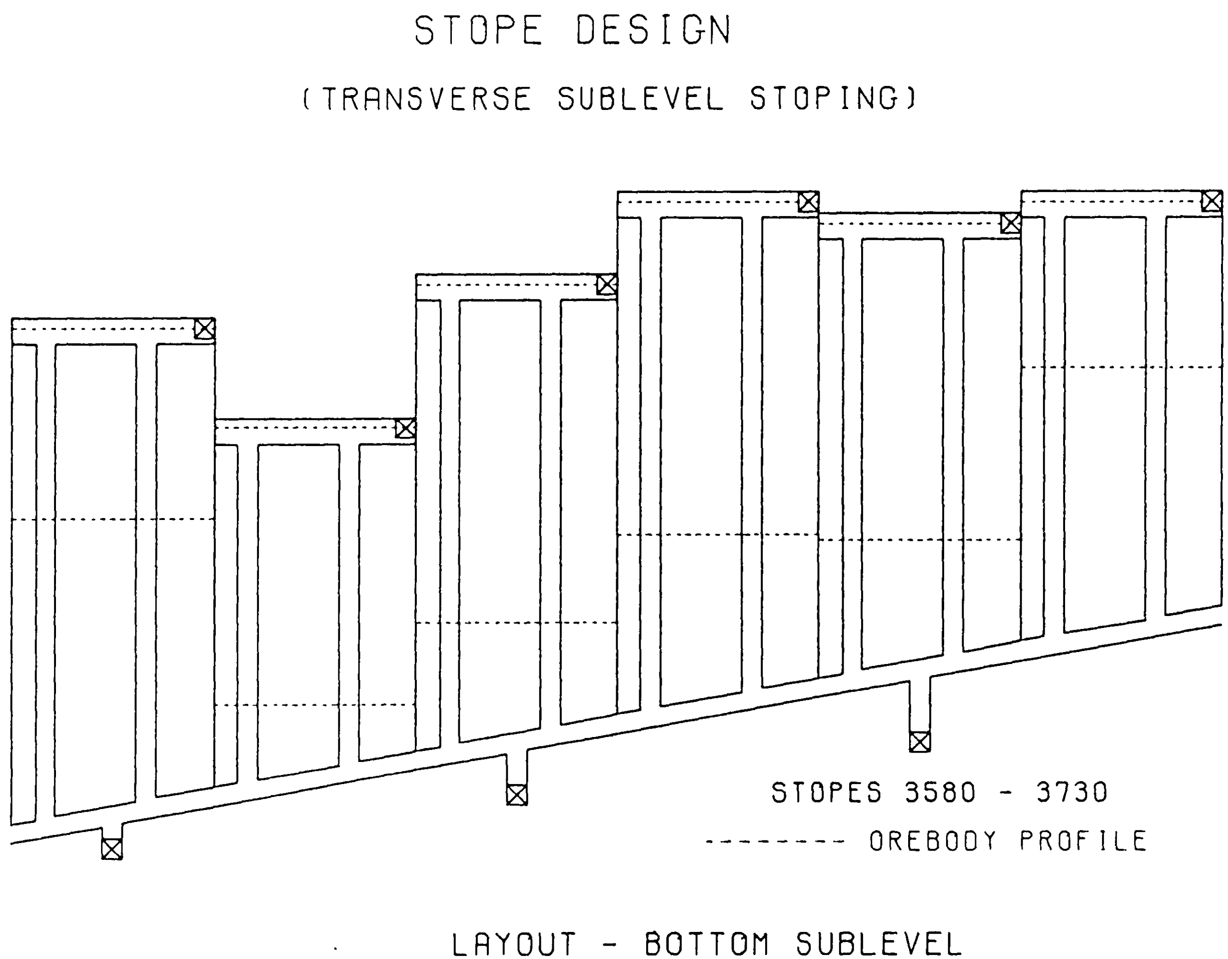
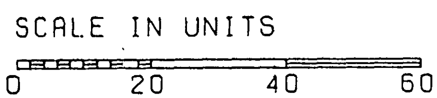


Fig. 10.5 Bottom sublevel layout for six continuous stopes - Transverse sublevel stoping.



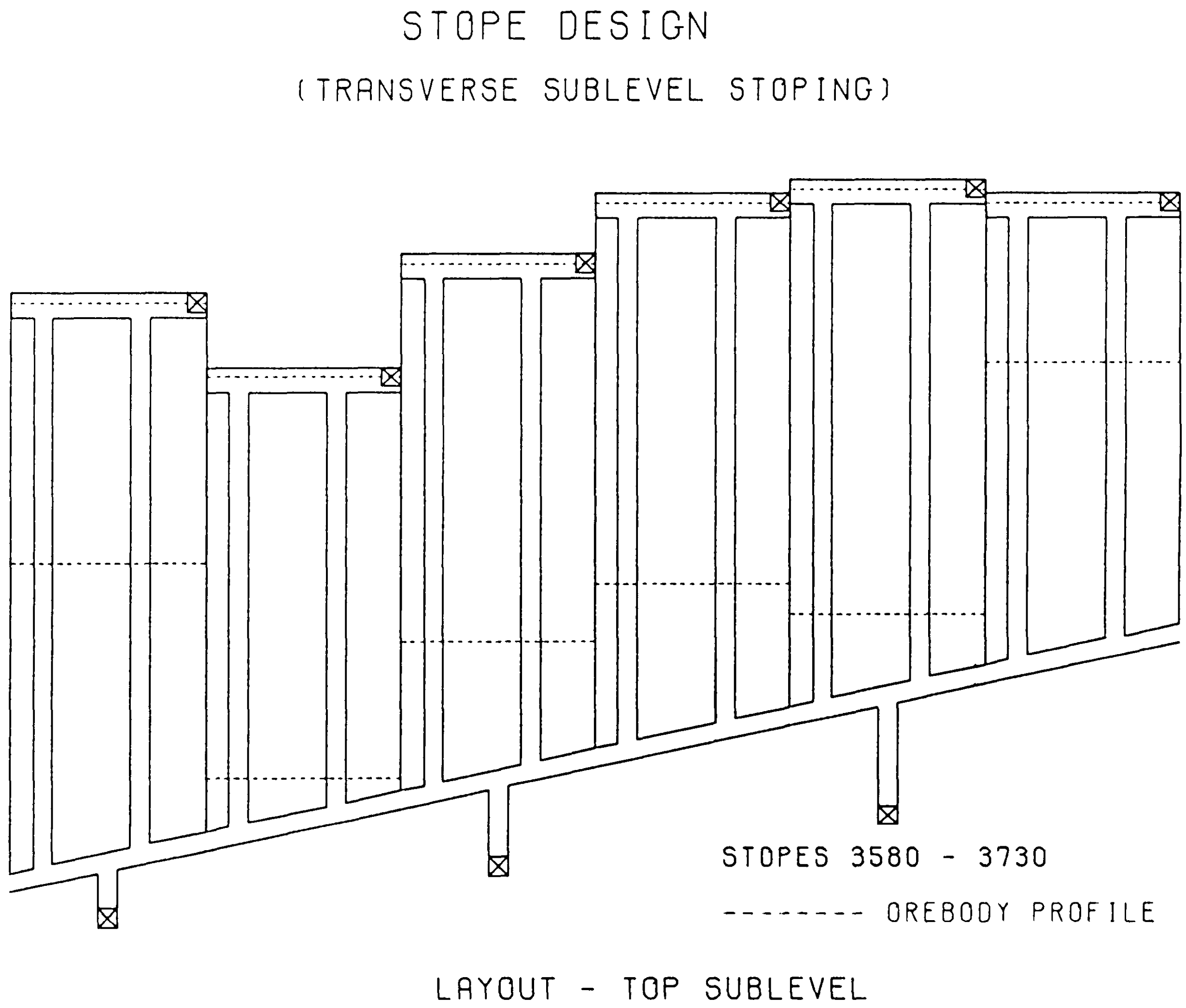
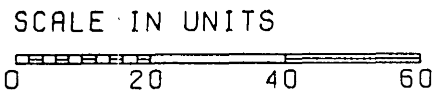


Fig. 10.6 Top sublevel layout for six continuous stopes - Transverse sublevel stoping.



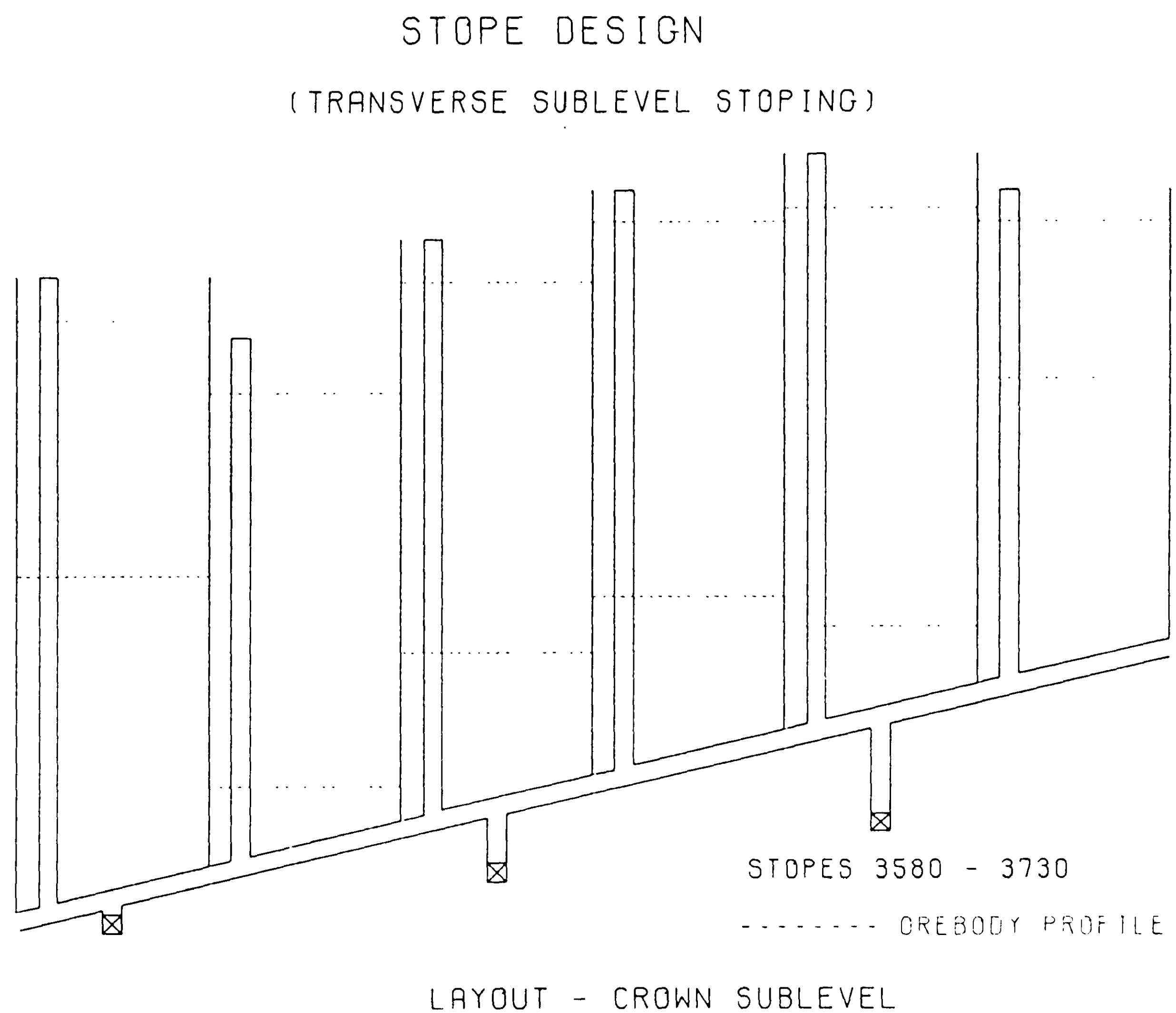
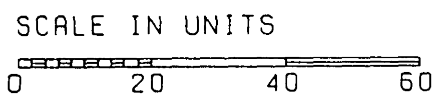


Fig. 10.7 Crown sublevel layout for six continuous stopes - Transverse sublevel stoping.



10.63 Algorithm - DTH stoping

The various steps involved to evaluate the DTH stope design have been illustrated by way of a flow diagram shown in fig. 10.8.

This algorithm is essentially for the wider orebodies with a stope length normally of 30m. Stope length can be more under suitable conditions.

This method is designed to use Down-the-hole (DTH) drills capable of drilling holes of 150-200mm dia. and length of +40m. With the use of such drills the blastholes can be drilled to a depth of 150m, depending upon ground conditions and capability of the machine to retrieve the steel and drill. The stope height can, therefore, be varied accordingly.

Drilling from only one horizon is planned. Access to the drilling horizon is through a common decline serving a number of stopes and commencing from the immediate upper haulage level.

Close drilling to create slot is essential, but stope drilling is designed with large hole spacing and burden. In this algorithm inclination of the holes is kept parallel to the hangingwall of the orebody.

The design of the extraction layout for stope mucking is the same as the one described for transverse sublevel stoping.

Design details (output) for the drill drive, extraction layout and cross-section along slot raise, are computed. The designs obtained from the input data shown in table 10.3, are shown in figs. 10.9 to 10.11.

In fig. 10.9, the extraction layout of six continuous stopes has been shown. This layout is similar to the one shown in fig. 10.4, except that provision of a service raise in this design is not made. This is due to the fact that access to the drilling level is by means of a decline commencing from immediate upper tramming level of the stope (not shown).

In fig. 10.12, the stope layout at the drilling horizon has been shown. The positions of slot and stope drilling holes have been shown using different symbols.

Fig. 10.13, presents cross-sections along the slot raises of the three stopes.

Input parameters	Dimensions	Remarks
Design parameters:		
stope height	60m	
stope length	30m	
Stope workings:		
Trough x-cut width	4.0m	
Extraction x-cut width	5.0m	
Draw point width	4.0m	
Drilling level x-cut width	3.5m	
Slot drive width	4.0m	
Footwall drive width	5.0m	
Height of workings	3.0m	at extraction level
Height of drilling level	3.5m	
Slot raise x-section	2m x 2m	
Equipment length	6.0m	
Turning angle	40.0	angle in degrees
Draw angle of muck	55.0	
Position of sublevels:		
Drilling level	51.0m	height above reference level
Upper level	60.0m	
Sill level height	9.0m	
Stope drilling:		
Slot drilling burden	3.0m	
Slot drilling spacing	3.0m	
Stope drilling burden	4.0m	
Stope drilling spacing	6.5m	
Model parameters		considering orebody profile at lower horizon
Orebody starting point-1	(3565,1256)	
Orebody starting point-2	(3565,1300)	
Orebody ending point-1	(3595,1256)	
Orebody ending point-2	(3595,1300)	
Dip at starting point	73.0	in degrees
Dip at ending point	66.0	in degrees
Reduced level main horizon	240.0m	

Table 10.3 Input parameters - DTH stoping

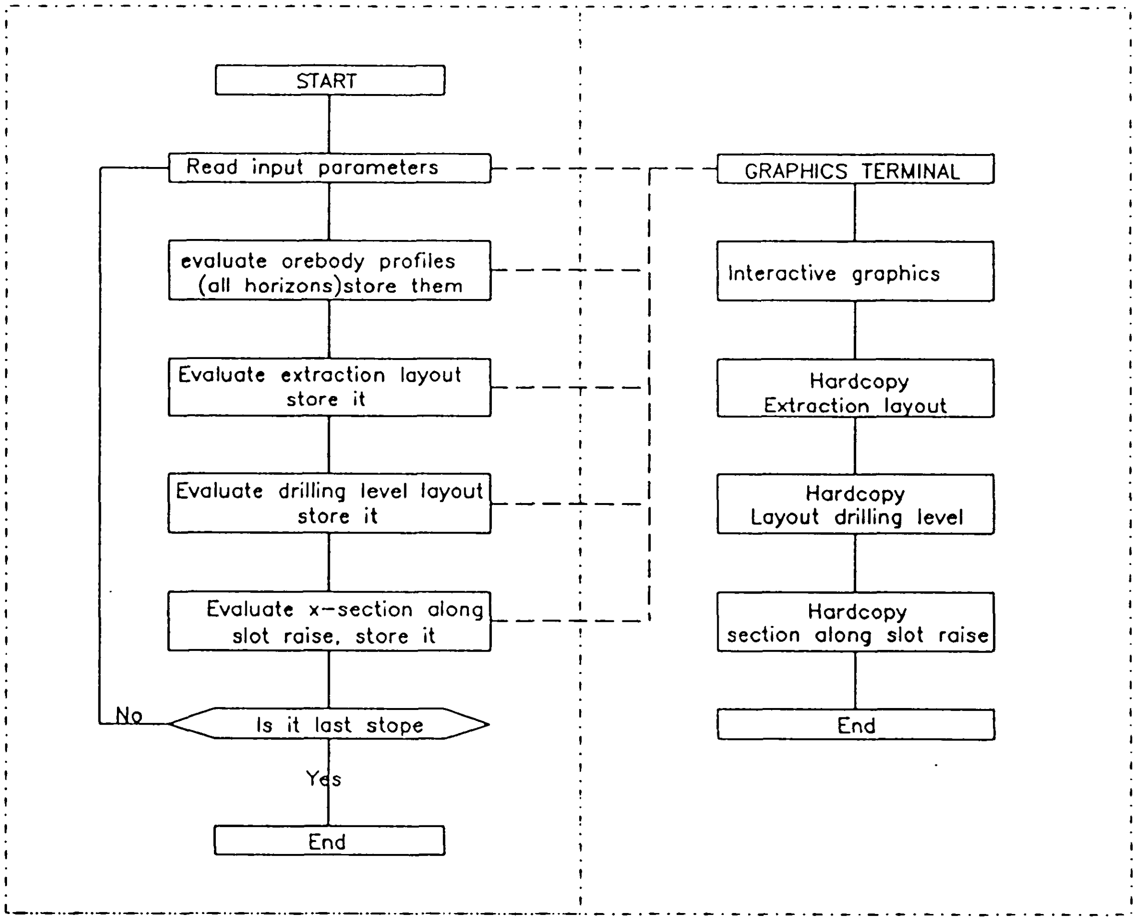


Fig. 10.8 Flow diagram stope design - DTH stoping.

STOPE DESIGN
(DTH STOPING)

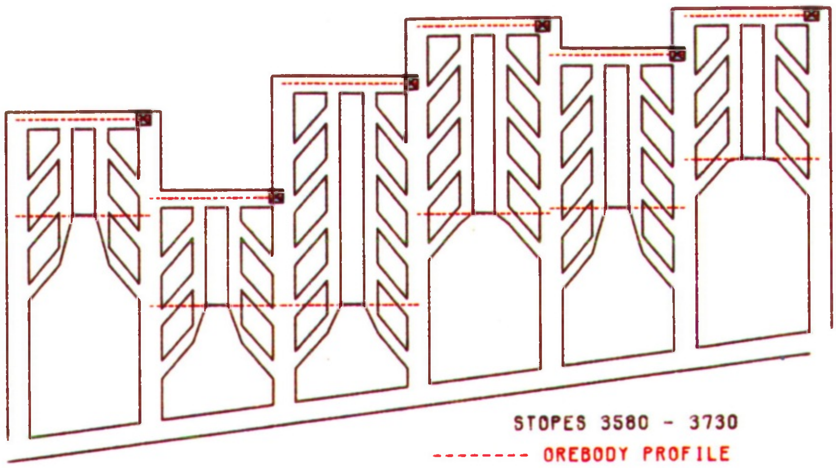


Fig. 10.9 EXTRACTION LAYOUT

STOPE DESIGN
(DTH STOPING)

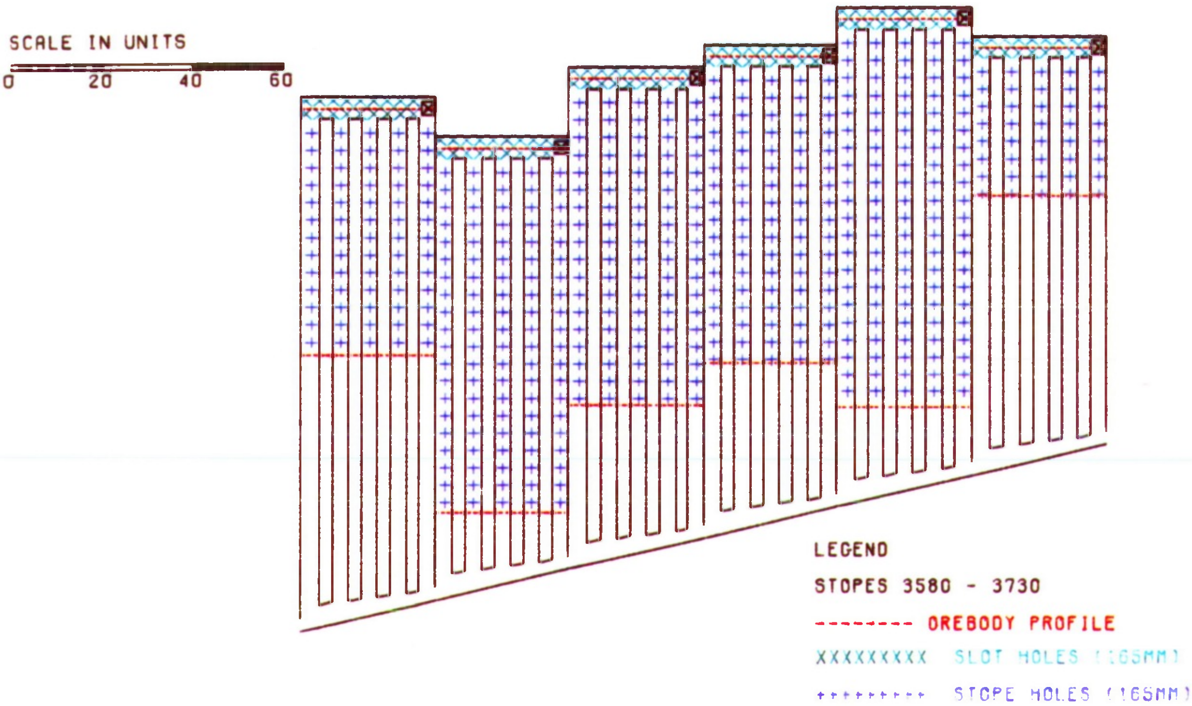


Fig. 10.10 LAYOUT - DTH DRILLING HORIZON

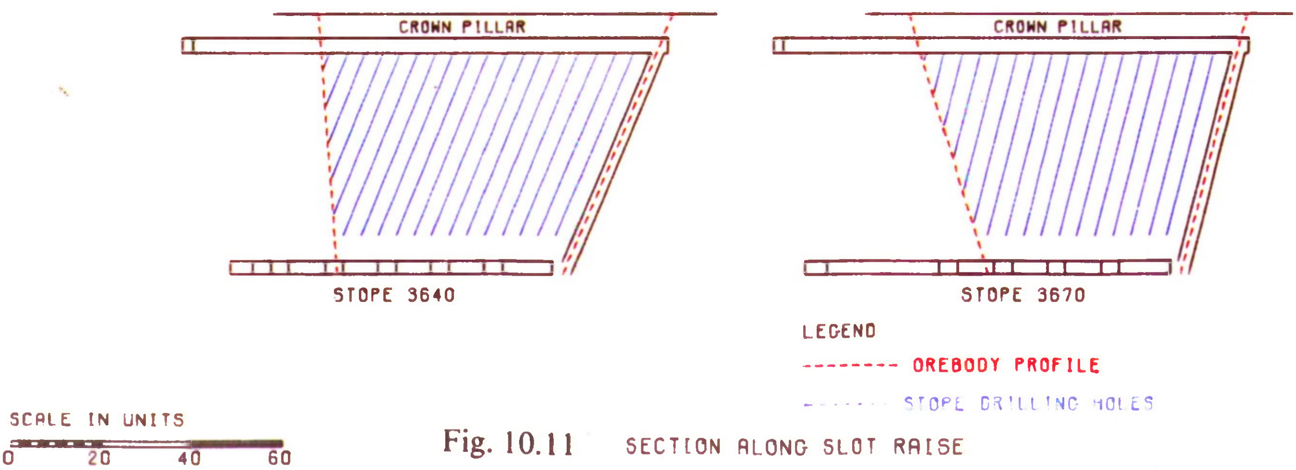


Fig. 10.11 SECTION ALONG SLOT RAISE

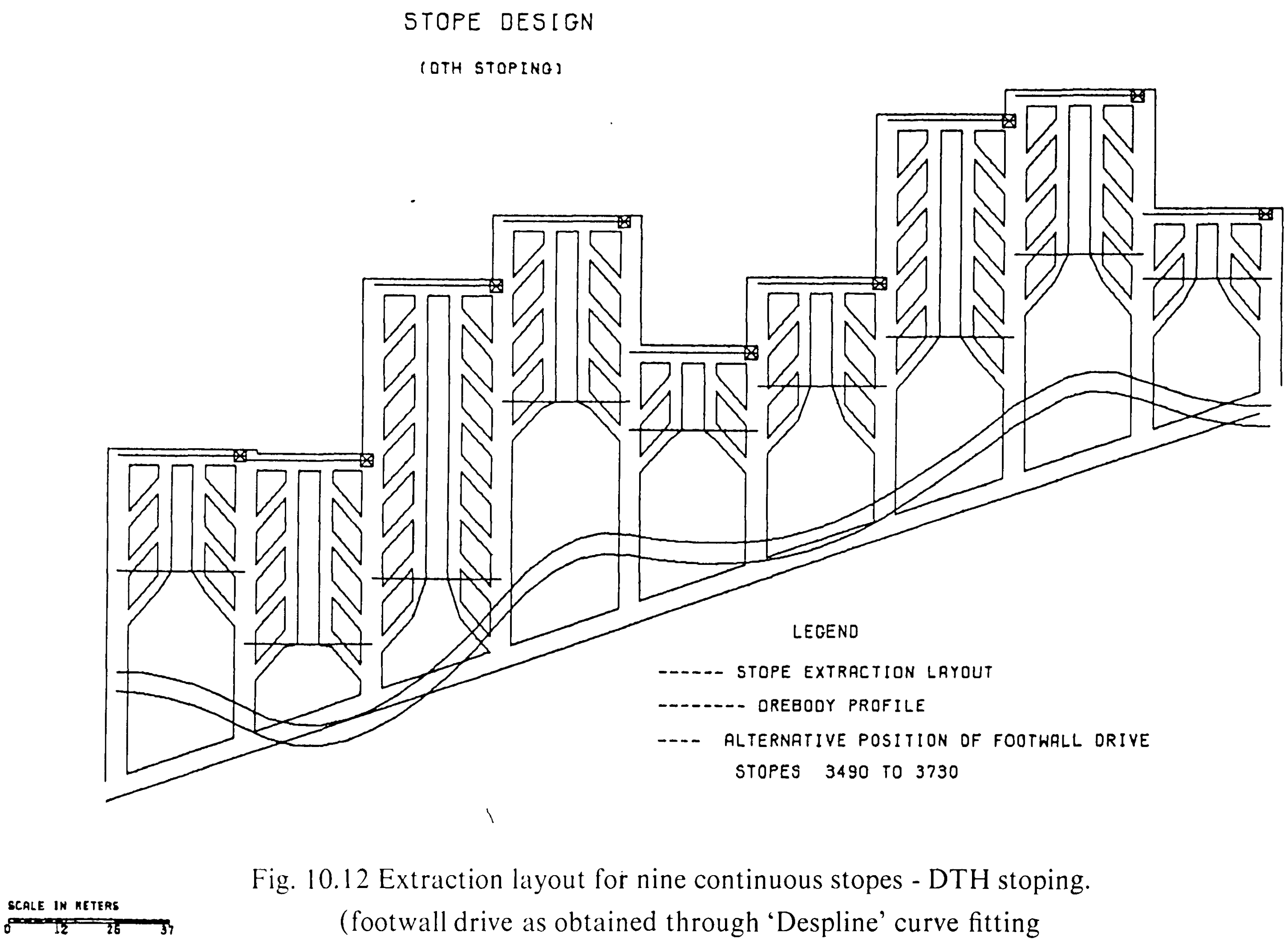


Fig. 10.12 Extraction layout for nine continuous stopes - DTH stoping. (footwall drive as obtained through 'Despline' curve fitting routines, shown).

10.64 Algorithm - Flat back Cut and Fill stopes with posts and pillars

An algorithm incorporating features explained below has been designed to obtain the design for this method. The flow diagram is outlined in fig. 10.13.

This method is suitable for wider orebodies with poor ground conditions necessitating artificial support. Due to excessive thickness of orebodies merely the fill is not adequate. Natural support of some kind is essential in addition. This is provided by vertical ore pillars spaced at regular interval within the stope boundaries, known as posts. At the extremities of a stope rib pillars are left. The rib pillars are recovered once the stope is worked out, but the post pillars are often not recoverable. Additional support in the form of cable bolting is sometimes required.

The stoping operation progresses from lower main level towards the upper level by successively mining ore slices usually of 3m height. Stope height up to 100 m or more is not uncommon.

The stope length varies, and the stope is often divided into three panels to obtain a continuous working cycle. In the first panel filling is in progress, whilst in the second preparatory work for filling and in the third production.

The filling material from the filling plant is conveyed to the working horizon through a network of pipeline.

The blasted muck is transferred to the stope ore passes. From there it is collected, either by trackless or tracked haulage system at the main level. A man pass is required for men and material access to the working horizon of the stope from the main level. Both the ore passes and man pass are built with cast iron tubings. Sometimes orebody profile dictates them to be in the rock mass also (as shown in some of the designs).

Stope drilling is undertaken using wagon drills. Ore fragmentation is achieved

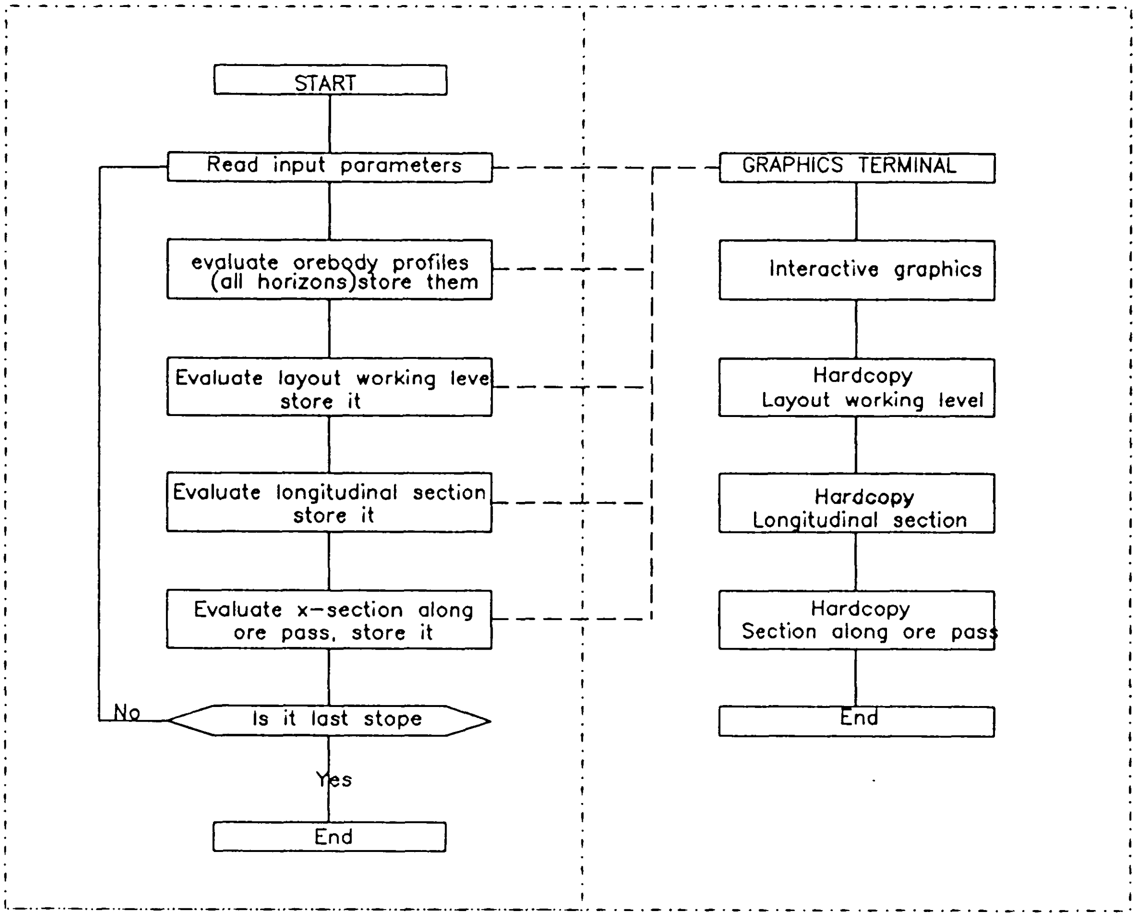


Fig. 10.13 Flow diagram stope design - Flat back cut and fill stope with posts and pillars.

by blasting these holes using conventional explosives. Stope mucking is by LHDs driven either by compressed air or diesel.

Design details for a working level in plan, cross-section along one of the ore passes and longitudinal section of the stope are computed. The algorithm is capable to design a number of stopes for a continuous orebody.

The design parameters, shown in table 10.4, similar to the one used at Dariba mines, India, where this method is used. The designs evaluated are shown in figs. 10.14 - 10.16. Fig. 10.15, is the longitudinal section of two adjacent stopes. The position of ore passes, man pass, service raise and rib pillars have been shown. A pattern of stope drilling, using drill jumbos, is also shown.

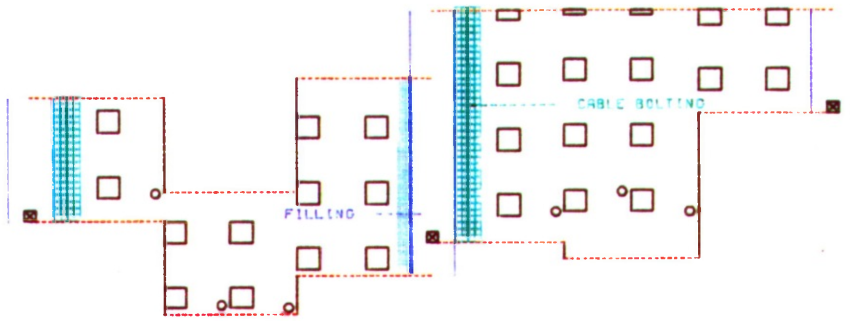
Fig. 10.14, is the stope plan at a working horizon. Along with the various workings, the post pillars have been shown. A pattern of cable bolting is also shown. The cross sections along the orepass of two stopes are shown in fig. 10.16.

Input parameters	Dimensions	Remarks
Design parameters:		
stope height	60m	
stope length	90m	
Panel length	30m	
Stope workings:		
Foot wall drive width	5.0m	
X-cut width	4.0m	
Height of workings	3.0m	at extraction level
Service raise x-section	2.5m x 2.5m	
Ore pass diameter	2.0m	
Man pass diameter	2.5m	
Slice height	3.0 m	at working level
Draw angle of muck	55.0	
Upper level height	60.0m	
Sill level height	9.0m	
Stope drilling:		
Stope drilling burden	1.0m	
Stope drilling spacing	1.2m	
Cable bolting spacing	1.5m	
Model parameters		considering orebody profile at lower horizon
Orebody starting point-1	(3565,1256)	
Orebody starting point-2	(3565,1300)	
Orebody ending point-1	(3595,1256)	
Orebody ending point-2	(3595,1300)	
Dip at starting point	73.0	in degrees
Dip at ending point	66.0	in degrees
Reduced level of main level	240.0m	

Table 10.4 Input parameters - Flat back cut and fill stoping with post and pillars.

STOPE DESIGN

(FLAT BACK CUT AND FILL WITH POSTS AND PILLARS)



LEGEND

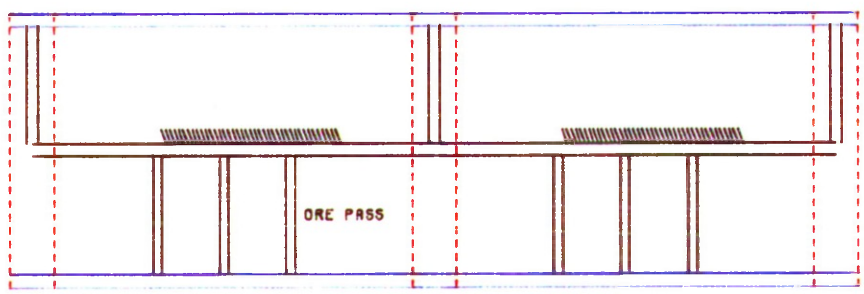
STOPES 3580 - 3730

-------- OREBODY PROFILE

Fig. 10.14 STOPE LAYOUT AT WORKING HORIZON

STOPE DESIGN

(FLAT BACK CUT AND FILL WITH POSTS AND PILLARS)



LEGEND

STOPES 3580 - 3730

-------- RIB PILLARS

//////// JUMBO DRILLING

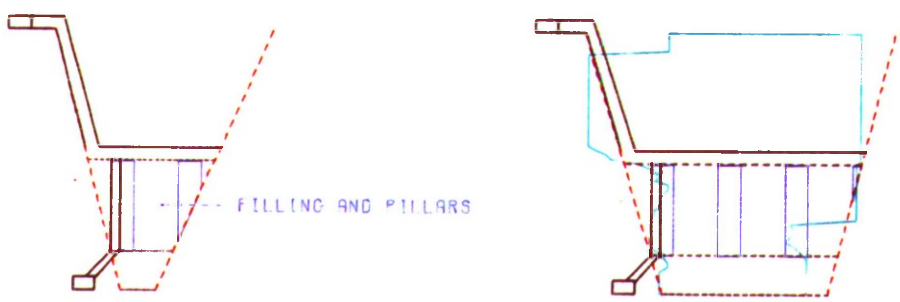
SCALE IN UNITS

0 20 40 60

Fig. 10.15 STOPES - LONGITUDINAL SECTION

STOPE DESIGN

(FLAT BACK CUT AND FILL STOPING WITH POSTS AND PILLARS)



LEGEND

- - - - - OREBODY PROFILE

----- CUTOFF GRADE PROFILE

Fig. 10.16 Cross-section along orepasses of two stopes - cut and fill stoping.

Using the algorithms described above, a number of alternative designs for each of the methods, can be quickly prepared. Thus the use of the computer in mine design can reduce many of the manual procedures and calculations associated with it. The output, in the form of quantity of work involved in each of the stope designs, can be used as one of the input parameters for the economic analysis (chapter 11).

10.7 Ring Design

The use of the computer for stope designing is demonstrated above. Its use could be extended to ancillary activities. Design of blasthole pattern for sublevel stoping is often a tedious and time consuming operation. A computer program developed to design a ring drilling pattern by Mining Research Centre, Canada (Hammond, 1968) has been used, with certain modifications.

The program can create a tabular output of hole lengths, dips and total metres for a ring drilling pattern. Orebody thickness, interval between upper and lower limits of the area covered by the ring measured along the dip of the orebody, toe-spacing and dip of the deposit forms the input data.

This program assumes a constant toe-spacing irrespective of hole length. This may not be a correct proposition in some conditions. The program was, therefore, modified. The following relation was used to calculate the toe-spacing:

$$TS = HL \times C1 + C2$$

Whereas

TS- toe-spacing in metres

HL- hole length in metres

C1- constant-1, this was given a value of 0.05

C2- constant-2, this was given a value of 0.75

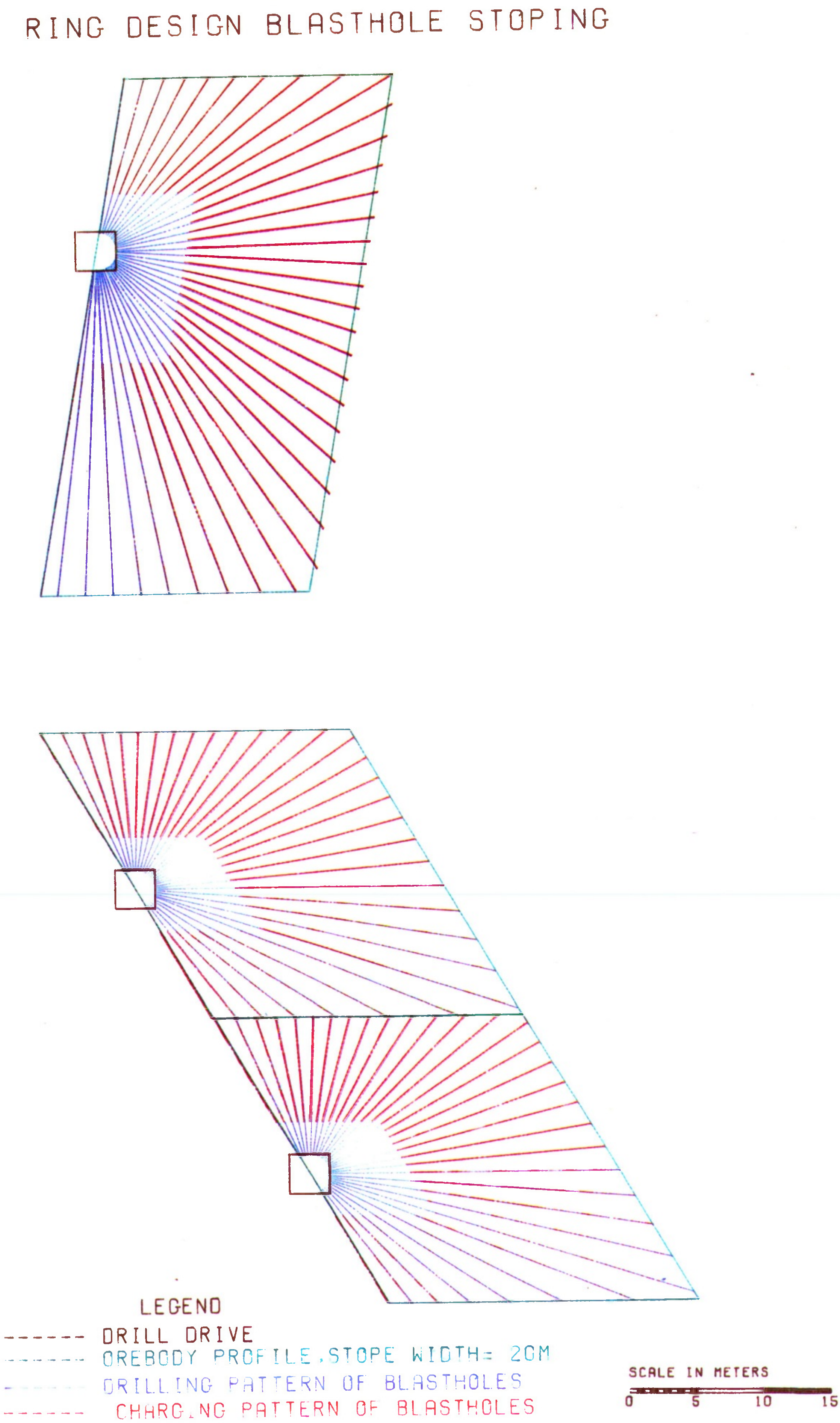


Fig. 10.17 Typical ring designs together with charging patterns.

By this relation toe-spacing for a hole of 1m length is 0.80 m and it increases as the hole length is increased. The author (1973) devised this formula for Khetri mine. The constants were derived through linear regression. However, their values can be altered to suit the rock characteristics.

Using the output from this program, a program to provide graphical output for the ring designs was prepared. Fig. 10.17 presents a typical ring design that was obtained using the algorithm described above. The drilling from two drives has been shown. It can be seen that the toe spacing is varying with the hole length.

The drill factor can be defined as tonnage produced per metre of drilling. In addition to the above aims, use of ring design can be made to estimate the drill factor to be used for the economical calculations.

10.8 Conclusions

It has been shown that computer-aided stope designs can automate many of the manual procedures and calculations involved in the mine planning. This can greatly reduce the the time and resources used in generating alternative stope layouts by hand.

The capability to compute the activities involved in each stope layout will provide more accurate predictions for future planning. This increased accuracy can eliminate or minimize the risks involved in investing large sums of money on the extension of existing mines or on a new mining venture.

Finally, these programs have been written using Fortran 77 with Tektronix terminals available at ICCC and are based on the basic data obtained from KCC mines. Minor modifications, however, may be necessary to input and output routines when used on other systems, and for other mines.

Chapter 11

11. Economic analysis

11.1 Introduction

This chapter outlines the economic analysis of the stope layouts for each extraction method, described in chapter 10. Hence, it provides a basis for selecting a stoping method which meets the desired objectives of an enterprise.

The system is tested by an economic analysis of the three mining methods modelled, for a small section at Khetri mine.

11.2 The economic analysis of a mineral deposit - literature review:

The development of a mineral deposit requires investigations into a number of aspects. Literature on the subject shows three major fields of study:

1. economic analysis,
2. financial analysis, and
3. political and environmental analysis.

Economic analysis is the yardstick by which proposed mining project and alternative investment opportunities are measured. Economic analysis is concerned with costs and potential profits, and can be applied at the various stages in the evaluation of a mineral deposit. There are two major parameters involved in economic analysis (Barnes, 1980, p. 128), these are:

1. known or knowable parameters
2. estimated economic parameters

Tonnage and grade of valuable mineral, metallurgical recoveries, production rate, royalty and tax structure are some the parameters which are either known or can be known.

Total investment costs, operating costs mining and processing plants, are the parameters which can be estimated. In addition the market price of the mineral or metal produced is also estimated. It is usually the most sensitive parameter in an economic analysis.

The normal objective of a financial analysis is to predict a cash flow over the period of investment and profitable production. If the return on the investment meets acceptable financial criteria project is said to be viable.

Johnson, et al. (1969) described some of the common methods used to evaluate a mineral resource. These are the average annual return, payback period, present worth, annual worth, future worth and discounted cash flow rate of return (DCFROR).

The payback and average annual return methods do not take into account the time value of money. A project can be evaluated in various ways, in terms of required minimum rate of return for two or more economic alternatives, by present worth (NPV) or future worth or annual worth calculations. Any one of which when correctly applied will give the right choice (Barnes, 1982, p.128).

NPV is one of the methods to compare the investment opportunities. Using different rates of production, an economic spectrum can be obtained. On the basis of which a decision can be taken. It provides a better means of comparison than operating margin (used in chapter 9) as it takes investment costs and time cost of money into consideration. Hence, in order to compare the profitabilities of different stoping methods the NPV approach has been adopted.

A widely used indicator is the DCFROR, in which actual rate of return can be determined by trial and error techniques. The DCFROR provides a more meaningful tool for analysing the mining venture than the average annual return or payback period. This is because it takes into account the time value of money. It is based on actual cash flow from initial capital investment to the end of the project and produces a common index of measurement. However in the case of an operating mine the initial investment is already a sunk cost. Therefore the NPV of future cash flows, at a suitable discount rate is a more appropriate economic evaluator.

Financial analysis has the reference to how, where, and at what cost the investment funds can be obtained to finance the proposed mine project.

Political and environmental analysis involve many other factors that may affect the making of a successful mine. These could include aspects of environmental impact legislation. In general they add a cost to the mine development (Barnes, 1980, p. 128).

11.3 Objective

For the purpose of this study it is assumed that the decision has already been taken to develop the mine, mill and other process plants. The primary development system of the mine has been so designed to give maximum flexibility in stoping system and layout and to permit changes if considered necessary.

The graphical output from the stope design models of transverse sublevel stoping, DTH stoping and flat back cut and fill stoping with posts and pillars have been illustrated in the preceding chapter. In addition, these models have been designed to compute the profitability in each case. Apart from this, an additional feature that has been incorporated with these models is the computation of equipment population together with their investment cost.

The objective of the study described in this chapter is to:

- predict the individual stoping activities and unit mining cost for a stoping method

- calculate profitability at a certain rate of mine production and to compare different mining methods on the basis of operating margins and present worth (NPV) calculations

- select a stoping method.

11.4 Input data

Details of input data for stope design purposes were discussed in the last chapter. Additional input data required for economic analysis will be discussed. The terms 'productivity', 'degree of mechanisation', 'design parameters' and 'model parameters' have been defined in the earlier chapters.

The in-situ reserves and grade have been considered as the model parameters, whereas the production rate, process recoveries, mine costs, process costs and metal prices as design parameters.

11.41 In-situ stope reserves and grade:

The mineral tonnage and average grade contained within the envelope selected to provide stope boundaries forms the in-situ ore reserves and grade. This approach takes account of the internal waste or subgrade material within a stope. If hypothetical stope boundaries are considered, the reserves and grade within these boundaries should be specified.

11.42 Mineable stope reserves:

In order to obtain the mineable reserves, the overall recovery of the in-situ reserves that is likely to be achieved during stoping, is considered. Percentage mineability depends on ground conditions, mining method and geometry of the orebody. No standard procedure is available to predict stope recoveries, but the cumulative percentage extraction with regard to reserves and grade based on experience of a working mine, should be used. In order to determine the mineable reserves for the sublevel and DTH stoping the percentage recovery data is based on the past performance of few years of KCC mines. For cut and fill stoping, considering the post pillars as not recoverable, the percentage extraction factor has been computed to assess the mineable reserves.

11.43 Rate of production and operating life:

The life of a stope, a section of the mine composed of number of regularly spaced stopes, a level or the entire mine, is assessed by specifying the rate of production and working days (or shifts) in a year. In this analysis the mine has been assumed to work for 300 days in a year and 3 shifts a day.

The rate of production can be varied to obtain different spectrums of mine economics.

11.44 Costs and Price

11.441 Operational costs - Mining:

These costs can be estimated. The procedure to estimate these costs is described in chapter 8. For a specific degree of mechanisation and productivity level, the unit operational costs are specified. In order to compute stope development costs the drilling, blasting, mucking and transportations costs per cubic metre of development work are used. Whereas drilling per metre, blasting, mucking and transportation costs per tonne of stoping tonnage are specified to compute stoping costs.

11.442 Fixed costs - Mining:

The costs that account for wages of indirect workers, mine services and all other costs which are not taken into account while considering the operational costs are grouped together and considered as a fixed cost. The fixed cost has been calculated for the year.

11.443 Process costs:

Both variable and fixed process costs per unit are specified as input parameters. The variable costs per unit, mining and process plants, have been used for cutoff grade calculations.

11.45 Metallurgical recoveries:

In order to assess the quantity of the finished product, the process recoveries (concentrator through wirebar or finished product) are specified. The process recovery data, as obtained through past performance of the mining complex has been used in this analysis.

11.46 Mineral/metal price:

Net metal price after accounting for metallic or gaseous by-products(if any), and deducting the excise duty, local taxes or royalty, selling and distribution expenses, is specified to assess revenues.

This analysis is for ore production of 5,000t/day. The costs and prices have been assumed to be constant throughout the life of the section considered. The cost and price data for the year 1985-86 of the mining complex have been used.

The annual fixed cost has been used to calculate the total mining cost per unit of ore produced. Using fixed and variable costs for mines and plants, the annual profitability has been evaluated. The profitability, so obtained, and the input interest rate provide the NPV of the section analysed. The NPV has been calculated at 10% p.a. discount.

11.5 Algorithm - stope design and economic analysis:

In fig. 11.1, the steps involved to arrive at the economic analysis through these models have been illustrated. In general, these algorithms have the following features:

They are capable of designing and performing economic analysis for one or more stopes for a continuous orebody. The stope design and economic analysis of each individual stope are first obtained. This is followed by the composite stope design layout and overall economics of the stopes under study.

The stope design provides the layout of different stope workings in the form of plans, sections or longitudinal sections, if needed. Based upon these stope design layouts, the computation of the work involved to develop and mine out the stope, proceeds.

Linear measurement in metres, of both horizontal and vertical developments, are first accounted. Using the specified size of each of the stope workings, the total volume of the horizontal and the vertical workings is computed. This volume is then deducted from the stope's total volume to provide stoping tonnage.

In sublevel stoping the stope drilling, blasting, mucking and transportation works involved to win the stoping tonnage are estimated.

Whereas in DTH stoping, in addition to above, the required amount of DTH drilling, is also computed.

In the case of cut and fill stoping, the works associated with rock fragmentation, mucking within the stope and at the main extraction level are evolved. Necessary filling and support works are also included.

Once all the works required to extract a stope are computed, using input operational costs, the total cost to mine out the stope is assessed. The operating cost per tonne of ore is also estimated.

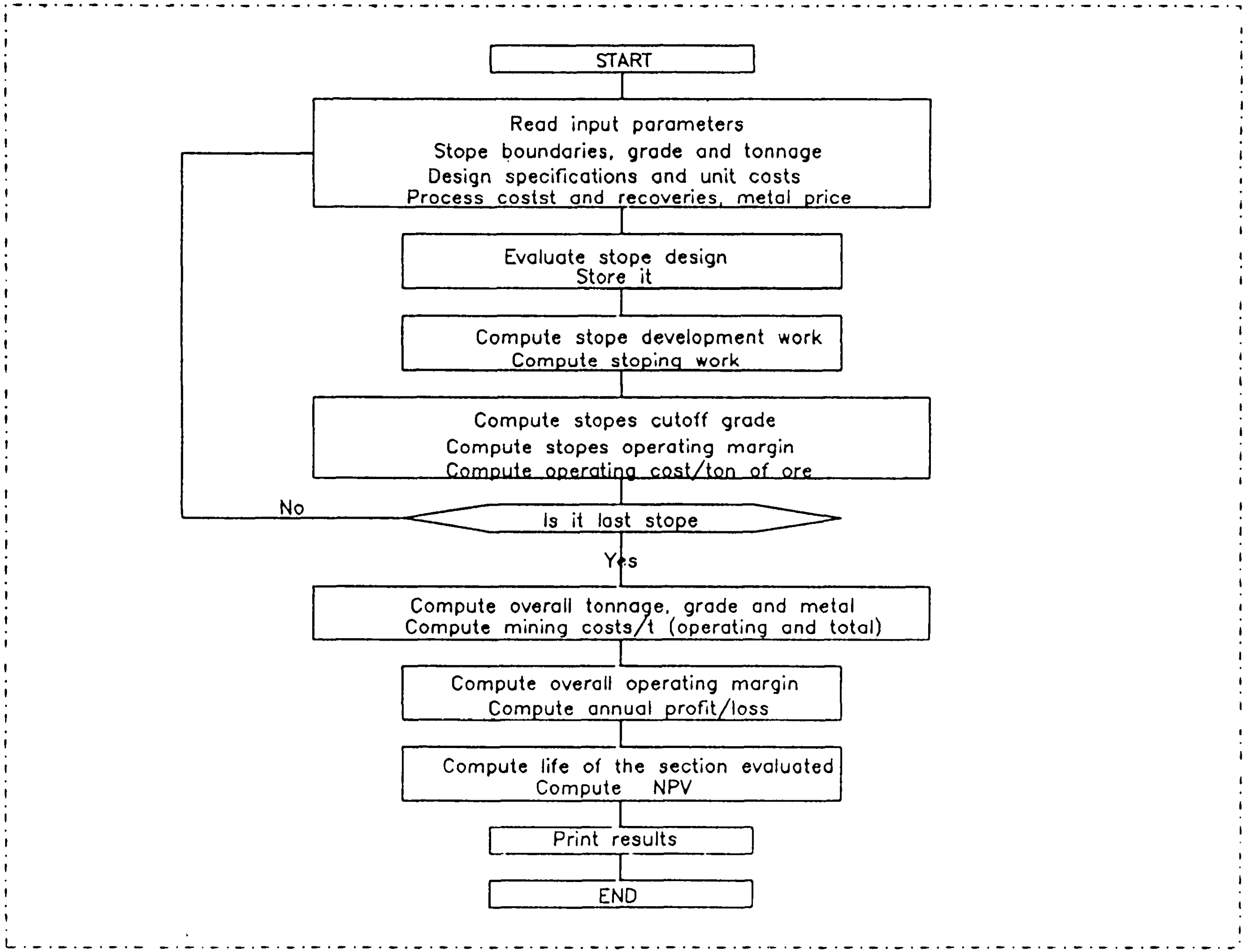


Fig. 11.1 General flow diagram for stope design and economic analysis - Sublevel, DTH and cut and fill stopings.

Using mine operating cost and the input process variable costs and recoveries, the calculations for cutoff grade follow.

The foregoing procedure is repeated till the analysis of the input data of the last stope is completed. Following this, the overall economics of the section, composed of number of stopes considered, is estimated. This includes the calculation of the overall tonnage (reserves), average grade, cutoff grade and operating cost/t of ore.

Using the annual rate of production the life of the section is determined. Calculations for total mining cost per tonne of ore and operating margin follow. The difference in the overall revenues and the costs for the year provides the annual profit.

Assuming the annual profit to be uniform throughout the life of the section (deposit), the net present value of the profit/loss at a specified rate of interest is ultimately assessed.

Using equipment performance norms and their costs, the equipment population (after allowing for the provision of standby equipment) and total capital investment required to acquire them can be computed. This computation is through a subroutine incorporated specifically for the purpose.

11.6 Economic analysis

For six continuous stopes the stope boundaries, grade and tonnage as obtained by the incremental analysis analysed. The input data related to stope design purposes were described in chapter 10. Additional data required for economic analysis are given in table 11.1. The input cost and price data related to mining operations have been described in chapter 8. The process costs are given in appendix 3.

STOPE NO.	Sublevel stoping		DTH stoping		Cut and fill	
	tons.	grade*	tons.	grade*	tons.	grade*
3580	169567	0.57	198554	0.55	159750	0.59
3610	263973	0.53	292960	0.52	286019	0.51
3640	271497	0.69	319809	0.65	253620	0.73
3670	270309	1.12	279972	1.10	296190	1.14
3700	283298	0.81	328284	0.76	318150	0.82
3730	127828	0.61	166478	0.57	134280	0.62

Table 11.1 Stope reserves and grades for the three stoping methods. * - grade in % Cu.

11.7 Output - Economic analysis:

The computer models provide the output of the economic analysis in the format outlined in the tables 11.2 to 11.15. First the output for each individual stope is produced. Overall economics are then assessed, for the whole of the section. Lastly, the equipment analysis is undertaken. It is not the aim to present the tables for each of the stopes analysed but the typical output for one of the stopes (stope 3670) is illustrated. The economic analysis at at any productivity level can be computed, but the results described below are at 70% productivity.

11.71 Transverse sublevel stoping:

Table 11.2, shows the development activities for one of the stopes (stope 3670). This includes both horizontal and vertical workings within the stope that are required to develop it.

Transverse sublevel stoping - stope 3670

Details of development work

Activities	Linear development work in m	
Total draw points length	88.00	
Extraction drives lengths	78.50	73.60
Trough drives lengths	36.50	36.50
Foot wall drive length	35.34	
Total extraction level development	121.84	
Total vertical development	121.84	
Bsl drilling x-cuts lengths	72.36	75.17
Bsl f/d and slot drive lengths	30.52	30.00
Bsl total development	208.05	
Tsl drilling x-cuts lengths	80.03	83.31
Tsl f/d and slot drive lengths	30.71	30.00
Tsl total development	224.04	
Crown s/l development	93.59	
Crown f/w drive	30.88	
Total development at crown level	124.47	
Total stope development	1056.85	

Table 11.2 Details of development activities for a stope - Sublevel stoping

Table 11.3, shows development work described in table 11.2 in cu.m (the volume) and the details of other activities such as stope drilling, blasting, mucking, transportation and hoisting. The unit costs for each of the activities shown are at 70% productivity. The total operating cost, cost per tonne of mining and cutoff grade, are also calculated.

Analysis operational costs

Degree of mechanisation- 1 Productivity level 70.0%

Operation	Quantum of work	cost/unit*	total cost*
Development(horizontal)			
Extraction level, cu.m.	4884.65	372.14	
Bottom sub level, cu.m.	1872.49	329.29	
Top sub level, cu.m.	2016.40	329.29	
Crown sub level, cu.m.	1120.23	329.29	
Total horizontal dev. , cu.m.	9893.77		3467218.8
Total vertical dev. , cu.m.	1096.53	443.04	485802.55
Long hole drilling, m	77231.14	102.14	7888609.59
Stoping(blasting, Mucking & transportation)	237338.10	33.87	8037783.84
Hoisting, tonnes	270309.00	7.50	2027317.50
Miscellaneous, tonnes	270309.00	8.10	2190673.23
Total variable cost			21906732.27
Variable mining cost/t			89.15
Cutoff grade, %Cu (with operating labour)			.460

Table 11.3 Activities and cost analysis for stope development and stoping for a stope - sublevel stoping. * - in Rs.

Apart from the quantum of work, and its associated cost for each of the stopes analysed, other technical details are estimated and shown in table 11.4. These include the quantity of metal, the revenues earned and operating margin (considering only the operating costs). Thus the economics of each individual stope is calculated in this manner.

Other operational details	
Percentage extraction	88
Stope reserves in tonnes	270309.00
Average grade in % Cu	1.13
Quantum of metal in tonnes	2553.72
Total revenues in Rs	108405500.95
Total cost of mining in Rs	24097405.50
Operating margin in Rs	84308095.46

Table 11.4 Techno-economical details for a stope sublevel stoping

Table 11.5 provides the details of the overall economics for the whole section analysed. It shows the overall cutoff grade, average grade, mineable reserves, total mining cost/ton of ore and total mining and processing cost/t of ore worked out. The life of the section, the present worth (NPV) of the earnings (profit/loss) that will be derived by mining this section, have been determined.

Degree of mechanisation - 1 Productivity level 70%

Cutoff grade = .433 Variable cost/tonne = 81.57

Average grade = .750 Total stopes reserve = 1386472.00

Ore production per day = 5000.0 Alternative - 1

Techno-economics	
Metal recovery in tonnes	8736
Operating margin (considering variable mining costs only) in millions Rs/yr	278.86
Operating margin (considering total mining costs) both fixed and variable) in millions Rs/yr	216.06
Operating margin (considering total variable and fixed costs (mine and processing plants) in millions Rs/yr	-248.44
Total mining cost/tonne of ore in Rs	123.44
Total cost/tonne of ore (mining and processing)	433.11
NPV of the section considered in millions Rs at 10% discount	-209.51
Life of the section in years	0.924

Table 11.5 Overall techno-economics of the section analysed sublevel stoping

The calculations, for equipment requirement, at the specified rate of production, are shown in table 11.6. For each category of equipment the capital expenditure and depreciation cost per year have been calculated. These calculations are based on the norms and cost figures obtained from the mines. The equipment performance norms have been shown in the appendix of chapter 8.

Equipment analysis

Equipment	Population	depreciation/yr*	total capital cost*
Jack hammer	46	519570	46000
Ring drilling jumbo	14	793800	2100000
Rocker shovel	22	2752200	15400000
Locomotives	22	2712600	17600000
Total		6778170	35146000

Table 11.6 Equipment analysis for a production of 5,000 t/day by sublevel stoping.
* - in Rs.

11.72 DTH stoping:

Table 11.7 gives an account of development work that will be needed in case of DTH stoping, for the same stope, considered for transverse sublevel stoping. It can be seen that the total development work in both the methods is almost the same. However, the workings of the sublevel stope are spread over four horizons where as that of the DTH stope are concentrated over two horizons. This feature helps in achieving better supervision and working efficiency.

Details of development work - stope 3670

Activities	Linear development(in m)
Extraction level	
Draw points	88.00
Slot x-cut	30.00
Extraction drive-1	84.33
Extraction drive-2	79.70
Trough drive-1	40.50
Trough drive-2	40.50
Foot wall drive	30.26
Drilling level	
Drill xcuts(all)	400.84
Slot xcut/drive	30.00
Foot wall drive	30.84
Decline(partly)	7.00
Crown level(optional)	75.16
Sub total	944.15
Vertical development(optional)	61.84
Total stope development	1005.98

Table 11.7 Details of development activities for a stope - DTH stoping

Similar to tables 11.3 and 11.4 described above, in tables 11.8 and 11.9, the economics of the same stope by DTH stoping has been shown. It can be seen that there is a decline in cutoff grade by this method, from 0.460% to 0.352% Cu.

Analysis operational costs - stope 3670

Degree of mechanisation- 3 Productivity level 70.0%

Operations	Quantum of work	cost/unit*	total cost*
Development(horizontal)			
Extraction level, cu.m.	5059.45	263.57	
Drilling level, cu.m.	5655.64	263.57	
Crown sub level, cu.m.	676.48	263.57	
Decline(common), cu.m.	122.50	263.57	
Total horizontal dev., cu.m.	11514.07		3034778.82
Vertical development, cu.m. (optional)	556.53	417.32	232252.46
Long hole drilling, m	20931.61	102.14	2138014.52
DTH drilling(165mm), m	3952.01	235.71	931545.35
Stoping(blasting,mucking & transportation)	243760.21	26.81	6535026.82
Hoisting tonnes	279972.00	7.50	2099790.00
Miscellaneous	279972.00	5.35	1497140.80
Total variable cost			16468548.76
Variable mining cost/t			58.82
Cutoff grade, %Cu (with operating labour)			.352

Table 11.8 Activities and cost analysis for stope development and stoping for DTH stoping. * - in Rs.

Other operational details

Percentage extraction	88
Stope reserves in tonnes	279972.00
Average grade in % Cu	1.10
Quantum of metal in tonnes	2586.23
Total revenues in Rs	109785649.93
Total cost of mining in Rs	16468548.76
Operating margin in Rs	93317101.17

Table 11.9 Techno-economical details for a stope DTH stoping

The influence of operating at a comparatively low grades can be analysed by comparing the results of the overall economics of the section, described in tables 11.5 and 11.10. It can be seen that lowering cutoff grade from 0.433 to 0.355% Cu, there is an increase in reserves by 0.2Mt (from 1.386Mt to 1.586Mt) and the corresponding increase in metal quantity is by 8%.

Degree of mechanisation - 3	Productivity level 70.0%
Cutoff grade = .355	Variable mining cost / ton = 59.80
Average grade = .709	Total stopes reserve = 1586057.00
Ore production per day = 5000.0	Alternative - 1

Techno-economics

Metal recovery in tonnes	9466
Operating margin (considering variable mining costs only) in millions Rs/yr	289.52
Operating margin (considering total mining costs only) in millions Rs/yr	226.72
Operating margin (considering total variable and fixed costs (mine and processing plants) in millions Rs/yr	-217.0075
Total mining cost/tonne of ore in Rs	101.67
Total cost/tonne of ore (mining and processing)	397.99
NPV of the section considered in millions Rs at 10%	-208.75
Life of the section in years	1.057

Table 11.10 Overall techno-economics of the section analysed DTH stoping

Table 11.11, lists the equipment required for mining by the DTH method. This method needs more capital investment on equipment than the other two methods as can be seen from tables 11.6, 11.11 and 11.14.

Equipment analysis

equipment	population	depreciation/yr*	total capital cost*
Jack hammer	20	225900	20000
Ring drilling jumbo	4	226800	600000
Drill jumbo	6	2786400	15600000
DTH drills	1	775800	5000000
LHDs	16	4579200	25600000
JDT trucks	16	7156800	40000000
Total		15750900	86820000

Table 11.11 Equipment analysis for a production of 5,000 t/day - DTH stoping * - in Rs

11.73 Flat back cut and fill stoping:

In the proposed flat back cut and fill stoping method three consecutive stopes(considered in other two methods) were merged to form a stope to give a stope length of 90m. The stope profiles were obtained using following alternative criteria:

1- keeping stope walls straight as in the other two methods
2- locating stope walls by moving panel by panel in from the extremities of the mineralisation until a block at (or above) the cutoff grade is encountered.

A comparison of the economic analysis for both these alternatives has been shown in table 11.15. This indicates that there is not a considerable difference in the quantity of metal that will be obtained by these two propositions considered for the particular section that has been analysed.

Tables 11.12 to 11.15 are the outputs of economic analysis for cut and fill stoping. Similar to tables 11.3 and 11.8, table 11.12 listing stope development and stoping activities together with their costs. The filling and support cost data of Dariba mine have been used. The results shown in this table indicates a higher cutoff grade compared with the other two methods described, is needed to extract the deposit by this method.

Post & pillar (flat back cut & fill)stoping

Analysis operational costs - stope 3670

Productivity level 70.0%

operations	quantum of work	cost/unit*	total cost*
Development(horizontal)			
Ceiling starting slice, cu.m.	9108.36	329.29	
Ore and manpass x-cuts, cu.m.	.00	329.29	
Service raise x-cuts, cu.m.	3495.27	329.29	
Foot wall drives, cu.m.	3000.00	329.29	
Total horizontal dev., cu.m.	15603.63	329.29	5138052.53
Manpass & orepasses, m	53.10	3000.00	477906.78
Service raise, cu.m.	331.60	400.18	132697.39
Drilling by jumbo, m	219397.98	49.14	10781843.79
Stoping(blasting,mucking and transportation)	629809.11	31.24	19676506.78
Hoisting, tonnes	676620.00	7.50	5074650.00
Filling & support, tonnes	676620.00	32.86	22231800.00
Miscellaneous, tonnes	676620.00	9.39	6351345.73
Total variable costs			69864803.00
Variable mining cost/t			103.26
Cutoff grade, %Cu (with operating labour)			.510

Table 11.12 Activities and cost analysis for stope development and stoping - cut and fill stoping. * - in Rs

The technical parameters shown in table 11.13, indicate that there is an increase in overall recovery (percentage extraction) of the ore by cut and fill over the other two methods considered.

Other operational details

note:
Method-1: by straightening stope boundaries
Method-2: without straightening stope boundaries

Percentage extraction	90.38
Stope reserves by method-1 in tonnes	676620.00
Stope reserves by method-2 in tonnes	536545.51
Average grade by method-1 in % Cu	.92
Average grade by method-2 in % Cu	1.12
Quantum of metal by method-1 in tonnes	5207.03
Quantum of metal by method-2 in tonnes	5024.26
Total revenues by method-1 in Rs	221038578.19
Total revenues by method-2 in Rs	213279796.90
Total cost of mining by method-1 in Rs	69864803.00
Total cost of mining by method-2 in Rs	55401327.31
Operating margin by method-1 in Rs	151173775.20
Operating margin by method-2 in Rs	157878469.59

Table 11.13 Techno-economical details for a stope 3670 cut and fill stoping

The equipment analysis, shown in table 11.14, indicates that comparatively less capital cost is incurred by this method over the other two methods analysed.

Equipment analysis

Equipment	population	depreciation/yr*	total capital cost*
Jack hammer	22	248490	22000
Wagon drills	10	567000	1500000
LHDs	16	4579200	25600000
Locomotives	10	1233000	8000000
Filling plant			10000000
Total		6627690	45122000

Table 11.14 Equipment analysis for a production of 5,000 t/day - cut and fill stoping. * - in Rs.

The overall techno-economics of the method are described in table 11.15. It can be seen that due to a higher cutoff grade, there is reduction in ore reserves by 16.7% and that of metal by 10% compared to those for DTH stoping.

Post & pillar (flat back cut & fill)stoping

productivity level 70.0%

Cutoff grade = .501 variable mining cost / ton = 100.86

Average grade = .767 total stopes reserve = 1322009.00

Ore production per day = 5000.0 alternative - 1

Techno-economics

Alternative 1

Metal recovery in tonnes	8516
Operating margin (considering total variable mining costs) in millions Rs/yr	258.90
Operating margin (considering total mining costs only) in millions Rs/yr	196.10
Operating margin (considering total variable and fixed costs (mine and processing plants) in millions Rs/yr	-276.58
Total mining cost/tonne of ore in Rs	142.72
Total cost/tonne of ore (mining and processing)	457.85
NPV of the section considered in millions Rs at 10%	-222.83
Life of the section in years	0.881

Alternative 2

Metal recovery in tonnes	8273
Operating margin (considering variable mining costs only) in millions Rs/yr	292.59
Operating margin (considering total mining costs only) in millions Rs/yr	229.79
Operating margin (considering total variable and fixed costs (mine and processing plants) in millions Rs/yr	-273.58
Total cost/tonne of ore (mining and processing)	478.31
NPV of the section considered in millions Rs at 10%	-198.71
Life of the section in years	0.791

Table 11.15 Overall techno-economics of the section analysed - cut and fill stoping

11.8 Comparison - economics of the three mining methods

The results of the overall economics of the three methods are summarised in table 11.16.

Techno-economical parameters	sublevel stoping	DTH stoping	cut and fill stoping
Stope reserves in million tonnes	1.386	1.586	1.322
Average grade in % Cu	0.75	0.70	0.76
Cutoff grade in % Cu	0.43	0.35	0.51
Metal recovery in tonnes	8736	9466	8516
Operating margin (considering total variable mining costs only) in millions Rs/yr	278.86	289.52	258.90
Operating margin (considering total mining costs both variable and fixed) in millions Rs/yr	216.06	226.72	196.10
Operating margin (considering total variable and fixed costs (mine and processing plants) in millions Rs/yr	-248.44	-217.75	-276.58
NPV of the section considered in millions Rs at 10%	-209.51	-208.75	-222.83
Life of the section in years	0.924	1.057	0.881

Table 11.16 Comparison of techno-economics of the section analysed by the three mining methods.

From this summary following inferences can be drawn:

DTH stoping allows a lower cutoff grade to be selected than the other two methods. This results in a higher tonnage, greater metal recovery and longer mine life.

Positive operating margins are provided by all three mining methods when the total variable mining costs are included. Adding fixed mining costs also gave positive operating margins in each method. In both cases DTH stoping gave the highest operating margin. However, when the total costs (fixed and variable) of process plants were included, all three methods give negative operating margins and NPVs at 10% discount rate.

The low overall grade (not greater than 0.77% Cu, see table 11.16) of this part of the deposit, is one reason for these negative operating margins. However, the high fixed costs of the process plants, could also be considered to share part of these losses. An improvement in the process recoveries could also help the situation. Also an increase in the rate of production over 5,000 t/day (at which this analysis has been undertaken) could further bring improvements, but again, the mine has yet to achieve this rated production from its current production of 3,500 t/day.

The cutoff grades at which the three methods have been analysed are in the range of 0.35% to 0.5% Cu. The grade-tonnage relationship established for this part of the deposit suggests that further rise in cutoff grade would reduce the reserves substantially, leading them to be unmineable. For the section analysed the annual losses by DTH stoping are the least.

Increasing productivity from the 70% level to 100% did not give positive operating margins. However, this proposition is not very practicable.

In chapter 8 the use of these models for designing sublevel stoping together with the economic analyses at varying stope width was illustrated. In a similar manner, other stope design parameters, including stope height, can be altered and tested to provided better results.

Equipment analysis indicates that DTH stoping is more capital intensive than the other two methods.

11.9 Conclusions:

These models provide a basis to select the mineral inventory, estimated through geostatistical techniques, which can be mined at profit. Reasonably accurate estimates of development and stoping activities can be made using them. The unit costs derived from the mine's cost reports have allowed estimates of total costs to be made. These have formed the basis for economic decisions regarding selection of a mining method.

Design of an economically feasible stope layout is a function of several factors. An ultimate stope design layout to answer certain economic constraints today may no longer be feasible under tomorrow's conditions and vice versa. For this reason efforts have been made to keep these stope design computer models flexible enough to run many times using a variety of economic and design parameters. This approach could provide sufficient alternatives to determine the best design for differing conditions.

Appendix - 3

Process	Production p.a. in tonnes	Variable costs p.a. in millions of Rs.	Fixed costs p.a. in millions of Rs.
Milling Ore	1836,000	71.60	47.35
Smelting Concentrates	109,781	119.46	117.93
Refining Anodes	17,066	15.07	19.74
Wirebars from Cathods	16,913	18.78	16.86

Based on the above data, the process costs on per tonne basis, as outlined below were calculated.

Process	Variable costs in Rs. per tonne	Fixed costs in Rs. per tonne
Milling Ore	40	26
Smelting Concentrates	1090	1075
Refining Anodes	880	1160
Wirebars from Cathods	1110	1000

The Revenues on per tonne Copper produced in Rs.

Copper price	44,500
Add for Metallic by-products	1,100
Add for gaseous by-products	600
Deduct for Duty + Selling costs	3,750
Net Copper price	42,450

(Source: Finance Department, KCC, 1986)
Note: These data are based on the performance of 1985-86 and have been used for the projections of future cash flows by the Finance Dept. KCC. The non-cash costs have not been included. The concentrates from Malanjkhand Project have not been considered. Only 60% of the annual fixed costs of Khetri mine were considered, rest 40% costs were accounted for the operating (variable) costs. This is because the allocation of mine workers on direct operations being 40% of the total labour strength.

The cost of an equipment and its depreciation cost as equipment and depreciation cost data as shown in tables 11.6, 11.11 and 11.14 were noted from the KCC and Rajpura Dariba mines.

Chapter 12

12. Conclusions, Recommendations and Suggestions for further research

The aim of this section is to bring together the various conclusions made in the different sections of this research based on the studies made for the two mines. However, the approach to ore estimation and mine design outlined in this thesis is general. The methodology could be applied to underground mines having similar geological characteristics. The conclusions made can be grouped into the following three areas:

12.1 Mineral inventory estimation using geostatistical techniques:

On the basis of the mineral inventories computed through kriging, using different sampling data, the following conclusions could be drawn:

No matter how close or sparse the samples are and in what-so-ever manner, the statistical analyses in all the cases have indicated that grade distribution of the deposit at Kolihan exhibit nearly log-normal habit.

Geostatistical methods can be used satisfactorily to evaluate low grade base metal deposits such as at KCC mines. However, the global estimation of the deposit by kriging using low density, irregularly spaced and inconsistently recorded sampling data could not provide the reliable results.

The semivariogram studies indicated that high density and regular sampling data (as in case of chip sampling) have shown lower nugget variance (Co) and low transition of amplitude (C), compared with the irregular and sparse drilling sampling data, from the same area.

The mineral inventories for the area between level 3 and level 4 of Kolihan mine, using underground drilling and chip sampling information were estimated using kriging. There was a good correlation between these estimates and the estimate made by the mine geologists using conventional method. Thus, it is possible to assess the mineral inventory satisfactorily through kriging using only the underground drilling information, provided a systematic sampling procedure is adopted. The estimation using chip sampling in a few mine workings could be used only as a check. Chip sampling of most of the mine workings, as is the practice currently followed at KCC mines, could be avoided. This would mean a considerable saving in sampling costs and time. However, it would be advisable to undertake similar comparisons in different parts of the mines before eliminating routine chip sampling.

The presence of nugget variance, observed while studying the semivariograms of drill samples suggests a need to improve upon the sampling procedures. The procedure of regular sampling of drilling information at 1m interval all along a drill hole, currently implemented at KCC mines, may prove useful in improving estimates if the current ore estimation technique used at KCC is replaced by kriging.

With the assistance of computers, geostatistical techniques have provided a quick, simple and effective way to evaluate an orebody. Using computer graphics the mineral inventory has been presented by way of grade-tonnage curves, plans, sections and other useful plots.

12.2 Cutoff grade decisions:

An orebody configuration decreases in size and increases in irregularity when raising the cutoff grades. In underground mines the variable costs (cutoff dependent costs) are very sensitive to a stoping method, orebody thickness, degree of mechanisation and productivity. Variable costs can fall as the cutoff grade is lowered. This allows stope outlines to be smoother and cheaper mining methods to be used. Thinner orebodies adversely affect the cutoff grades. When a thickness of 30m is attained, the cutoff grade becomes almost stable. An allowance for productivity and degree of mechanisation should be made in cutoff grade calculations. There is a sharp decline in cutoff grades with an increase in process recoveries.

12.3 Modelling stope boundaries, stope design models and economic analysis:

The methodology developed to evaluate the stope boundaries for the sublevel, DTH and cut and fill stopings has been used satisfactorily in a situation where the continuity of mineralisation in the direction of exploitation existed. Using this methodology the internal waste or the sub-grade material contained within a stope could be assessed. In addition, it provided a basis to compare the three methods.

The validity of results depends primarily on the quality of the technical and economical parameters used as well as the orebody modelling. Small 1m x 1m size panels were chosen to achieve a selectivity of ore sections.

The computer assisted stope design and mine planning has automated many of the manual procedures and calculations involved in mine planning. This enabled the evaluation of many layouts and mining strategies.

The stope design computer models are flexible enough to run many times using a variety of economic and design parameters. This approach provides many options and alternatives to determine the best design for differing conditions.

Mine production costs have been built up from stope design and estimation of individual mine activities. Those have formed the basis for the mineral exploitation strategy.

The results from analysing one section of the Khetri mine indicated that a much higher cutoff grade than that established by only variable costs should be investigated. It is possible that in doing so the orebody configuration may reduce to the extent that it may become unmineable, as it was observed when raising the cutoff grade from 0.1% to 1% Cu for Kolihan mine (chapter 6).

12.4 The recommendations and suggestions for the future research work:

The problem of 'disappearing tonnage' as pointed out in chapter 4, could be examined using multigaussian kriging, currently being developed at the Stanford University (Verly, 1983).

A specific study to examine the influence of increasing depth of mine workings on cutoff grade may be undertaken. This aspect is of significant importance particularly where multi-stage hoisting is essential.

Future work in the area of defining the optimum stope boundaries could be the investigation of more elegant ways to demarcate the optimum stoping boundaries. This includes designing of an algorithm which eliminates specifying of wall angles manually. A three dimensional approach in a situation where it is not possible to assume continuity of mineralisation in the third direction may be necessary.

In the area of stope design, the algorithm for other mining methods, not considered in this research, including the caving methods can be developed. In addition, evaluation of stope dimensions taking strata conditions and mechanical properties of rocks into consideration is also an important area where the future research could be focused.

Further work should be directed at investigating the economic effect of mining at higher cutoff grade at KCC mines.

Possible areas for the use of computers in the context of underground metal mines were outlined in chapter 10. In this research the use of computers in the following areas have been demonstrated:

- assimilation and display of base data

- orebody modelling
- determination of feasible and efficient mineral exploitation limits.
- design of the exploitation methods
- economic evaluation of the deposit
- ancillary activities such as - blasthole pattern design etc.

In addition to the further refinement in the areas where use of computers in the above context has already been explored, the following are the some of the activities which can be undertaken, in future, with the assistance of computers:

- calculation of a schedule for the deposit exploitation
- design and display of exploitation sequence
- calculation of resources consumed and production generated during the exploitation.

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