ECONOMIC CONSEQUENCES OF HOLE DEVIATIONS IN MINING OPERATIONS

SAM KANGWA

JUNE 2000

ECONOMIC CONSEQUENCES OF HOLE DEVIATIONS IN MINING OPERATIONS

By

SAM KANGWA

A Dissertation Submitted to the

UNIVERSITY OF ZAMBIA

In Fulfilment of the Requirements for the Degree of

MASTER OF MINERAL SCIENCES IN MINING ENGINEERING

Department of Mining Engineering
School of Mines
University of Zambia
LUSAKA

JUNE 2000

DECLARATION

This is to declare that this dissertation has been written in accordance with the rules and regulations governing the award of the Degree of Master of Mineral Sciences of the University of Zambia.

It is further declared that the dissertation has not been submitted for a Degree in another University or similar institution and that the contents in this dissertation are my original work. Where other people's work has been drawn upon, acknowledgement has been made.

Candidate's signature: ____________________ Date: 23rd June 2000.

APPROVAL

The University of Zambia approves this dissertation of Sam F. Kangwa as fulfilling the requirements for the award of Degree of the Master of Mineral Sciences in Mining Engineering.

Examiners:

1. ____________ Date: 20/10/2000
Prof. Thomson Sinkala

2. ____________ Date: 20/10/2000
(Prof. R. KRISHNA)

3. ____________ Date: 20/10/2000
(Dr P R K CHHOSSAF)
External Examiner

ABSTRACT

In hard rock mining, drilling and blasting still remain the main method of fragmenting rock. In mining methods involving the use of long holes, extra costs are incurred as a result of consequences associated with hole deviations.

The consequences include build-ups, hang-ups and poor rock fragmentation. These consequences usually lead to extra drilling, loss of drill strings, ore dilution, ore loss, increased explosive consumption, time wastage and delays in the chain of production operations, accelerated wear of equipment, extra crushing, extra mineral dressing work, and increased labour costs.

Many studies do exist on hole deviations. Whereas sources and effects of the deviations have been investigated, no work has been done on quantification of extra costs due to hole deviations. The objectives, therefore, of this study were to identify consequences associated with hole deviations in mining operations and to quantify the consequences in economic terms. For the study to be meaningful, it was decided to carry out a field investigation at an operating mine. The investigation was carried out at South ore-body shaft of Zambia Consolidated Copper Mines, Nkana Division. Nkana Division operates an underground mine and produces copper and cobalt.

The detailed studies of hole deviations and their consequences involved two selected operations, namely stope drilling and raiseboring of Vertical Crater

Retreat (VCR) mining method. The studies involved examination of drilling plans, observation of drilling operations, measurement of hole deviations, observation of blasting activities, examination of fragmentation and handling of ore.

The studies showed that almost all the holes observed had some degree of deviation. The linear deviation ranged from 0.2m to 12.0m for hole lengths of 30m to 105m. It was subsequently established that there is a strong link between hole deviations and many problems faced in the chain of mining operations.

The consequences were identified and correspondingly quantified in economic terms. The cost estimations were done using both rudimental and computer aided calculations.

From the estimates, the VCR stope studied accrued approximately US$274,000 extra cost, representing 62% of the total operational cost, while the raise pilot hole accrued approximately US$6,000 extra cost representing 44% of the total operational cost. These extra costs were as a result of hole deviations.

These results clearly show that hole deviations have a significant negative effect on the mining operations and that they lead to poor mining economy.

If the quality of production and service holes can be improved, a saving of up to 70% may be achieved in the VCR stope. For an underground mine such as Nkana Division, it may become possible for the mine to compete favourably with other underground and surface mines.

For this to be achieved, it is recommended that management should:

a) Set up a steering group to look into minimising hole deviations
b) Rehabilitate or purchase drilling machines and accessories to improve machine productivity
c) Train operators and supervisors incharge of drilling on the principles of achieving improved quality of holes, and
d) Reinforce the use of computer program for planning blasting sequences to improve fragmentation.

ACKNOWLEDGEMENTS

The work is a contribution in the area of precision drilling. This research is a follow up to the work carried out on hole deviations, control measures and drilling techniques. I would therefore like to acknowledge previous researchers in the field of precision drilling.

My sincere gratitude to my Supervisor Prof. Thomson Sinkala, not only for his invaluable guidance and support for this work, but also for seconding two 4th year mining students as research assistants. These assistants worked tirelessly in vexatious conditions. I also wish to acknowledge contributions of Prof. Radhe Krishna, acting Dean of School of Mines for his valuable suggestions during the early presentations of progress reports and for moral support rendered to me throughout my study period.

I am indebted to the former Zambia Consolidated Copper Mines, Nkana Division management for affording me an opportunity to carry out fieldwork at the Division and for providing research equipment, which included drills and drilling accessories, and for allowing me to use company labour and facilities. In particular I would like to thank Mr. George Chibuye, the then General Manager; Mr. Alex Manda, the then Manager-Mining and Mr. Jacob Banda, the then Underground Manager, Technical; Mr. Peter Haken, the then Underground Manager, Projects and Mr. Duncan Tembo, the then Assistant Underground Manager Technical who scrutinised the contents of this dissertation, especially topics related to mining operations at Nkana Division; and Mr. Davie Partridge and later Mr. Elias Mkandawire, the then Assistant Underground Managers at SOB Shaft for facilitating underground trips and field data collection.

Thanks also go to Capt. Felix Kanungwe, Dean of School of Technology and Mr. Wallace Samiselo, Head of Mining Department, both of the Copperbelt University, for allowing me to use School and Department facilities.

I wish to express my thanks to my family for the support and patience, enduring my absence from home. Life would have been unbearable without the many friends at SOB shaft, the Copperbelt University and the University of Zambia who supported me during my studies.

Finally, I gratefully acknowledge the financial assistance from the Copperbelt University who sponsored my study. I also wish to thank the United Nations University Institute for Natural Resources in Africa - Mineral Resources Unit (UNU/INRA-MRU) for providing material help and use of facilities.

TABLE OF CONTENTS

CONTENTS **PAGE**

Declaration.......... I
Approval.......... Ii
Abstract.......... Iii
Acknowledgements.......... Vi
Table of contents.......... Viii

List of Tables.......... Xiv
List of Figures.......... Xvi
Nomenclature/Symbols.......... XvIii

CHAPTER ONE: INTRODUCTION.......... 1

1.1 Specific Objectives and Contents of this Study.......... 3
1.1.1 Specific objectives of this study.......... 3
1.1.2 Contents of this study.......... 4

CHAPTER TWO: LITERATURE REVIEW.......... 6

2.1 Types and Sources of Hole Deviation.......... 6
2.1.1 Hole collar position deviation.......... 7
2.1.2 Hole alignment deviation.......... 8
2.1.3 Hole trajectory deviation.......... 8

2.2 Effects of Hole Deviations.......... 11
2.2.1 Brief discussion of some effectsof hole deviations.......... 14

2.3 Economic Consequences of Hole Deviations.......... 17

CHAPTER THREE: THE FIELD STUDY AT ZCCM-NKANA DIVISION MINE.......... 18

3.1 Selection of the study Area.......... 18

3.2 Approach to the Study Area.......... 18
3.2.1 Initial Visit.......... 19
3.2.2 Detailed Study.......... 19
3.2.3 Hole deviation consequences in monetary terms.......... 20

3.3 Background Information on Nkana Division and SOB Shaft, the Study Area........ 21
3.3.1 Nkana Division........ 21
3.3.1.1 Location........ 21
3.3.1.2 The brief general geology of Nkana........ 21
3.3.2.3 Production and metallurgical processes........ 22
3.3.3.4 Mine records........ 22

3.3.2 SOB shaft, the study Area........ 23
3.3.2.1 Regional geology of SOB shaft........ 25
3.3.2.2 Mining Methods........ 25

3.3.2.2.1 Brief description of SLC mining method........ 26
3.3.2.2.2 Brief description of SLOS mining method........ 30
3.3.2.2.3 Brief description of VCR mining method........ 31

3.3.2.3 Hole deviations associated with the mining methods........ 35
3.4 Choice of VCR mining method for detailed study........ 37

CHAPTER FOUR: HOLE DEVIATIONS AND THEIR CONSEQUENCES ASSOCIATED WITH VCR MINING METHOD........ 39

4.1 The General Principle of VCR Mining Method........ 39
4.2 The VCR Stope Studied at SOB Shaft........ 42
4.2.1 Geology of the study area........ 45
4.2.2 Rock survey of the study area........ 45
4.2.3 The general development of the stope........ 47

4.3 The Drilling of the Stope and Raise Pilot Holes........ 53
4.3.1 The stope drill holes........ 53
4.3.1.1 Examination of drilling plans........ 53
4.3.1.2 Observation of drilling operations........ 54
55
4.3.1.3 Measurements of hole deviations........ 62
4.3.1.3.1 Computer 'monitoring' program........ 69

4.3.1.4 Observed causes of hole deviations........ 73
4.3.1.4.1 Consequences identified during operations........ 78

4.3.1.5 Observation of blasting activities........ 80
4.3.1.5.1 Charging........ 80
4.3.1.5.2 Blasting sequence of VCR holes........ 82
4.3.1.5.3 Problems encountered during blasting of 2590N stope........ 83

4.3.1.5.4 Consequences identified during blasting activities............ 84

4.3.1.6 Examination of fragmentation resulting from the blasts........ 86

4.3.1.7 Examination of the handling of ore crushing and metallurgical processes.. 90

4.3.1.7.1 Other consequences identified during and after stope operations.. 92

4.3.2 The raise pilot hole.. 93

4.3.2.1 Examination of drilling plans and site preparation............... 93
4.3.2.2 Observation of pilot hole drilling..................................... 96
4.3.2.3 Measurement of pilot hole deviation................................ 102
4.3.2.4 Interception of the hole using unplanned development......... 113
4.3.2.5 Consequences associated with pilot hole deviation............. 115

CHAPTER FIVE: HOLE DEVIATION CONSEQUENCES IN MONETARY TERMS.................................... 116

5.1 Methods Used in Cost Estimation of Extra Costs Due Hole. Deviations Consequences.. 118

5.2 Cost Estimation.. 121

5.2.1 Estimation extra costs for stope drill holes consequences...... 121
5.2.1.1 Manual calculations.. 121
6.2.1.2 Computer aided calculations.. 130

5.2.2 Estimating extra costs for raise pilot hole.......................... 131
5.2.2.1 Manual calculations.. 131
5.2.2.2 Computer aided calculations.. 134

5.2.3 Itemised cost analysis... 135

5.3 'MINCOST' Computer Program.. 135

5.3.1 Concepts used in the 'MINCOST' program......................... 139

5.3.1.1 Dilution.. 137
5.3.1.2 Recovery.. 140
5.3.1.3 Recovery and costs.. 140
5.3.1.4 Accelerated wear of equipment.................................... 142
5.3.1.5 Labour and Services... 143

5.3.2 Data input.. 143
5.3.3 Execution of the program.. 145

CHAPTER SIX: CONCLUSIONS AND RECOMMENDATIONS....... 149

6.1 Conclusions.. 149
6.2 Recommendations... 151
6.2.1 Areas of operational and managerial improvement................ 151

6.2.2 Further Studies.. 152

7.0 REFERENCES... 154

8.0 APPENDICES.. 161

Appendix A... 161

Table A Analysis of SOB shaft Cost for May and June.................. 161
Table B Analysis of SOB Shaft Cost for July and August................ 162
Table C Analysis of SOB Shaft Cost for September and October...... 163

Appendix B... 164

Table D Purchase Price of Drills and Drilling Accessories.............. 164
Table E Drilling Accessories Used at SOB Shaft.......................... 165
Table F Showing Long-hole Drilled Metres Per Machine................ 165

Appendix C... 166

Table G Explosive Price List for Kafironda Products..................... 166
Table H Explosive Price List for ZAMDET Products...................... 166

Appendix D... 167
Table I The Average Unit Mining Operational Cost at SOB Shaft...... 167
Table J Showing Monthly Production Figures at SOB Shaft............ 167

Appendix F
The Model For Itemized Cost Estimating Method........................... 168

Appendix G... 170

Table K. Showing London Metal Exchange Copper Price................ 170
Table L. Showing Market Cobalt Prices.. 171

Appendix H – Rock compressive strength.................................... 171

Appendix I.. 172
Graph 1 Showing Dilution Trend for 2590BCN Stope.................... 172
Graph 2 Showing Ore and Waste from 2590BCN Stope Per Month... 172

Appendix J.. 173
Stope and service hole drilling.. 173

Appendix L.. 177
Quantification of consequences associated with hole deviations........ 177

Appendix M.. 178
Examples of 'MINCOST' calculations... 178

Appendix N.. 180
Structure of the Mining Department at Nkana Division and SOB Mine Sections 180

Appendix O. Location of holes as monitored by the computer program 181

Appendix P. Plates showing types of fragmentation........................ 183

Appendix Q. Plates showing worn-out Cubex ITH drill..................... 184

Graph 3 Metres Drilled and Total Cost of Drilling Per Month............ 185
Graph 4 Ore Extracted and Total Cost of Extraction Per Month........ 185

Graph 5 Primary Development Performance and Total Cost Of Development Per Month... 186

Graph 6 Secondary Development Performance and Total Cost Of Development Per Month... 186

LIST OF TABLES

TABLE PAGE

3.1 Showing measured linear hole deviations........................... 36

4.1 2590N / 2880L Stope Data... 43

4.2 2590BCN Ore body characteristics.................................. 43

4.3 Rock Survey Results.. 46

4.5 Planned Drilling Information for 2590N Stope...................... 55

4.6 Recommended and actual parameters............................... 60

4.7 Actual Hole Lengths and Inclinations for Chamber Rings.......... 66

4.8 Actual Burdens of Chamber Rings.................................... 67

4.9 Planned and Actual Lengths, and Inclinations of Recovery Ring 1.. 68

4.10 Planned and Actual Lengths, and Inclinations of Recovery Ring 2.. 68

4.11 Planned and Actual Lengths, and Inclinations of Wrecking Ring 69

4.12 Chamber Blasting Monitoring Report for 2590N/2880L Stope.... 84

4.13 Summary of metres drilled and explosives used.................. 85

4.14 Metres and explosives utilisation.................................... 86

4.15 Percentage of fragments going through the pre-determined opening... 87

4.16 Classification of Rock Fragmentation Based on Average Fragment Size (K_{50}) and Size Distribution............................ 88

4.17 2590BCN/2880L Stope Performance................................ 92

4.18 Data for 2430N Tip Raise... 94

4.19 Progress report showing 'feet' drilled per shift..................... 98

4.20 Recommended drilling and rates of penetration..................... 101

4.21 Instrument Data.. 102

4.22 Correction of magnetic bearing using declination.................. 105

4.23 Camera Survey Film Readings.. 108

4.24 Film Readings and Calculated Results............................. 109

5.1 Costs of mining operations associated with hole deviations......... 117

5.2 Summary of costs due to consequences of hole deviations......... 128

5.3 Estimating extra costs for stope drill holes.......................... 130

5.4 Summary of costs for drill holes consequences...................... 133

5.5 Estimating extra costs for raise pilot hole............................ 134

5.6 Data input for 'MINCOST' program.................................... 144

5.7 Cost estimation computation for planned costs...................... 146

5.8 Cost estimation computation for extra costs.......................... 147

LIST OF FIGURES

FIGURE PAGE

1.1 Operating unit costs as percentage of total cost........................ 2

2.1 Diagrammatic representation of collar (d_c), alignment (d_a), and trajectory (d_t) deviations... 7

2.2 General comparison of deviations.. 9

2.3 Extras due to effects of hole deviations on mining operations........ 12

2.4 Illustrating the formation of hang-ups (a) and build-ups (b) as a result of hole deviations.. 13

3.1 Sketch of the section along 2590N of Section 300..................... 24

3.2 Sublevel caving mining method showing drilling level and draw-points.. 29

3.3 Illustrating Sublevel Open Stoping Mining Method..................... 33

3.4 Illustrating the VCR mining method..................................... 34

4.1 VCR mining method layout.. 41

4.2 2590N stope dimensions.. 44

4.3 2590N stope drilling position.. 49

4.4 Stope under-cut and proposed raise pilot hole position on 2840 Level... 50

4.5 Raise pilot hole drilling position...................................... 51

4.6 2590N drilling level and undercut...................................... 52

4.7 Stope rings on 2700 drilling level...................................... 57

4.8 2590N section showing drilling rings................................... 58

4.9 Holes positions on chamber level....................................... 59

4.10 Holes positions on (a) drilling and (b) holing levels.................... 62

4.11 Aligning of the Instrument when measuring the hole collar dip...... 64

4.12 Improvised instrument for measuring of hole alignment.............. 65

4.13 Sketch illustrating application of trigonometry for computer monitoring program calculations.. 71

4.14 Illustrating worn out parts causing one side to take up more thrust than the other side.. 74

4.15 Illustrating change in hole paths due to uneven floor................ 75

4.16 Shift in development floor elevation.................................... 75

4.17 Illustrating the charging procedure...................................... 82

4.18 A general illustration showing that good quality holes and good blast geometry will result in good fragmentation thus leading to low cost of loading, hauling and crushing.......................... 89

4.19 Plans of (a) pilot hole drilling and (b) holing layouts................ 95

4.20 Illustrating TN, MN, declination, true and magnetic WCB........... 106

4.21 Illustrating the terms nadir and zenith.................................. 106

4.22 Planned and actual starting positions................................... 111

4.23 Planned and actual holing positions.................................... 112

5.1 Drilling Cost.. 120

5.2 Illustrating Extra Costs Associated with Hole Deviations in Mining Operations.. 136

5.3 Illustrates the effect of grade on ROM, processing cost and metal output.. 139

NOMENCLATURE & SYMBOLS

AH/W	Assay Hangingwall
AF/W	Assay Footwall
AUGM	Assistant Underground Manager
ANFO	Ammonium Nitrate Fuel Oil
cm	Centimetre
Co	Cobalt
Conv.	Conventional
Cos.	Cosine
Cu	Copper
Dev.	Deviation
Det.	Detonator
Dia.	Diameter
DTH	Down-The-Hole
Dr'd	Drilled
E	East
EXC.	Excluding
Ext.	Extension
ft	Foot
F/W	Footwall
GFW	Geological Footwall
HWA	Hangingwall Argillite
H/W	Hangingwall
HWQ	Hangingwall Quarzite
Hz	Hertz
INC.	Including
Incl.	Inclination
IRS	Intact Rock Strength
JS	Joint Set Spacing
Kg	Kilogram
L.	Mine Level
Lb	Pound
LHD	Load, haul and Dump
m	Metre
M/C	Mine Captain
MIMS	Mincon Information Management System
Min.	Mining
mm	Millimetre
MPa	Megapascal
N	North
Nm	Newton-metre
NWS	Near Water Sediments

pp	Page
R/B	Raiseborer
Rec.	Recovery
ROM	Run-on-mine
r.p.m	Rotations per minute
RMC	Rock Mass Classification
RMS	Rock Mass Strength
RQD	Rock Quality Designation
S	South
S. G	Specific Gravity
Sin.	Sine
SLC	Sublevel Caving
SOB	South Ore-Body
SLOS	Sublevel Open Stoping
SOSH	South Orebody shale
T	Metric tonne
t	Imperial ton
Tan.	Tangent
UGM	Underground Manager
Var.	Variance
VAT	Value Added Tax
VCR	Vertical Crater Retreat
W	West
w	Watt
Wrec.	Wrecking
ZK	Zambian Kwacha (Currency)
ZK'M	Million Zambian Kwacha
$	Dollar (US$)
#	Mine Shaft

CHAPTER ONE

1.0 INTRODUCTION

Precision in mining engineering, compared to other engineering industries, lags behind resulting in high operational costs in areas such as development, stoping, transportation, mineral dressing and services. Reasons for this lag range from the type of mining environment to limitations in technology.

In hard rock mining, drilling and blasting still remains the main method of fragmenting rock, and extra costs are incurred partly as a result of consequences associated with hole deviations.

A drill hole is considered to have deviated if it has not (i) been collared at the intended point, (ii) followed the designed path and/or (iii) holed at the intended destination.

In the definition of hole deviation, the phrase "designed path" acknowledges the fact that sometimes holes may not be designed to take straight paths to required destinations. In such cases holes are intentionally deviated to follow curved paths. This is common practice, for example, in drilling for hydrocarbon and hydrothermal activities [Sinkala, 1989].

In blasting hole deviations can drastically alter blast geometries in terms of burden and spacing, especially at the toes of the holes. Hole deviations can also result in extra drilling in an attempt to correct a burden, or control build-ups and hang-ups.

In mining, there are many economic factors influencing operational costs. For example, poor mining results in extra operational costs, which are accrued during the various stages of production. Figure 1.1 is an example showing unit operational costs of mining in Sweden [Lindvall, 1983].

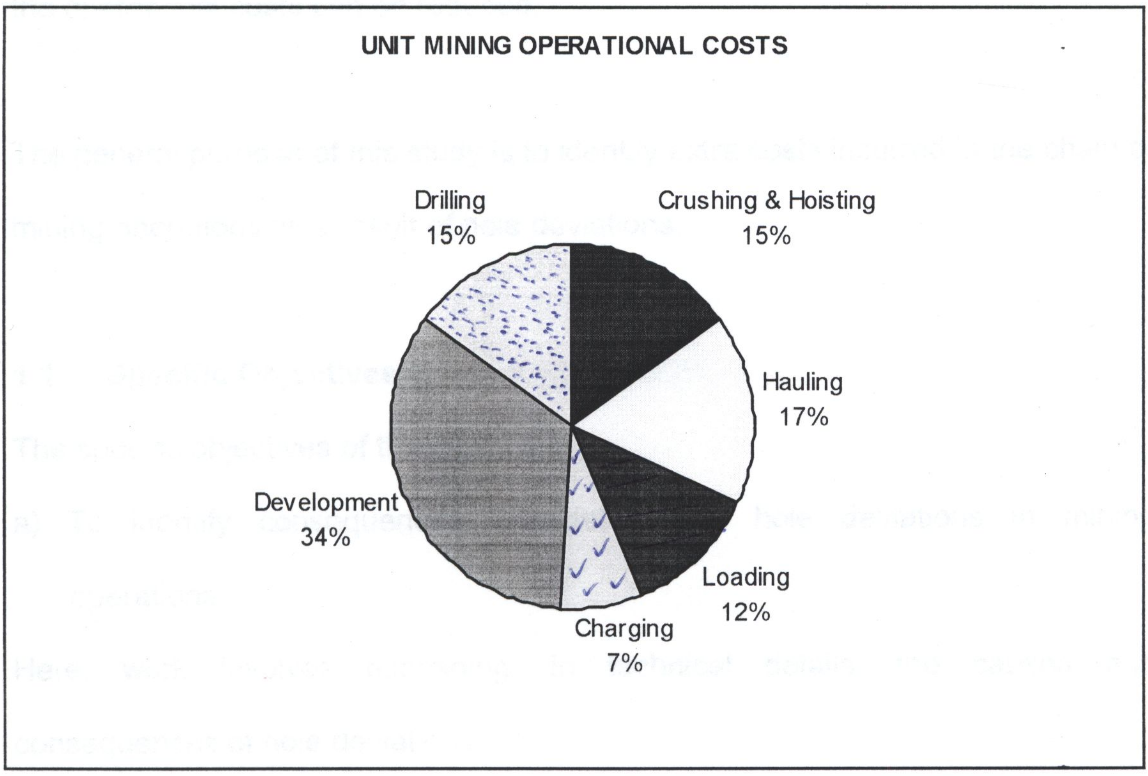


Figure 1.1. Operational unit costs as percentage of total cost [After: Lindvall, 1983].

The accuracy in drilling has a significant effect on the results of mining operations, and is one of the major factors influencing mining operational economy *(Figure 1.1). For example, poor drilling and blasting subsequently* affect fragmentation. Fragmentation will in turn affect loading, hauling, crushing, hoisting and mineral dressing costs. Large hole deviations, therefore, can lead to poor mining economy.

Variations as to drilling costs and other associated operations depend on prevailing rock properties, mining method and the drilling equipment employed. If these factors are taken into consideration when planning for *'optimised drilling'*, the operational costs can be reduced.

The general purpose of this study is to identify extra costs incurred in the chain of mining operations as a result of hole deviations.

1.1 Specific Objectives

The specific objectives of this study are:

a) To identify consequences associated with hole deviations in mining operations.

Here, work involves examining, in technical details, the causes and consequences of hole deviations.

b) To translate the consequences in economic terms.

Here, the study attempts to estimate the costs of the observed consequences associated with hole deviations.

1.2 Contents of this Study

This dissertation is presented in seven chapters as follows:

Chapters One and Two

The background of this study, objectives and contents of this study are covered in Chapter One. Literature on hole deviations is reviewed in Chapter Two and includes types, sources, and effects of hole deviations.

Chapter Three

This chapter gives general background information on Zambia Consolidated Copper Mines (ZCCM) Nkana Division where fieldwork for this study was undertaken. The approach to the field study is also explained in this chapter.

Regional geology of the study area and mining methods currently in use at ZCCM-Nkana Division, South Ore-body (SOB) shaft, are described in this chapter. Hole deviations associated with these mining methods are examined. Based on data gathered, a mining method is selected for detailed study.

Chapter Four

The general principle of the selected VCR mining method is described in this chapter and reasons for selecting 2590N VCR stope for the case study are explained. The relevant geology of 2590N area and the general development of the stope are described. Detailed field studies of two types of drill holes; namely

(a) stope production holes and (b) raise pilot hole are presented. The detailed studies included the examination of the stope and raise pilot plans, observation of drilling operations, measurement of hole deviations causes of the deviations and observation of blasting activities. Examination of fragmentation, handling of ore, crushing and metallurgical processes are also described here. Quantification of consequences associated with hole deviations are given in relevant sections.

Chapter Five

Hole deviation consequences in monetary terms are determined in this Chapter. Estimation of 'stope drill' and 'raise pilot' economic consequences is carried out. The estimation is carried out both rudimentally and by computer, where the computer program called 'MINCOST' is used. This computer program has been developed during this study.

Chapter Six

Conclusions and recommendations for future work are given in this Chapter.

CHAPTER TWO

2.0 LITERATURE REVIEW

The subject of hole deviations can be found in industries such as mining, petroleum and exploration.

The discussion which follows focuses on hole deviations as experienced in mining industry, with special reference to underground operations.

2.1 Types and Sources of Hole Deviations

From literature survey, different investigators have classified the sources of hole deviations differently [Lindvall, 1983; Medda, 1983; Walker, 1997 and Sinkala, 1990]. According to Sinkala [1989], there are three main types of hole deviations in long hole drilling, namely; collaring, alignment and trajectory deviations, as illustrated in Figure 2.1.

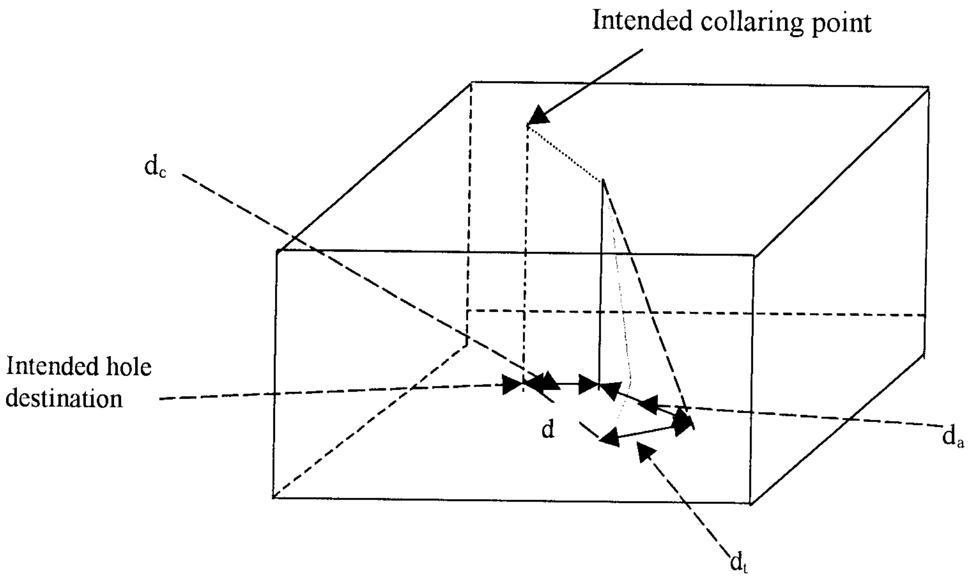


Figure 2.1. Diagrammatic representation of collaring (d_c), alignment (d_a) and trajectory (d_t) deviations [After Sinkala, 1989]

2.1.1 Hole Collar Position Deviation

Collaring deviation is a lateral displacement from a planned location. It is therefore a constant for any hole length. In general, it should not exceed one bit diameter [Tamrock Surface Handbook, 1984]. Thus $d_c < D$, where D = bit diameter.

The sources of collaring deviation include topography of drilling area and poor setting and positioning of drilling rig.

2.1.2 Hole alignment deviation

Deviation due to alignment (d_a) arises from inaccuracies in setting the feed on which a rock drill is mounted, in a planned direction. It is obvious from Figures 2.1 and 2.2 that the deviation increases linearly with an increase in hole length. Causes of alignment deviations include instability of drilling rigs, lack of precision in setting-out, tools/techniques used to align a feed beam, the topography at collaring point, and drilling operator's experience and care [Sinkala,1989].

2.1.3 Hole Trajectory Deviation

Both quantitative and qualitative experiences have shown that some power law can, in general, describe trajectory deviation with an increase in hole length. While collaring and alignment deviation errors arise from sources outside drill holes, trajectory based errors arise from sources inside drill holes.

According to Sinkala [1986], of all the holes studied in road cuts and open pits in Sweden, Norway and Finland, 81% showed trajectory deviation (d_t) to be greater than alignment deviation (d_a). 61% of rock faces had 70% of holes with $d_t > d_a$, 47% of rock faces had all holes in the faces with $d_t > d_a$. Therefore, trajectory deviation (d_t), is a more serious problem than collaring and alignment deviations, as illustrated in Figure 2.2.

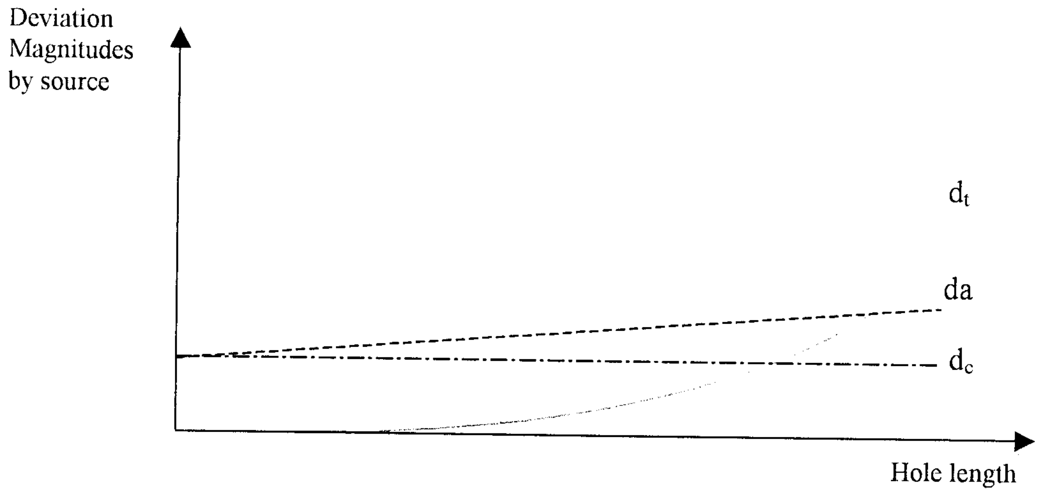


Figure 2.2. A general comparison of deviations [After Sinkala, 1989]

Essentially, there are four source categories of factors contributing to trajectory deviation, thus:

A. Hole design - hole inclination
- hole diameter
- hole length

B. Drill parameters- thrust
- percussion
- rotation
- flushing velocities (and medium)

C. Equipment - drilling machine and type
- drill string
- bit
- stabiliser

- couplings

D. Rock properties- rock structure

- rock hardness
- interval between laminations or unidirectional joints
- degree of cohesion between laminations or joint boundaries.
- variation of properties within the rock mass

Apart from source D, an operator or machine designer directly or indirectly contributes to deviations through drilling parameters and/or equipment. For instance (i) an operator may apply an unnecessarily large thrust, which could cause deviations, (ii) a machine designer may specify unnecessarily large ranges of drilling parameters, which might be misused by an operator [Sinkala, 1989].

Since alignment and collaring deviations are caused outside of holes, they are easier to investigate and understand [Karlsson 1983; Engvall 1984; Sinkala and Granholm 1987]. On the other hand it is very difficult to study those sources of trajectory deviation arising inside drill holes. Previous studies [Brown et al. 1981; Almgren 1981; Sinkala 1986; Medda et al. 1983] have also shown that this deviation is also the most difficult to understand.

2.2 Effects of Hole Deviations

According to Sinkala [1989] and Atlas Copco Manual [1983], inaccurate stope drilling has a very negative effect on stope performance and affects stoping economy, due to any, or all, of the following:

1. Ore losses and poor recovery
2. High dilution
3. Stope build-ups
4. Poor fragmentation
5. Instability of opening
6. Increased explosive consumption
7. Time wastage due to resulting hang-ups and build-ups
8. Extra drilling to make up for the consequent build-ups
9. Loss of man/machine hours in the chain of operations as a result of delays due to extra drilling and re-blasting
10. Loss of drill rods and bits due to jamming.
11. Extra primary crushing and milling, and
12. Extra mineral dressing work.

Examples of consequences due to hole deviations in the chain of mining operations, resulting in extra operational costs, are demonstrated in Figure 2.3.

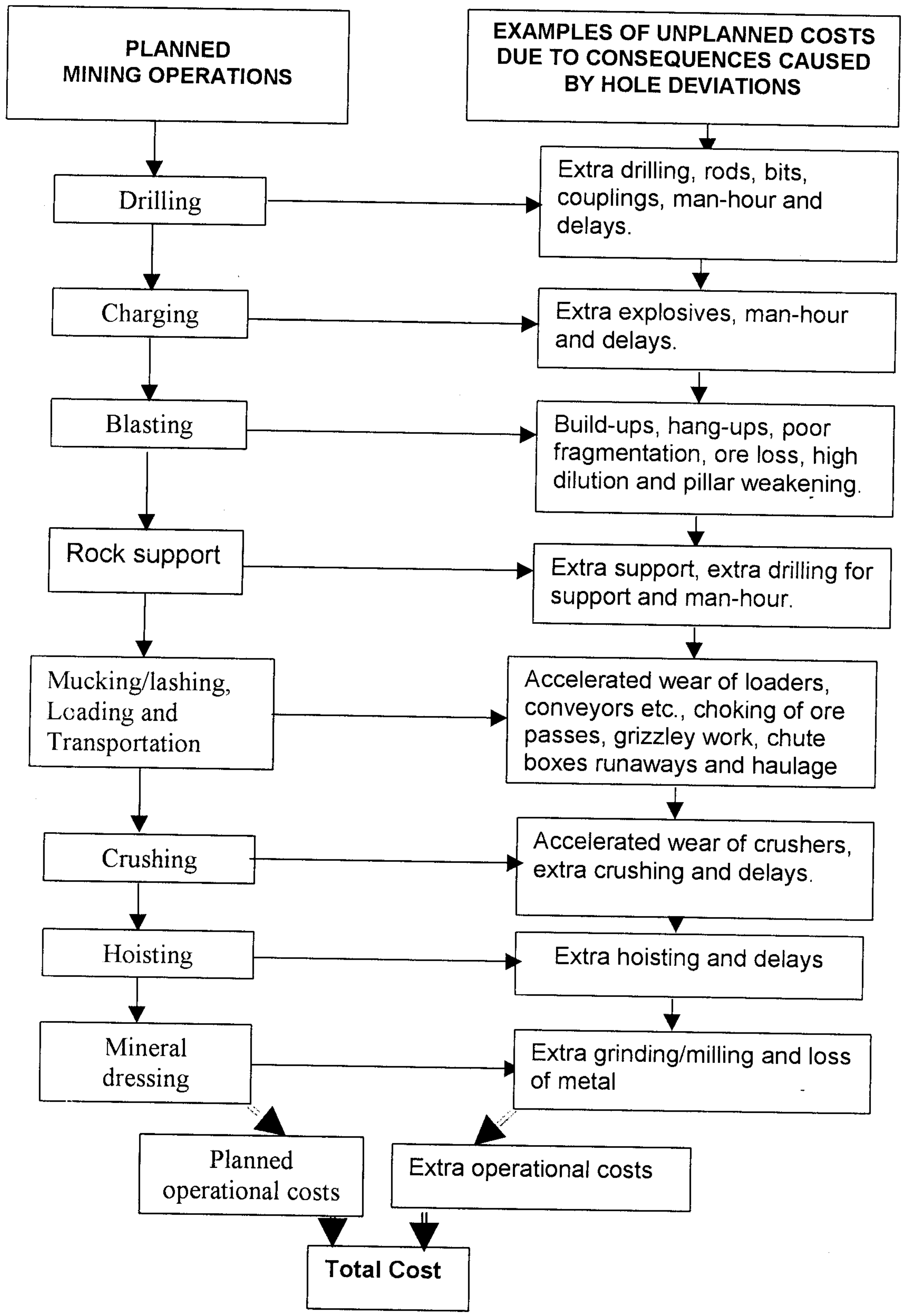


Figure 2.3. Extras due to effects of hole deviations on mining operations.

Figure 2.4 illustrates some consequences of hole deviations.

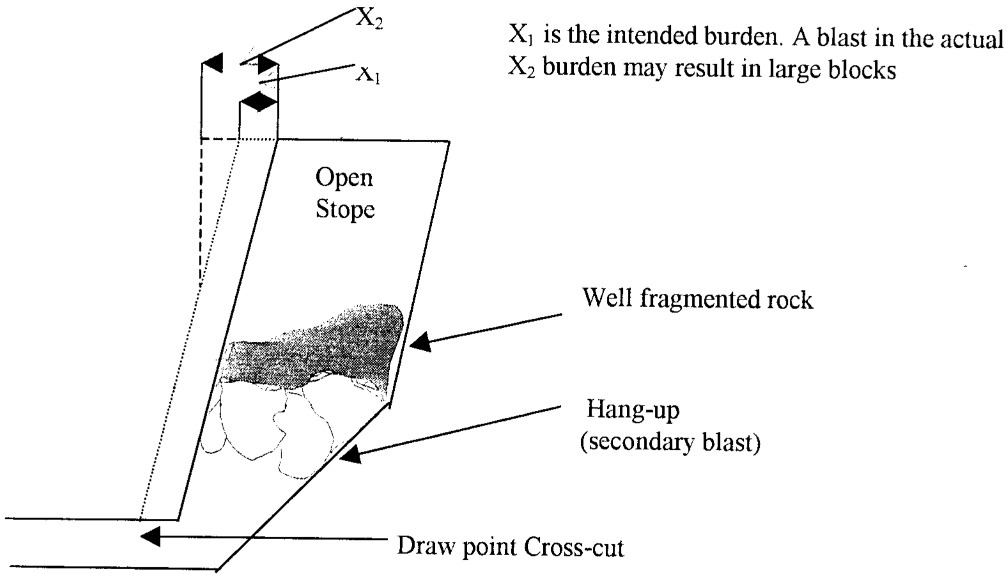


Figure 2.4. (a)

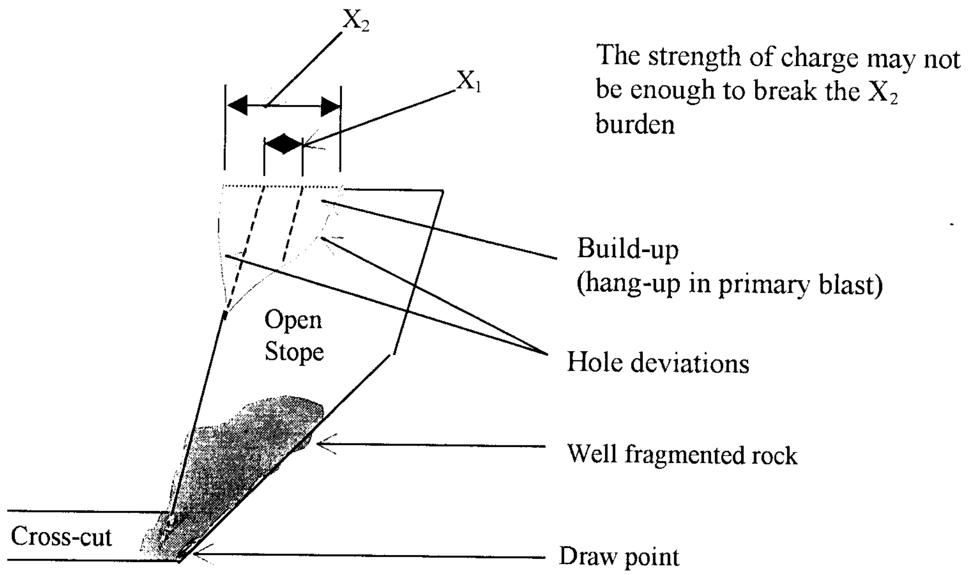


Figure 2.4 (b)

Figure 2.4 Illustrating the formation of (a) hang-ups and (b) build-ups as a result of hole deviations [After Sinkala, 1989].

2.2.1 Brief discussion of some effects of Hole Deviations

The following are brief discussion of effects of hole deviations [Sinkala, 1990]:

1. Increased explosive consumption

Blasting of build-ups, hang-ups and large size rocks in secondary blasting arising from unforeseen increased burden increase the consumption of explosives.

2. Time wastage

Frequent breakdowns associated with handling rough muck lowers the availability of ore handling equipment and facilities as well as the utilisation of man/machine hours.

3. Extra drilling

Build-ups and jammed holes entail re-drilling of extra holes. In the case of a jammed hole, where another hole is usually drilled next to the jammed one, the burden with the holes in the adjacent ring is altered.

4. Accelerated wear of ore handling equipment and facilities

The handling of poorly fragmented rock accelerates wear of ore handling equipment, especially on the contact surfaces. Accelerated wear reduces the life of the equipment and ore handling facilities. Ore handling equipment should handle well-fragmented rock rather than big chunks of rock.

5. Delays in the chain of production operations

Re-drilling and blasting of build-ups as well as hang-ups and large size rocks in secondary blasting interrupt the chain of production operations and result in increased cycle time. Sometimes, these delays can take days or weeks to rectify, and are costly.

6. Ore dilution

Holes deviating into waste rock or low-grade ore below cut off induce dilution. In some cases, tonnes of ore may be left in the stope due to a decision not to tram low grade, resulting from dilution. This may significantly lower the recovery from the stope.

7. Ore loss

Excessive diluted broken ore is usually left in stopes together with unbroken ore in situ. The quantities involved may be enormous especially where excessive dilution occurs at the early stages of stope draw.

8. Loss of drill strings

Jamming of drill strings in drill holes due to hole deviations is not uncommon in long drilling. The drill string, which includes drill rods, couplings and bits, is often lost together with the jammed hole. The lost drill string may involve some new components and a significant length. The loss of drill strings may also result in loss of production time due to attempts to claim the drill strings.

9. Extra mineral dressing work

A mineral processing plant is designed to operate efficiently for a certain range of ore grade and size. Poorly fragmented rock require more energy to liberate the minerals from gangue. Larger fragments require more stages of communition than those within the designed size range. Also, extra waste rather than ore tonnage is treated here.

10. Poor stability of rock excavation and pillars

Over-break increases the dimensions of excavations and reduces the sizes of pillars. The stress state induced by over-break is often manifested by unstable excavations and pillars especially at the abutments.

11. Reduced ventilation efficiency

The cross-sectional area of an airway affects the volume flow rate. Over-break increases the cross-sectional area of the airways and thus lowers the flow rate. The power and thus more energy required to achieve the desired flow rate are increases.

12. Safety

Drilling and blasting of hang-ups, build-ups and large rocks are hazardous operations. These operations have in some cases resulted in fatal and serious injuries to mining personnel. Minimising hole deviations may reduce some of these safety hazards.

13. *Others*

In development, inaccurate drilling of cut holes and perimeter holes results in poor advance and over-break/under-break. Over-break may result in wrong spatial positioning of development ends and re-designing the wrongly positioned ends altogether. Induced instability of development openings, holing into dangerous excavations or geological features, extra scaling down, inefficient mine ventilation are some of the other effects of hole deviations.

All these undesirable effects incur costs and require rectification for efficient and cost-effective mining operations.

2.3 Economic Consequences of Hole Deviations

As discussed in Section 2.1, many studies do exist on hole deviations. Possible sources and effects of the deviations have also been investigated. To this author's knowledge, however, no work has been done on quantification of extra costs due to hole deviations. It is envisaged that once quantification is done, unplanned 'hidden' mining operational costs would become apparent and efforts to save costs would be better appreciated.

CHAPTER THREE

3.0 THE FIELD STUDY AT ZCCM-NKANA DIVISION MINE

For this study to be meaningful, it was decided to carry out a field study at an operating mine. Through contacts with Zambia Consolidated Copper Mines (ZCCM), Nkana Division agreed to host the project.

3.1 Selection of the Study Area

The field study was undertaken at South Ore-body (SOB) shaft. SOB shaft has five operating Sections (Appendix N), each Section being under the charge of a mine Captain. The main study was carried out in Section 300. Nkana Division management selected the area under study based on hole deviation and related problems experienced in the Section. According to the management, the problems include:

a) delayed production due to extra drilling
b) jamming and extra use of bits and rods
c) poor rock fragmentation, and
d) high dilution.

3.2 Approach to the Field Study

The sequences of the field study and subsequent quantification of economic consequences were as follows:

3.2.1 Initial visit

Initial visit to ZCCM, Nkana Division to familiarise oneself with:

a) underground section of the study area

b) geology, mining methods and the related mining activities

c) monitoring of drilled holes, including measuring of hole deviations

The purpose of this was to collect necessary information to plan a detailed field study.

3.2.2 Detailed study

A detailed investigation of two selected operations as case studies. The selected operations were:

a) Stoping

This involved:

(i) examination of stope and drilling layout plans

(ii) observation of drilling operations

(iii) measurement of hole deviations

(iv) observation of blasting activities

(v) examination of fragmentation resulting from the blasts, and

(vi) examination of handling ore, crushing and metallurgical processes.

b) Raise-boring

This study involved monitoring raise boring operations including:

(i) examination of drilling plans and site preparation

(ii) observation of pilot hole drilling operations

(iii) measurement of pilot hole deviation.

The study of both activities (a) and (b) was aimed at understanding the drilling operations. The data gathered from these case studies were used to identify consequences associated with hole deviations. These consequences were then used to determine their associated economic implications.

3.2.3 Hole deviation consequences in monetary terms

Here, the study attempted to estimate the cost of hole deviation consequences and involved:

(i) identification of the method to be used in cost estimation

(ii) estimation of costs associated with 'stope drill' and raise pilot hole deviation consequences, and

(iii) development of models and 'MINCOST' computer program.

3.3 Background Information on Nkana and SOB Shaft, the Study Area

3.3.1 Nkana Division

3.3.1.1 Location

Nkana is situated approximately in the geographical centre of the Zambian Copperbelt. The Nkana mining licence, ML 3/2 covers 11,217 hectares in extent, and lies immediately to the west of the City of Kitwe. The mine area is located at latitude 12 degrees 54 minutes south and longitude 28 degrees 6 minutes east.

Nkana Division comprises two mines, namely Mindola and Nkana. Mindola mine has two shafts; Mindola number 1 and North Shafts, while Nkana mine comprises Central and South Ore-body shafts. Both these mines are supported by Technical and Geological departments (Appendix N).

3.3.1.2 The brief general geology of Nkana

The ore-bodies are primary sulphides, averaging 8 metres in width where the folding occurs and dipping to the west at an average of 60 degrees. The Mindola ore-body is regular with no folding, while the Nkana ore-body is invariably folded, especially at depth. The deposits are contained in a group of sedimentary strata in the Lower Roan group of the Katanga system.

The Lower Roan is made up predominantly of a footwall formation consisting of archaceous material, conglomerates, quartzites and sandstone followed by the ore formation of argillites which comprises dolomites, dolomitic shales and quartzite. The dominant ore minerals are chalcopyrite, bornite and carrollite.

3.3.1.4 Production and metallurgical processes

Nkana Division produces copper and cobalt from predominantly sulphide ores mined underground from four areas designated by shafts. The Division produces about 350,000 tonnes of ore a month of which more than half comes from Mindola mine.

Nkana Division has a concentrator, which operates differential flotation of copper and cobalt minerals. The smelter treats about 38 million tonnes a month of sulphide and oxide concentrates (including those from other Divisions). The refinery produces about 142,000 tonnes of electro-copper a month and casts about 16,000 tonnes of wire-bars. The cobalt plant produces up to 1,300 tonnes of electrolytic cobalt a year.

3.3.1.4 Mine Records

Production figures and records are under the auspices of Production Control Section. The mine records include:

a) Production estimates.
b) Production tonnage including stope production; tonnes of ore and waste trammed.
c) Stope control including reserve grade, box grade, dilution and recovery.

d) Contained copper and cobalt.

e) Development, production control of ore and waste.

f) Mine ore reserve estimates including drilled tonnage, fully developed tonnage and active tonnage.

g) Long hole drilling estimates including metres drilled per machine type.

h) Analysis of shaft production and costs for primary and secondary development, ore mined, stoping and services.

The Production Control Section in conjunction with operating officials, Planning and Survey Sections ensure the collection and accuracy of the figures and information related to production. The figures, when reconciled are sent to the central office as mine records.

3.3.2 SOB Shaft, the Study Area

The initial visit to ZCCM - Nkana Division was to familiarised with SOB Sections, particularly Section 300 which was selected (Section 3.1) as the study area.

In this section, background information on Section 300 which runs from 2620 ft-level to 3140 ft-level (Figure 3.1), north of SOB shaft is presented.

The information includes:

a) geology of SOB shaft area

b) mining methods and associated activities, and

c) hole deviations associated with stoping.

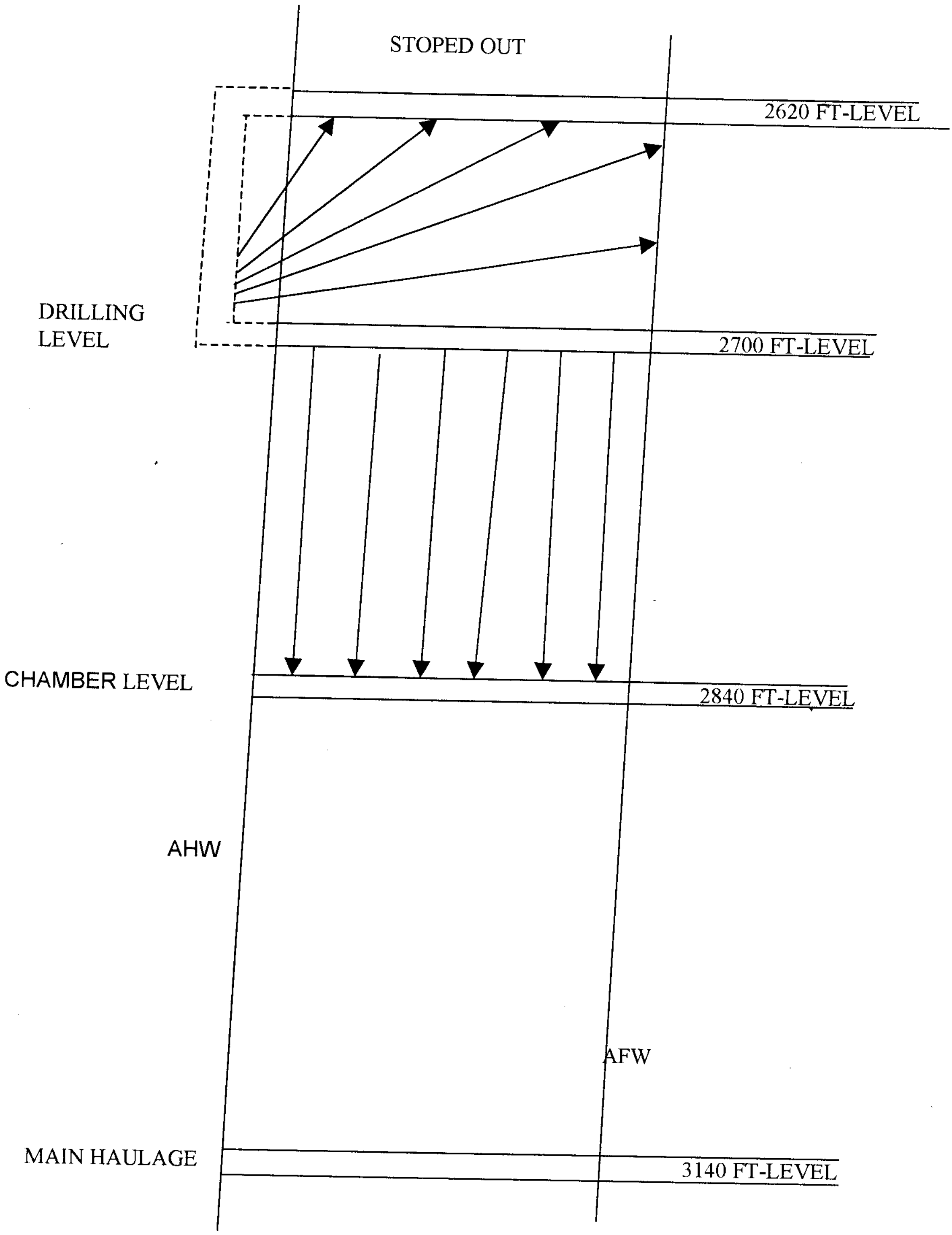


Figure 3.1 Sketch showing section along 2590N of Section 300

3.3.2.1 *Regional Geology of SOB Shaft*

The regional geology of SOB mining area is very complex and comprises deformed rocks of the late Precambrian and Katanga systems. The Granite and Lufubu Systems comprise Precambrian basement complex. Lufubu consists schists, gneisses and quartzites [Mendelsohn, 1972].

The Katanga system consists mainly of dolomitic limestone, dolomites, argillites and shales, with sandstone, quartzites and conglomerates.

Many minor folds complicate the South Ore-body near the nose of the syncline. The folds are tight and isoclinal. The grey contact shales range from a tremolitic shale or schist to a dense quartzitic argillite. In general it is a dense fine-grained quartizitic with beds of crystalline carbonate 25mm to 50mm thick [Jordan, 1972].

The contact with the overlying South orebody shale is generally sharp, but where the contact shale is a more massive argillite, the contact is gradational. The ore minerals, chalcopyrite and carrollite, are distributed as specks and fine grains in the carbonate beds. Some sulphides are disseminated throughout the rock.

The South ore-body shale is a black fissile shale with numerous quartz-carbonate veins and lenses ranging from paper-thin to several centimetres in thickness. The black colour is imparted by graphite carbon. The fissility is not parallel to any particular direction. Near the base, it is wavy and contorted, and associated with it are a number of veins. The wavy fissility and veining decrease upwards and the rock becomes a finely laminated black shale. This in turn grades into a massive black pyritic argillite.

The veins comprise quartz and carbonate, but in places contain felspar and mica. Most of the veins follow the fissility, but several cut it at high angles and some are irregular and curved. In many instances elongate carbonates have crystallised perpendicular to the walls of the vein in a comb-like structure.

The ore minerals, chalcopyrite and carrollite, occur mainly in the veins as interstitial blobs and stringers. A little sulphide is disseminated throughout the shale [Jordan, 1972].

3.3.2.2 Mining Methods

SOB shaft is currently using three main underground mining methods to extract sulphide copper and cobalt ores, namely; Sublevel Caving (SLC), Sublevel Open Stoping (SLOS) and Vertical Crater Retreat (VCR). These are briefly described below.

3.3.2.2.1 Brief description of SLC Mining Method

Conditions for use

At SOB shaft, SLC (Figure 3.2) method is used in areas where the ore-body is steeply dipping (>60°), and where both the ore and surrounding rock are weak to moderately strong. The design parameters in SLC are largely a function of caving mechanics. The mineralised and surrounding rocks fracture under controlled conditions. The breakage, collapse and gravitational flow of rock are all controlled.

Drifts

The sublevel Caving method used at SOB involves developing 2.4m by 3.5m drilling levels in between 3.5m by 3.5m main levels spaced at 12m to 14m metres vertically (Figure 3.2). Regularly spaced 2.4m by 2.4m cross cuts at the bottom of the stope are used as ore draw-points using loaders. The initial free face for stoping is created by developing a cut-through slot raise. This is later enlarged into a slot. Hand-held jackhammer drills are used in developing drifts.

Stope drilling

Gardner Denver and CH drifter machines are used for drilling long blast holes. The above drills work on percussive principle, using pneumatic energy. Stope blast holes are drilled in a fan pattern and vary in length, ranging from 3m to 17m.

Blasting

The blasting of the first fan of holes, breaks into the slot. Several rounds may later be blasted simultaneously to initiate caving. The explosives used to blast holes are ammonium nitrate fuel oil (ANFO) with ammonium gelignite as a primer.

Ore Handling

The ore handling from stopes consists of loading the ore from sublevel drives and cross-cuts. The ore drawn is transported to ore-passes and released into the same. LHD machines are used for these operations. The ore released in the ore-passes gravitates down to box chutes at the main haulage, after which, the ore is loaded on locomotive trains and transported to shafts for hoisting to the surface.

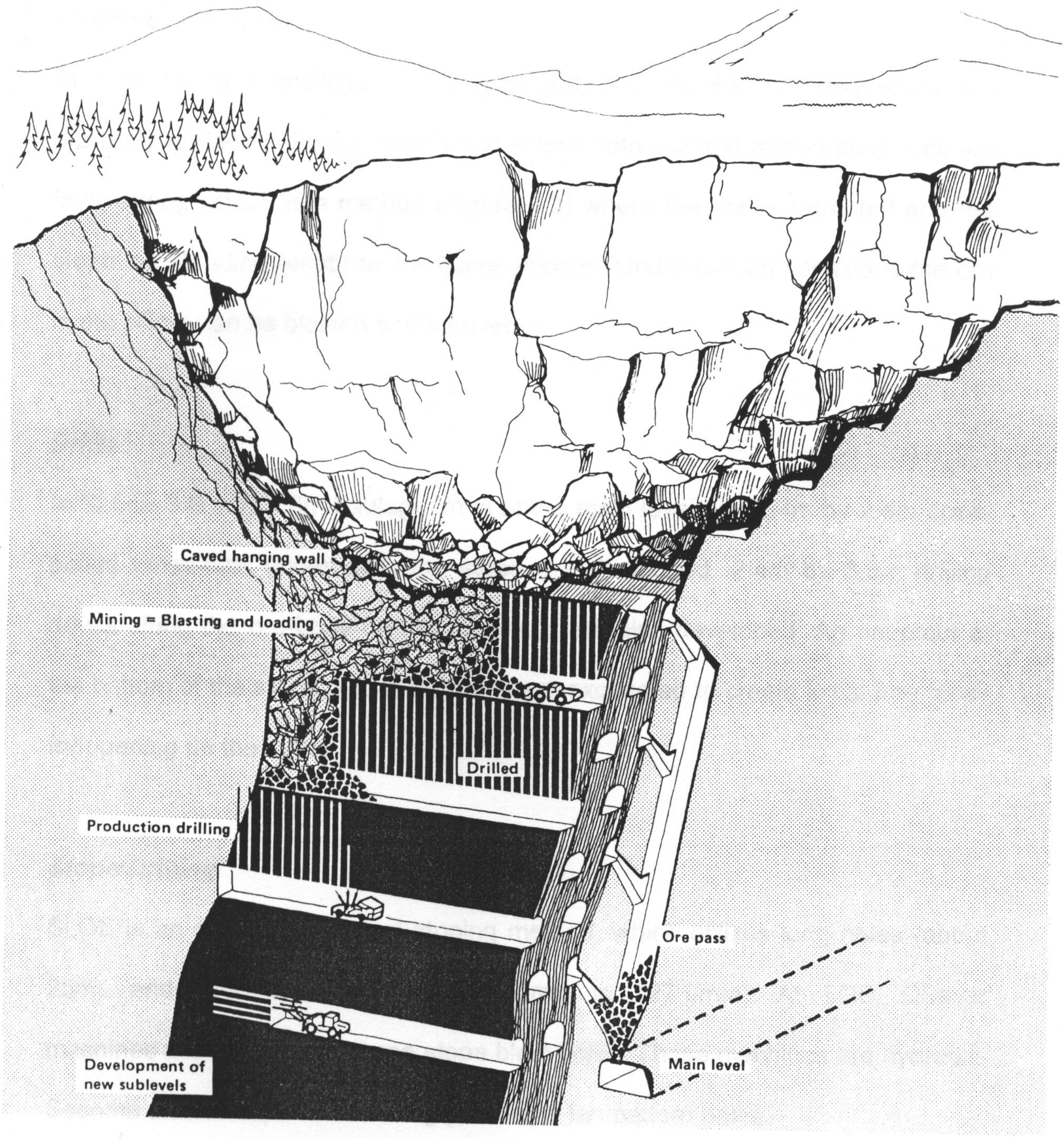


Figure 3.2 Sublevel caving mining method showing drilling level and draw-points [Source: Atlas Copco, 1987].

3.3.2.2.2 Brief Description of SLOS Mining Method

Conditions for use

At SOB, SLOS is employed in Sections 300, 700 and 800, in areas where the ore-body is steeply dipping (>55°) and where both ore and surrounding rock are fairly strong. SLOS is a method (Figure 3.3) where the ore is extracted and the stope left standing empty as the name implies. Under certain conditions the ore in the pillars can be blasted and recovered.

Drifts

In SLOS, 3.8m by 3.8m haulage drive at the main level and 2.4m by 2.4m draw-points cross-cuts underneath the stope are developed. 2.4m by 3.5m drilling drives along the ore-body on the sublevels are also developed. An undercut at the bottom of the stope, and a slot raise are excavated to create initial free faces for opening up the stope.

Stope Drilling

SLOS is an overhand, vertical stoping method, which utilizes long holes (about 25m), and large diameter holes (63.5mm to 127.0mm). At SOB, Quasar machines are used to drill these stope blast holes. These machines are hydraulic Secoma drifters capable of drilling parallel or fan pattern holes.

Blasting

Blasting of ore is carried out from sublevels. The blasting rounds break into the undercut and the slot. The explosives used to blast holes are slurry with ammonium gelignite as a primer.

Ore Handling

The blasted ore flows through the stope by gravity in the customary way to the haulage level. The ore is drawn at the haulage level through the draw points cross cuts at the bottom of the stope. The ore is mechanically loaded using LHD machines. The ore drawn is transported to ore passes and released into the same. The ore released in the ore-passes gravitates down to box chutes at the tramming level, after which the ore is loaded on locomotives trains and transported to shafts for hoisting to the surface.

3.3.2.2.3 Brief Description of VCR Mining Method

The VCR mining method (Figure 3.4) is similar to SLOS since both methods are variations of sublevel stoping.

Conditions for use

The VCR is the dominant stoping method at SOB shaft and is used in areas where the ore-body is steeply dipping (60^{o}). Both ore and surrounding rock should be fairly strong.

Drifts

In VCR, 3.8m by 3.8m haulage drive at the main level and 2.4m by 2.4m draw-points cross-cuts underneath the stope are developed. 2.4m by 3.5m drilling drives along the ore-body on the sublevels are also developed. An undercut at the bottom of the stope is excavated to create initial free face.

Stope Drilling

The stope is drilled from the drilling level at the upper level. Parallel holes are drilled through to the undercut (Figure 3.4). The Cubex machine is used to drill blast holes. The Cubex machine is an in-the-hole (ITH) hydraulic drill, which applies percussion directly on the bit inside the drilling hole. The blast holes vary in diameter from 115mm to 215mm.

Blasting

The holes are charged and initiated from the top at the drilling level. The VCR is an overhand method and the blasting of ore is done in slices. The blasting rounds break into the undercut. The explosives used to blast holes are slurry power plus with magnum 365 as primer.

Ore Handling

The ore handling from the draw-points to surface is the same as explained for SLOS in Section 3.3.2.2.2.

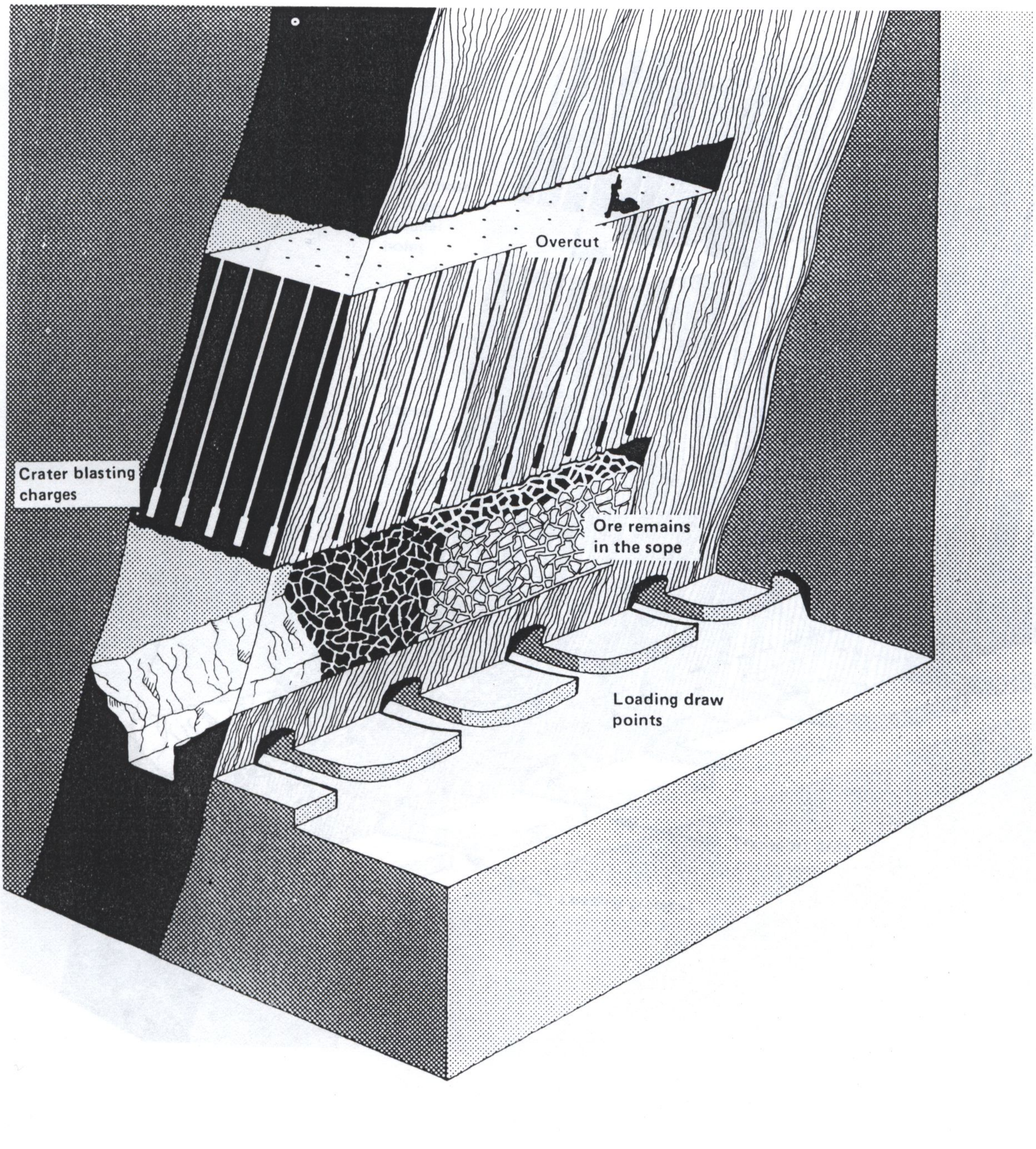


Figure 3.3 Illustrating sublevel open stoping mining method [Source: Atlas Copco, 1987]

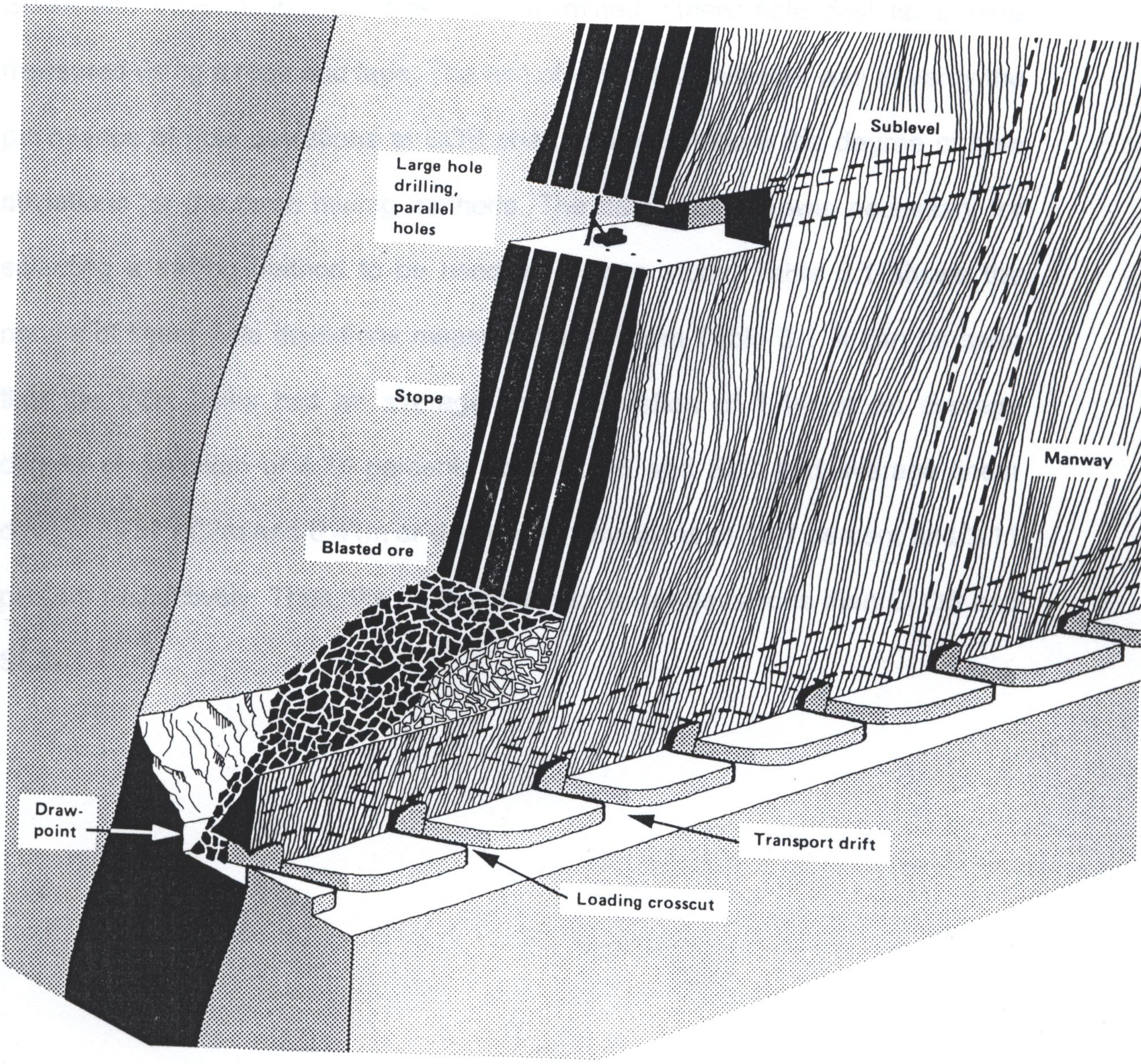


Figure 3.4 Illustrating the VCR mining method [Source: Atlas Copco, 1987]

3.3.2.3 Hole deviations associated with the mining methods

Intended and actual holing points for some stope production holes for SLC, SLOS and VCR mining methods were examined. Linear hole deviations were measured using a rope and tape. The aim of this initial fieldwork was to verify the prevalence of hole deviations at SOB shaft and to compare the degree of hole deviations for the three mining methods. The data obtained was necessary for selecting a mining method to be used as a case study. Table 3.1 shows the results of linear hole deviations measurements. From Table 3.1, it can be seen that the VCR holes had an average linear deviation of 1.4m or an average degree of deviation of 4.36 percent, while SLOS and SLC had average linear deviations of 1.02m and 0.41m or degrees of deviations of 3.67 percent and 3.11 percent respectively. Thus, the VCR holes had larger deviations than SLOS and SLC.

Table 3.1 Measured linear hole deviations

Activity	Period	Hole No.	Hole Length (m)	Linear Deviation (m)	Degree of Deviation (%)
2540 BCN Cubex VCR Stope Drilling	August to October 1998	A1	30.8	1.4	4.54
		A2	31.0	0.4	0.01
		B3	29.8	2.1	7.05
		B4	29.5	1.7	5.76
		C5	30.7	0.8	2.61
		C6	31.0	1.9	6.13
		D7	29.7	2.2	7.41
		D8	30.4	0.2	0.66
		E9	30.5	2.0	6.56
		E10	29.8	1.2	4.03
		F11	42.5	1.8	4.23
		F12	41.8	1.4	3.35
Averages			**32.2**	**1.4**	**4.36**
2260L Quasar SLOS Stope Drilling	October to Nov. 1998	1	23.5	0.9	3.83
		2	23.0	0.2	0.87
		3	22.6	1.0	4.42
		4	22.2	1.2	5.40
		5	21.9	0.6	2.74
		6	31.4	0.8	2.55
		7	31.2	1.3	4.17
		8	30.9	1.4	4.53
		9	30.0	0.7	2.33
		10	31.7	0.5	1.58
		11	30.8	1.7	5.52
		12	31.0	1.9	6.13
Average			**27.52**	**1.02**	**3.67**
2880 to 2840L Quasar SLC Stope Drilling	Nov. to Dec. 1998	S1	8.0	0.2	2.25
		S2	8.8	0.3	3.41
		S3	11.5	0.2	1.74
		S4	12.9	0.4	3.10
		S5	13.2	0.5	3.79
		L1	15.4	0.7	4.54
		L2	15.2	0.6	3.95
		L3	15.0	0.5	3.33
		L4	14.6	0.4	2.74
		L5	14.4	0.6	4.17
		C1	14.1	0.5	3.55
		C2	13.6	0.3	2.20
		C3	13.2	0.4	3.03
		C4	12.4	0.2	1.61
		C5	12.1	0.4	3.31
Average			**12.96**	**0.41**	**3.11**

3.4 Choice of VCR Mining Method in Section 300, SOB Shaft for Detailed Study

From the data gathered during the initial visit to SOB shaft, it was concluded that hole deviations and their consequences associated with VCR mining method be selected for detailed study due to the following reasons:

a) The method exhibited higher degree of hole deviation (Table 3.1) than SLC and SLOS. With respect to hole length and rock type, it was expected that VCR holes would exhibit low degree of deviation as compared to SLC and SLOS because VCR holes were vertically drilled and had larger diameter (165mm) as compared to SLC (57mm) and SLOS (127mm). The magnitude of hole deviation decrease with an increase in hole verticality and diameter [Sinkala, 1989]. The observed occurrence of large hole deviations in VCR method confirms the concerns by Nkana Division management over the VCR stoping problems caused by hole deviations.
b) With reference to SOB regional geology (Section 3.3.2.1), the ore-body characteristics and geotechnical data are favourable for the use of VCR (Section 3.3.2.2.3). If hole deviations can be minimized, the VCR stope height can be increased to reduce sublevels, thereby significantly minimising operational costs (Figure 1.1) compared to the other methods.
c) Because of large stope heights associated with VCR stopes, any hole deviations give rise to much greater consequences than other methods.

d) Nkana management has decided to promote the use of VCR method because of its associated advantages over other methods. The advantages include:

- Reduced and simple development
- Less labour intensive
- Easy to mechanise
- High recovery
- Broken ore gives wall support, and
- Safe working areas.

CHAPTER FOUR

4.0 HOLE DEVIATIONS AND THEIR CONSEQUENCES ASSOCIATED WITH VCR MINING METHOD

Detailed studies of hole deviations of the two selected operations and their consequences associated with the VCR mining method as case studies were carried out at SOB shaft in Section 300 in VCR stoping area (Section 3.3.2.2.3). The selected operations were 2590N stope drilling and 2430N raise pilot hole drilling.

4.1 The General Principle of VCR Mining Method

In addition to the description in Section 3.4.3, the principles behind the VCR method utilize a unique blasting technique, the crater blasting. In VCR stoping, the ore is excavated in horizontal slices, and the stoping starts from below and advances upwards. The broken ore remains in the stope to support the walls. The ore is recovered at the bottom of the stope, through a draw-point system (Figure 4.1). The development for VCR stoping consists of:

a) Over-cut of the full stope area at the upper level of the stope (Figure 4.1). The over-cut serves as drilling excavation for DTH machines. The drill holes are normally parallel and vary from 127mm to 170mm in diameter.

b) Under-cut of the complete stope area, at the bottom of the stope. This serves as initial free face.

c) Haulage or loading drift along the ore-body at the draw-points level. The draw-points loading arrangement at the bottom of the stope involve cross-cuts, drives and the main haulage.

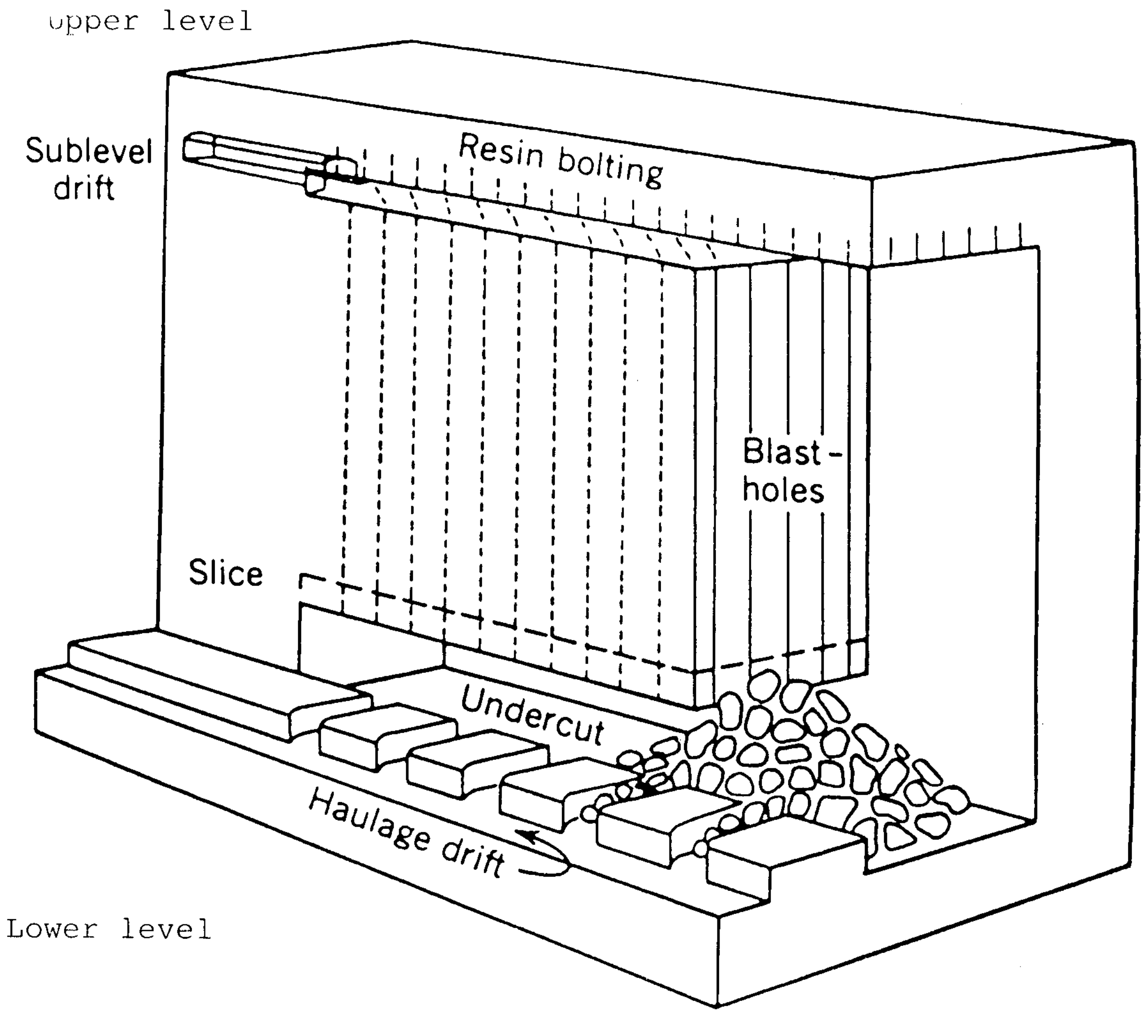


Figure 4.1 Illustrating the VCR mining method stope layout
[After Hartman, 1987].

4.2 The VCR Stope Studied at SOB Shaft

The stope studied in detail at SOB shaft was 2590BCN (see stope data in Tables 4.1 and 4.2). The reasons for choosing to study this particular stope were:

a) Preparatory work had just started giving the author an opportunity to follow all stages of ore extraction.
b) The VCR mining operations at 2590N/2880L were a reflection of general VCR procedures of ore extraction taking place at SOB shaft.
c) The area was accessible, making it easy to monitor the operations.
d) The preparatory drilling work for 2430N ore tip raise to service ore extraction from the stope had just started giving the author an opportunity to monitor the raise pilot hole drilling.

Table 4.1 2590N/2880L Stope Data [Source: ZCCM, Nkana Division, Mine Planning section, 1998]

Strike length:	28.0m
Stope length	17.0m
Crown pillar length	11.0m
Back stope length	52.0m
Thickness	15.8m
Rib pillar tonnage	19,633
Crown pillar tonnage	12,812
Stope tonnage	23,486
Total reserve tonnage	55,931
Total drilled tonnage	55,931
In situ contained copper	1,454.2T
In situ contained cobalt	44.7T
Reserve copper grade	2.60%
Reserve cobalt grade	0.08%

Table 4.2 2590BCN Ore body characteristics

Ore body size	≈ 16m
Ore body shape	Tabular and moderately regular
Ore body strength	Fair
Ore body depth	886.0m
Dip	82 degrees
Rock strength	Good

The 3-Dimensional diagram of 2590N/2880L ore-body is shown in Figure 4.2. The diagram shows the stope and pillar dimensions and ore-body dip.

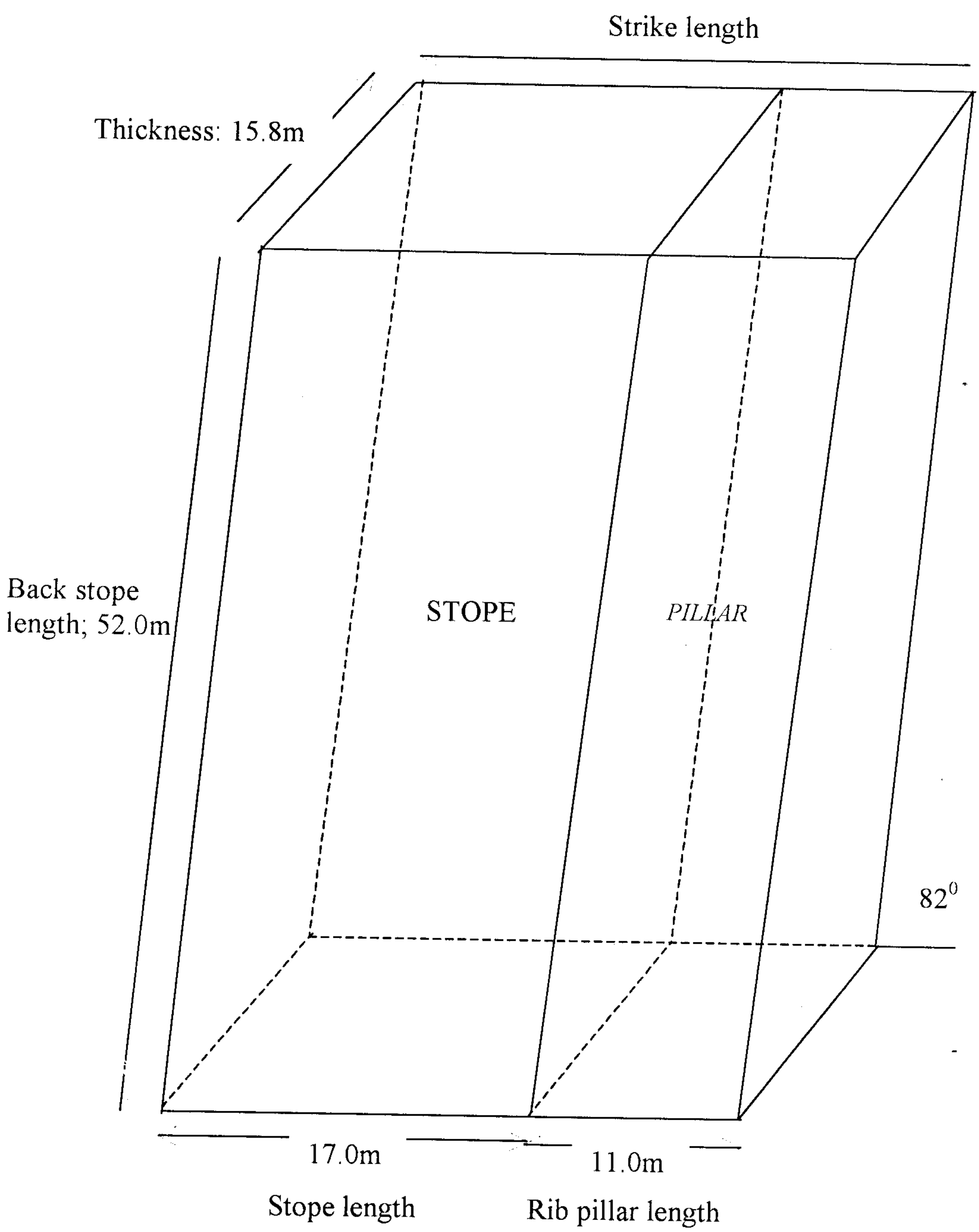


Figure 4.2 Sketch showing 2590N stope dimensions

4.2.1 Geology of the study stope area

The geology of the study area is similar to the regional geology of the SOB shaft as described in Section 3.3.1.4.2.

In this area, the rock formations are mainly banded shale and footwall sandstone with average rock compressive strength of 140MPa (varying from 90MPa for shale to 190MPa for sandstone). The formations are almost wholly composed of mica grains having a preferred orientation. Some fine-grained quartz occurs interstitially with the mica. The ore-body shale contained 38.41 percent quatrz and 26.31 percent calcite while basal sandstone contained 27.37 percent and 38.27 percent calcite [Nkana Division, Geology Department, 1998].

4.2.2 Rock Survey of the Study Area

The rock survey was carried out at SOB mine in the study area to determine Rock Quality Designation (RQD), Intact Rock Strength (IRS), Rock Mass Strength (RMS), Joint sets and Joint number (Js and Jn). The results were used for Rock Mass Classification (RMC) and Rock Rating (RR). Table 4.3 shows the survey results of rock classification and rating.

Table 4.3 Rock Survey Results

ADJUSTED DATA INPUT					
ROCK	RQD	IRS	JS_1	JS_2	JS_3
HWA	97	100	0.27		
HWA	82	120	0.47		
HWA	93	121	0.22	50	
HWA	98	89	0.29		
SOSH	88	56	0.07	19	
SOSH	100	83	0.29		
SOSH	100	93	0.31	1.3	

ROCK UNADJUSTED RATING								
ROCK	RQD	IRS	JS_1	JS_2	CLASS	RATE	RMS	DESCR.
HWA	15	10	20	22	2b	66	56	Good
HWA	12	12	22	23	2b	69	68	Good
HWA	14	12	14	24	3a	63	62	Fair
HWA	15	9	20	23	2b	67	52	Good
SOSH	13	6	9	23	3b	50	25	Fair
SOSH	15	8	20	23	2b	66	48	Good
SOSH	15	9	16	23	3a	64	51	Fair

The geology and the rock survey show varying rock formation, RQD and RMS. One of the problems that might occur when drilling through rock of irregular hardness is that if the same thrust is used (or if constant penetration rate is maintained), differential rock disintegration at formation boundaries is likely to occur when traversing from soft (shale) to hard (sandstone) rock. The differential rock disintegration causes centre-line of drill bit (and hence drill rods) to be eccentric to line of thrust acting on shank [Sinkala, 1987], resulting in earlier buckling or bending of drill strings. The result is often hole deviations.

4.2.3 The general development of the stope

The drilling area for 2590N stope was conventionally developed south of footwall drive at 2700 level (Figure 4.3). Instead of one opening, the area consisted of the main excavation for rings 3 to 8 and a cross cut for rings 1 and 2 (Figure 4.7). This was to avoid a large opening in a moderately weak formation. Ideally a VCR stope should have only parallel holes and thus the drilling level should be completely opened as a slice to allow full freedom of drilling positions. Ground conditions at SOB do not allow such wide openings and therefore compromise was made as required. The undercut was developed between the two cross cuts (Figure 4.5) which also sufficed as draw points on 2840 chamber level.

Because of the weak rock formation, the 2700 level was supported with 4.5m long cement grouted cable bolts at 2m intervals and with 2.4m long grouted wire ropes/split sets at 1m centres. In addition, all excavations were supported with grouted wire ropes, wire mesh and lacing.

The 2430N tip raise chamber or drilling excavation on 2840 ft-level was mined by drilling with jackhammer and blasting. The excavation was developed north of footwall drive (Figure 4.4). The drilling area was enlarged from 3.5m by 3.5m to 9.6m wide and 6.1m high to provide space to accommodate derrick, electrical and hydraulic raise-borer assemblies, and for cylinders to move up and down freely. The holing position was planned to be on the side of already developed 3.5m by 3.5m main haulage on 3140 ft-level (Figures 4.6 and 4.23).

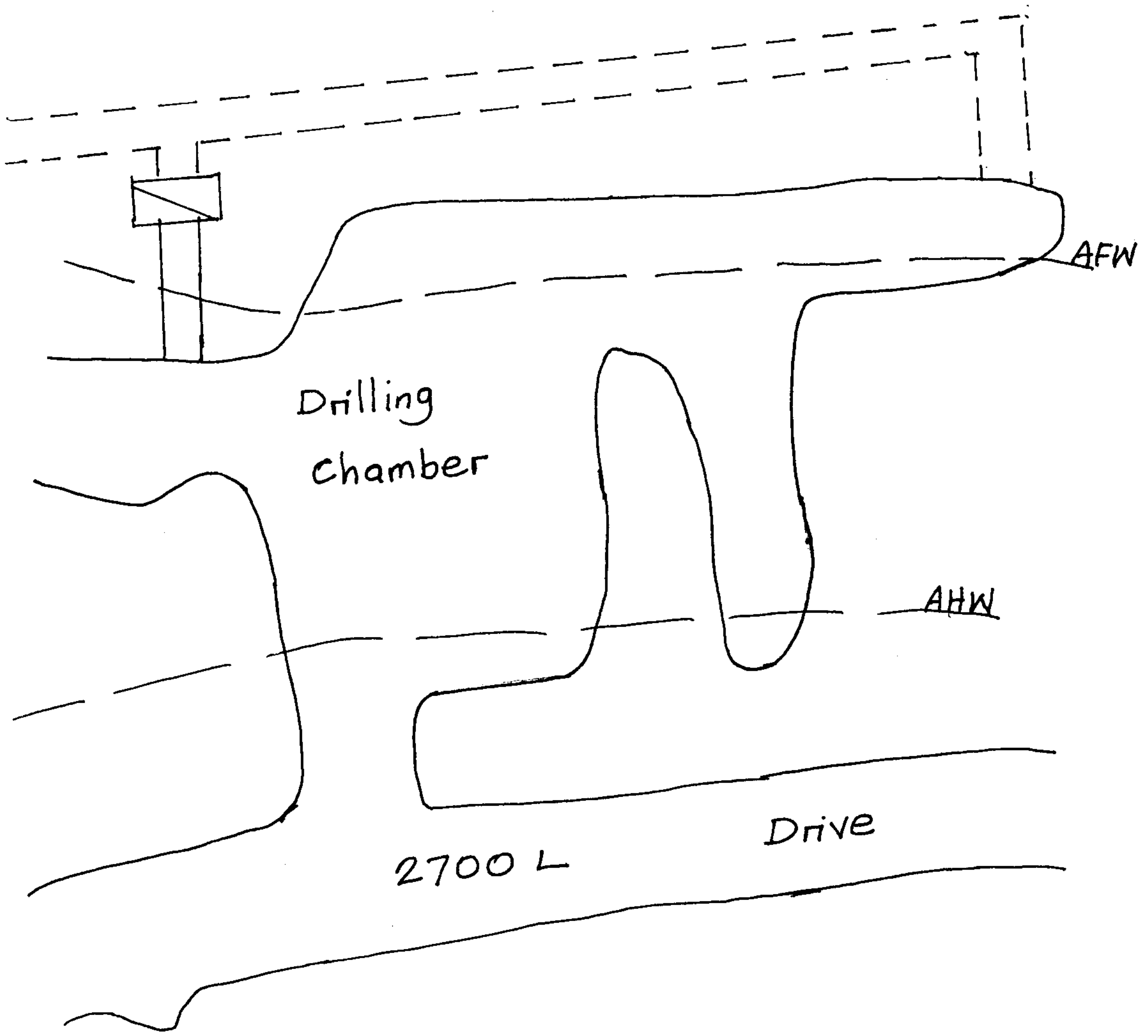


Figure 4.3 Drilling position for 2590N stope

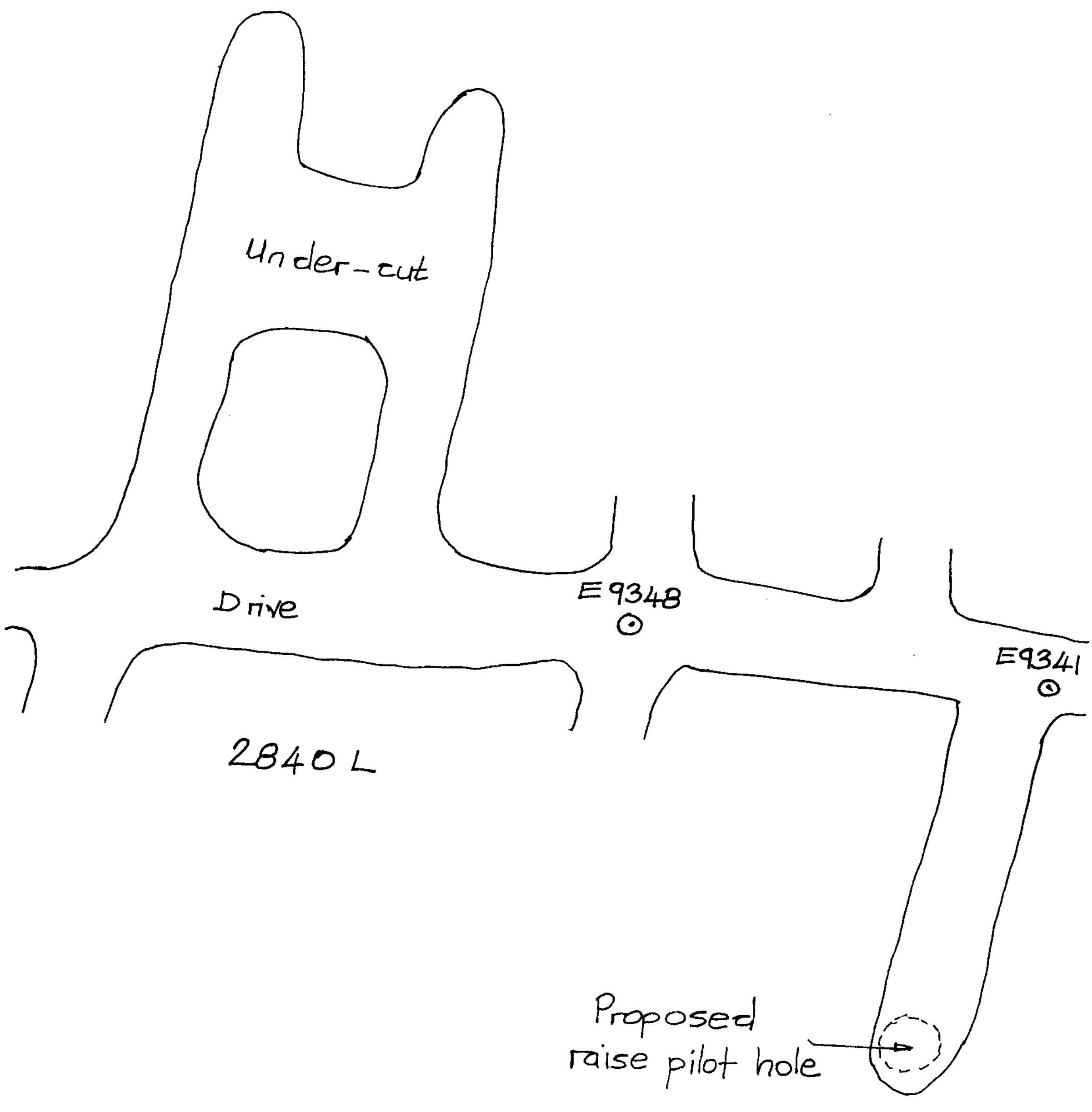


Figure 4.4 Stope under-cut and proposed raise pilot hole position at 2840ft Level

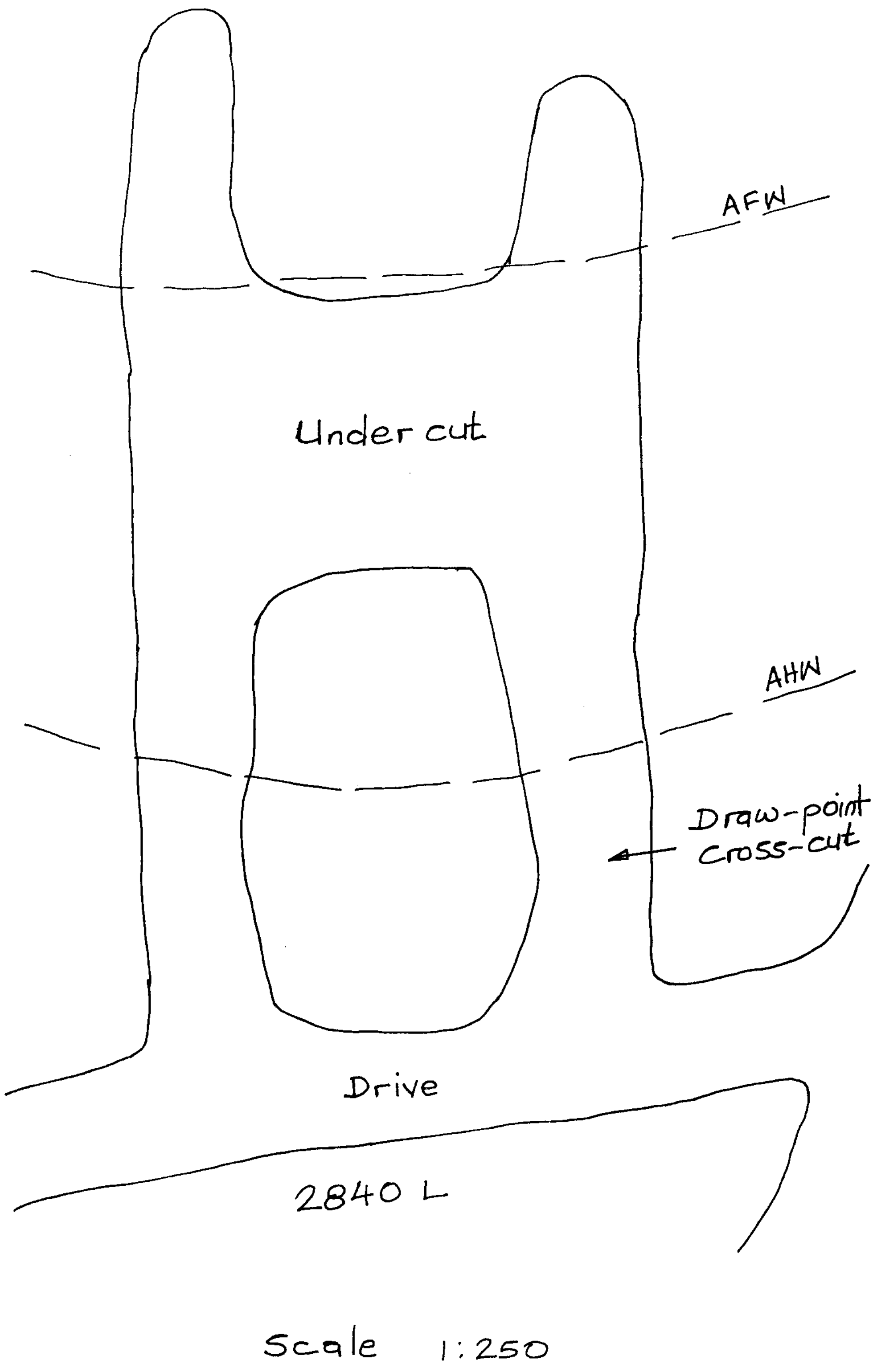


Figure 4.5 Raise pilot hole drilling position

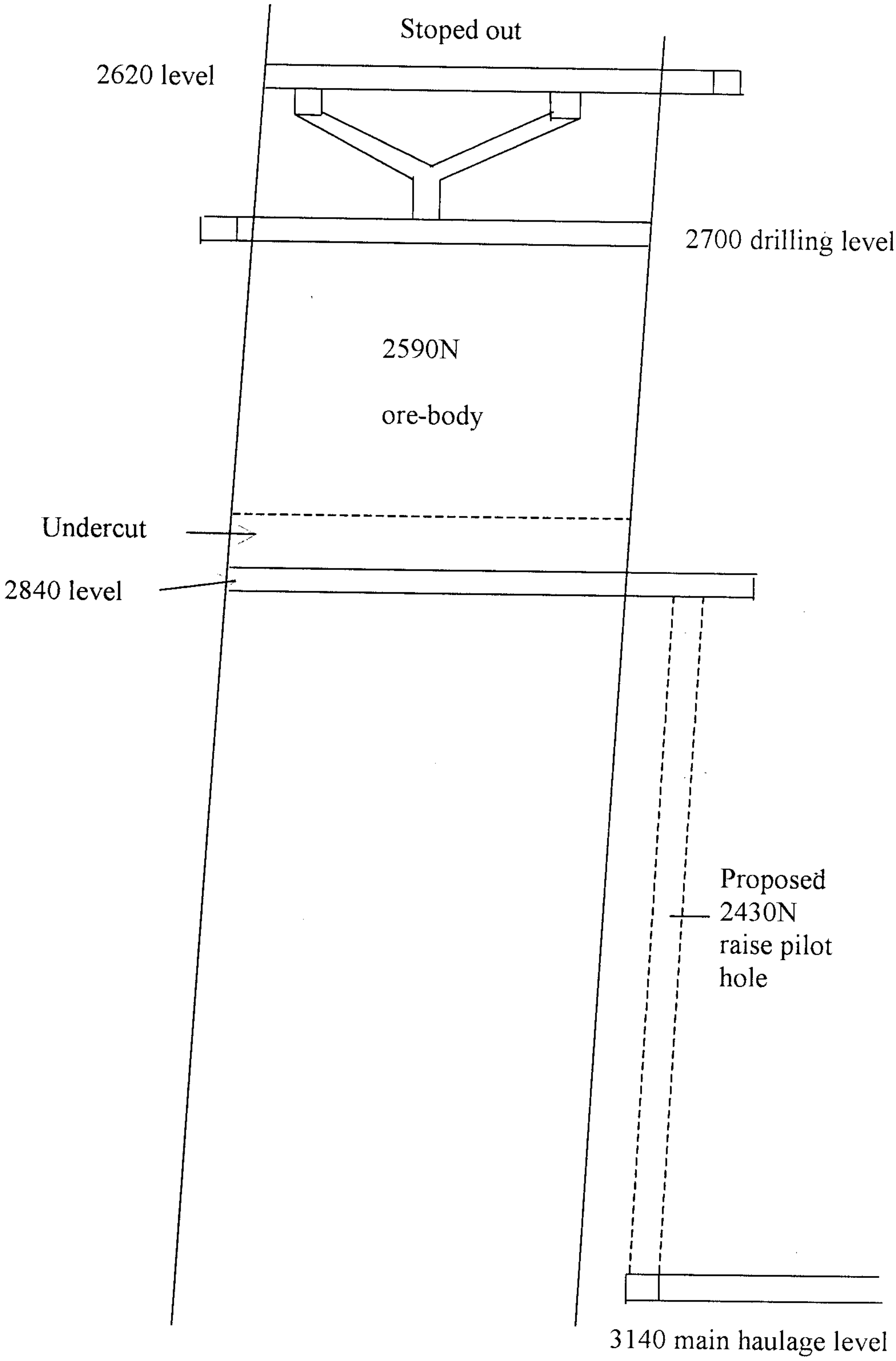


Figure 4.6 2590N section showing drilling level and undercut

4.3 The Drilling of the Stope and Raise Pilot Holes

The actual study considered two types of drill holes:

a) Stope production holes, and

b) A raise pilot hole.

The drifts in the study area were developed prior to commencement of the study. Nevertheless, information was obtained from Survey Department on the actual profile of the drifts for comparison with planned ones. This information was required to relate the planned and actual positions of drill hole rings. It was observed that the actual and planned profiles were not the same. The difference between the actual and planned profiles was scaled and the maximum off-centre distance was 0.6m. Also examined were the following:

a) drill hole plans prior to drilling of the stope and pilot hole

b) setting out of drilling equipment.

On completion of drilling, holes were measured for deviations. Consequences of the drilling operation were observed both during and after their executions.

4.3.1 The stope drill holes

4.3.1.1 *Examination of drilling plans*

Examination of the 2590N stope drilling plans showed that ring 2 (with 4 holes) and ring 3 (with 5 holes) were to be drilled at incline (Figure 4.8) since the drilling *level was not fully sliced (Section 4.2.3). In VCR stoping, holes are normally parallel and vertically drilled (Section 4.1). Apart from rings 2 and 3, the rest were* planned to be drilled vertically. The individual hole inclinations and lengths (Table 4.5) of all the rings were checked and were found to match the stope dimensions, as shown in Figure 4.2. On scaling the toe-burdens on the drilling plans for the stope, it was found that the dimensions matched those provided in the 4th column in Table 4.8 (planned toe burden). 63 holes were planned for the whole stope. 32 holes with total length of 1088.26m (Table 4.5) were planned to be drilled for VCR chamber. Recovery rings had 18 holes while wrecking ring had 13 holes.

4.3.1.2 *Observation of drilling operations*

The drilling area was lashed to solid before setting-out ring positions. The drill hole ring positions, which for the 2590N VCR stope were planned at intervals of 3.5m, were set-out by stope planners. The ring positions were set-out using survey pegs to establish ring lines. The marks for each line were painted by free hand on the chamber excavation walls. Before drilling of holes started, the drill, Cubex ITH machine was transported to the site and positioned on ring 1 centre line using the marks on the walls. Because of uneven foot-wall, it was difficult to rigidly rig the drill. After rigging the drill, the inclination for each particular hole was set using an improvised instrument Figures 4.11, 4.12 and plate 1). The techniques used were examined. It was found that the protractor on this improvised instrument was graduated in 5 degrees, making it difficult to obtain the exact readings in between graduations. It was also observed that when adjusting the drill to the required inclination, the drill used to shift off the centre line. The shifting ranged from 0.05m (minimum) to 0.15m (maximum).

Table 4.5 Planned drilling information for 2590N stope

Chamber Rings	No. of Holes	Planned Length (m)	Planned Inclination (Degrees)
1	4	36.58	90
2	4	36.58	Varying
3	5	20.12	Varying
4	4	36.58	90
5	4	36.58	90
6	4	36.58	90
7	3	36.58	90
8	4	36.58	90
Sub Total	**32**	**1088.26**	
Recovery Rings			
1	9	Varying	Varying
2	9	Varying	Varying
Wrecking Ring			
1	13	Varying	Varying

Planned Stope Information

Toe burden: 1.5m for chamber rings

3.5m for chamber peripheral holes

4.0m for recovery ring 1

2.0m for up holes of recovery ring 2

4.0m for down holes of recovery ring 2 and up holes of wrecking ring

6.0m for down holes of wrecking ring

Ring burden:	3.5m for chamber rings and recovery ring 1
	3.5m for down holes of recovery ring 2
	1.9m for up holes of recovery ring 2
	4.0m for up holes of wrecking ring
	7.0m for down holes of wrecking ring
Hole diameters:	165mm for chamber rings, recovery ring 1, down holes of recovery ring 2 and down holes of wrecking ring.
	51mm for up holes of recovery ring 2
	127mm for up holes of wrecking ring
Drilling machines:	DTH-Cubex for 165mm and 127mm diameter holes
	CH Drifter for 51mm diameter holes.

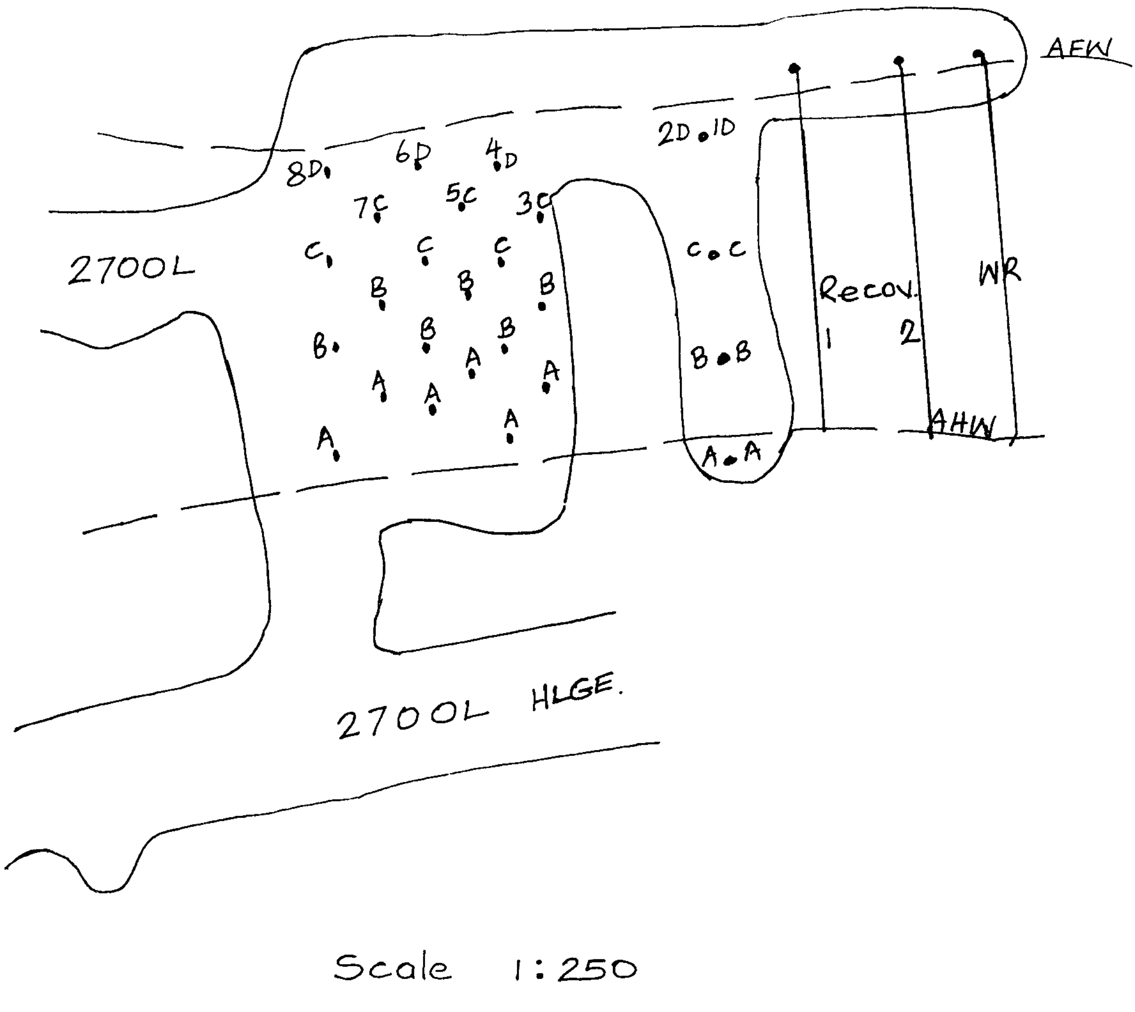
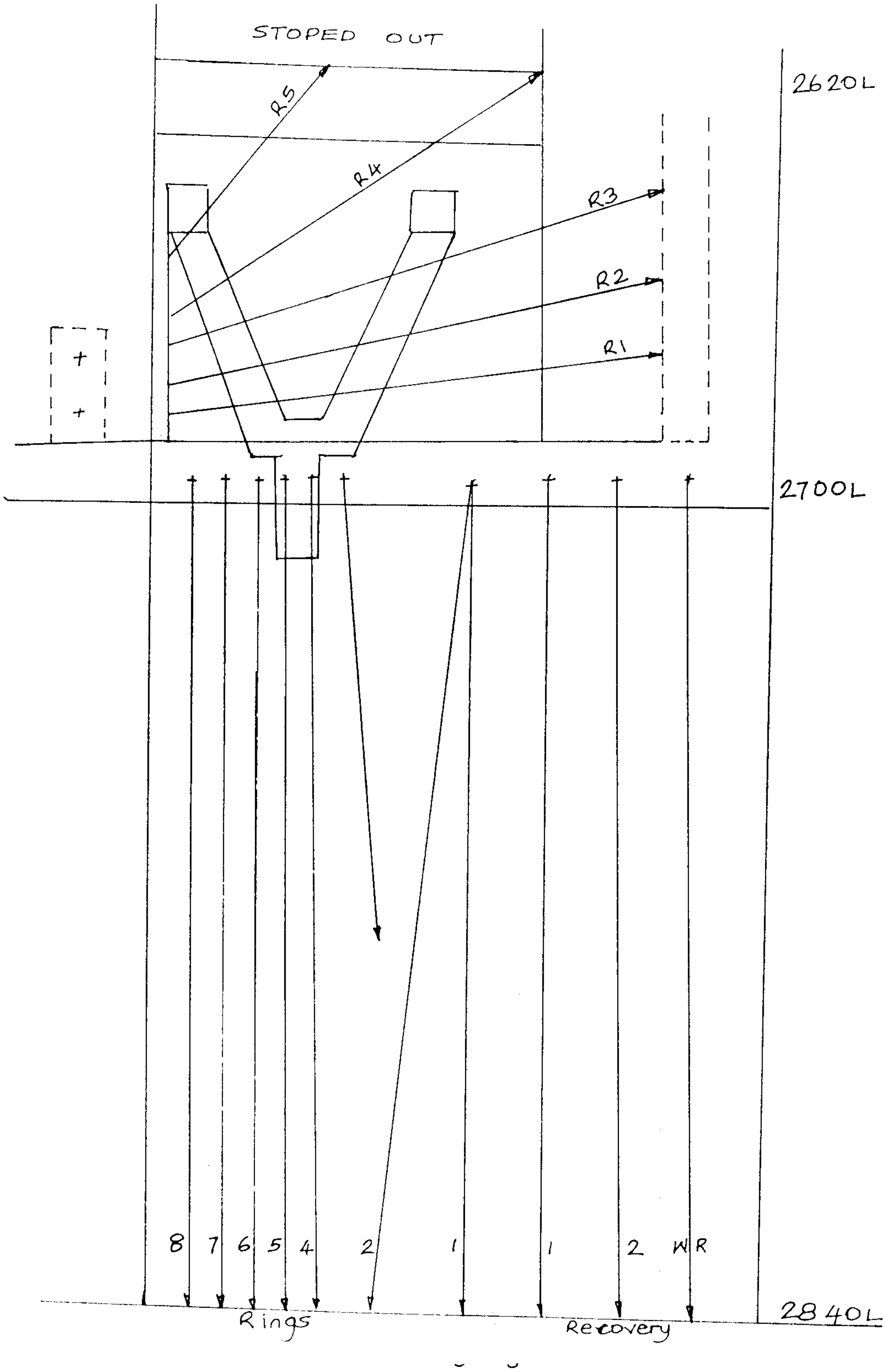


Figure 4.7 Stope ring positions on 2700 level


STOPED OUT
2620L
R5
R4
R3
R2
R1
2700L
8
7
6
5
4
2
1
1
2
WR
2840L
Rings
Recovery

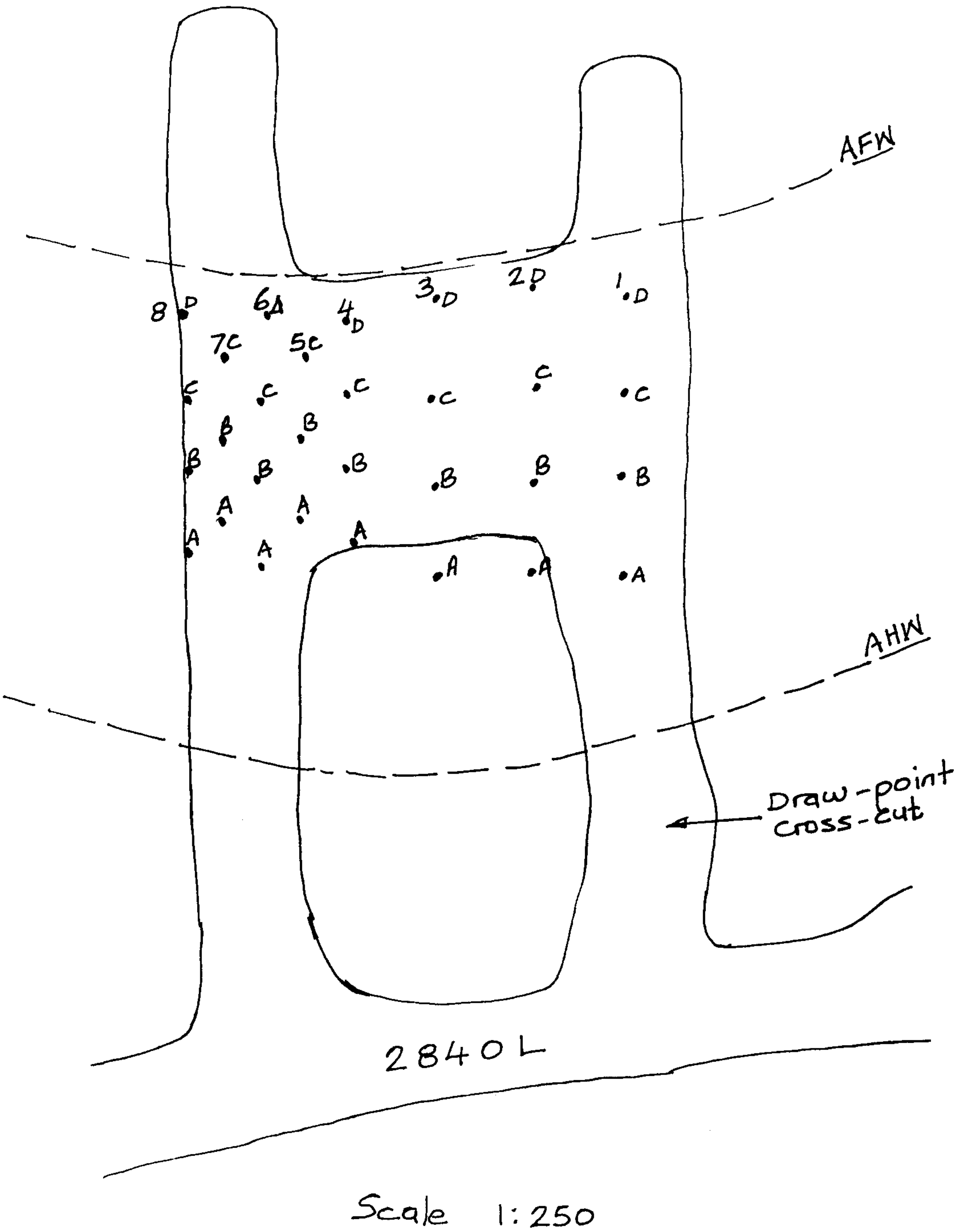


Figure 4.9 Hole positions on chamber level

After setting-up the Cubex drill, the actual drilling of the stope holes commenced, starting with chamber ring 1, hole number 1A (Table 4.5 and Figure 4.7). The holes were drilled with 165mm button bits. The drilling pressure for collaring the holes was provided by the drill string weight of about 85kg. A mixture of water and air were used as a flushing fluid.

The drilling of all the holes was observed and drilling parameters (Table 4.6) were recorded.

Table 4.6 Showing Recommended and actual Parameters.

PARAMETER	RECOMMENDED	ACTUAL
Operating air pressure	15 bar	8 bar (average)
Thrust range (minimum and maximum)	850 - 1,800kg	700 - 2200kg
Flushing - Air velocity (minimum)	20.5 metres/sec.	12m/s (average)
Water flowrate (minimum)	25 litres/minute	10 - 15 l/minute
Rotational speed	10 - 100 R.P.M	5 - 90 R.P.M
Rate of penetration (maximum)	54metres/shift.	32metres/shift
Hydraulic oil used per shift	5 litres	10 - 20 litres

The actual air pressure and thrust were monitored at least 5 times per shift by reading the gauges attached to the machine and the average pressure and range of thrust force were then obtained. Air velocity and water flow-rate were measured along the air and water columns at points next to the drill. Rotational speeds were monitored by marking a point on the rod and counting the times it rotated per minute. This was repeated every after two rods and the average range was obtained. The rates of penetration were monitored by observing the time taken to drill one rod. This was done when the drilling was at full throttle. The average penetration rate was obtained from 19 observations.

From Table 4.6, it can be seen that the air pressure, air velocity and water flow-rate were below the recommended levels. The maximum effective rate of penetration observed was 32m per shift (0.09 metres per minute), as compared to the expected 54m per shift (0.15 metres per minute) for the rock formation in the study area (Section 4.2.1). In addition to the above deficiencies, high consumption of hydraulic oil was observed (Table 4.6). The high consumption was due to leakage from the cylinders and hydraulic assembly. This resulted in delays, as drilling had to be stopped whenever refilling of oil was taking place. Each shift was losing 7 to 15 minutes due to this alone. Apart from the oil leaks, it was also observed that the Cubex machine which was employed for drilling stope holes, had the following defective components:

a) lags, spacer and keeper plates were worn-out
b) stinger cylinders were not working, allowing the rig to wobble
c) drilling table and centralizer were abraded, and
d) slip spanner and drive head shaft were not effectively clamping to drive the rods.

4.3.1.3 *Measurements of hole deviations*

During the fieldwork, only linear hole deviations were measured using a flexible metric tape. The following procedure was used to determine the accuracy of holes:

Using a drilling layout and survey pegs, the planned collaring and holing points were established. The holing points were marked. After holes were drilled, the deviations (the linear distance between the actual and intended holing positions) were measured. Figure 4.10 shows an example of marked holes on drilling and holing levels.

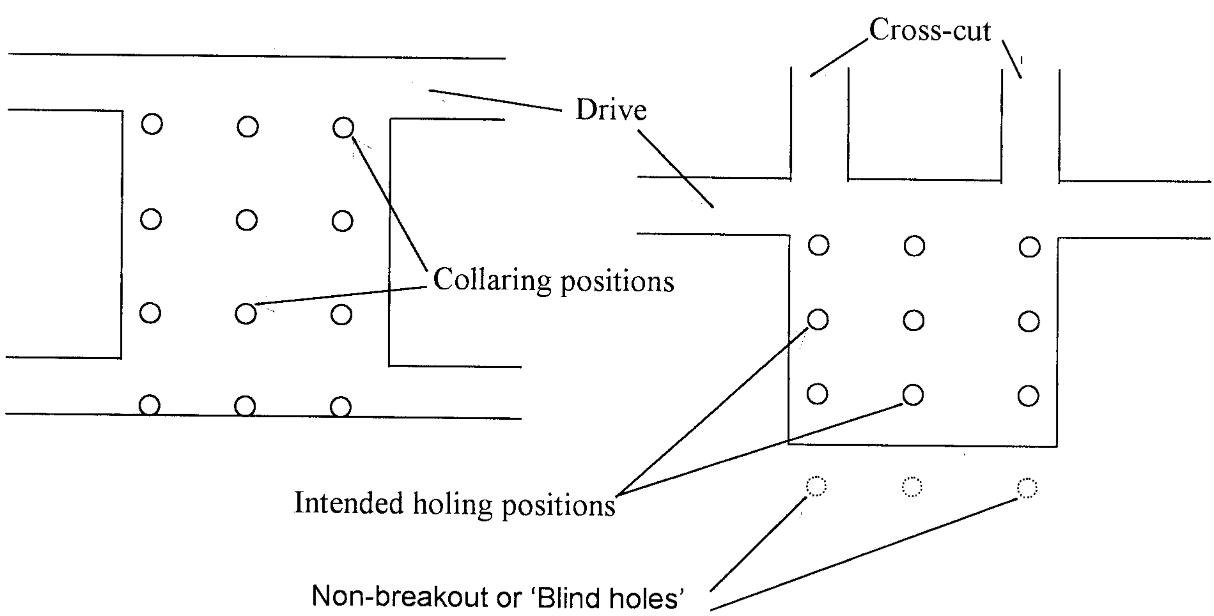


Figure 4.10 Holes positions on (a) drilling and (b) holing levels.

The actual drilled hole lengths, toe-burden and inclinations were also measured. The lengths were measured by dropping a pre-marked twine with a weight in each hole. The actual toe-burden was a linear distance measured at two hole ends. The inclinations were measured using an improvised instrument (Figure 4.12 and plate 1). The plastic tube was pushed into the hole and aligned with hole 'foot-wall' (Figure 4.11). The inclination was then read on the protractor fixed on the top part of the tube. The measured inclinations, lengths and toe-burdens are shown in Tables 4.7 to 4.11. Note that hole numbers starting with E represent remedial (extra) drill holes, mainly to reduce large burdens caused by inadvertent hole deviations.

There were two types of unplanned extra drilling, which were studied:

1. Remedial, and
2. Unauthorized.

Remedial Extra Drilling

The remedial extra drilling was the additional drilling to the initial planned drilling layout. This happened when the stope was not properly drilled because of problems such as severe hole deviations, jammed drill strings and change of collaring position. The extra drilling was planned by the stope planning section and a supplementary drilling layout was issued. The metres drilled were recorded and taken into account in costing.

Unauthorized Extra Drilling

In addition to remedial drilling, there was ad-hoc unrecorded extra drilling. Like for remedial, unrecorded drilling was done in order to reduce large toe burdens caused by hole deviations and to replace blocked holes. But, unlike remedial drilling, unrecorded drilling was not reflected in the plans kept by stope planning section. These holes were drilled upon realizing that the drilling was not accurate by the driller or the operating official, such that the holes were not accounted for in costing.

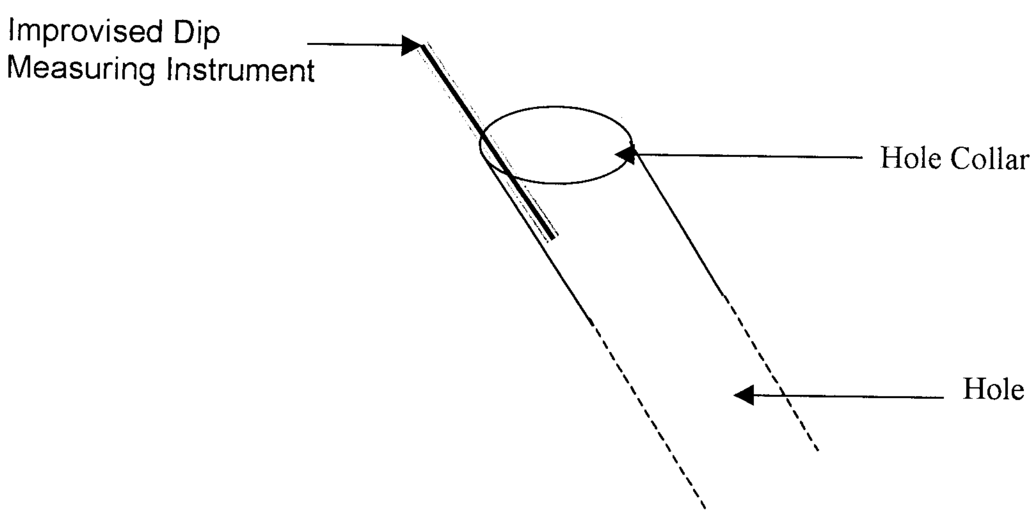
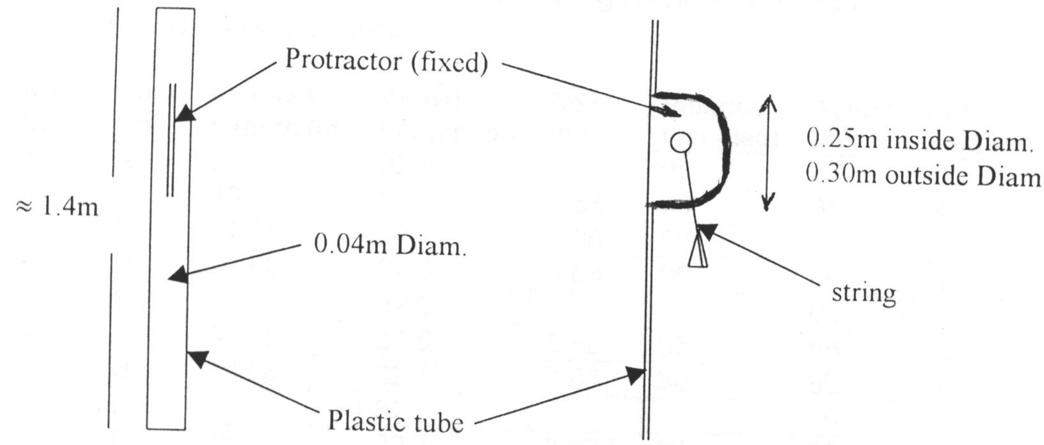


Figure 4.11 Aligning of the instrument when measuring the hole collar dip.



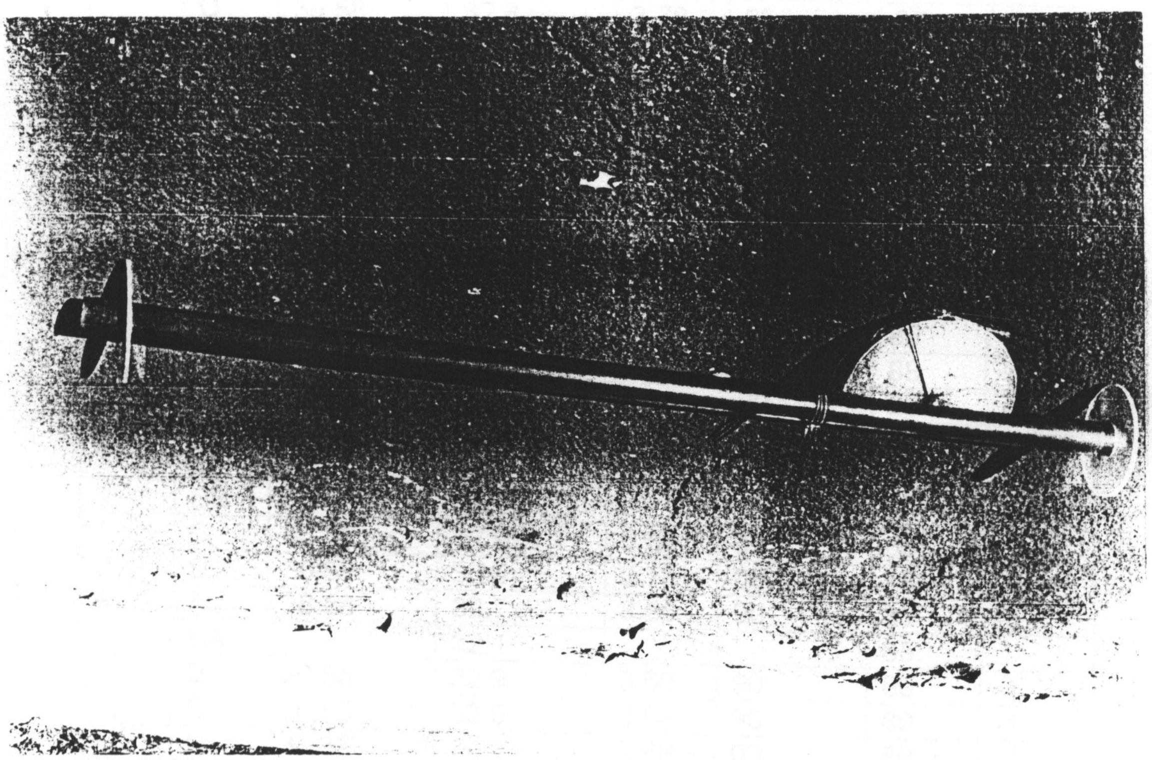


Plate 1
Photo by S. Kangwa

Figure 4.12 Improvised instrument for measuring hole alignment.

Table 4.7. Planned versus actual hole lengths and inclinations for chamber rings.

Ring No.	Hole No.	Planned Length (m)	Actual Length (m)	Dev. (m)	Planned Incl (deg)	Actual Incl.	Differ. (deg.)
1	A	36.58	32.8	3.78	90	87	-3
	B	36.58	32.9	3.68	90	86	-4
	C	36.58	32.7	3.88	90	89	-1
	D	36.58	32.9	3.68	90	86	-4
	ED		32.6			85	
2	A	36.58	33.0	3.58	85	88	+3
	B	36.58	32.7	3.88	90	89	-1
	EB		32.9			86	
	C	36.58	32.7	3.88	90	87	-3
	EC		32.9			89	
	D	36.58	32.8	3.78	85	85	0
3	A	20.12	19.8	0.32	85	84	-1
	AA	20.12	17.9	2.22	80	82	+2
	B	20.12	12.1	8.02	90	87	-3
	C	20.12	19.8	0.32	90	90	0
	CC	20.12	16.5	3.62	90	88	-2
4	A	36.58	32.4	4.18	90	89	-1
	B	36.58	32.9	3.68	90	87	-3
	C	36.58	32.9	3.68	90	89	-1
	D	36.58	33.0	3.58	90	85	-5
	ED		32.8			88	
5	A	36.58	32.2	4.38	90	86	-4
	EA		32.7			87	
	B	36.58	32.9	3.68	90	88	-2
	EB		32.9			86	
	C	36.58	32.9	3.68	90	89	-1
	D	36.58	32.8	3.78	90	88	-2
6	A	36.58	32.7	3.88	90	85	-5
	EA		32.1			88	
	B	36.58	32.9	3.68	90	89	-1
	C	36.58	32.9	3.68	90	89	-1
	D	36.58	32.9	3.68	90	89	-1
7	A	36.58	32.3	4.28	90	85	-5
	B	36.58	32.6	3.98	90	86	-4
	C	36.58	10.0	26.58	90	88	-2
	D	36.58	32.4	4.18	90	87	-3
8	A	36.58	34.8	1.78	90	88	-2
	EA		34.1			86	
	B	36.58	32.8	3.78	90	86	-4
	C	36.58	33.8	2.78	90	88	-2
	D	36.58	32.3	4.28	90	90	0
Total Metres		**1,088.26**	**1,246.0**				

Table 4.8. Planned versus actual burdens of chamber rings

Ring No.	Hole No.	Actual Burden (m)	Planned Toe Burden (m)	Deviation (m)
1	A	1.4	1.5	-0.1
	B	2.1	1.5	+0.6
	C	4.3	1.5	+2.8
	D	5.3	1.5	+3.8
	ED	1.5	1.5	0
2	A	1.5	1.5	0
	B	3.5	1.5	+2.0
	EB	1.9	1.5	+0.4
	C	2.8	1.5	+1.3
	EC	3.9	1.5	+2.4
	D	0.7	1.5	-0.8
3	A	1.9	1.5	+0.4
	AA	3.0	1.5	+1.5
	B	2.6	1.5	+1.1
	C	3.4	1.5	+1.9
	CC	0.6	1.5	-0.9
4	A	4.2	1.5	+2.7
	B	0.6	1.5	-0.9
	C	2.2	1.5	+0.7
	D	4.3	1.5	+2.8
	ED	2.7	1.5	+1.2
5	A	3.1	1.5	+1.6
	EA	1.2	1.5	-0.3
	B	2.1	1.5	+0.6
	EB	4.4	1.5	+2.9
	C	2.3	1.5	0.8
	D	1.0	1.5	-0.5
6	A	4.5	1.5	+3.0
	EA	2.5	1.5	+1.0
	B	0.6	1.5	-0.9
	C	1.9	1.5	+0.4
	D	0.3	1.5	-1.2
7	A	0.9	1.5	-0.6
	B	2.2	1.5	+0.7
	C	3.2	1.5	+1.7
	D	1.7	1.5	+0.2
8	A	4.4	1.5	+2.9
	EA	2.8	1.5	+1.3
	B	2.4	1.5	+0.9
	C	0.6	1.5	-0.9
	D	1.1	1.5	-0.4

Table 4.9. Planned versus actual lengths and inclinations of recovery ring 1

Hole No.	Planned length (m)	Actual length (m)	Deviation (m)	Planned incl.	Actual incl.	Deviation (degrees)
1	36.58	36.60	+0.02	87	85	2
2	36.88	36.95	+0.07	86	85	1
E2		36.75			75	
3	36.88	36.70	-0.18	75	75	0
4	36.88	36.90	+0.02	69	70	1
E4		36.65			65	
5	23.17	23.20	+0.03	56	55	1
6	16.46	16.50	+0.04	47	50	3
7	10.36	10.25	-0.11	26	25	1
E7		10.50			30	
8	8.23	8.20	-0.03	+9	+10	1
9	8.84	8.90	+0.06	+30	+30	0

Total planned length = 214.28m and total actual length = 298.10m

Table 4.10. Planned versus actual lengths and inclinations of recovery ring 2

Hole No.	Planned length (m)	Actual length (m)	Deviation (m)	Planned incl.	Actual incl.	Dev. (deg.)
1	35.97	36.00	+0.03	87	85	2
E1		35.80			83	
2	36.27	36.10	-0.17	82	80	2
3	37.49	37.50	+0.01	74	75	1
4	39.01	39.00	-0.01	66	65	1
E4		38.50			60	
5	22.25	22.00	-0.25	58	60	2
6	15.54	15.70	+0.16	48	50	2
7	10.06	10.00	-0.06	25	25	0
8	8.53	8.45	-0.08	+29	+30	1
9	8.53	8.20	-0.33	+6	+5	1

Total planned length (165mm diameter holes) = 196.69m

Total actual length (165mm diameter holes) = 270.60m

Total planned length (51mm diameter holes) = 17.06m

Total actual length (51mm diameter holes) = 16.65m

Table 4.11. Planned versus actual lengths and inclinations of wrecking ring

Hole No.	Planned length (m)	Actual length (m)	Deviation (m)	Planned incl.	Actual incl.	Deviat. (deg.)
1	36.58	36.70	+0.12	85	85	0
2	37.79	37.50	-0.29	76	75	1
E2		36.60			70	
3	38.10	36.80	-1.3	67	65	2
E3		37.90			60	
4	18.29	18.40	+0.11	50	50	0
5	10.67	10.70	+0.03	24	25	1
E5		10.10			20	
6	6.10	7.20	+1.10	+5	10 (up)	5
E6		6.80			25	
7	9.75	10.50	+0.75	+26	30	4
8	11.89	12.40	+0.51	+46	50	4
9	17.68	18.10	+0.42	+63	65	2
10	23.16	22.70	-0.46	+73	75	2
E10		23.10			80	
11	22.56	21.50	-1.06	+83	85	2
12	22.25	22.80	+0.55	+89	90	1
13	22.56	23.20	+0.64	+79	80	1

Total planned length (165mm diameter holes) = 141.43m

Total actual length (165mm diameter holes) = 224.70m

Total planned length (127mm diameter holes) = 135.95m

Total actual length (127mm diameter holes) = 168.30m

4.3.1.3.1 *Computer 'monitoring' program*

Apart from measuring linear deviations at the bottom of the holes, an attempt was made to 'trace' hole paths. A computer program was developed jointly with Nkana Division, Technical Section. The program is based on co-ordinate system. The relative positions of pre-marked hole collar points were located with reference to mine grid (the easting, E and northing, N). The perpendicular distances or partial co-ordinates (eastings, ΔE and northings, ΔN) give the co-ordinates of the hole collar. ΔE and ΔN of the hole collar points were measured from the mine plans using scale rule. Using collar information (E, N, ΔE and ΔN) as input data, the program automatically calculates ΔE and ΔN of each hole at any given length below the collar by trigonometric formulae, as follows:

$\Delta E = L_h \sin (WCB)$, and

$\Delta N = L_h \cos (WCB)$

Assuming constant bearing for stope and assay boundaries (Figure 4.13):

$Y \text{ at } D_p = E + \Delta E \times D_p / L_h$

$X \text{ at } D_p = N + \Delta N \times D_p / L_h$

For holes at constant bearing (Figure 5.13):

$Y \text{ at } D_p = E + (\Delta E \times D_p) / [\sqrt{(L_h)^2 - (\Delta E)^2 - (\Delta N)^2}]$

$X \text{ at } D_p = N + (\Delta N \times D_p) / [\sqrt{(L_h)^2 - (\Delta N)^2 - (\Delta E)^2}]$

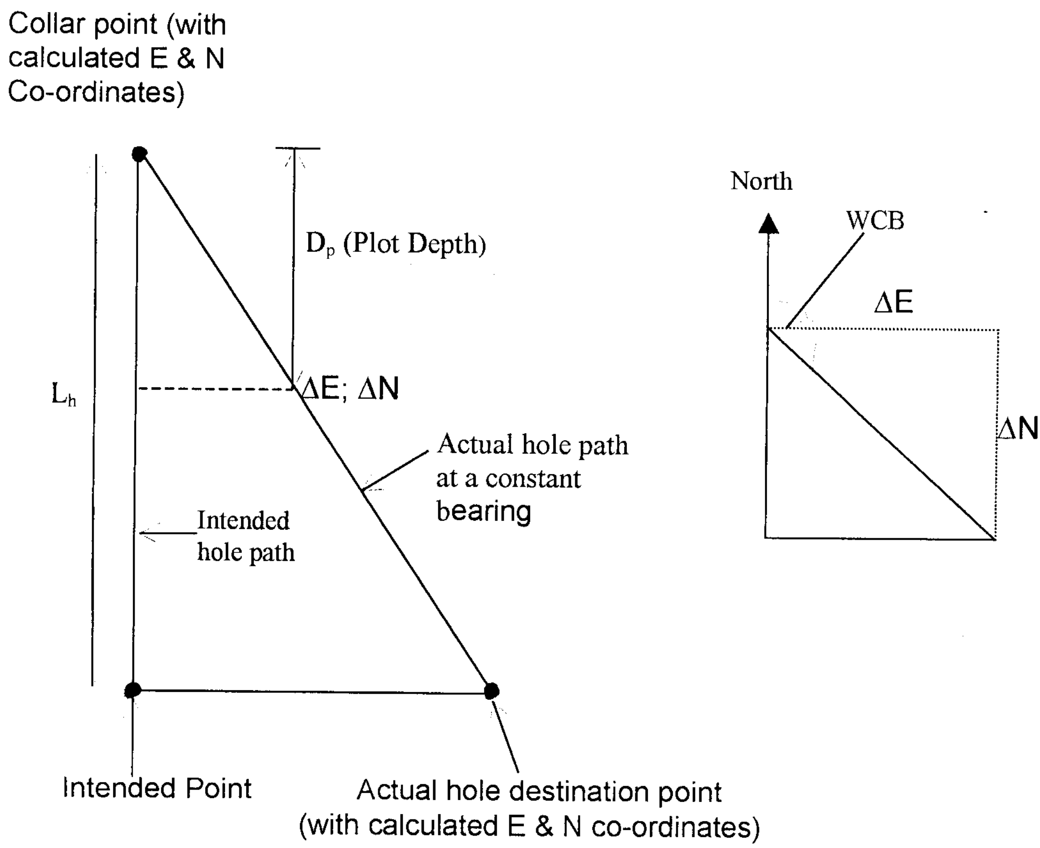


Figure 4.13 Sketch illustrating the trigonometry applied in the computer monitoring program calculations

Where: L_h = Hole length

WCB = Whole Circle Bearing

D_p = Plot depth (any position between collar and hole destination)

E = Collar co-ordinates (easting)

N = Collar co-ordinates (northing)

ΔE = Partial co-ordinates of collar and hole destination (easting)

ΔN = Partial co-ordinates of collar and hole destination (northing)

Y = Co-ordinates (easting) at D_p

X = Co-ordinates (northing) at D_p

Examples of Programmed Calculations.

For stope boundaries and ore-body limits i.e. at point AFW (Figures 4.7 and 4.9), ΔE and ΔN were calculated as follows:

Y at at $D_p = E + \Delta E \times D_p / L_h$

Using the measured values and taking D_p as 39m (maximum stope drilled length)

Y at D_p $= 18.2 + \{(2.6 - 18.2) \times 39 / 39\}$

$= 18.2 - 15.6 =$ **<u>2.6m</u>**

X at D_p $= N + \Delta N \times D_p / L_h$:

$= 30.2 + \{(21.4 - 30.2) \times 39\} / 39$

$= 30.2 - 8.8 =$ **<u>21.4m</u>**

For holes i. e. hole number 1C (Table 4.7 and Figure 4.7), ΔE and ΔN were calculated as follows:

X at D_p $= 16.7 + (14.4 - 16.7) \times 39 / \sqrt{} 39^2 - (14.4 - 16.7)^2 - (3.5 - 10.8)^2$

$= 16.7 - 2.3 =$ **<u>14.4m</u>**

Y at D_p $= 10.8 + (3.5 - 10.8) \times 39 / \sqrt{} 39^2 - (3.5 - 10.8)^2 - (14.4 - 16.7)^2$

$= 10.8 - 7.4 =$ **<u>3.4m</u>**

With reference to the mine grid, the calculated ΔE and ΔN were then used to automatically plot the points (*Appendix O*). The calculations may be done manually as well, but it is laborious.

The program has, however, some limitations as listed below:

a) It assumes 'constant' hole deviation

b) The holes are assumed to be straight (no deflections)

c) Blocked holes are assumed open to the holing level, and

d) Blind holes are assumed as holing

Despite the limitations listed above, the program was used for planning blasting sequence i.e. choosing which holes to blast at a particular depth. The program was useful as some peripheral holes deviated into the central portion and vice versa, some central 'VCR' holes deviated outwards into peripheral area. The program was able to show the assumed (predicted) status of the holes at any point below the collar.

4.3.1.4 *Observed causes of hole deviations*

The following were observed as the main causes of hole deviations:

1. Defective drilling machine: The defective components i.e. worn out spacer plates outlined in Section 4.3.1.2, are likely to have contributed greatly to hole deviations. When pressure is applied, the lags, which are tightly fitted between the spacer plates, are supposed to centralise the feed beam (and hence thrust) through the machine centre. But, if there is space between the lags and the plates, the thrust will be eccentric, resulting in one side of the lag taking up more thrust than the other (Figure 4.14). This may cause hole

deviations. Appendix P, Plate 2 show the worn-out machine that was used for drilling production holes.

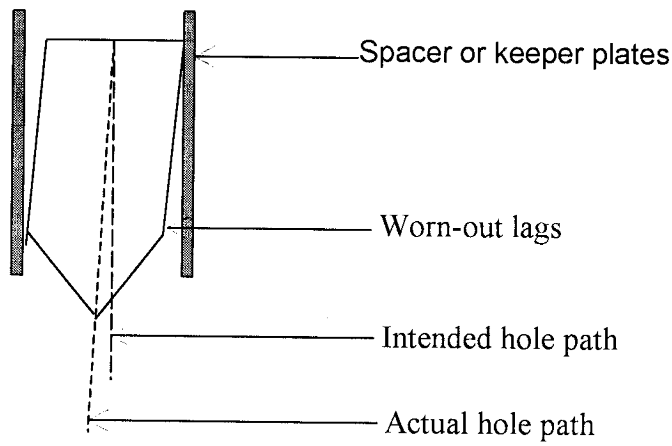


Figure 4.14 Illustrating worn out parts causing one side to take up more thrust than the other side.

2. Poor rigging: It was observed that the machine was not being properly rigged, mainly due to lack of appropriate rigging tools such as the instrument for measuring inclination. This may have given rise to both collaring and alignment deviation.
3. Uneven floor as illustrated in Figure 4.15: In this case planned and actual hole paths would be in the same vertical plane, but differ in lateral positions. The lateral displacement (δ) would be proportional to the vertical displacement (y). This displacement is the distance which the machine centre has moved above or below the grade or centre line (Figure 4.15).

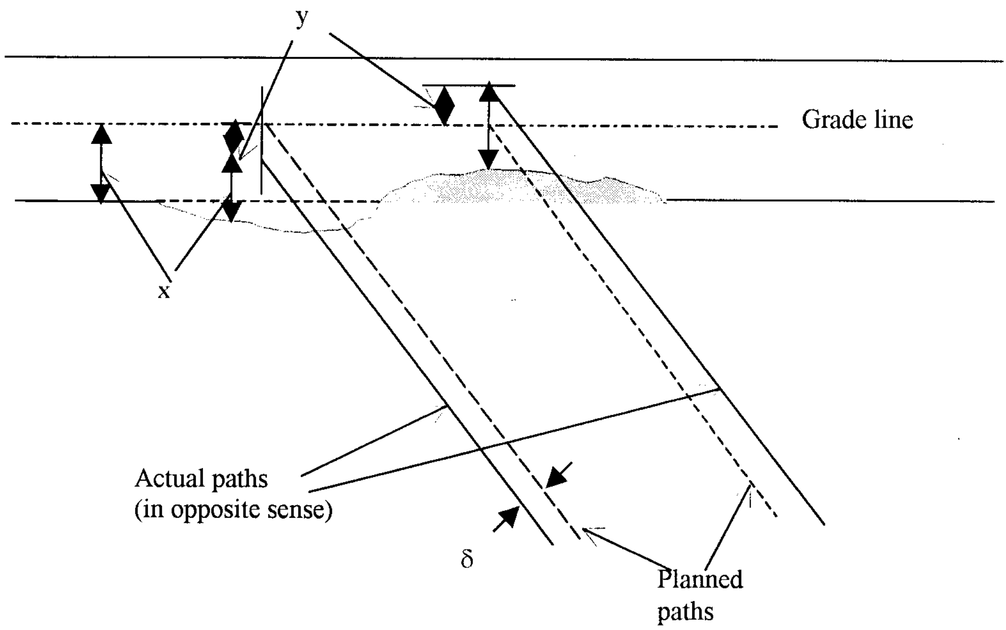


Figure 4.15 Illustrating change in hole paths due to uneven floor

4). Poor setting of the inclination: This resulted into hole alignment deviations (Section 4.3.1.2).

5). Poor development or poor development survey as illustrated in Figure 4.16. (see also Section 4.3 - *drifts profiles*).

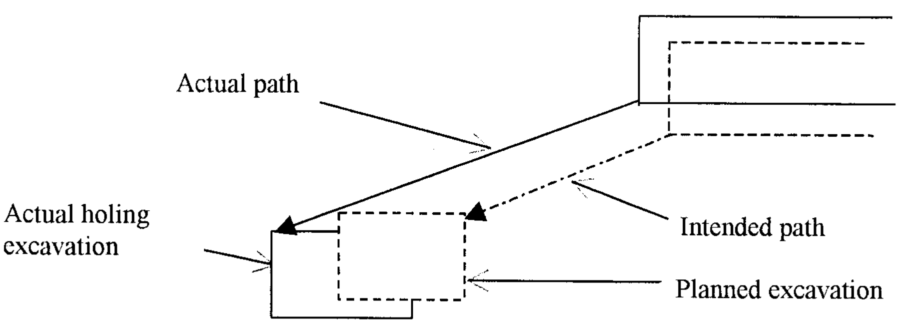


Figure 4.16 Illustrating shift in development floor elevation

6). Non-homogeneous rock formation: When drilling in different rock formations with different strength and structure, drilling parameters were not adjusted according to rock type (Section 3.3.2.1 - *geology of the study area*).

7). Inappropriate application of drilling parameters (see also Section 4.3.1.2 and Table 4.6).

Non-technical causes

Apart from the technical causes, the following non-technical causes of hole deviations were observed:

1). Management style: It was observed that the management emphasised drilling more metres per shift instead of emphasising on both quantity and quality. The operators were compelled to disregard recommended drilling parameters and 'force' the 'defective' machine to drill the demanded metres. This may have contributed to hole deviations.

2). Lack of co-operation among the work force: Co-operation among workers and units was lacking. Instead of sharing tools like spanners and inclination instruments, workers tended to be 'selfish'. It was also observed that units were not consulting each other, for example in cases of machine rigging and marking of drilling positions. This affected the work performance and the quality of holes.

3) Lack of full implementation of Mincon Information Management System (MIMS): The MIMS is a computer financial system developed by Management Science America, and comprises a comprehensive set of modules designed to cater for mining business requirements.

The MIMS enables responsible holders to access the chart of accounts. The chart of accounts structure consists of cost centre, expense element, cost code, general ledger code, account code and account segmentation.

The system enables the Supervisors to monitor and analyse their operational costs at any time thereby making decisions based on actual figures.

Lack of full appreciation of MIMS may have contributed to drilling supervisors not to appreciate the increased drilling costs. This is because responsible holders were unable to access the system in a timely manner. Rather, it was taking more than a month before the accounting section produced general cost figures.

4.3.1.4.1 *Consequences identified during operations*

Some consequences were identified during operations and these included:

a) Delay time

b) Extra drilling, and

c) Loss of production

Delay time:

For purpose of cost estimation, this study considers delay time as the time lost when a machine falters or when labour and/or machines are re-directed from planned activity. The time taken to do repeat jobs (e.g. extra drilling, secondary blasting, clearing of boulders and mucking of extra ground) is also regarded as delay time.

For the purpose of deviation-related downstream costs, the following hole deviation-related delays were considered:

a) remedial extra drilling (Section 4.3.1.3)

b) secondary blasting

c) clearing of boulders

d) clearing of 'run-away' ground

For the purpose of cost estimation, the consequences have been analysed as follows (for calculations, see Appendix L):

Chamber Rings *(Refer to Table 4.9)*

Cost item	Planned Quantity	Actual Quantity	Variance
Blast holes	32	41.0	9
Metres	1,088.26	1,246.0	157.74
165mm button bits	1.5	2.0	0.5
Shifts (2/day)	22	34	12
Labour (total workers)	88	136	48

Recovery Ring 1 *(Refer to Table 4.11)*

Cost item	Planned Quantity	Actual Quantity	Variance
Blast holes	9	12	3
Metres	214.28	298.1	83.82
165mm button bits	0.3	0.5	0.2
Shifts (2/day)	4.0	6.0	2.0
Labour (workers)	16	24	8

Recovery Ring 2 *(Refer to Table 4.12)*

Cost item	Planned Quantity	Actual Quantity	Variance
Blast holes	9	11	2
Metres	196.69	270.99	74.3
165mm button bits	0.3	0.4	0.1
Shifts (2/day)	4.0	8.0	4.0
Labour (workers)	16	32	16

Wrecking Ring *(Refer to Table 4.13)*

Cost item	**Planned Quantity**	**Actual Quantity**	**Variance**
Blast holes	13	18	5
Metres	277.38	393.0	115.62
165mm button bits	0.3	0.4	0.1
127mm button bits	0.2	0.3	0.1
Shifts (2/day)	8.0	12.0	4.0
Labour (workers)	32	48	16

4.3.1.5 *Observation of blasting activities*

Prior to blasting of the stope holes, blasting plans were examined. Charging of blast holes and the subsequent blasting were observed.

4.3.1.5.1 *Charging*

Before charging commenced, all the holes were visually checked. The following information was also noted:

a) Hole blockages and closures. Where these had occurred and where necessary, holes were cleaned.

b) Planned lengths of holes (Section 4.3.1.3)

c) Planned burdens of holes. Because of deviations the planned hole burden of 1.5m was not achieved (Table 4.8), and it was observed that blast-holes with burdens between 1.8m and 0.6m were charged.

The Charging of blast-holes was carried out as follows:

a) Holes were exposed and marked.

b) The distances between the ground level and the stope back were determined using a pre-marked rope and weight.

c) The wooden plug assembly was lowered to the marked position on strong twine (i.e. the plug was placed at approximately 0.5m from the bottom of each hole as shown in Figure 4.17).

d) Coarse sand was dropped down the hole as stemming material (of approximately 0.2m length of 165mm diameter hole).

e) Tubes of slurry powder plus (110mm by 560mm) explosives were dropped into the hole (one case of slurry was charging about 0.85m of 165mm diameter hole).

f) The primer was lowered into the hole. Magnum 365 (32mm by 200mm) and non-electric millisecond (Nonel MS) detonators were used for timing. .Nonel MS detonators had delay times with 18 intervals from 3 to 20.

g) More slurry explosives were dropped into the hole.

h) Fine sand of about 1.5m high as stemming on top of the charge was added.

i) The Nonel MS detonator tubes were connected using Nonel snapline (SLO) surface connectors units so that all detonators in the round were initiated simultaneously.

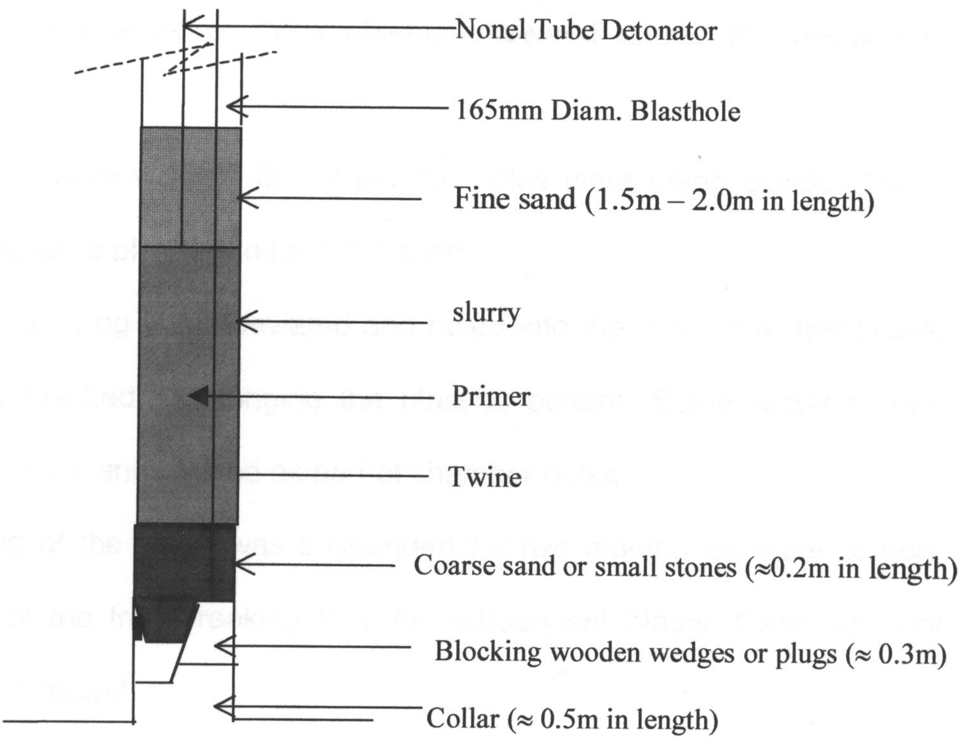


Figure 4.17 Illustrating the charging procedure (not to scale)

4.3.1.5.2 *Blasting sequence of VCR holes*

The VCR holes were divided into two portions:

a) the central portion for 'VCR' blasting, and

b) the periphery portions for cylindrical blasting.

The periphery holes were blasted at least two milliseconds (delay) behind to ensure that they had the option of blasting into the opening created by the central holes, or into the undercut.

4.3.1.5.3 *Problems encountered during the blasting of 2590N stope*

The following were some of the problems observed during the blasting of 2590N/2880L:

a) Holes, which were within 1.0m of blasted holes were being closed. These closed holes were planned to be 1.5m apart.

b) Some recovery ring holes deviated and holed into the main chamber (Table 4.12). This resulted in changing the blasting pattern. Some recovery ring holes were taken and blasted as part of chamber holes.

c) The blasting of the stope was suspended for two months because of non-availability of the free breaking face for subsequent blasts. Swell was not drawn as scheduled.

The delay in completing blasting of 2590N stope within the anticipated period of one month resulted into difficulty in accounting for draw tonnage and general monitoring of stope performance. This was because the 2880 ft-level SLC stope (in the next Section) started producing, resulting in mixing of ore from the two stopes. This required strict monitoring in order to account for ore from 2590N stope.

Table 4.12 Chamber Blasting Monitoring Report for 2590N/2880L Stope

DATE (Blasted)	Number of Slurry Cases Used	Total Hole Vertical Advance (m)	Case / Metre	Remarks
05. 12. 98	27.0	54.0	2.00	Blasted 18 holes
06. 12. 98	29.0	55.0	1.93	Repeat blast to widen the crater, 19 holes were blasted
06. 03. 99	42.0	71.9	1.71	5 recovery holes and 20 chamber holes were blasted.
13. 03. 99	60.0	83.6	1.39	24 chamber holes and 5 recovery holes were blasted
17. 03. 99	60.0	90.1	1.50	22 chamber holes and 5 recovery holes were blasted.
18. 03. 99				Found the hangingwall had collapsed
19. 03. 99	55.0	78.4	1.43	21 chamber holes and 4 recovery holes were blasted
29. 03. 99	45.0	76.1	1.69	18 chamber holes and 5 recovery holes were blasted
15. 04. 99	40.0	69.7	1.74	16 chamber holes and 5 recovery holes were blasted
30. 04. 99	18.0	35.0	1.94	Approximately 10m remained.
03. 05. 99	260.0			Final Blast (wrecking round)
Total cases = 636.0				

4.3.1.5.4 ***Consequences identified during blasting activities***

Some consequences were identified during blasting activities and include:

a) extra consumption of explosives

Explosives included slurry power plus (25Kg cases), bunch connectors and Nonel MS detonators.

b) extra man-hours

c) delay time

d) loss of production

The field observations showed that almost all the above mentioned consequences were due to hole deviations (Sections 4.3.1.2 and 4.3.1.3) rather than poor blasting. Blasting was appropriate for the operations observed and same techniques were used for all blasting rounds.

For the purpose of cost estimation, the consequences have been quantified as follows:

Stope Area	**Cost item**	**Planned Quantity**	**Actual Quantity**	**Variance**
VCR chamber	Slurry (25Kg cases)	575	636	61
Rib pillar	Slurry	433	486	53
	Magnum busters	260	307	47
Crown pillar	Slurry	144	236	92
	Magnum busters	87	138	51

Results Analysis

Table 4.13 Summary of metres drilled and explosives used

	TONNAGE	METRES DRILLED		EXPLOSIVES (25Kg Cases)	
		PLANNED	ACTUAL	PLANNED	ACTUAL
Stope	23,486	1,088.26	1,246.0	575 x 25 = 14375	636x25 = 15900
Rib pillar	19,633	694.94	852.3	693 x 25 = 17325	793x25 = 19825
Crown	12,812	632.76	974.4	231 x 25 = 5775	374x25 = 9350

Table 4.14 Metres and explosives utilization

	TONNES/CASE (EXPL.)		POWDER FACTOR		TONNES/METRE DRILLED	
	PLANNED	ACTUAL	PLANNED	ACTUAL	PLANNED	ACTUAL
Stope	23486/575 = 40.8	23486/636 = 36.9	14375/23486 = 0.612	15900/23486 = 0.677	23486/1088 =21.59	23486/1246 = 18.85
Rib	19633/693 = 28.3	19633/793 = 24.8	17325/19633 = 0.882	19575/19825 = 0.987	19633/694.9 = 28.25	19633/852 = 23.0
Crown	12812/231 = 55.5	12812/374 = 34.3	5775/12812 = 0.451	9350/12812 =0.730	12812/632.8 =20.25	12812/974 = 13.15

4.3.1.6 *Examination of fragmentation resulting from the blasts*

After the blasting of 2590N stope was completed, rock fragments were measured to determine the quality of rock fragmentation.

Fragmentation in this study is referred to as the breaking of ore or rock by blasting [Nelson, 1964]. Fragmentation would be at its best when the broken material is not smaller than necessary for handling and not so large as to require hand breaking or secondary blasting.

The quality of fragmentation was based on the type of transport to handle the ore from the stope. It was planned [Nkana, Division, Production Section, 1998] that the 2590N stope ore handling was to be done mechanically by LHD 250D loader (with a loading capacity of 3.0m^3) or 300D loader (with a loading capacity of 3.5m^3) at 2840 ft-level and by locomotive at 3140 ft-level. The fragments suitable for these type of transport ranges between 500mm and 1200mm [Kuznetzo, 1973; Nkana Division, Production Section, 1992] as shown in Table 4.16. Therefore the majority of the fragments were supposed to be in this range.

To determine the size of fragments, two methods were used:

a) Measuring tape: Representative samples of rock fragments were picked after visually checking the blasted ground. A tape was used to measure the dimensions of the fragments. The longest side was considered for this study.
b) Sieve technique: For this method, a 'grizzley' with adjustable steel bars to regulate the openings was used. A scoop by the loader, from each of the four draw-points was released on it. The percentage of the undersize (fragments passed through the opening) and oversize was determined visually by at least two observations per shift for the life of the stope and the average was taken as rough approximation of size distribution. Table 4.15 was compiled basing on these estimations. The results shown in Table 4.15 implied that the majority (75%) of the fragments were under 100mm in size.

Table 4.15 Showing the percentage of fragments going though the pre-determined opening.

Opening size (mm)	**Undersize (approx. %)**	**Oversize (approx. %)**
25	40	60
100	75	25
500	85	15
750	90	10
1200	98	2

Table 4.16 shows the classification of rock fragmentation based on average fragment size (K_{50}) and size distribution. The classification of fragmentation is based on the size of fragments. The debris is examined and the majority (> 50%) size of fragments or pieces of rock, is taken as more distributed in that particular debris. K is used to denote the size type of fragments and K_{50}, therefore represents the average or the majority fragment size in a particular pile.

Table 4.16 Classification of Rock Fragmentation Based on Average Fragment Size (K_{50}) and Size Distribution

K_{50} (mm)	**Type of Fragmentation**	**Loading/Transport Method**	**Remarks**
< 25	Powder	No suitable u/g mechanical transport	Under size, difficult to load and transport resulting into spillage, dust, run-aways and ore loss.
25 – 100	Fine	Suitable for surface conveyor belt transport.	Results into spillage and dust
100 – 750	Medium	LHD loaders and u/g conveyor belts	Suitable for u/g transport
500 – 1200	Coarse	LHD loaders and haulage (locomotive)	Suitable for haulage transport
> 1200	Boulders	No suitable method	Oversize resulting into loading box blockages and secondary fragmentation.

For the ideal situation, the K_{50} for 2590N stope ore body should have been between 500 – 1200mm, but it was observed that because of inaccurate drilling of production holes, and concentrated blasting (Table 4.14 and Appendix O) the K_{50} was 25 to 100mm. The few boulders introduced into the ore pile may have been due to sloughing off of the stope hanging wall and sides as observed during and after blasting. The pilling off of the hanging wall was probably caused by pulling empty of the stope before blasting the last two rounds.

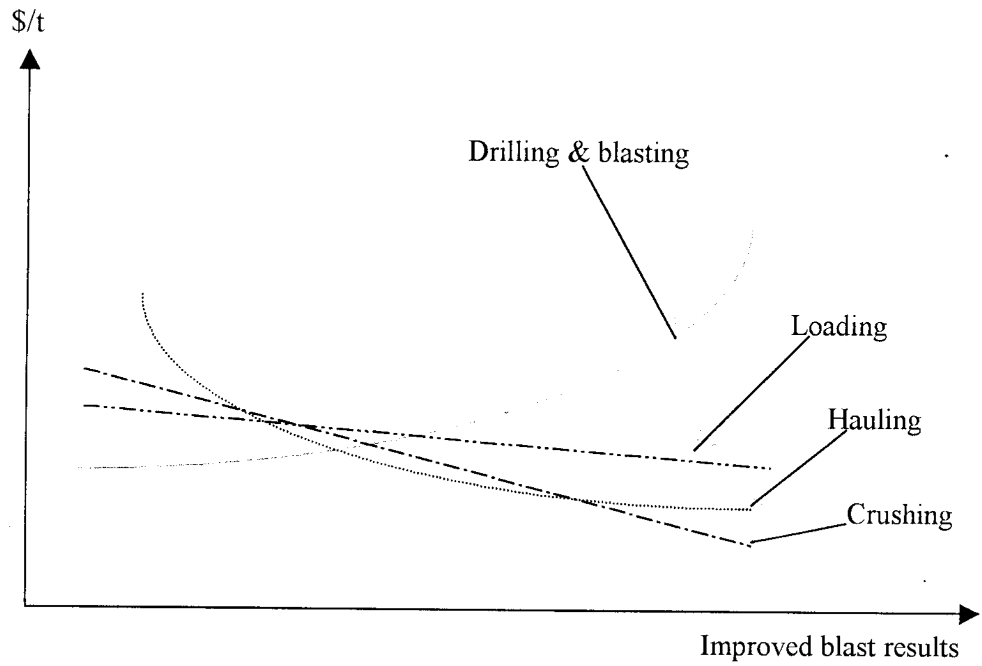


Figure 4.18 A general illustration showing that good quality holes and good blast geometry will result in good fragmentation, thus leading to low cost of loading, hauling and crushing.

It is possible to some extent to compensate for hole deviation [Lindvall, 1983 and Sinkala, 1986] with increased specific drilling. How much can be compensated, is a matter of which fragmentation can be tolerated and of course the costs for the increased drilling (Figure 4.18).

4.3.1.7 *Examination of the handling of ore, crushing and metallurgical processes*

The ore handling from 2590N stope was done mechanically. The ore was drawn from the draw points (Figure 4.5) by 250D LHD loaders (Section 4.3.1.6). Since the ore size was mainly between 25mm and 100mm (Table 4.15), it was not suitable for LHD loaders (Table 4.16). This resulted in transportation problems. The problems of transporting ore included loading, spillage and low loader speed. These problems caused delays.

The ore drawn from the draw-points, was transported to 2430N ore-pass and released into the same. The ore released in this ore-pass gravitated down to a box chute at 3140ft main haulage level where the ore was loaded on locomotive trains. The trains transported the ore to the shaft for hoisting to the surface.

Because the ore size was fine (25mm to 100mm), spillages and run-aways were experienced at the chute box and during tramming of ore. This resulted in delays. The delay time was caused by failure to control the flow of ore at the chute box, derailments of locomotives and mine cars, clearing of spillage and run-aways from the rail tracks and at the chute box area.

For the purpose of cost estimation, the consequences have been analysed as follows:

Consequences due to ore handling and other processes

Cost item	Planned Quantity	Actual Quantity	Variance
Total delay time	32 days	45 days	13
Total extra tonnage	55,931	73,842	17,911
Labour (total workers)	288	405	117

At SOB shaft, the primary crushers are located at the surface, therefore the ore is hoisted and directly tipped into the crushers. The crushed ore is then transported by the belt conveyor into the surface storage bins. The ore from these bins is transported as run-of-mine (ROM) tonnage by rail for metallurgical processes. These processes involve differential flotation of copper and cobalt minerals. The copper concentrates are then smelted and refined to produce electro-copper, which is casted into wire-bars. The cobalt concentrates are transported to cobalt plant. The plant treats these concentrates to produce electrolytic cobalt.

4.3.1.7.1 *Other consequences identified during and after stope operations*

Table 4.17 shows the overall stope performance and consequences, which were identified during and after stope operations.

Table 4.17 2590BCN/2880L Stope Performance

Month (1999)	G	g	GF	TT	T_{cu}	T_{ore}	T_{waste}	Dil.	Ext.	Rec.
March	2.60	2.00	77.1	3123	62.46	2407.8	716.0	29.7	5.3	4.1
April	2.60	1.75	67.2	4227	73.9	2843.0	1383	48.7	15.4	9.1
May	2.60	1.75	68.7	21813	389.6	14,986	6827	45.6	52.6	34.6
June	2.60	1.96	75.4	12950	253.8	9752.0	3198	32.8	75.9	50.8
July	2.60	1.68	64.8	10565	177.9	6842.0	3723	54.4	95.0	60.2
Aug.	2.06	1.81	87.9	10101	183.0	8881	1220	13.7	134.4	108.1
Sept.	2.06	1.64	79.6	3042	49.9	2422	620	25.6	141.5	113.0
Oct.	2.06	1.01	49.0	436	4.4	212	224	106	142.8	113.5

Examples of calculations (taking figures for the month of March) are shown below, where:

G = Planned reserve grade from block exploration core logging [Nkana Division, Geological Section, 1997)

g = Actual box grade as sampled from the ore box

GF = Grade factor = g/G = 2.004/2.600 = 0.771

TT = Tonnes trammed, monitored at the tally points = 3123T

T_{cu} = Tonnes of contained copper = (TT x g)/100 = (3123 x 2.00)/100 = 62.46T

T_{ore} = Tonnes of ore trammed = (TT x GF)/100 = (3123 x 77.1)/100 = 2407.8T

T_{waste} = Tonnes of waste = TT – T_{ore} = 3123 – 2407 = 716T

Dil. = Dilution = (T_{waste}/T_{ore}) x 100 = (716/2407) x 100 = 29.7%

Ext. = Extraction ratio at each month ending

Rec. =Recovery ratio at each month ending

Overall stope ore recovery = 48,345.8 / 55,931 = 86.45%

Metal (contained) recovery = 1,194.96 / 1,454.20 = 82.17%

The dilution variations, ore and waste pulled from 2590BCN stope per month (Table 4.17) are graphically illustrated in Appendix I.

4.3.2 The raise pilot hole

4.3.2.1 Examination of drilling plans and site preparation

Examination of the 2430N ore raise tip drilling plans showed that the inclination and the length matched those provided in Table 4.18. This was checked by scaling the plans. The location of the collaring point (Figures 4.19) was scaled from the survey pegs and it was found to agree with dimensions shown on the layouts (Figure 4.19).

Table 4.18 Data for 2430N tip raise [Source: ZCCM-Nkana Division, survey Department]

Rigging level	2840L
Location	2430N
Purpose	Tip raise
Holing level	3140L
Drilling machine	1141 Robins 61R Raiseborer
Planned inclination	65 degrees
Planned length	105m
Hole diameter	22.86cm (Pilot) 183.00cm (Reamed)

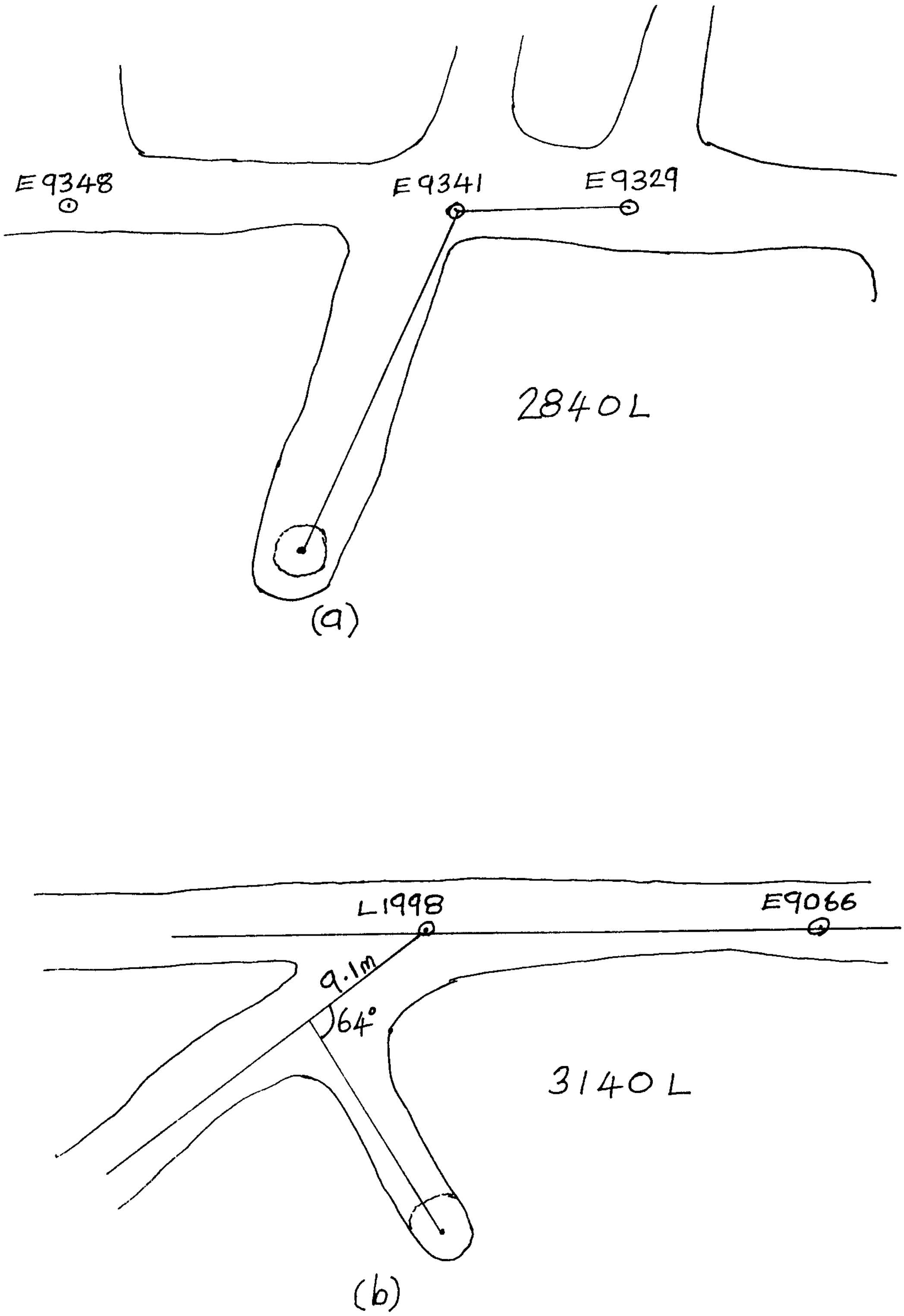


Figure 4.19 Plans of (a) pilot hole drilling and (b) holing layouts

Site preparations

The drilling area was lashed down to bed-rock. The roof was supported with 4.5m long cement grouted cable bolts at 1.0m intervals and with 2.4m long grouted wire ropes and split sets at 0.8m centres. In addition, the chamber was reinforced with wire mesh and lacing.

A thin concrete skin was laid and long bearers installed for the base plates. The bearers and base plates were positioned and levelled with the aid of the Survey department. The channel between the two bearers facilitated easier installation and removal of an apparatus (blooie-seal) used as cuttings collector and has provision for sludge drainage. Hold-down bolts were then bolted to the bearers and concrete laid, leaving a trench for the blooie-seal. The raise-borer, rods and drilling accessories were moved into place. With the help of the Survey department, the inclination was finalised using slight adjustment of the turnbuckles. The pilot hole drilling commenced after installation and setting the raise-borer to the planned inclination.

4.3.2.2 *Observation of pilot Hole drilling*

The pilot hole was collared through the concrete pad using a 22.86cm (11inch.) CH roller bit with roller-ball bearings. The collaring through the pad and drilling to the depth of 7.5m was done by the operator's 'feel' (this depended on the operator's experience, and the bit weight was supposed to provide the drilling pressure otherwise no recommended pressure or thrust force was given). During collaring, it was recommended not to exceed the rate of penetration of 0.3m (1ft) per hour. After 7.5m depth was drilled, the drilling was stopped and the bit pulled out in order to install the bit-sub. The bit-sub was installed approximately 6m above the bit. The bit-sub is a tool mounted on a drill string to maintain the pressure on the bit and to stabilize the bit. The blooie-seal was installed and later removed because it was defective. Since the blooie-seal was not working, water and air were directly supplied to the bottom of the hole as flushing circulation fluid. Water was supplied from the mine water system to the raise-borer machine through a water-hose pipe. The planned and actual water flow-rate and air pressure are shown below:

Parameter	**Planned**	**Actual**
Water (Litres per minute)	25	<20.
Air (Psi)	200 to 250 (14 to 17 bar)	Approx. 150 (10 bar)

The pilot hole drilling re-commenced, and the drilling was done down to the depth of 106.68m (350ft) as shown in Table 4.19.

Table 4.19 Progress Report Showing 'Feet' Drilled per Shift *

Date	Feet Drilled		Accumulation
	Morning Shift	Afternoon Shift	
11.10.98	10		135
		20	155
12.10.98	5		160
		20	180
13.10.98	10		190
		15	205
14.10.98	10		215
		15	230
15.10.98	6		236
		5	241
16.10.98	7		248
		10	258
17.10.98	10		268
		12	280
19.10.98	10		290
		10	300
20.10.98	10		310
		10	320
21.10.98	10		330
		10	340
22.10.98	5		345
23.10.98	5		350

**Drilling prior to 11.10.98 took place before the commencement of the field study by the author.*

The recommended drilling pressure (Table 4.20) was not accurately applied because the pressure gauges were malfunctioning. The pressure required at different depths was calculated as explained below:

To determine the hydraulic thrust pressure, necessary to be applied along with dead weight in the downward direction, and in order to arrive at a particular bit force, calculations were done as follows:

Length of drill rods in the hole	= Number of rods x length per rod
Total drill string weight	= Length of rods in the hole x weight per unit length
Total apparent dead weight	= Total drill string weight + Derrick dead weight

The derrick dead weight is the derrick travelling weight for 61R Raise-borer machine number 1146. The derrick dead weight is specified by the manufacturer and for this particular machine it was 11,000lbs [Raise-borer manual, Technical specification, 1986].

Since the raise inclination was 58 degrees to the horizontal, the correction factor was applied. Therefore, Total effective dead weight = Total apparent dead weight x Correction factor

The desired total bit force was approximately 45,000lbs (20,408Kg). The desired total bit force was the required force for Nkana Division SOB rock, for effective rock cutting with minimal bit wear. It was empirically determined [Nkana Division, Geotechnical Department, 1996]. Therefore to determine hydraulic thrust force necessary, the effective total dead weight was subtracted from the desired bit force.

To determine the necessary hydraulic pressure reading, the force was divided by the effective rod end area of hydraulic cylinder.

Case example:

If 50 rods are in the hole;

∴ Total length = 50 x 5 = 250 feet (or 50 x 1.524 = 76.2m)

Total drill string weight = 250 x 131 = 32,750lbs (145.672KN)

Total apparent dead weight = 32,750 + 11,000 = 43,750lbs (194.6KN)

Total effective dead weight = 43,750 x 0.848 (Sin 58^0) = 37,100lbs (165.02KN)

Hydraulic thrust force necessary = 45,000 - 37,100 = 7,900lbs (35.14KN)

Hydraulic pressure (P) reading = P = Force/Area

Since the hydraulic cylinder diameter used was 9.5 inches (241.3mm)

∴ P = 7,900/70.8822 = 111.45Psi or 7.686bar

Since the 61R raise-borer 1146 was calibrated in pounds per square inch, the pressure readings were calculated in Psi as shown in Table 4.20.

Note that the actual hydraulic thrust force is not indicated because the pressure gauges were often not functioning.

Table 4.20 Showing recommended drilling pressure and rate of penetration (ROP)

No of Rods	Total Length (ft)	String Wt (lbs)	Total Apparent Dead Wt.	Total Effective Dead Wt.	Hydraulic thrust force (lb)	Hydraulic pressure (Psi)	Pressure Applicat. (max/min)	Maximum ROP (m/hr)
0 – 5						'By feel"	Collaring	0.3 (1ft)
5	25	3,275	14,275	12,105	32,895	464	Slow down	1.5 (5ft)
10	50	6,550	17,550	14,882	30,118	425	"	"
15	75	9,825	20,825	17,650	27,340	386	"	"
20	100	13,100	24,100	20,437	24,563	347	"	"
25	125	16,375	27,375	23,214	21,786	307	"	"
30	150	19,650	30,650	25,991	19,009	268	"	"
35	175	22,925	33,925	28,768	16,232	229	"	"
40	200	26,200	37,200	31,546	13,454	190	"	"
45	225	29,475	40,475	34,323	10,677	151	"	"
50	250	32,750	43,750	37,100	7,900	111	"	"
55	275	36,025	47,025	39,877	5,123	72	"	"
60	300	39,300	50,300	42,654	2,346	33	"	"
61	305	39,955	50,955	43,210	1,790	25	"	"
62	310	40,610	51,610	43,765	1,235	17	"	"
63	315	41,265	52,265	44,321	679	10	"	"
64	320	41,920	52,920	44,876	124	2	"	"
65	325	42,575	53,575	45,432	-432	6	Slow up	"
66	330	43,230	54,230	45,987	-987	14	"	"
67	335	43,885	54,885	46,542	-1,542	22	"	"
68	340	44,540	55,540	47,098	-2,098	30	"	"
69	345	45,195	56,195	47,653	-2,653	37	"	"
70	350	45,850	56,850	48,209	-3,209	45	"	"
71	355	46,505	57,505	48,764	-3,764	53	"	"
72	360	47,160	58,160	49,320	-4,320	61	"	"
73	365	47,815	58,815	49,875	-4,875	69	"	"
74	370	48,470	59,470	50,431	-5,431	77	"	"
75	375	49,125	60,125	50,986	-5,986	84	"	"

4.3.2.3 *Measurement of pilot hole deviation*

Since the planned pilot hole length was 105.0m (344.5ft), and at 106.68m (350ft) the hole did not reach the intended holing point at 3140ft-level, the drilling was stopped. The rods were pulled out and the management, at the request of the author, sanctioned the survey for possible deviation.

4.3.2.3.1 *Measurement technique used*

The multi-shot in-hole camera was used for the hole survey. Technical specifications (Table 4.21) and principle of operation are explained below.

Table 4.21 Data on hole deviation measurement Instrument [Source: Instruction booklet, Bullet 681]

Instrument	Directional Survey Instrument Type DT, Germany made
Camera length	432mm
Camera diameter	31.75mm
Ranges of angle unit	$0 - 12^0$, $0 - 17^0$ and $5 - 90^0$
Battery unit	8 cells by 1.5V = 12V
Barrel	Non-magnetic high tensile tube
Light bulbs	G.E No. 330 Bulb T 1¾ S.C Midflanged 0.08 Amps, 14 volt – 15.9mm overall length
Film	16mm Positive Safety, cut to 10mm
Capacity of film spool	2.6m
Size of photos	6.1mm
Photos per foot of film	50
Film capacity in hours	6.5 with 1 minute cycle of operation 13 with 2 minutes cycle of operation
Exposure time	9 to 10 seconds

The camera employed uses photo techniques based on optics. The instrument consists of:

a) A battery section to provide power for lights which illuminates the compass in order to provide power for the electronic timer.
b) A camera section which contains a chamber for the multiple shot film disc, and a simple pre-focussed lens and lamp unit. This is the main part of the instrument.
c) A timer which synchronises the film advance and light illumination to enable a continuous film advance and exposure at a set time interval, from instrument on set to switch-off. The camera which was used had a timer set at one minute interval, meaning that a shot was taken every after one minute.
d) An angle/compass unit in combination.

The instrument is powered by dry cell batteries, which operate a small electric motor. The electric motor and the bulbs are supplied with current by a battery of 12volts, which is installed in a battery section of the outer barrel. Through a contact timing watch and gear train, the movement of the film and the timing of the exposure lights are actuated so as to take a series of pictures of an angle unit showing the direction and the inclination of the bore hole. Type DT instrument takes up to 400 exposures during one survey operation.

To carry out a survey, the instrument is assembled for use. The instrument is then lowered into the bore hole in a protective barrel. The barrel is normally 42mm diameter and is pressure proof against 330 bar ($\approx$ 4500Psi). The instrument can function up to about 250^0 F ($\approx 107^0$C).

The instrument records a magnetic bearing, but by orienting the instrument on drill pipe or tubing, a correct bearing can be obtained and photographed in cased holes and in disturbed magnetic formations.

The accuracy of the instrument depends upon:

a) The distance between the individual surveying points, and
b) The alignment of the instrument with the bore hole. This is influenced by the hole size, the angle of the bore hole, and stiffness of the surveying mechanism.

The accuracy is within one degree if not affected by magnetic disturbances. Location of the bottom of the hole is within a radius of approximately 1.5m for 305m of depth.

The $5^0 - 90^0$ angle unit, which was used, had a line through the centre of the disc photos and graduations around the circumference. The inclination of the bore hole is read from the intersection of the horizontal cross hairs and the calibrated vernier scales. The magnetic bearing or azimuth is read from the intersection of the vertical cross hairs, and the compass card.

For reading and tabulation of the results, the following instruments were used:

a) Stop watch
b) Measuring tape
c) Magnifying glass, and
d) Waterproof marker.

In this study, *True North* at any point is the direction towards the geographic North Pole of the earth, i.e. the direction of the meridian through the point. *Magnetic North* or *magnetic meridian* at any point is the direction taken up by a freely suspended compass needle and it differs from the true North. The angle between true North and magnetic North is called the magnetic *variation or declination* (Figure 4.20) and it varies periodically depending on the strength of the magnetic field. The angle is East or West according to whether the compass needle points East or West of true North. The declination to be used in calculations can be found using isogonic chart (lines of equal declination) or can be obtained from Survey General's office.

To correct the magnetic readings, the declination is either added or subtracted (Table 4.22). The correction factor is applied to the bearing according to whether the declination lies East or West of true North (Figure 4.20).

Table 4.22 Correction of magnetic bearing using declination

QUADRANT	EAST DECLINATION	WEST DECLINATION
NE	Add to bearing	Subtract from bearing
SW	Add to bearing	Subtract from bearing
SE	Subtract from bearing	Add to bearing
NW	Subtract from bearing	Add to bearing

MM = TN +/– (Declination) or TN = MM +/– (declination)

Where: MM = Magnetic meridian
TN = True North

The bearing of an observed distant point is the angle between North and the line from the observer's position to the point. The bearing is stated, however, as not merely the angle between the two directions, but as the amount of angular rotation made turning from the northerly reference direction to the direction of the observed point. The northerly reference direction may be true or magnetic north, and the rotation is measured in a clockwise direction.

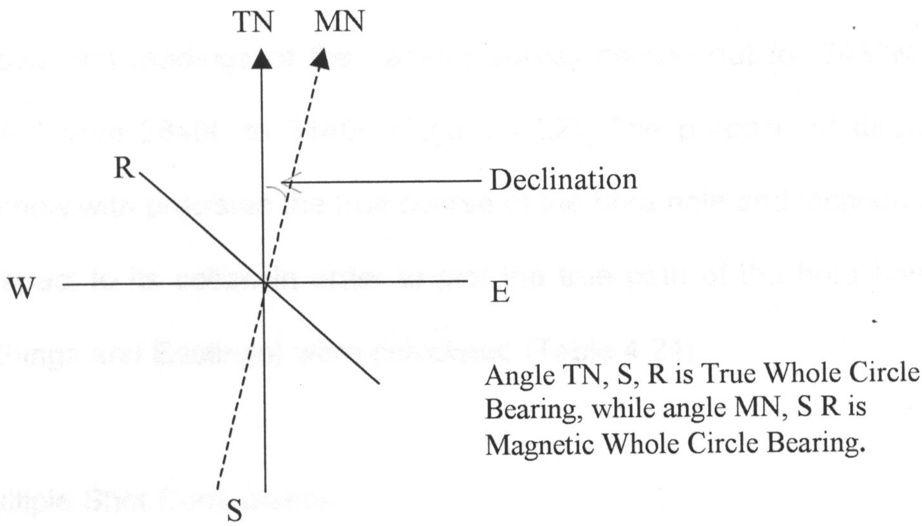


Figure 4.20 Illustrating the TN, MN, declination, True and Magnetic whole circle bearing (WCB).

Nadir is defined as a point on celestial sphere directly below an observer and diametrically opposite the *zenith*. The direction of the line through the instrument centre to the centre of the earth (downwards) is termed the *nadir*, and the opposite direction vertically upwards from the instrument centre is the *zenith* (Figure.4.21). Results of measurements presented here were taken underground.

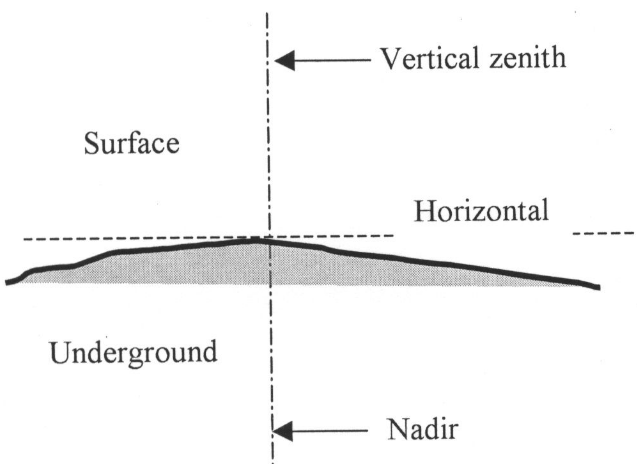


Figure 4.21 Illustrating the terms nadir and zenith

4.3.2.3.2 *Data obtained from hole survey*

Table 4.23 shows film readings of the camera survey carried out for 2430N raise borer hole drilled from 2840L to 3140L (Figure 4.22). The purpose of directional survey was to know with precision the true course of the bore hole and location of the bottom with respect to its collar. In order to plot the true path of the bore hole, co-ordinates (Northings and Eastings) were calculated (Table 4.24).

Example of Multiple Shot Computation:

Vernier scale reading: 32 degrees from Nadir

Magnetic bearing (Azimuth): 15 degrees SW

$\therefore$ true bearing = 90 – 32 = 58 degrees

Magnetic bearing (WCB) = 180 + 15 = 195 degrees

True bearing (WCB) = 195 – 8 (declination) = 187 degrees.

Table 4.23 Camera survey film readings

Shot No.	Hole length (m)	Inclination From Nadir (degrees)	Magnetic Bearing (degrees)	Inclination True (degrees)	Bearing (°T)
1 – 10	Moving				
11	5.60	32	15 SW	58	187
12	9.40	32	15	58	187
13	13.20	32	15	58	187
14	17.00	32	15	58	187
15	20.80	32	16	58	188
16	24.60	32	16	58	188
17	28.40	32	16	58	188
18	32.20	32	16	58	188
19	Moving				
20	36.00	34	18	56	190
21	39.80	34	18	56	190
22	43.60	34	18	56	190
23	47.40	34	18	56	190
24	51.20	34	20	56	192
25	55.00	34	20	56	192
26	58.80	34	20	56	192
27	62.60	34	20	56	192
28	66.40	35	20	55	192
29	70.20	35	20	55	192
30	74.00	35	20	55	192
31	77.80	35	20	55	192
32	81.60	35	20	55	192
33	85.40	35	20	55	192
34	89.20	40	28	50	200
35	94.00	40	28	50	200
36	97.80	40	32	50	214
37	101.60	41	34	47	216
38	105.40	42	35 SW	48	217
39	Moving				
40	Moving				

Using collar co-ordinates (33873.618mE and 31859.548mN), elevation of 410.00 and data in Table 4.23, the co-ordinates at different depth of the pilot hole were calculated. The results are shown in Table 4.24.

Table 4.24 Film readings and calculated results

FILM READINGS			RESULTS		
Hole length (m)	Dip (degrees)	Bearing (degrees)	Hole length (m)	Northings (co-ordinates)	Eastings (co-ordinates)
5.60	58.0	187.00	4.7	31,864.3	33,873.3
20.80	58.0	188.00	17.6	31,856.3	33,872.1
36.00	56.0	190.00	33.4	31,847.9	33,870.7
51.20	56.0	192.00	48.6	31,839.5	33,868.9
66.40	55.0	192.00	63.7	31,831.0	33,867.1
89.20	50.0	200.00	83.9	31,816.6	33,862.1
97.80	50.0	214.00	95.8	31,811.4	33,859.0
101.60	47.0	216.00	100.6	31,809.3	33,857.5
105.40	48.0	217.00	104.4	31,807.2	33,855.9
			105.4	31,807.2	33,855.9

Using the co-ordinates at 105.4m (Table 4.24), the bottom of the hole was calculated to be 10.8m south and 4.5m west of the intended holing point.

Figure 4.22 shows the 'actual path' of the pilot hole (continuous line). The path was plotted using the data in Table 4.24. The planned path is shown in broken line.

4.3.2.4 *Interception of the hole using unplanned development*

The hole deviation led to developing a cross cut in waste in an effort to try and intercept the hole. The cross-cut was estimated to intercept the hole at 8m, but it advanced 12m without intercepting the hole. Using the 'sounding method', the pilot hole drilling sound was heard in the water drive. The hole was then deepened and holed at the length of 108.6m.

Figure 4.22 shows the section of planned and actual starting positions of the pilot hole while Figure 4.23 shows the plan of planned and actual holing positions. Crosscut D2 (Figure 4.23) was developed in an attempt to intercept the hole.

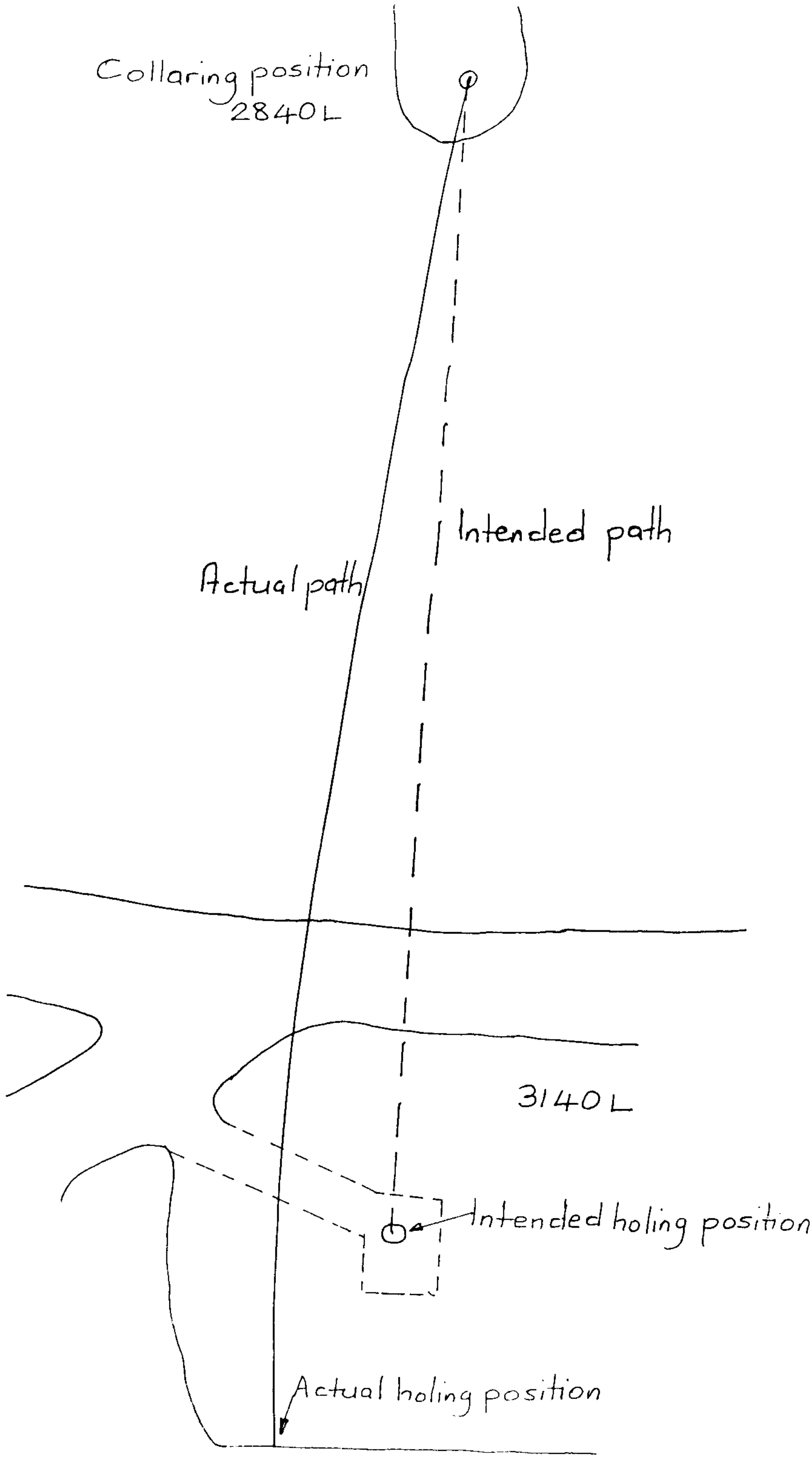


Figure 4.22 Planned and actual starting and holing positions

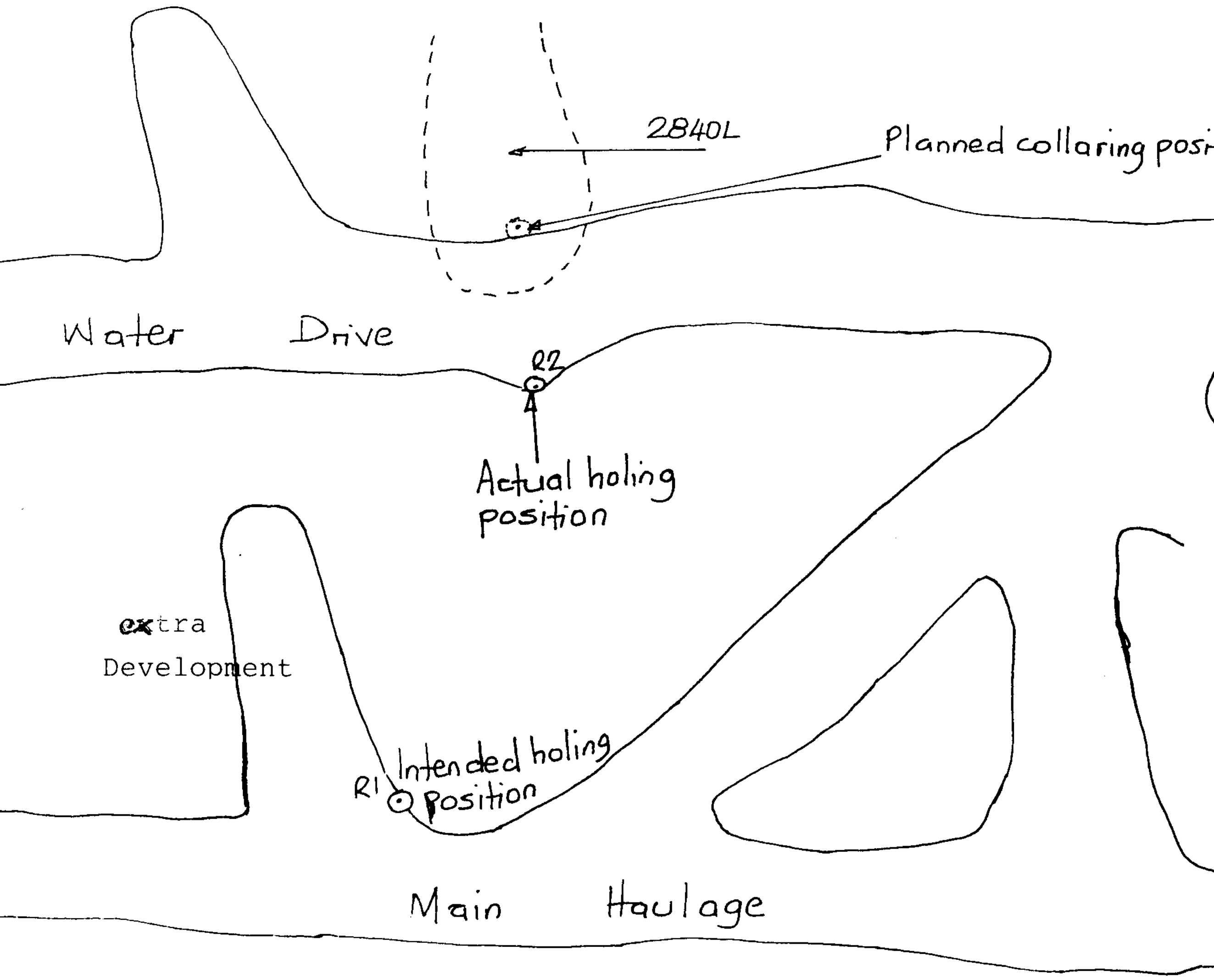


Figure 4.23 Planned and actual holing positions

4.3.2.5 *Likely causes of pilot hole deviation*

The following were observed as the main causes of the pilot hole deviation:

a) Inadequate number of stabilizers.

Drilling of the pilot hole for 2430N ore tip raise required at least six stabilizers (ZCCM-Nkana Division, Technical Department, 1998), but because of shortage of stabilizers, only two were installed. Stabilizers maintain the inclination angle of a hole by controlling the location of contact points between the hole and the drill-string. The stabilizer 'rods' also reduce the deflection of drill-string as they are stiffer than 'normal' drilling rods.

b) Drilling pressure

The recommended pilot drilling pressure (Table 4.20) was often not attained because the pressure gauges were often not functioning. This meant that more pressure than necessary may have been applied resulting in the deflection of the drill-string, which might have contributed to the hole deviation.

c) Insufficient flushing of circulation fluid

Water circulation never exceeded 20 litres per minute (Section 4.3.2.2) throughout pilot drilling. The recommended flow-rate was 25 litres per minute. The air pressure was on average 150Psi (10.3bar) instead of the recommended 200 to 250Psi (14 to 17bar). Flushing of circulation fluid is important in rock drilling because it has a direct bearing on drilling. Insufficient flushing of the fluid may result in hole deviation if the bottom of the hole is not properly cleaned, as the bit will not evenly cut the rock. The fluid also contributes to supporting the weight of drill-string, resulting in less string weight and hence hole deflection.

d) Non-uniform rock formations

The change in rock formations might have contributed to hole deviation. The top 42m of the hole was drilled in banded shale with compressive strength of about 90MPa (Section 4.2.1) while the rest of the hole was drilled in sandstone with compressive strength of 190MPa. The change in rock strength could have contributed to the deviation.

e) Inclined pilot hole

The pilot hole was drilled at 58 degrees inclination. This might have contributed to gravity-induced deflection of drill-string resulting in hole deviation. It has been observed that large diameter holes deviate downwards in the vertical plane [Sinkala, 1996], due to gravitational forces.

f) Defective gears

Robbins 61R raise-borer 1141 has three driving gears. A reverse and two forward driving gears. The two forward gears (low and high speeds) were jamming. This might have resulted in inappropriate application of thrust for different rock formations and different depths, thus contributing to the observed hole deviation.

4.3.2.6 *Consequences associated with pilot hole deviation*

The pilot hole deviation resulted in the following consequences:

a) Unplanned extra drilling. Instead of 105m, the raise holed at 108.6m

b) Time wastage

c) Extra use of drilling accessories – bits, couplings, oil, etc.

d) *Extra development – drilling, blasting and development operations*

e) Extra mucking and transportation of waste

f) Accelerated wear of equipment

g) Extra hoisting of waste, and

h) Extra man-hours.

These consequences are quantified in Section 5.2.1.1

CHAPTER FIVE

5.0 HOLE DEVIATION CONSEQUENCES IN MONETARY TERMS

The consequences associated with hole deviations have an adverse economic significance on mining economy and contribute to high production costs [Sinkala,1989]. An example of cost items for some of these consequences is shown in Table 5.1, where x denotes an occurrence of a cost. This part of the study attempts to estimate the cost of consequences identified during:

a) the drilling of stope and raise pilot holes, and

b) subsequent operations.

Table 5.1 Extra costs in mining operations associated with hole deviations [Adapted from Sinkala, 1990].

Extra Cost Item	Unit Base cost	SOURCES OF EXTRA COSTS				
		Build-ups	Hang-ups	Poor Fragment	Extra Develop.	Others
Drilling						
Cubex	X	X	X	X	X	
Quasar	X	X	X	X	X	
LHD Drifter	X	X	X	X	X	
Bits: 57mm	X				X	
127mm	X	X	X	X		
165mm	X	X	X	X		
Rods: 915mm	X				X	
959x42mm	X				X	
1759x41mm	X	X				
2559x40mm	X	X				
Couplings R32	X				X	
Pilot Adaptor	X				X	
Explosives						
ANFO	X					
Slurry	X	X	X	X		
Magnum	X	X	X	X		
Accessories	X	X	X	X	X	
Ore dilution	X			X	X	
Ore loss	X	X			X	
Delay time	X	X	X	X	X	X
Loaders						
250D	X			X	X	
300D	X					
350D	X					
Mucking	X			X	X	
Conveyors	X			X		
Crushers	X			X		X
Rock support	X					X
Hoisting	X					X
Min processing	X					X
Labour	X	X	X	X	X	X
Maintenance	X	X	X	X	X	X
Services	X	X	X	X	X	X
Others	X	X	X	X	X	X
Total Cost	Σ	Σ	Σ	Σ	Σ	Σ
OVERALL TOTAL COST			=	(Σ Total)		

A clear quantification of the costs of these consequences can be useful in targeting unit operations where improvements through either reducing or eliminating the causal effects would make significant savings in the overall mining operational costs.

A model for quick or 'on-the-spot' estimates of the unit operational costs would be very useful for mines, thus enabling a timely decision towards minimising operational costs. This may be achieved by quick decision on alternatives included in 'timely decision' in order to reduce the anticipated extra costs in downstream operations. The alternatives include:

a) Controlled drilling aiming at reducing hole deviations.

b) Re-planning of blasting patterns

c) Re-drilling of problem areas.

d) Combination of one or more of the above alternatives.

5.1 Methods Used in Cost Estimation of Extra Costs Due to Consequences of Hole Deviations

In estimating cost of consequences associated with hole deviations, the cost items for the consequences (Table 5.1) were identified. The unit costs of items were obtained from various sources, including equipment manufacturers, accounts section of ZCCM-Nkana Division and London metal exchange (LME) market. The unit prices were based on the price levels of 1998.