

LIST OF TABLES

<u>TABLE</u>	<u>Page</u>
1.3.1 Authorized capital expenditure for TM3 projects	10
1.3.2 Employee requirements for a 35 000 ton project	11
1.3.3 Average total labour for REGM	12
1.3.4 Waste development cost savings	13
1.3.5a Tonnage split from underground sources	14
1.3.5b Tonnage split from underground sources	15
1.3.5c Tonnage split from underground sources	16
1.3.6a Cost per ton and recovered grade	17
1.3.6b Cost per ton and recovered grade	18
1.3.6c Cost per ton and recovered grade	19
1.3.7 Life of mine reserves as at 31 March 1989	20
1.4.1 Average increase in cost for machinery vs. labour	22
2.1.1 Fatal and reportable statistics 1975-1994 (REGM)	34
2.1.2 Fatal and reportable statistics 1969-1994 (Gold members, Chamber of Mines)	35
2.1.3 Reportable statistics 1985-1994 (Cooke 1 & 2)	36
2.2.1 Labour complements, rockdrill shop	39
2.6.1 Removal of TM3 equipment from u/g. Labour and shift requirements	51
2.6.2a TM3 machines taken out from underground	53
2.6.2b TM3 machines taken out from underground	54
2.8.1 Composition of mine overheads	67

LIST OF TABLES

<u>TABLE</u>	<u>Page</u>
1.3.1 Authorized capital expenditure for TM3 projects	10
1.3.2 Employee requirements for a 35 000 ton project	11
1.3.3 Average total labour for REGM	12
1.3.4 Waste development cost savings	13
1.3.5a Tonnage split from underground sources	14
1.3.5b Tonnage split from underground sources	15
1.3.5c Tonnage split from underground sources	16
1.3.6a Cost per ton and recovered grade	17
1.3.6b Cost per ton and recovered grade	18
1.3.6c Cost per ton and recovered grade	19
1.3.7 Life of mine reserves as at 31 March 1989	20
1.4.1 Average increase in cost for machinery vs. labour	22
2.1.1 Fatal and reportable statistics 1975-1994 (REGM)	34
2.1.2 Fatal and reportable statistics 1969-1994 (Gold members, Chamber of Mines)	35
2.1.3 Reportable statistics 1985-1994 (Cooke 1 & 2)	36
2.2.1 Labour complements, rockdrill shop	39
2.6.1 Removal of TM3 equipment from u/g. Labour and shift requirements	51
2.6.2a TM3 machines taken out from underground	53
2.6.2b TM3 machines taken out from underground	54
2.8.1 Composition of mine overheads	67

LIST OF FIGURES

<u>Figure</u>	<u>Page</u>
1.3.1 Operating parameters for 35 000 ton per month	6
2.8.1 Approach to workshop.	62
2.8.2 Competitor analysis.	65
2.8.3 Theoretical business model.	72
2.8.4 As is model.	73
2.8.5 Opportunity model.	74
2.8.6 Opportunity model.	75
2.8.7 Paterson banding.	76
2.8.8 Present structure	77
2.8.9 Proposed structure	78
2.8.10 Horse shoe effect	88

<u>APPENDIX</u>		<u>Page</u>
APPENDIX 51	JCI ANNOUNCES RATIONALISATION AT REGM	143
APPENDIX 52	MANAGEMENT BRIEF TO ALL EMPLOYEES	144
APPENDIX 53	SURFACE TM3 TRAINING SCHOOL	145
APPENDIX 54	SURFACE TM3 TRAINING SCHOOL	146
APPENDIX 55	SECTION SHOWING TRAINING STOPES	147
APPENDIX 56	PLAN SHOWING TRAINING STOPES	148
APPENDIX 57	UNDERGROUND CONVENTIONAL TRAINING SCHOOL	149
APPENDIX 58	UNDERGROUND CONVENTIONAL TRAINING SCHOOL	150
APPENDIX 59	UNDERGROUND CONVENTIONAL TRAINING SCHOOL	151
APPENDIX 60	TYPICAL TM3 LAYOUT	152
APPENDIX 61	TYPICAL NARROW TM3 LAYOUT	153
APPENDIX 62	TYPICAL CONVENTIONAL LAYOUT	154
APPENDIX 63	JCI MINES VS COMPETITORS	155
APPENDIX 64	REGM ORE DISTRIBUTION	156
APPENDIX 65	EXTERNAL DRIVERS MARKET ANALYSIS	157
APPENDIX 66	REGM WORKING COST COMPONENTS	158
APPENDIX 67	TIME UTILISATION DAY-SHIFT STOPING	159
APPENDIX 68	TIME UTILISATION NIGHT-SHIFT STOPING	160
APPENDIX 69	TIME UTILISATION DAY-SHIFT DEVELOPING	161
APPENDIX 70	TIME UTILISATION NIGHT-SHIFT DEVELOPING	162
APPENDIX 71	kWh POWER USED	163
APPENDIX 72	INTERNAL DRIVERS FOR CHANGE	164

<u>APPENDIX</u>		<u>Page</u>
APPENDIX 26	NUMBER OF FATALITIES, CHAMBER OF MINES	118
APPENDIX 27	FATALITY RATE GRAPH, CHAMBER OF MINES	119
APPENDIX 28	NUMBER OF REPORTABLES, CHAMBER OF MINES	120
APPENDIX 29	REPORTABLES RATE GRAPH, CHAMBER OF MINES	121
APPENDIX 30	HAZWISE CAMPAIGN SEQUENCE	122
APPENDIX 31	HAZWISE SYMBOL	123
APPENDIX 32	PRE-LAUNCH POSTERS :WHAT NOW ?	124
APPENDIX 33	PRE-LAUNCH POSTERS :WHAT'S HAPPENING	125
APPENDIX 34	PRE-LAUNCH POSTERS :EAGLE KILLING SNAKE	126
APPENDIX 35	HAZWISE CENTRE COOKE 1	127
APPENDIX 36	HAZWISE CENTRE COOKE 1	128
APPENDIX 37	HAZWISE CENTRE COOKE 1	129
APPENDIX 38	HAZWISE CENTRE COOKE 1	130
APPENDIX 39	HAZWISE CAMPAIGN POSTER	131
APPENDIX 40	HAZWISE CAMPAIGN POSTER	132
APPENDIX 41	HAZWISE STICKERS AND RUBBER STAMPS	133
APPENDIX 42	ACTION PLAN JHA	134
APPENDIX 43	JOB HAZARD ANALYSIS WORKSHEET	135
APPENDIX 44	CARDS HANDED OUT TO EMPLOYEES	136
APPENDIX 45	COOKE 1 NUMBER OF REPORTABLES GRAPH	137
APPENDIX 46	COOKE 2 NUMBER OF REPORTABLES GRAPH	138
APPENDIX 47	PAYLIMIT CALCULATION	139
APPENDIX 48	PAYLIMIT CALCULATION	140
APPENDIX 49	PAYLIMIT CALCULATION	141
APPENDIX 50	PAYLIMIT CALCULATION	142

<u>APPENDIX</u>		<u>Page</u>
APPENDIX 1	LOCALITY PLAN OF REGM	93
APPENDIX 2	PLAN OF LEASE AREA	94
APPENDIX 3	STRATIGRAPHICAL COLUMN OF COOKE SECTION	95
APPENDIX 4	WEST-EAST SECTION OF COOKE AND DOORNKOP	96
APPENDIX 5	EAST-WEST SECTION COOKE 2 SHAFT	97
APPENDIX 6	EAST-WEST SECTION COOKE 3 SHAFT	98
APPENDIX 7	EAST-WEST SECTION COOKE 1 SHAFT	99
APPENDIX 8	NORTH-SOUTH SECTION DOORNKOP SHAFT	100
APPENDIX 9	AVERAGE MONTHLY TONNAGE FROM UNDERGROUND	101
APPENDIX 10	AVERAGE MONTHLY TM3 PERCENTAGE MINED	102
APPENDIX 11	AVERAGE MONTHLY GRADE FROM UNDERGROUND	103
APPENDIX 12	AVERAGE MONTHLY COST PER TON FROM U/G	104
APPENDIX 13	COST INCREASES LABOUR VS MACHINERY	105
APPENDIX 14	NARROW TRACKLESS AT COOKE 3	106
APPENDIX 15	DILUTION OF ASG VS ASG	107
APPENDIX 16	STOMEC	108
APPENDIX 17	STOMEC	109
APPENDIX 18	STOMEC	110
APPENDIX 19	STOMEC COST PER HOUR	111
APPENDIX 20	STOMEC AVAILABILITY VS. UTILIZATION	112
APPENDIX 21	ANNUAL TONNAGE PRODUCED BY STOMEC	113
APPENDIX 22	NUMBER OF FATALITIES, REGM	114
APPENDIX 23	FATALITY RATE GRAPH, REGM	115
APPENDIX 24	NUMBER OF REPORTABLES, REGM	116
APPENDIX 25	REPORTABLE RATE GRAPH, REGM	117

	<u>Page</u>
2.4.5 The letter of agreement	45
2.4.6 Conclusion to section 2.4	46
2.5 Retraining a new workforce	48
2.5.1 The cost of setting up a new training centre	48
2.5.2 Conclusion to section 2.5	50
2.6 Scrapping and removal of trackless equipment	51
2.6.1 Conclusion to section 2.6	52
2.7 Excessive regional support due to trackless mining	55
2.7.1 Extraction ratio	55
2.7.2 Dilution and grade	56
2.7.3 Conclusion to section 2.7	58
2.8 Reengineering REGM's workforce	59
2.8.1 Why reengineer?	59
2.8.2 Who will reengineer?	59
2.8.3 Past results of reengineering	60
2.8.4 Selection of test site	61
2.8.5 The Process :External drivers for change	61
2.8.6 Internal drivers for change	66
2.8.7 BPR visions	85
2.8.8 Conclusion to section 2.8	89
3 <u>CONCLUSIONS AND RECOMMENDATIONS</u>	90
 APPENDIX	 93
REFERENCES	165

CONTENTS	Page
TITLE	i
DECLARATION	ii
ABSTRACT	iii
ACKNOWLEDGEMENTS	iv
CONTENTS	v
LIST OF APPENDICES	vii
LIST OF FIGURES	x
LIST OF TABLES	xi
LIST OF SYMBOLS	xii
LIST OF NOMENCLATURE	xiii
 1 <u>INTRODUCTION</u>	 1
1.1 History of REGM	1
1.2 Geological overview of REGM	2
1.2.1 Structure	2
1.2.2 Economical reefs mined on Cooke Section	3
1.3 The necessity of changing to trackless mining	5
1.4 The rationale for changing the mining mix	21
1.4.1 Sequence of mining	23
1.4.2 The STOMEC H.25.2	24
1.4.3 Reasons for the removal of the STOMEC H.25.2	25
1.4.4 Chapter summary	27
 2 <u>FORMULATION OF THE PROBLEMS ENCOUNTERED AND SOLUTIONS TO THESE PROBLEMS.</u>	 29
2.1 Expected deterioration in safety statistics	29
2.1.1 The sequence of events	30
2.1.2 The "Hazwise programme"	30
2.1.3 Conclusion to section 2.1	32
2.2 The cost of changing back to conventional mining	37
2.2.1 Conclusion to section 2.2	38
2.3 Implications of a reduced tonnage	40
2.4 Dealing with the "redeployment" of a highly skilled Workforce	41
2.4.1 Introduction	41
2.4.2 Measures taken to avoid retrenchment	41
2.4.3 The sequence of events	42
2.4.4 The disruption to the Mine	45

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Abstract

The purpose of this project is to analyse the problems encountered when converting from a mix of trackless mechanised mining methods, which at one time produced up to 80 percent of tons mined, to a more conventional type of mining. Some of the problems encountered were, the safety aspect, the cost of converting back to conventional, the implications of a reduced tonnage on ore-reserves and paylimits, excess skilled labour and redundant equipment.

It is said that trackless mining is safer than conventional. This became apparent when the mining mix changed to a more conventional type and the safety performance deteriorated. This in turn motivated a new safety programme (Hazwise) to be introduced at Randfontein Estates Gold Mine. As a result safety actually improved in some areas where the change back to conventional mining took place.

The details of how the redeployment of some 700 skilled employees was done with relatively minor disruptions will be shown.

Also of concern is the cost effectiveness of scrapping and removing trackless equipment to surface. A complete breakdown is shown and depending on the type of machine, whether it is profitable or not.

Trackless roadways are large excavations and therefore require protection, which normally has to be left for the life of the project. Many times this protection is by way of regional pillars, the disadvantage of these being the lock up of gold. The saving in regional pillars more than pays for the conversion back to conventional mining due to recovering the high grade reef.

As a consequence of the down-scaling of trackless mining, Randfontein Estates is also embarking on a "Re-engineering" programme in order to stay competitive. A more conventional type of mining means a lower volume and higher cost per ton which could seriously affect the present large ore reserves of low grade reef and life of the mine. The Re-engineering concepts which have been proposed will be discussed.

DECLARATION

I declare that this project report is my own, unaided work. It is being submitted for the degree of Master of Science in Engineering in the University of the Witwatersrand, Johannesburg. It has not been submitted before for any degree or examination in any other University.

30 day of November 1995

Bonn

AN ANALYSIS OF THE PROBLEMS
ENCOUNTERED WHEN CONVERTING
TRACKLESS MINING BACK TO A
MORE CONVENTIONAL TYPE OF
MINING ON RANDFONTEIN ESTATES
GOLD MINE

Andrew Brown

A project report submitted to the Faculty of Engineering,
University of the Witwatersrand, in the fulfillment of the
requirements for the degree of Master of Science in
Engineering.

Johannesburg, 1995

TM3 PROJECTS, CAPITAL EXPENDITURE

SHAFT	LEVEL PROJECT	DATE AUTHORIZED	AMOUNT AUTHORIZED
COOKE 1	90 E 8	20-05-86	7 720 000
	101 W/B	25-04-86	10 381 000
	106 K9A	23-05-86	8 958 000
COOKE 2	90 W	22-05-86	7 115 000
	95 N	25-04-86	10 620 000
COOKE 3	118 N	20-04-86	6 121 000
	106 N	26-05-86	6 166 000
	106 S	26-05-86	14 131 000
T O T A L			R71 212 000

table 1.3.1 Authorized Capital expenditure

Referring to table 1.3.5c, it can be seen that there was a significant drop in the percentage trackless mining from January 1995. This was due to the enforcing of the correct classification of mining method, that is, by trackless means or conventional means.

In the next section 1.4 it will be shown why the decision to move away from a high percentage of trackless mining was taken.

ix). This mining method results in lower working costs, improved productivity, greater flexibility of mining, the cost effective mining of lower grade reef and the diminished exposure to labour unrest.

x). Improved environmental control due to large areas of the mine being sealed off.

xi). Savings on general underground equipment.

By referring to appendix 62 it will be seen that a large portion of the ore-reserve lies between 2.0 and 4.0 g/t. This was another consideration taken when changing to TM3. The only way to drop the paylimit at the time was to go for volume. By referring to the graph on grade (appendix 11) and tons (appendix 9), it can be seen that the grade dropped to a low of 3.5 g/t and the total Mine's tonnage increased to 600 000 tons per month.

Referring to appendix 10 the build up of TM3 took almost four years from 1986 to 1989. Trackless mining got up to a high of 518 000 tons per month in mid 1990 then started a decline down to the present day ie. mid 1995 of some 170 000 tons per month (table 1.3.5a/b/c). The reasons why this occurred will be discussed in section 1.4.

The cost per ton (table 1.3.6a/b/c read with appendix 12) shows a steady rise with the introduction of trackless mining. However from 1992 to June 1995 there were larger increases in the cost per ton due to reducing tonnages.

Referring to table 1.3.7 it can be seen that the tremendous increase in tons and kilograms are largely due to the introduction of trackless mining. The overall grade of REGM only drops by 6 % and the tons made available increases by 113 % (H.Scott-Russell, 1989).

Scott-Russell also made four very good points on the change to trackless.

- 1) From a technical and financial aspect, the decision to adopt a TM3 strategy was correct.
- 2) The fundamental asset of the company namely the ore reserve, is preserved through availability of ores which otherwise are sub-economic by conventional methods.
- 3) The life of mine is extended by TM3 methods.
- 4) It is absolutely necessary to fill the plant (and hoisting) capacities from underground sources of ore.

Referring to the table 1.3.3 the average labour complement prior to the introduction of trackless mining was approximately 17 000 people. This dropped down to an average of some 14 500 people when trackless mining was at its highest level of production.

Another significant reduction was made in the amount of waste development which had to be done. This enabled existing shafts to hoist more reef tonnage. For the equivalent projected tonnage to be produced by the TM3 projects some 92 808 metres of conventional development would have had to be completed. This would have cost R68 690 000 in 1988 terms. The TM3 only required 21 646 metres of development at a cost of R23 700 000. This meant a saving of R45 030 000 (table 1.3.4)(R.I.Klintworth, L.J.Cox,1988).

The safety aspect was a valid reason for changing to TM3. If one does an analyses of the graphs (appendix 24) on reportable injuries, it can be seen that there was a huge drop from 1986, then a steady climb until 1993 when the "hazwise" programme was introduced.

According to H.Scott-Russell (1987), there were many non-quantifiable benefits derived from TM3. They were.

i). Increased shaft availability resulting from reduced waste hoisting resulting in increased reef hoisting and throughput if extra milling capacity installed.

ii). The reduced, but highly-skilled, labour force results in greater stability and a need for local labour as opposed to migratory labour.

iii). Concentrated working areas will result in improved underground environment and supervision.

iv). The concentration of backfilling will reduce fire hazards and, therefore, insurance premiums.

v). Better supervision and the concentration of working areas will reduce gold losses.

vi). Tonnages previously inaccessible to conventional mining methods will now be available resulting in an extended mine life.

vii). Mining operations may now be planned to deliver a constant head grade to the mills.

viii).Trackless mining will result in reduced compressed air and water consumption.

in May 1986 with an authorized capital of R7 115 000. For the total Mines capital refer to table 1.3.1.

35 000 tons per month was determined as an ideal tonnage for a project and this was manned by 241 people as opposed to 557 for a conventional section of equal tonnage (table 1.3.2). This reduction in labour amounted to a saving of some R6.69 per ton (R.I.Klintworth, L.J.Cox, 1988).

The fleet of equipment necessary for a 35 000 ton per month TM3 project consisted typically of the following. Referring to figure 1.3.1 (H.Scott-Russell, 1989).

Operating parameters - 35 000 tons per month

- a) Double shift operation.
- b) 3 x Atlas Copco Boomer H.127
Rated 12 500 tons per rig per production month.
- c) Load Haul Dump (LHD)
Primary movers: 3 x 4,6 cu.m
Maximum tram: 150 metres one way.
- d) Articulated dump truck (ADT)
3 x Volvo BM 861 or Wagner/Toro equivalent.
- e) Utility vehicles
4 x PK 1 000 / 4 000 prime movers.
- f) Ventilation
Minimum air supply 0.06 cu.m/second/rated KW of diesel equipment.

figure 1.3.1 Operating parameters for a TM3 section

In the mid 1989 when trackless machinery was at its maximum the total complement of equipment was as follows.

- a) Drillrigs = 70
- b) Load haul dump = 89
- c) Articulated dump trucks = 45
- d) Utility vehicles = 172
- e) Road-way construction vehicles and supervision vehicles are included in the total for utility vehicles.

1.3 The necessity of changing to trackless mining

Johannesburg Consolidated Investments Company, as the managing company of REGM had made projections into the 1990's about the uncertainty of gold market prices, a fluctuating exchange rate, escalating working costs, lower grades and the right of every business to make sure of profitability. This had prompted some mining houses to investigate trackless mining (H.Scott-Russell, 1988).

From the description of the geology of Cooke section, it is obvious that a wide range of conditions existed as far as reef widths, dip and structure are concerned to warrant an investigation into trackless mining. In the mid 1980's the need for a different mining method became apparent. A feasibility study on each shaft of the most suitable area to introduce trackless mining was requested by Head Office.

It was recognized that the ideal areas would require the following prevailing conditions:

- 1) Structurally as undisturbed as possible.
- 2) Reef widths preferably in excess of 3 metres to minimize dilution as the ends had to accommodate the large equipment.
- 3) Large enough ore reserves to warrant the establishment of a trackless project.

The areas that could fulfill most of the above were:

Cooke 1, 95 bowl area, on the UE1a.

Cooke 2, 90 level E8 conglomerates.

Cooke 3, 106 UE1a multireef package on the anticline which was up to 20 metres in thickness.

In addition to geological conditions that lend themselves to trackless mining, there are many other considerations that are taken into account. The projected working costs of conventional mining were seen as a very distinct threat to profitability in the 1990's (Smith and Cox, 1988). This definitely applied to REGM as the grades were of a very low value.

The E8 reef on Cooke 2 had all three of the above characteristics, and trackless mining of that area started

in fact are called the a2 and a3 bands.

At Cooke 3 the UE1a consists of a single reef to the west of anticline. On the anticline the reefs diverge and to the east of the anticline three distinct conglomerate bands are developed, ie. the a1, a2, a3, a3b and a5 conglomerates. To the east a quartzite middling is developed between each of the reefs, and obtain a total thickness from the bottom of the a1 to the top of the a5 of some 35 metres. Refer to appendix 6 for a east-west section through Cooke 3. Each of these conglomerates has its own payshoot pattern superimposed on each other. The significance of the diverging conglomerates is that immediately to the east of the anticline the total package can obtain a total thickness of some 21 metres, which has a profound bearing on the mining methods that can be used to extract the entire package.

3) E9gb may occur directly below the UE1a where it is still preserved. These conglomerates occasionally may have pay values and are usually mined with the UE1a.

4) E8 conglomerates which sub-outcrops against the UE1a in the west is some 60 metres below the UE1a in the east (appendix 7). The E8 reefs occur in well defined payshoots, and two distinct phases of sedimentation can be recognised, ie the a and b phases. The a phase is usually developed being the bottom band, and the b phase is sporadically. The conglomerate band varies from 80 cm. to 3 metres. Again the thickness of the reef band to some extent would dictate the mining method used.

5) Kimberley reefs have been mined to a lesser extent on Cooke section. The Kimberly reef package consists of the K7 conglomerates which is made up from the K7a and K7b with a quartzite middling, the K8 shales and conglomerates varies from 1 cm. to several metres. In channels as much as 40 metres thick. The K9 conglomerate, the most important as an auriferous unit may consist of a K7a and K7b band. If the K8 shales is not developed, the K7 may overlay the K9 conglomerate directly. In the latter case the K7 and K9 can be mined as a unit depending on the payability and gold distribution within the conglomerates. Refer to Section of Doornkop (appendix 8).

the hypothesis that the Panvlakte fault is listric and that extensional tectonics was responsible for the faulted and folded nature of the area immediately to the east of the fault and the steep dips to the east.

Gold bearing conglomerates was introduced from the west on all the auriferous bearing reefs, with different fluvial fans feeding into the basin, deposition being in typical braided fluvial fan fashion. The eastern extent of the conglomerate deposition differing greatly for different conglomerates.

1.2.2 The economical reefs mined on Cooke section are:

1) The Ventersdorp Contact reef. This is directly underlying the Ventersdorp lavas and is developed to the south-west of Cooke 2 shaft (appendix 5). It is believed that the well developed reef in this area is due to the reworking of the overlying UE7 conglomerates that is still preserved in this area due to a paleo-synclinal feature prior to the deposition of the VCR. The UE7 conglomerate sub-outcrops against the VCR in the west, and increase in thickness to the east to attain a thickness of some 4 meters at the sub-outcrop against the overlying Black Reef. Currently the VCR/UE7 can be mined to a vertical distance of 50 metres below the Dolomite due to the danger of water ingress into the workings. The dips are generally steep at 45 degrees and are severely faulted.

2) The UE1a conglomerate is the most important economical auriferous conglomerate mined and is developed over the entire property. The eastern extent of the reef is the sub-outcrop against the lavas, while the western extent is the payability which decreases down the paleoslope to the east.

The lithology of the reef differs drastically from north to south due to the fact that a few different fans and entry points can be recognised and that the different paleo-environments had a decisive influence on the reefs' lithology. The base of the UE1a rests on a major unconformity.

The northern part of the area, the UE1a, consists of a single conglomeratic deposit that varies in thickness from 40cm to some 200cm, with dips generally eastwards at 10 to 20 degrees, with local variations. To the south of Cooke 2 are one or two small pebble conglomerate or grit bands which are developed in some areas, and can be equated and

1.2 Geological overview of REGM

Cooke section is situated south of Randfontein and immediately north of its sister mine, Western Areas Gold Mining company (appendix 1, 2). The entire area is overlain by the Melmanie Dolomite that increase in thickness from 0 metres at the edge of the basin to some 600 metres at Cooke 1 to the south of the basin edge. To the west of the lease boundary the north south striking Panvlakte fault is situated with an average upthrow to the west of some 1 000 metres, thus removing the auriferous Elsburg reefs that is mined to the east of the fault. The eastern boundary being the payable extent of the reefs down dip of the paleo slope. Refer to appendix 3 for stratigraphic column of Cooke Section.

1.2.1 Structure

Overlying the Elsburg Quartzites to the west is a wedge shaped Ventersdorp lava, truncating onto the Panvlakte fault, thinning to the east until the dolomite is directly overlying the Elsburg Quartzites. Referring to the east-west sections through Cooke 1, 2 and 3 shafts, a prominent anticline is apparent in the south ie. Cooke 3, extending partly up into the Cooke 2 area, with steep dips immediately to the east and west of the anticline, up to 80 degrees to the west and some 35 to the east. The eastern dips flatten out to an average of 10 degrees, while the western dips remain steep at 25 degrees close to the lava sub-outcrop. Further to the north the dips are gentler at 20 degrees, in the west to gently undulating dips in the far east. However local variation occur where the dips may increase to some 45 degrees (appendix 4). Three different phases of igneous intrusions can be recognised ie.

1) Ventersdorp age dykes that is generally nnw-sse and wnw-ese trending, is a fine grained diabase with a fair amount of alteration.

2) Pilansberg age dykes generally wnw-ese striking, often associated with water, and is often altered and porous.

3) Karoo age dykes, the youngest of the three and are the only dykes that extend into the dolomite and are thus good water barriers that form well defined hydrological compartments. Faulting is found over the entire property, the majority being normal faults, however reverse faults are fairly common. The western part of the property is more faulted than the eastern areas, which is in line with

1 INTRODUCTION

1.1 History of REGM

Gold was said to have been mined at Randfontein firstly by the Pullinger family in 1885.

In August 1886 Cecil John Rhodes investigated Randfontein and was offered the claims for a sum of 7 000 Pounds. However Rhodes did not think it was a viable proposition and he lost an opportunity to get into the West Rand Goldfields.

J.B. Robinson started buying up options and on 7 March 1889 with capital of 2 000 000 Pounds in 1 Pound shares the Randfontein Estates Gold Mining Company, Witwatersrand Limited was registered.

Gold was first produced in September 1892 when 15309 tons at a grade of 13,38 g/t was milled for that year. The Randfontein Section, as it was known, had a high production in 1940 of 4 267 441 tons at a grade of 5,00 g/t. Production reduced gradually until 1966 when it stopped and cleaning up started. However a new lease on life occurred when in September 1970 Dick Cooke turned the first sod at Cooke 1 shaft and the Cooke Section was born.

Just before Cooke 1 was ready for full production, Cooke 2 shaft was started with the first blast in January 1974. Cooke 2 was primarily mined for its uranium content. Cooke Section was now producing 335 000 tons reef and 90 000 tons waste per month.

Poor results came in from the old Randfontein Section and in order to maintain and increase uranium production the first sod was turned for Cooke 3 during May 1980.

Production got up to a high of almost 6 million tons per annum in 1985 and at this time trackless mining started at Cooke 2 (Hocking, 1986).

LIST OF NOMENCLATURE

ADT	Articulated dump truck
ARA	Authority, responsibility, accountability
ASD	Advance strike drive
ASG	Advance strike gully
CIC	Central Indaba Committee
GME	Government Mining Engineer
IS	Integrated systems
JHA	Job Hazard Analysis
LHD	Load haul dump
LIFO	Last in First Out
NUM	National Union of Mineworkers
MBO	Manage by objective
PAT	Paterson Grading of Employees
PC	Personnel computer
RDP	Reconstruction and Development programme
REGM	Randfontein Estates Gold Mine
STOMECH	H25-2 Electro hydraulic stope drill rig
TM3	Trackless Mechanised Mining Method
TPM	Tons per month
UE	Upper Elsburg
VCR	Ventersdorp Contact Reef

LIST OF SYMBOLS

<u>Quantity</u>	<u>Symbol</u>
METRES	m
CENTIMETRES	cms
GRAMS	g
GRAMS PER TON	g/t
METRES SQUARED	m ²
PERCENTAGE	%
OUNCE	oz
CUBIC METRE	cu.m

AVERAGE INCREASES OF MACHINERY VS LABOUR FROM 1987-1995

AVERAGE MACHINERY COST INDICES

AVERAGE LABOUR

INCREASES		INCREASES	
PERCENTAGE		PERCENTAGE	
1987	14.7		15.0
1988	14.5		12.0
1989	21.6		13.5
1990	18.1		13.5
1991	14.2		4.0
1992	9.3		5.0
1993	8.5		8.0
1994	8.0		9.75
1995	7.8 (FORECAST)		10.5 ?

table 1.4.1 AVERAGE INCREASES OF MACHINERY VS LABOUR

1.4 The rationale for changing the mining mix

There were many valid reasons for certain projects cutting back on totally trackless mining. Some of them are.

- i) Devaluation of the Rand. This resulted in a higher than projected cost for TM3 equipment.
- ii). Higher capital cost of TM3 equipment made locally.
- iii). Higher cost of TM3 spares and maintenance.
- iv). Low availability of skilled artisans and foremen necessary to keep up the high standard of maintenance that REGM was used to.
- v). Higher cost of TM3 spares inventory.
- vi). Higher cost of TM3 training.
- vii). Relatively low labour cost increases for conventional mining as opposed to TM3 equipment and spares cost increases (table 1.4.1)(appendix 13).

The initial expectation of 35 000 tons per month for a TM3 project soon became 70 000 tpm after the TM3 projects went through a learning curve. Trackless mining had become more and more efficient at Cooke 3. One of the projects at Cooke 3 had an original ore-reserve of some 8 000 000 tons, which included pillars. It should have lasted some 12 years, however it was largely mined out in 6 years due to efficient TM3 operations.

During 1991 in a section of Cooke 3, a project which was producing some 100 000 tons per month suddenly had a stope width reduction of 3 metres to 1 metre. There was no information available through surface boreholes or footwall development to warn of this eventuality.

The dilution that was being incurred in mining this project was not acceptable and a change in mining method had to be found. Fortunately the gold grade was higher at the reduced stope width. Another consideration was that sufficient backfill was not available for both the South and North sections of Cooke 3. The TM3 method at Cooke 3 relies on backfill for a safe operation.

A new layout was introduced in the North section. This consisted of trackless roadways developed 60 metres apart on strike and conventional raises developed 30 metres apart between these roadways. Panels 30 metres in length

LIFE OF MINE RESERVES AS AT 31 MARCH 1989

GOLD R32 000 PER KG.

CONVENTIONAL			
SECTION	TONS	G/T.	KG
COOKE 1	2 826 000	8.7	24 586 200
COOKE 2	5 733 000	6.1	34 971 300
COOKE 3	8 418 000	5.6	47 140 800
D/KOP.	29 240 000	7.0	204 680 000
REGM	46 217 000	6.7	309 650 000
TRACKLESS			
SECTION	TONS	G/T.	KG
COOKE 1	4 718 000	5.6	26 420 800
COOKE 2	11 384 000	5.4	61 473 600
COOKE 3	20 074 000	5.9	118 430 000
D/KOP.	62 373 000	6.7	417 890 000
REGM	98 549 000	6.3	624 214 000

table 1.3.7 comparison between mining methods

COST PER TON AND RECOVERED GRADE

	TOTAL COST PER TON MILLED SURFACE & U/G	COST PER TON MILLED FROM U/G	GRADE RECOVERED FROM U/G
MAY 92	93.60	120.53	4.49
JUNE	97.02	127.22	4.76
JULY	102.68	133.82	4.76
AUGUST	105.19	139.62	5.38
SEPTEMBER	101.26	134.82	5.42
OCTOBER	103.56	139.90	5.50
NOVEMBER	101.67	134.80	5.26
DECEMBER	103.12	137.64	5.37
JANUARY 93	102.11	144.16	5.41
FEBRUARY	108.45	149.47	5.38
MARCH	107.82	149.40	5.66
APRIL	110.91	154.71	5.72
MAY	109.01	150.48	6.03
JUNF	103.86	143.69	5.33
JULY	115.81	153.91	5.41
AUGUST	110.54	148.77	5.19
SEPTEMBER	107.28	148.42	5.46
OCTOBER	111.13	154.10	5.63
NOVEMBER	114.10	159.74	5.55
DECEMBER	108.90	151.75	6.00
JANUARY 94	112.63	161.73	6.11
FEBRUARY	112.00	157.47	6.46
MARCH	122.09	169.79	6.18
APRIL	122.28	174.43	6.35
MAY	125.60	193.08	5.75
JUNE	126.63	184.75	6.26
JULY	125.65	177.28	5.68
AUGUST	122.07	172.96	5.50
SEPTEMBER	126.32	181.32	5.22
OCTOBER	127.32	186.20	5.28
NOVEMBER	125.35	183.83	5.03
DECEMBER	125.97	185.30	4.97
JANUARY 95	130.80	200.03	4.69
FEBRUARY	122.34	179.75	5.03
MARCH	124.47	179.75	4.80
APRIL	134.98	196.72	4.86
MAY	133.15	198.99	4.66
JUNE	139.19	202.41	4.40

table 1.3.6c Cost per ton and recovered grade

COST PER TON AND RECOVERED GRADE

	TOTAL COST PER TON MILLED SURFACE & U/G	COST PER TON MILLED FROM U/G	GRADE RECOVERED FROM U/G
NOVEMBER	76.95	88.31	3.79
DECEMBER	82.02	96.97	3.86
JANUARY 89	76.65	94.96	3.90
FEBRUARY	76.25	88.90	3.19
MARCH	78.84	91.81	3.34
APRIL	78.92	96.49	3.68
MAY	75.54	89.31	3.45
JUNE	79.23	94.71	3.31
JULY	84.03	94.37	3.37
AUGUST	89.67	100.01	3.54
SEPTEMBER	89.61	105.04	3.92
OCTOBER	90.81	104.47	3.86
NOVEMBER	86.40	101.87	3.73
DECEMBER	87.04	101.83	3.67
JANUARY 90	76.61	101.01	3.70
FEBRUARY	85.49	103.21	3.53
MARCH	92.46	112.49	4.30
APRIL	78.53	99.34	3.57
MAY	84.65	105.31	3.87
JUNE	87.80	104.42	3.69
JULY	88.16	100.52	3.88
AUGUST	89.51	104.24	3.89
SEPTEMBER	94.44	111.97	4.00
OCTOBER	87.69	104.89	3.97
NOVEMBER	90.42	107.49	3.95
DECEMBER	91.93	112.42	4.02
JANUARY 91	82.11	121.17	3.69
FEBRUARY	88.51	114.50	3.85
MARCH	90.29	116.94	4.09
APRIL	90.38	112.19	4.06
MAY	88.84	115.42	4.31
JUNE	95.34	117.05	4.01
JULY	97.31	121.73	4.67
AUGUST	97.49	121.82	4.68
SEPTEMBER	96.45	124.30	4.69
OCTOBER	96.55	124.27	4.87
NOVEMBER	97.68	126.32	4.78
DECEMBER	92.98	119.60	4.51
JANUARY 92	91.96	116.69	4.62
FEBRUARY	99.86	127.20	4.72
MARCH	95.15	124.34	4.72
APRIL	98.85	127.75	4.72

table 1.3.6b Cost per ton and recovered grade

COST PER TON AND RECOVERED GRADE

	TOTAL COST PER TON MILLED SURFACE & U/G	COST PER TON MILLED FROM U/G	GRADE RECOVERED FROM U/G
APRIL 85	41.00		5.17
MAY	45.13		5.08
JUNE	45.77		5.10
JULY	48.23		4.46
AUGUST	43.58		4.64
SEPTEMBER	44.10		4.64
OCTOBER	47.67		4.58
NOVEMBER	46.77		4.26
DECEMBER	44.74		4.13
JANUARY 86	46.42		4.02
FEBRUARY	49.92		3.95
MARCH	47.32		4.01
APRIL	48.27		4.53
MAY	51.29		4.24
JUNE	59.52		4.65
JULY	54.35		4.14
AUGUST	55.21		3.99
SEPTEMBER	53.48		4.17
OCTOBER	52.72		4.34
NOVEMBER	57.25		4.67
DECEMBER	58.46		4.57
JANUARY 87	58.32		4.51
FEBRUARY	59.70		4.58
MARCH	55.75		4.02
APRIL	53.88		4.11
MAY	56.82		3.86
JUNE	55.08		4.08
JULY	60.33		4.02
AUGUST	65.55		4.28
SEPTEMBER	72.19		3.93
OCTOBER	59.79		4.64
NOVEMBER	59.09		4.10
DECEMBER	59.91		4.34
JANUARY 88	65.08		4.41
FEBRUARY	64.44		4.42
MARCH	66.34		4.70
APRIL	65.16		4.45
MAY	64.73		3.94
JUNE	65.70		3.82
JULY	73.99	86.08	3.65
AUGUST	74.98	84.61	3.52
SEPTEMBER	78.95	84.72	3.46
OCTOBER	80.69	91.36	3.66

table 1.3.6a Cost per ton and recovered grade

TONNAGE SPLIT FROM UNDERGROUND SOURCES

	TOTAL MILLED	CONVENTIONAL	TM3	% TM3
JULY 92	505 000	104 395	400 605	79.3
AUGUST	476 000	138 564	337 436	70.9
SEPTEMBER	478 999	143 801	335 198	70.0
OCTOBER	483 999	135 598	348 401	71.5
NOVEMBER	503 891	127 825	376 066	74.6
DECEMBER	500 000	128 157	371 843	74.4
JANUARY 93	459 999	103 436	356 563	77.5
FEBRUARY	447 000	113 208	333 792	74.7
MARCH	458 000	125 560	332 440	72.6
APRIL	416 000	108 742	307 258	73.9
MAY	435 001	112 785	322 216	74.1
JUNE	460 000	137 793	322 207	70.0
JULY	473 000	145 384	327 616	69.3
AUGUST	479 001	95 783	383 818	80.1
SEPTEMBER	480 999	103 009	377 990	78.6
OCTOBER	462 001	100 592	361 409	78.2
NOVEMBER	447 000	97 329	349 671	78.2
DECEMBER	472 000	96 781	375 219	79.5
JANUARY 94	435 000	99 277	335 723	77.2
FEBRUARY	462 000	101 337	360 663	78.1
MARCH	443 000	93 521	349 479	78.9
APRIL	441 000	100 927	340 073	77.1
MAY	363 999	74 374	289 625	79.6
JUNE	371 000	86 704	284 296	76.6
JULY	422 000	112 540	309 460	73.3
AUGUST	429 002	114 014	314 988	73.4
SEPTEMBER	412 000	154 829	257 171	62.4
OCTOBER	396 000	114 704	281 296	71.0
NOVEMBER	398 998	103 107	295 891	74.2
DECEMBER	391 000	94 465	296 535	75.8
JANUARY 95	340 000	174 947	165 053	48.5
FEBRUARY	405 001	201 975	203 026	50.1
MARCH	393 999	192 434	201 565	51.2
APRIL	363 000	175 041	187 959	51.8
MAY	364 000	201 339	162 361	44.6
JUNE	356 001	195 077	160 924	45.2
JULY	379 000	208 986	170 014	44.9

table 1.3.5c Milled tonnages and percentage TM3

TONNAGE SPLIT FROM UNDERGROUND SOURCES

	TOTAL MILLED	CONVENTIONAL	TM3	% TM3
JANUARY 89	551 605	160 921	390 684	70.8
FEBRUARY	609 130	150 262	458 868	75.3
MARCH	594 518	142 122	452 396	76.1
APRIL	560 588	144 728	415 860	74.2
MAY	613 412	150 457	462 955	75.5
JUNE	610 902	155 154	455 748	74.6
JULY	634 767	168 876	465 891	73.4
AUGUST	615 731	154 489	461 242	74.9
SEPTEMBER	590 281	162 905	427 376	72.4
OCTOBER	595 902	164 647	431 255	72.4
NOVEMBER	630 654	154 277	476 377	75.5
DECEMBER	629 849	162 177	467 672	74.3
JANUARY 90	587 966	152 139	435 857	74.1
FEBRUARY	591 540	152 487	439 053	74.2
MARCH	563 798	148 399	415 399	73.7
APRIL	611 418	166 751	444 667	72.7
MAY	573 462	157 810	415 652	72.5
JUNE	580 031	153 494	426 537	73.5
JULY	666 745	148 968	517 777	77.7
AUGUST	639 737	136 432	503 305	78.7
SEPTEMBER	608 480	127 428	481 052	79.1
OCTOBER	621 268	133 892	487 376	78.4
NOVEMBER	611 164	126 883	484 281	79.2
DECEMBER	567 421	131 458	435 963	76.8
JANUARY 91	524 975	108 206	416 769	79.4
FEBRUARY	547 100	116 251	430 849	78.8
MARCH	543 996	110 762	433 234	79.6
APRIL	552 397	114 378	438 019	79.3
MAY	553 004	123 262	429 742	77.7
JUNE	532 002	137 780	394 222	74.1
JULY	1 787	130 394	411 393	75.9
AUGUST	552 793	129 123	423 670	76.6
SEPTEMBER	542 062	115 044	427 018	78.8
OCTOBER	548 626	131 554	417 072	76.0
NOVEMBER	520 143	124 259	395 884	76.1
DECEMBER	545 308	105 336	439 972	80.7
JANUARY 92	564 224	116 952	447 272	79.3
FEBRUARY	554 139	125 555	428 584	77.3
MARCH	542 022	120 640	421 382	77.7
APRIL	528 000	100 962	427 038	80.9
MAY	525 998	114 313	411 685	78.3
JUNE	510 000	98 179	411 821	80.7

table 1.3.5b Milled tonnages and percentage TM3

TONNAGE SPLIT FROM UNDERGROUND SOURCES

	TOTAL MILLED	CONVENTIONAL	TM3	% TM3
APRIL 85	487 000			0
MAY	475 000			0
JUNE	494 000			0
JULY	488 000			0
AUGUST	545 000			0
SEPTEMBER	545 000			0
OCTOBER	522 000			0
NOVEMBER	548 000			0
DECEMBER	572 000			0
JANUARY 86	524 000		?	?
FEBRUARY	524 000		?	?
MARCH	566 000		?	?
APRIL	540 000		?	?
MAY	542 000		?	?
JUNE	510 000		?	?
JULY	528 000	393 067	134 933	25.6
AUGUST	500 000	347 872	152 128	30.4
SEPTEMBER	526 000	359 983	166 017	31.6
OCTOBER	499 000	311 306	187 694	37.6
NOVEMBER	534 000	337 000	197 000	36.9
DECEMBER	505 000	331 000	174 000	34.5
JANUARY 87	467 000	323 000	144 000	30.8
FEBRUARY	500 000	315 000	185 000	37.0
MARCH	511 000	306 000	205 000	40.1
APRIL	440 000	222 000	218 000	49.5
MAY	416 000	228 000	188 000	45.2
JUNE	453 000	246 000	207 000	45.7
JULY	469 000	226 000	243 000	51.8
AUGUST	355 000	170 000	185 000	52.1
SEPTEMBER	280 000	163 935	116 065	41.5
OCTOBER	387 465	165 144	222 321	57.4
NOVEMBER	479 340	223 069	256 271	53.5
DECEMBER	472 324	226 371	245 953	52.1
JANUARY 88	450 569	190 248	260 321	57.8
FEBRUARY	474 856	181 447	293 409	61.8
MARCH	528 839	314 615	314 224	59.4
APRIL	521 478	175 221	346 257	66.4
MAY	587 473	187 351	400 122	68.1
JUNE	535 916	152 767	383 149	71.5
JULY	572 657	175 189	397 468	69.4
AUGUST	612 943	165 606	447 337	73.0
SEPTEMBER	626 070	165 094	460 976	73.6
OCTOBER	599 147	163 164	435 983	72.8
NOVEMBER	618 032	176 988	441 044	71.4
DECEMBER	561 733	168 275	393 458	70.0

table 1.3.5a Milled tonnages and percentago TM3

WASTE DEVELOPMENT COST SAVINGS

PROJECT	CONVENTIONAL			TM3			SAVING MR
	WASTE DEV M	R/M	COST MR	WASTE DEV M	R/M	CCST MR	
COOKE 1							
106 K9A	10 800	750	8,10	2060	1850	3,81	4,29
106 W/B	26 278	750	19,71	2578	1850	4,77	14,94
90 E8	12 762	750	9,71	1370	1850	2,53	7,04
COOKE 2							
90 WIDE	10 039	750	7,53	1000	1850	1,85	5,67
COOKE 3							
106 SOUTH	7 057	750	5,29	1919	1850	3,55	1,74
106 NORTH	5 100	750	3,83	NIL	1850	NIL	3,83
118 NORTH	2 469	750	1,85	597	1850	1,07	0,78
DOORNKOP	18 303	750	12,81	12140	500	6,07	6,74
TOTAL	92 808	740	68,69	21646	1093	23,7	45,03

table 1.3.4 Waste development cost savings

AVERAGE TOTAL LABOUR FOR REGM FROM 1985 - 1995

1985	17073
1986	16750
1987	14850
1988	14569
1989	14704
1990	14242
1991	14346
1992	15409
1993	15742
1994	15951
1995	14159

table 1.3.3 Average total labour

Employee Requirements for a 35 000 TPM Project

<u>Pat E,D,C</u>	<u>TM3</u>	<u>Conventional</u>
<u>Mining</u>		
Production Manager	0.5	0.5
Project Mining Engineer	0.5	1
Mine Overseer	1	1
Shift Bosses	3	7
Stoppers	2	9
Developers	0	4
Cleaners	1	4
Drill rig supervisors	2	0
General Miners/Construction	1	3
	11	29.5
<u>Engineering</u>		
Project Engineer	0.5	0.5
Mechanical Equipment Overseer	0.5	0
Foreman electrician	1	1
Foreman fitter	2	1
Electrician	3	3
Fitter/Diesel-hydraulic fitter	11	5
Boilermakers	2	3
Rigger	0.5	1
Storeman	3	0
	23.5	14.5
<u>Service department</u>	4	5
<u>Pat.B</u>		
Production personnel	101	417
Service and Support	140	140
Total	241	557
Grand total	255	606

table 1.3.2 Employee requirements

RANDFONTEIN ESTATES GOLD MINE

LIST OF FATALS AND REPORTABLES FROM 1975 TO 1994

YEAR	FATALS	RATE	REPORTABLES	RATE	
1975	2	0.44	199	40.00	
1976	10	2.11	209	47.00	
1977	19	2.13	347	41.00	
1978	14	1.12	516	35.00	
1979	11	0.77	502	35.00	
1980	12	0.78	425	26.00	
1981	20	1.33	372	24.00	
1982	18	1.28	308	24.00	
1983	19	1.47	292	23.00	
1984	14	0.99	253	18.50	
1985	9	0.56	191	12.07	
1986	21	1.31	179	9.80	INTRODUCTION OF TM3
1987	22	1.43	89	5.79	
1988	8	0.52	70	4.82	
1989	4	0.26	71	4.84	
1990	8	0.52	81	5.20	
1991	3	0.19	98	6.67	CONVENTIONAL MINING STARTED INCREASING
1992	6	0.37	110	6.74	
1993	7	0.41	133	7.83	HAZWISE TRAINING STARTED
1994	7	0.40	84	4.81	

Table 2.1.1 list of fatalities and reportables

Then a steady rise until 1994 , after the introduction of the "Hazwise" programme, when the rate dropped to a new all time low of 4.81 per 1 000 employees (table 2.1.1).

A cost cannot be put on safety, however the cost to implement this programme was negligible compared to the benefits obtained. No additional people were required for the implementation or the training of the programme as it was totally driven by REGM employees at the separate shafts and plants.

The Hazwise programme has definitely had a remarkable effect on the safety performance of REGM. The employees have become more safety conscience than before and the training in hazard identification has been a success.

As is with any new concept the hazwise programme must be followed up with a different approach in the near future. This is to ensure the continued improvement in safety and to keep employees minds focussed on the hazards of their jobs.

The future follow-up programme is likely to be implemented in 1996 at REGM.

Hazard centres were established at each shaft, workshop and plant, where areas physically display mock-ups depicting deviations and correct methods next to each other. At Cooke 1 shaft it is located in such a position that 90 % of workers have to walk past it at least twice a day (appendix 35, 36, 37, 38).

Every quarter, feedback is given at the recreation club. The Section Manager, Engineer or Plant Superintendent gives a presentation on the progress and achievements of objectives.

The recognition of achievement is an important part of the campaign and the team who are judged the winners are given a luncheon at the recreation club. All employees down to team leader and engineering aide level are invited. The other winning members of the team will receive a small award which is issued at the workstation. There is a floating trophy awarded to the winning team on a quarterly basis.

The training of each employee has taken in the region of ten months, mainly due to a limited space constraint in the training centre. However having a smaller class (15 to 20) definitely had the advantage of a better impact. All employees were sensitized in hazard recognition and elimination. They were trained in the principles of the JHA concept and shown how to effectively implement the JHA programme in their workplace.

The results for 1994 for Cooke 1 and Cooke 2 are outstanding as the graphs show (appendix 45, 46)(table 2.1.3). For two shafts with a labour force of 5000 employees producing 230 000 tons per month, the average reportable rate for the year was 2,96 per 1 000. If you compare it to the previous year there was an improvement of 310 %. If one compares this to the rest of the Gold Mining industry, a rate of 19.60 per 1 000, the conclusion can be easily be made that this hazwise campaign was a resounding success.

2.1.3 Conclusion to section 2.1

Although it is a difficult statement to make, there was an expected deterioration in safety when more conventional mining took place and the REGM safety graphs confirm this (appendix 22 to 25). However referring to these graphs it can be seen the remarkable drop in the number of reportable injuries when trackless mining started in 1986.

ii). Safety statistics of REGM

- total mine, showing reportable and lost time injuries
- shafts/plants, same as above
- pie-chart, showing what type of injuries have taken place as a percentage.

iii). The lectures

There are four basic steps of a Job Hazard Analysis or (JHA).

- Select a job to be analysed (a job that has repeatedly produced injuries is a focal point for an immediate JHA)
- Separate the job into basic steps (briefly describe each job step or activity)

- Identify the hazards associated with each step ie. can the employee be?

Struck by
Struck against
Contacted by
Contact with
Caught on
Caught in
Caught between
Fall on same level
Fall to below
Over-exertion
Exposure

- Control each hazard (develop a recommended safe job procedure to eliminate or reduce potential)

iv). Workshop on "Hazwise"

- divide employees into groups
- select a job to be analysed
- review/study the job guideline
- observe sequence of job steps in hazard centre
- analyse job worksheet
- recommend safe job procedures

v). Summary (appendix 42, 43, 44).

The JHA not only prevents injuries by anticipation but also provides new employees with guidelines and safe working procedures. The implementation of the JHA process will mean continuous safety improvements at REGM with the ultimate goal of zero injuries.

vi). A closing video address by the General Manager on commitment and support from Senior management for the job hazard analysis programme, was then shown to the delegates.

designed a new campaign giving a new outlook towards safety, this was "Hazard identification". It was based on being pro-active instead of reactive.

This was to be called the "Hazwise Campaign". The main communication symbol would be an eagle killing a snake which would be displayed in three different languages namely English, Afrikaans and Fanagalo (appendix 31).

2.1.1 The sequence of events

The first step at introducing this new campaign was through a meeting organized by the General Manager and the Central Indaba Committee or CIC as it is known. This was done to a large number of guests which included most Head Office personnel and a number of General Managers from the surrounding mines. Each member of senior management had a portion of the campaign to explain. The General Manager called on all Managers, Supervisors and Workers to join him in the seeking out and eliminating of hazards in the work-place that maim and kill our fellow workers (appendix 30).

The second step was to display pre-launch posters, this was done by showing the campaign message in three stages, so as to arouse the curiosity of the workers (appendix 32, 33, 34).

The third step was to display banners at strategic points ie. entrance to the shafts, plants and workshops. In the hostels, at the crushes and at the main offices. Then underground at critical areas ie. notice boards, waiting places, workshops and main trackless roadways.

Hazard centres were established at the shafts and metallurgical plants (appendix 35, 36, 37, 38). Hardhat stickers were issued and a campaign stamp was used on all promotional material, memorandums and letters (appendix 39, 40, 41).

The secondary launch of the Hazwise programme was then decentralized to the shafts and plants. At this secondary launch the target population was from Production manager down to Gang Supervisor. The programme followed is explained in the next section.

2.1.2 Hazwise programme

1). Introduction to the "Hazwise video"

2 Formulation of the problems encountered and solutions to these problems.

The gradual transition, from trackless mining, to a more conventional type of mining has not been without its problems. Safety started to deteriorate, through-put in the plants was falling, costs were rising at an alarming rate, and labour unrest was becoming common all over REGM.

In this chapter eight major topics will be discussed. They are.

- 2.1 Expected deterioration of safety statistics.
- 2.2 The cost of changing back to conventional mining.
- 2.3 Implications of a reduced tonnage.
- 2.4 Dealing with the redeployment of a highly skilled workforce.
- 2.5 Retraining a new workforce.
- 2.6 Scrapping and removal of trackless equipment.
- 2.7 Excessive regional support due to trackless mining.
- 2.8 Re-engineering REGM's workforce.

2.1 Expected deterioration of safety statistics

With the down scaling of trackless mining there was a definite deterioration in safety at REGM (appendix 22 to 25)(table 2.1.1). If one compares REGM's safety trend to the rest of the Chamber of Mines Gold Members (appendix 26 to 29)(table 2.1.2) then it can be seen something had to be done. The safety department had used tried and trusted campaigns, but injuries continued rising. It was mainly attributed to a different type of and poorly trained workforce.

If you refer to table 1.3.4 or appendix 10 it can be seen how the trackless mining built up, in 1991, to 80 % of total underground tonnage milled, and has now fallen to the present, ie June 1995, tonnage of 45 % of total underground. In terms of people there are now more people working in conventional mining sections.

To deal with the above circumstances the safety department

difficulties were encountered when the fall off in trackless mining occurred and what changes had to take place.

It was expected that the safety rate would deteriorate due to more conventional mining.

The life of mine ore reserves would decrease at an alarming rate due to a higher paylimit.

The question was asked, what can we do with the over strength TM3 operators?

The training of a new type of worker was suddenly a major issue. Would the training continue on surface, as was the case with TM3, or would the training be conducted underground in a conventional environment?

The main solution to REGM's problem soon became apparent as the tonnages started to drop. The Mine had to conduct business differently and this lead to the introduction of Business Process Re-engineering.

i). The parallel relationship of the STOMEK drilling feed beams and the fixed feed lengths prevented fanning of the drill holes at the tight end of the face.

ii). The drilling of blast holes for the ventilation holing in the gully pillars was not possible.

iii). The drilling of ancillary (non face) holes is not possible due to the fixed attitude of the STOMEK turret and feed beams.

iv). Drilling of the free face at the gully is extremely difficult and requires careful control of the gully shape to permit access of the STOMEK.

v). The traction potential of the machine on run of mine footwall indicates a maximum of 8 degrees.

vi). The traction requires footwall cleaned to shovel clean standard prior to insertion on the stope face thus extending panel preparation times.

vii). The lack of flexibility of the turret and feed beams makes fault negotiation extremely difficult and the machine cannot cope with a throw in excess of 1.0 metre without incurring abnormally high levels of dilution.

viii). Setup / disassembly / interface transfer times represented significant delays in the production process.

ix). The STOMEK is not capable of drilling a ledge, siding or gully.

1.4.4 Chapter summary

It is very easy to see why the decision to go the trackless route was taken. The benefits in ore reserves alone is justification enough. It extended the life of Randfontein Estates Gold Mine by a number of years.

Due to the high tonnage mined the paylimit was reduced significantly thus allowing the mining of previously uneconomical ore reserve.

However, what was not taken into account was the 35 000 ton per month projects, which due to very good efficiencies soon rose to a production level of some 70 000 to 80 000 tons per month and shortened the life of the sections sooner than expected.

In the following chapter it will be shown what

Referring to appendix 15 it can be seen what the difference is in dilution and what the affect it makes on the grade. At a 1.0 metre stoping width the theoretical cost saving would be as follows.

waste tons ASD / metre	19.25
waste tons ASG / metre	3.30
difference in tons / metre	15.95

Cost per ton milled at Cooke 1 for 1994/1995 financial year was R165.

At 15m. face advance per month the saving would be:

$15.95 * 15 = 239$ tons of waste not blasted and milled

$239 * 165 = R\ 39\ 435$ per month per ASG

Blasting of the ASG would not require any other labour, however the increase in cost would be an additional winch driver and an extra winch for the ASG.

A winch costs in the region of R 20 000 therefore its payback would be approximately 12 shifts.

2). Requires a minimum and maximum height in the stope face.

The minimum and maximum working height of the STOMEK was 1.20 and 1.70 metres. It was found that if we tried to go lower damage to the hoses and fittings occurred. This would result in down-time and pushed up the cost per ton. With the channel width going below 1.0 metre, the dilution which had to be carried in order to physically accommodate the STOMEK, was no longer acceptable.

3). Maintenance costs

Because of the nature of the machine, maintenance had to be of a high standard. The costs of spares was becoming exorbitant and with a decreasing stope width, the cost per ton rose considerably.

4). Restrictions in the stope face

There were a number of restrictions in the panels themselves. They were.

The feed beams and turntable controls are situated behind the two feed beams. The crawler undercarriage controls are situated on the undercarriage between the tracks.

c) The transport carrier and power pack

A turntable which can rotate through 90 degrees with extendible ramps (appendix 16, 18) to allow the drill rig to be driven onto and off the stope face.

A hydraulically powered hose reel at the rear of the unit with two hydraulic hoses connecting the power pack to the drill rig.

Four hydraulic jacks, one on each corner of the transport carrier to level and lift the carrier to the required height so that the drill rig can be driven onto the stope ledge.

A 55kW electric motor driving a hydraulic pump which circulates oil at approximately 150 litres/ minute at a working pressure of 200 bar.

1.4.3 Reasons for the removal of the STOMEC H.25.2

In February 1993 it was decided to remove the STOMEC H.25.2 from underground operations. The cost per hour was unacceptably high at R600 (appendix 19). The availability of the machine was over 80 percent, however the utilization was approximately 20 percent (appendix 20). The tonnage produced by the four units went from a high of 150 000 tons per annum in 1989 to 60 000 tons per annum in 1992 (appendix 21).

When it was in operation the performance was very encouraging, however the machine had certain drawbacks.

They were.

1). Limited access into the stope face.

The machine was transported on a trailer pulled by a tractor. This tractor and trailer had to reverse into an ASD which was 3.5m wide by 3.0m high, up to the stope face. The only access was between the face and the blasting barricade. The main concern was the amount of dilution which had to be blasted to accommodate the transportation of the STOMEC to the face.

approximately 70m. apart.

ii). Every 30m. a loading cubby was blasted and a conventional raise developed through to the roadway above.

iii). Mining of the first raise commenced with two down-dip panels of 15m. each.

iv). Mining of the third raise commenced as per iii above.

v). A bracket pillar was left on the bottom roadway.

vi). When both were completed and vamped, a backfill barricade was constructed in both loading cubbies.

vii). Backfill was then pumped into No.1 and No.3 paddocks.

ix). The panels were left to cure for one month.

x). The down-dipping of No.2 raise would then commence.

1.4.2 The STOMEC H.25.2

The need to mechanise conventional drilling was the primary aim of the introduction of the STOMEC H.25.2 at Cooke 2 in December 1987. It was designed to operate in stope widths of between 1.2 to 1.7 metres.

a) Description of the unit

The unit consists of three separate units, namely the crawler mounted drill rig, the transport carrier and powerpack with a four wheel drive tractor to tow the transport carrier (appendix 18).

b) Drill Rig

The feed beams are mounted on a turntable, which in turn is mounted on a crawler undercarriage, driven by hydraulic motors and controlled by hydraulic brakes. The two feed beams are fixed at a burden of 52cm. The burden can be altered to suit customer requirements. The feed beams may be independently operated, forwards and backwards (appendix 16, 17).

The machines hydraulics are driven by an electro-hydraulic power pack placed at a safe distance from the face. Hoses from the power pack are coupled to a manifold to enable easy manoeuvring of the machine and hoses.

were then mined in a down dip direction (appendix 14).

The change in the type of mining created a number of problems. The safety reportable rate started to deteriorate as opposed to the virtual straight line performance of the rest of the Gold Mine Industry.

The number of employees was slowly creeping up from a low of 14 242 in 1991 back to almost 16 000 in 1994. This was obviously impacting on profitability.

It was soon realized that conventional development would have to be restarted and this would then also seriously affect the paylimit.

If reference is made to the cost per ton graph (appendix 12) you will see a steady rise from 1987 to 1992, then a tremendous increase from 1993 to present ie. June 1995. It was in light of this that a new way of doing business had to be found. This led to the start of a Business Process Reengineering (BPR).

1.4.1 Sequence of mining

a) Trackless method

Referring to appendix 58 the sequence of mining a section totally trackless was as follows.

- i). Firstly the grid was developed 150m on strike and 100m on dip.
- ii). Stoping then commenced with 7 metre wide ASD's and 7 metre wide pillars.
- iii). On completion of a block the area was vamped and backfill barricades were built.
- iv). Pumping of backfill took place and the backfill left for a period of time depending on hardness to be obtained.
- v). Pillars were then mined out and final vamping took place.

b) Narrow trackless method at Cooke 3

Referring to appendix 14 the sequence of mining a section using narrow trackless means was as follows.

- i). The TM3 grid was developed on strike with roadways

e) Employees on leave

i. All employees that were on leave would be handled on their return to the Mine.

f) Consultation

i. Arrangements to interview with other mines was made where vacancies existed.

ii. Affected individuals were advised by their head of department, personnel practitioners and the relevant Union representative where it was applicable.

iii. The affected employees were then transferred to Phumlani (an under utilised hostel away from the production shafts).

iv. The finalisation of the retrenchment lists indicating all voluntary, early retirement, and affected employees was completed with total Union involvement.

v. All non-Union employees who were affected were told who their respective Jabulani member was.

vi. Management then sent out a brief to all other employees informing them that those not interviewed were not affected.

g) Consultation with the affected Unions

The consultation with the affected Unions was to continue for some four weeks until a final agreement was finalised.

h) Implementation of rationalisation

i. Selection of interview panels per workstation was done.

ii. The panel held a briefing session with each affected employee and given 48 hours to think about the alternatives to retrenchment.

i) Re-training

i. Arrangements were made for training of Employees who accepted lower level occupations.

v. Affected employees were identified according to agreed upon total working places.

vi. Summary per job level of excess labour was sent to the Managing Director.

b) Notification to Press/Unions/Employees

i. Approval of a draft press statement from Chairman.

ii. Notification was given of scheduled meetings to all Unions.

iii. Approval of General Manager's briefing session document to all Union representatives.

iv. Finalise Management Brief to all employees.

c) Negotiation team

i. Strategy Meeting - technical and financial overviews were finalised.

ii. A schedule with provisional dates for meetings with the Unions was drawn up.

d) Press release/meeting

i. Announcement of rationalisation to the press and shareholders was done on the 3-2-94 (appendix 51).

ii. A briefing session was held with all company car holders which were D2 level and above.

iii. A briefing session with the Workers Forum and Jabulani and Isibonda representatives was held.

iv. A briefing session was held with all other Unions and Associations.

v. To back this up a brief to all employees was distributed at the workstations.

vi. A verbal briefing to all employees via the "flat structure" was also done to avoid any misunderstanding of the brief.

vii. Group placement of excess labour was then started.

- f) Offer an alternate occupation, where there are vacancies, to the affected workers.
- g) Extend the leave periods (paid or unpaid).
- h) Ask for voluntary retrenchments.
- i) Ensure manpower planning has been done correctly.
- j) Contact other Mining Houses and the rest of the mining industry for possible shortages.

2.4.3 The sequence of events

The redeployment programme was split up into ten sections. They are:

- a) Preparation exercise
- b) Notification to Press/Unions/Employees
- c) Negotiation team
- d) Press release/meeting
- e) Employees on leave
- f) Consultation
- g) consultation with the Unions
- h) Implementation of Rationalisation
- i) Retraining
- j) General

a) Preparation exercise

- i. 2 year plan was finalised and approved by Management and Head Office.
- ii. Proposed redeployment packages were approved by Senior Management and agreed to by the Unions.
- iii. Excess number of occupations per working place were identified in relation to 2 year plan.
- iv. Principles for the redeployment were determined as per working place.

- f) Offer an alternate occupation, where there are vacancies, to the affected workers.
- g) Extend the leave periods (paid or unpaid).
- h) Ask for voluntary retrenchments.
- i) Ensure manpower planning has been done correctly.
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- b) Notification to Press/Unions/Employees
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- d) Press release/meeting
- e) Employees on leave
- f) Consultation
- g) consultation with the Unions
- h) Implementation of Rationalisation
- i) Retraining
- j) General
- a) Preparation exercise

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ii. Proposed redeployment packages were approved by Senior Management and agreed to by the Unions.

iii. Excess number of occupations per working place were identified in relation to 2 year plan.

iv. Principles for the redeployment were determined as per working place.

2.4 Dealing with the "redeployment" of highly skilled Workforce

2.4.1 Introduction

During the two years prior to retrenchment of some 700 skilled and semi-skilled workers in 1994, REGM carried employees over strength in a bid to try and avoid retrenchment. However it became increasingly evident that these employees were becoming a disruptive element to the rest of the workforce.

The total number of trackless machines had come down from a high of 376 in 1989 to the figure of in 1994.

The 700 employees were mainly trackless machine operators who, due to the down scaling of trackless mining were no longer required. Numerous attempts to use these workers in a productive manner was not successful due to most employees having a negative attitude. It was decided by Management that these workers would be given the choice of redeployment to another occupation or face retrenchment.

Unlike the previous retrenchment of some 1 100 employees in 1990, where there was no recognition agreement with the National Union of Mineworkers. Management now faced a recognised Union at the time of the first general election in April 1994.

2.4.2 Measures taken to avoid retrenchment

There are several options that Management can take before turning to the last resort of retrenchment. They are:

- a) Reduction in people by natural attrition, that is, not to fill any posts that go vacant due to resignations or disciplinary reasons
- b) Ask for voluntary early retirement from 55 years old and above.
- c) All employment to stop except where absolutely essential.
- d) Reduce overtime.
- e) Contact other group mines to place as many people as possible.

2.3 Implications of a reduced tonnage

If one refers to appendix 47 you can see from the paylimit calculation what the production rate of 1990 in 1995 monetary terms would do to the paylimit. It drops the actual of 4.40 g/t to a theoretical grade of 3.86 g/t. This equates to some 15 000 000 tons of ore reserve (appendix 64) or roughly 2.5 years extra life of mine.

Looking at appendix 48 it can be seen in the column on the right-hand side what the effect on the paylimit an extra 30 % production would make. The tonnage increases from an average of 389 000 tons per month to 481 000 tons per month. This is in line with the BPR concepts which is discussed in section 2.8.

It drops the paylimit grade from the current 4.42 g/t to 4.14 g/t, this gives additional ore reserves of some 10 000 000 tons or nearly two years extra life of mine.

There can be no doubt that REGM must reduce overheads or overhead cost per ton to remain competitive and to increase the life of the mine. The infrastructure is geared for 600 000 tons per month from underground operations and are currently now only doing in the region of 400 000 tons per month.

PAY LIMIT CALCULATION

Working Cost R/ton / Revenue R/gram = Pay limit gram/ton

For 481 000 tons per month

$$X / 45 = 4.14 \quad X = 186.30 \text{ R/ton}$$

For 370 000 tons per month

$$Y / 45 = 4.2 \quad Y = 198.90 \text{ R/ton}$$

assumption : Gold = R 45 000 / Kilogram

21.8 % of total cost consists of overheads, both on mine and off mine. For the financial year 1994/1995 this amounted to R179 532 000 or R38.46 per ton. The estimated cost of overheads at the increased rate of 481 000 tons per month will be R31.85. This shows a decrease of R6.61 per ton which is clearly not sufficient for REGM's target of an overall drop in cost of 30 % .

LABOUR COMPLEMENTS

Rockdrill, Button bit and drill steel shop

<u>Occupations</u>	<u>Old complement</u>	<u>New complement</u>
Section Manager	1	0
Senior Salvage Officer	1	1
Shift Boss	1	0
Drillsteel Supervisor	3	1
Foreman	1	1
Fitters	2	2
Extra Heavy Duty Drivers	1	0
Senior Engineering Gang Supervisor	1	0
Senior Supervisor Records	1	1
Heavy Duty Driver	1	1
Maintenance Gang Supervisor	1	1
Senior services recorder	6	9
Supervisor Records	2	0
Electrical Aide	0	1
Boilermaker Aide	1	1
Fitter Aide	7	7
Services Recorder	1	1
Lambis Bolt Thread operator	33	23
T.C.Reconditioner	4	0
Rockdrill fitter helper	5	5
	<u>73</u>	<u>50</u>

table 2.2.1 labour complements

This decrease in complement equates to a saving of R70 000 per month in 1995 monetary terms.

the additional 565 rockdrills and 730 airlegs (table 2.2.1).

Winches that were abandoned in old areas when trackless mining started were reclaimed and recommissioned, however there was a shortfall and some 100 winches were ordered over a period of three years. This cost approximately R 1 800 000.

2.2.1 Conclusion to section 2.2

Essentially the geology and TM3 costs forced REGM back to more conventional mining methods. Fortunately it was done with relatively little capital cost, which was largely due to the fact that not many of the assets that became redundant were in fact sold off when trackless mining started.

A lesson that has been learnt with the change to trackless mining and back to a more conventional mode is the folly of not developing ahead. The mistake made, and for many reasons, in stopping footwall development is now being felt on REGM's costs. However with hind sight there are certain areas which, had prior knowledge of the geological structure been known, would probably have not converted to trackless mining in the first place.

2.2 The cost of changing back to conventional

The direct costs are easily quantified and are given in detail below. However the cost of poor productivity during the change back to conventional, if worked out, would be staggering.

When trackless mining started in the mid 1980's the estimated savings due to equipment becoming redundant at REGM was as follows (H.Scott-Russell,1987).

Equipment type	No	cost each	total saving
Compressors	2	R1 500 000	R3 000 000
Rockdrills	640	1 200	768 000
Air-legs	200	350	70 000
Scraper winches	188	9 500	1 786 000
Air loaders	32	16 000	512 000
Locos	16	20 000	320 000
Hoppers	96	1 500	144 000
Cost in 1977 terms			R6 582 000

Referring to the above list of equipment, the savings on air loaders was essentially a refurbishment cost. The savings on locomotives and hoppers came about by the concentration of trackless mining thus less rolling stock was required.

One of the main constraints that was found when changing back to conventional mining, was the inadequate size of the compressed air columns. When trackless mining started air columns were reduced in size because of the use of hydraulic drill-rigs. Main compressors on surface had not been decommissioned so the main cost would be the additional power every month. Note the power verses tons graph (appendix 71).

Pipe columns were reinstalled at great cost and more importantly the time taken to reinstall them caused a delay in production.

Another cost incurred was the purchase of rockdrills and airlegs. This was done over three years at a cost of approximately R 2 300 000.

The rockdrill shop was not increased in size or complement as a result of increasing conventional mining. This was largely due to a decision to use knock-off bits as opposed to integral steel. A study was also done on the rockdrill shop and the result was that it was over staffed even with

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Air loaders	32	16 000	512 000
Locos	16	20 000	320 000
Hoppers	96	1 500	144 000
Cost in 1987 terms			R6 582 000

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One of the main constraints that was found when changing back to conventional mining, was the inadequate size of the compressed air columns. When trackless mining started air columns were reduced in size because of the use of hydraulic drill-rigs. Main compressors on surface had not been decommissioned so the main cost would be the additional power very month. Note the power verses tons graph (appendix 7.).

Pipe columns were reinstalled at great cost and more importantly the time taken to reinstall them caused a delay in production.

Another cost incurred was the purchase of rockdrills and airlegs. This was done over three years at a cost of approximately R 2 300 000.

The rockdrill shop was not increased in size or complement as a result of increasing conventional mining. This was largely due to a decision to use knock-off bits as opposed to integral steel. A study was also done on the rockdrill shop and the result was that it was over staffed even with

REPORTABLES FOR COOKE 1 AND COOKE 2 FROM 1985-1994

	<u>COOKE 1</u>	<u>COOKE 2</u>
1985	46	48
1986	33	31 INTRODUCTION OF TM3
1987	19	22
1988	11	24
1989	9	17
1990	17	15
1991	24	29
1992	14	28
1993	19	29 HAZWISE TRAINING STARTED
1994	8	8

table 2.1.3 Reportables for Cooke 1 and Cooke 2

GOLD MEMBERS, CHAMBER OF MINES

LIST OF FATALS AND REPORTABLES FROM 1969 TO 1994

YEAR	FATALS	RATE	REPORTABLES	RATE
1969	624		25358	
1970	512		24580	
1971	542		24010	
1972	500		23393	
1973	529		23155	
1974	485		22222	
1975	495		18970	
1976	547	1.38	20494	51.89
1977	587	1.41	19843	47.56
1978	645	1.48	20664	47.47
1979	563	1.25	19218	42.66
1980	619	1.32	17693	37.66
1981	603	1.26	15530	32.42
1982	596	1.26	15242	32.00
1983	630	1.30	14548	29.93
1984	592	1.23	13528	28.49
1985	528	1.13	12866	27.31
1986	681	1.46	11351	23.67
1987	529	1.15	9633	20.93
1988	495	1.09	9430	18.86
1989	516	1.24	8922	20.08
1990	522	1.25	8055	19.21
1991	443	1.21	7399	19.80
1992	400	1.13	7475	21.04
1993	398	1.23	7230	21.34
1994	357	1.08	6793	19.96

table 2.1.2 list of fatalities and reportables

Abstract

A typical sized block of ground pillars left due to rock mechanics constraints

111 * 157 =	17 427m2
	= 1 170m2

extraction % = 93.3%

2.7.2 Dilution and grade

All external waste, both above and below the profitable or economic channel limits of the reef, inclusive of condoned or non-economic hanging or footwall required for clean mining or equipment access.

If one takes a grade of 7.2 g/t over a channel width of 100cms

Refer to appendix 60

```
(111m * 157m) - 4 000m2 = 13 427m2
13 427m2 * 1m * 2.75      = 36 924 reef tons
13 427m2 * 2m * 2.75      = 73 849 waste tons
                           = 110 773 total tons
```

Therefore % dilution is $73\ 849 / 110\ 773 = 66.7\%$

$$265\,853 / 110\,773 = 2.4 \text{ g/t}$$

2.7 Excessive regional support due to trackless mining

In a normal trackless section with a stoping width of 3,0m and a channel width of 2,8m it would definitely be payable to endorse trackless mining. However in our situation in some sections of REGM, the channel width had reduced to less than a 1,0m. In this situation totally trackless mining could not be justified and the conversion to conventional or narrow trackless had to be considered.

Trackless roadways are large excavations and therefore require protection, which normally has to be left for the life of the project. Many times this protection is by way of regional pillars, the disadvantage of these being the lock up of gold. the saving in regional pillars more than pays for the conversion back to a more conventional type of mining due to recovering the high grade reef.

This next section 2.7.1 will look at the topic from the point of view of extraction rates, dilution and grade.

2.7.1 Extraction ratio

If one considers the extraction ratio under normal conditions, you can see for the same size block of ground conventional has a higher rate (appendix 62) of 93.3 % compared to narrow trackless (appendix 61) of 90.3 % and trackless (appendix 60) of 77.1 %.

Trackless

Refer to appendix 60

A typical sized block of ground	$111 * 157 = 17\,427\text{m}^2$
Pillars left due to rock mechanics constraints	$10 * 10 * 40 = 4\,000\text{m}^2$
extraction %	<u>$= 77.1\%$</u>

Narrow Trackless

Refer to appendix 61

A typical sized block of ground	$111 * 157 = 17\,427\text{m}^2$
pillars left due to rock mechanics constraints	$= 1\,675\text{m}^2$
extraction %	<u>$= 90.3\%$</u>

<u>MACHINE NUMBER</u>	<u>TYPE OF MACHINE</u>	X MACHINE CONTAMINATED
		> HAS BEEN APPROVED
		? NOT TESTED TO DATE

SUBROCK DUMPTRUCK

TS 1007
TS 1003
TS 1024
TS 1014
TS 1010
TS 1023
TS 1020
TS 1002
TS 1009
TS 1032
TS 1021
TS 1025
TS 1027
TS 1016
TS 1023
TS 1012
TS 1001

>
>
X
X
X
X
X
X
X
X
X
X
X
X
?
?
?
?

UTILITY VEHICLES

UN 1816
UN 1867
UN 1787
UN 1316
UN 1559
UN 1412
UN 1319
UN 1872
UN 6085
UN 1101
UN 1416
UN 1315
UN 1863
UN 1261
UN 1313

>
>
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>
>
>
>
>
>
>
>
X
?
?
?

LHD SCOOPTRAMS TORO 250

LW 2303
LW 2252
LW 2289
LW 2290
LW 1506
LW 1433
LW 2250

ENGINE TAKEN

X
X
X
X
X
X
?

table 2.6.2b machines taken out from underground

MACHINES AT DOORNKOP PROJECT STORE YARD AS AT 06-09-95

<u>MACHINE NUMBER</u>	<u>TYPE OF MACHINE</u>	X MACHINE CONTAMINATED > HAS BEEN APPROVED ? NOT TESTED TO DATE
<u>ATLAS COPCO BOOMER DRILL RIGS</u>		
DA 1022		>
DA 1232		>
DA 1025		>
DA 1622	SOLD TO BLACK MOUNTAIN MINE	>
DA 1234		>
DA 1665		X
DA 1675	BOOM & FEED RAIL OF BOOMER SOLD TO BLACK MOUNTAIN MINE	?
DA 1032		X
DA 1230		X
DA 1003		X
DA 1824		X
DA 1610		X
DA 1025		?
<u>COMPAIR TAMROCK DRILL RIGS</u>		
DT 2000		X
DT 3150		>
DT 2885	PIECES	X
DT 3134		>
DT 1130		?
<u>SECOMA ROOF BOLTERS</u>		
RE 1519		>
RE 1446		X
RE 1463		X
RE 1462		X
RE 1461		X
RE 1486		?
<u>CASE SCALERS</u>		
UG 1884		X
UG 1008		?
<u>CATERPILLAR DOZER</u>		
VC 1183	SOLD TO BUFFALO EXCAVATORS	X

table 2.6.2a Machines taken out from underground

Cost per shift is therefore $2\ 500 / 24 = R104.17$

11 Artisan shifts	$R328.75 * 11 = R3\ 616.25$
27 Aide/Helper shifts	$R104.17 * 27 = R2\ 812.59$
Total	<u>R6 428.84</u>

If one considers that just one new tyre on these machines cost R14 000 then it appears that this exercise would be very profitable.

Another consideration is the shaft availability for slinging. With the down scaling of trackless mining there has been a reduction in tonnage hoisted therefore there would be no loss of revenue due to the shaft standing for one shift while a TM3 unit was slung to surface.

2.6.1 Conclusion to section 2.6

REGM had a definite policy on removing all redundant equipment from underground and reassembling this equipment on surface.

The total number of units that came out permanently was as follows.

a) Drill rigs	=	1
b) Load haul dump	=	22
c) Articulated dump trucks	=	25
d) Utility vehicles	=	35

It should be noted that a large portion of the equipment taken out was replaced by new equipment. Referring to table 2.6.2a and b it can be seen that in fact very little machines were ever sold when brought out the mine. There were many other machines that came out to surface for refurbishment and sent back underground when completed. The redundant machines are now assembled and parked at Doornkop shaft store.

2.6 Scrapping and removal of trackless equipment

Many schools of thought on the viability of scrapping and removing redundant trackless equipment from underground, have been discussed on the mine.

One school of thought advocated leaving the redundant TM3 equipment underground because it was said to be too expensive and a waste of resources to bring it out of the mine. On the other hand if one breaks down the cost it can be seen how expensive it really is to leave the machines underground.

The following table 2.6.1 is a breakdown of the time taken, and the cost thereof, to strip and transport out of the mine a Toro 350 LHD or a 25 ton articulated dumptruck.

SHIFTS	TASK	ARTISANS	AIDES	HELPERS
2	STRIP MACHINE AND LOAD ON CARS	2 FITTERS=4	1	2
1	TRANSPORT OUT OF SECTION	1 RIGGER =1	3	2
1	SLING MACHINE OUT OF MINE	2 RIGGERS=2	6	4
2	ASSEMBLE ON SURFACE	2 FITTERS=4	1	2
6		7 =11	11	10

table 2.6.1 labour and shift requirements

Total cost of an artisan for 1 month is R7890

(average cost first 8 months of 1994/95 financial year Cooke 2 #).

Cost per shift is therefore $7890/24 = R328.75$

Total cost of an aide or a helper for 1 month is R2500

(average cost first 8 months of 1994/95 financial year Cooke 2 #).

(average cost for a worker is R2 500 per month at Cooke 1 for 1994/95 financial year).

$$R2\ 500 / 24 = R\ 104.16 \text{ per day}$$

$$1\ 600 * R104.16 = R\ 166\ 656 \text{ per month per total man shifts}$$

fixed cost	<u>R 74 000</u>
	R 240 656

therefore total cost per man to be trained is:

$$R240\ 656 / 240 = R1\ 002.73$$

2.5.1 Conclusion to section 2.5

Underground training is an absolute must for conventional mining. However it was proven that training for TM3 operators was best done on surface.

The training of an artisan in TM3 equipment was becoming very costly due to the time required. It took 54 days to fully train an artisan on a machine. This in 1995 monetary terms cost approximately R17 755.

To train a TM3 operator would cost in 1995 monetary terms approximately R2 500 each. This was equivalent to one month at the training school.

- 1) Opening up of x/c's
- 2) Wire-mesh and lacing of x/c's and raises
- 3) Building of lecture rooms both surface and underground

The number of training personnel is as follows:

Senior Training Officer	1
Team Leader trainers	14
Average number of trainees per month	240

	R and
Cost of equipping 1	20 000
2	65 000
3	25 000
	110 000

The average shifts spent in the underground training centre (appendix 57, 58, 59) per non-supervision trainee is six days. The supervisors are on average required for ten days. Therefore the cost of training one man is as follows.

Cost of training

Senior Training Officer	R 12 000
Team Leader Instructors: 14 @ R3 000	R 42 000
Stores	R 20 000
	R 74 000

Average number of trainees per month is 240 of which approximately 40 are supervisors and the balance in a non-supervisory capacity.

Therefore $200 * 6 = 1\ 200$ shifts per month.
 $40 * 10 = \underline{400}$ shifts per month
 Total $= 1\ 600$ man shifts per month

2.5 Retraining a new workforce

From 1986 to 1993 REGM's training department placed most of its efforts into the TM3 area. The trackless machines were expensive to buy and to operate therefore requiring a higher level of training. Conventional mining training continued at a much lower level.

All the training was done on surface firstly to minimise the danger of inexperienced trainee operators damaging the machines or worse injuring someone, and secondly it was easier for instructors to teach from a mock-up surface installation (appendix 53, 54).

With the downscaling of TM3 operations there came a need for conventional training to be stepped up. With the redeployment in 1994, redundant TM3 operators were asked if they wanted to accept a lower occupation or alternately take a package.

There were some who opted to stay, however a large percentage were not interested and took the retrenchment package. New employees had to be signed on to fill the vacant conventional posts. It became obvious to Mining personnel on the shafts that the level of conventional training was inadequate. New employees needed more on the job conventional training.

A request was made to Senior Management to move all conventional training to underground areas. This was agreed to by all parties concerned.

Initially each shaft would be responsible for its own training, however due to the number of instructors required, it was decided to centralize Cooke 1 and Cooke 2's training at Cooke 2 shaft. Cooke 3 would do its own training and that of all Affirmative Action training of Learner Miners.

At Cooke 2 an old area was identified close to the station. The grade in this area was unpay and was unlikely to change in the foreseeable future.

2.5.1 The cost of setting up a new training centre

There was a choice of two reef horizons which were the E8 and the UE1a. The main advantage being the wide and narrow reef next to each other (appendix 55, 56). Wire mesh and lacing of the cross-cuts were the major cost of equipping the training centre. The estimated cost is as follows.

financial affairs and running of the Company. Unfortunately the perception is if the Company is making a profit why must retrenchment take place in the first place.

Management of Companies would take away this type of thinking if they embarked on an educational programme for training the Union representatives about the basics of finance. This applies to both "White" and "Black" Unions.

If the sequence of events are followed there will seldom be a comeback at a later stage. Retrenchment at any stage is undesirable, however in REGM's case there was absolutely no choice. The main factor which was attributed to the successful retrenchment of these 700 employees was the excellent communication and negotiation between Management and Union.

2. Selection criteria, LIFO per Mine.
3. Severance benefits and other assistance to retrenched Employees.
 - a). Notice period.
 - b). Severance payment.
 - c). Leave.
 - d). Removal expenses.
 - e). Provident fund benefits.
 - f). Unemployment insurance fund cards.
 - g). Documentation.
 - h). Assistance to affected Employees residing in Company accommodation and participants in the housing bond subsidy scheme.
 - i). Vocational training.
4. Note regarding the industry death benefit scheme.
5. Retrenchment of incapacitated employees.
6. Notification to affected employees who will be on leave at the time of retrenchment.
7. Notification to and counseling of effected employees.
8. Timetable for retrenchment.
9. Recall of retrenched employees in the event of manpower requirements changing in the future.
10. Full and final settlement.

2.4.6 Conclusion to section 2.4

The fact that the National Union of Mineworkers were a recognised Union on REGM at the time of this retrenchment in 1994, greatly assisted Management in completing a well negotiated package for the Employees who would not accept redeployment to another occupation.

One of the main problems encountered when negotiating with the Unions is the "apparent" lack of knowledge of

ii. These employees were immediately sent back to the shaft for training.

j) General

i. Finalisation of the Retrenchee's documentation was completed.

ii. All returning Employees from leave not affected by retrenchment were notified.

2.4.4 The disruption to the Mine

There can be no Management that enjoys retrenching employees, and REGM's Management was no exception. There were disruptions to operations prior to and during the retrenchment period which are discussed below.

After the General Manager's brief to all employees (appendix 52) there were suddenly dissatisfied people all over the Mine. The main reason was obviously because employees were unsure of their future and this was mainly attributed to the disinformation given at certain workstations by various Unions.

The Unions played an active part in the disruption of production at REGM. The Trackless Operators embarked on a go slow strike at some of the affected working places which led to an increase in tension with their Supervisors.

Once the people to be redeployed were identified and transferred away from the shafts, the situation returned to normal. However this caused a loss in production for approximately ten days. Fortunately when the situation returned to normal the loss in production was made up.

The cross transferring of people on the Mine was also a very disruptive factor. This was due to the agreement with the Unions to "LIFO" Mine wide and not only at the individual shafts.

2.4.5 The Letter of Agreement

Due to the sensitive nature of the agreement I will only give the headings of the points discussed. They were:

1. Steps taken to minimise the extent of retrenchment.

- Proven ore reserves.
- Life of mine.
- Mining flexibility.

c) Core process and competencies.

i). Development and stoping.

The core process was analysed by the BPR team and they came up with some startling figures.

- Time utilisation on day-shift stoping was as follows (appendix 67).

43 % operate
19 % travel
38 % other

- Time utilisation on night-shift stoping was as follows (appendix 68).

27 % operate
17 % travel
56 % other

- Time utilisation on day-shift developing was as follows (appendix 69).

29 % operate
19 % travel
52 % other

- Time utilisation on night-shift developing was as follows (appendix 70).

49 % operate
20 % travel
31 % other

What these four graphs (appendix 67 to 70) show is just how ineffective and unproductive our core business has become. It is in this department that a major contribution to profitability can be made. Certain areas of concern were identified, the main one being logistics.

ii). Manage logistics

- Administrative downtime : procedures surrounding authorisation, paperflows and other specified requirements delay the whole supply process and create scope for errors and multiple checking.

- The impact of "buffer" stores (ie stock already expensed but not yet use on stores cost. The lack of

FINANCIAL REVIEW

CATEGORY	% OF TOTAL OVERHEADS	% OF TOTAL WORKING COST
ON MINE	14	1
OFF MINE	86	8
TOTAL	100	9
-Gold realisation	6	0.6
-Insurance	11	1.0
-Head Office	28	2.6
-Group training	15	1.4
-Chamber of Mines	17	1.6
-Computer costs	12	1.1
-GD stores	11	1.0
	100	9.3

table 2.8.1 composition of mine overheads

- Prescribed supervisory hierarchy.
- Restriction of duties (certificate of competency).

There are also various mine obstacles in our way.

- Complicated bonus systems.
- Affirmative action targets.
- Management / Union relations.
- Communication to the labour force.

2.8.6 Internal drivers for change

There are five internal drivers for change to discuss.

- a) Financial review.
- b) Business strategy.
- c) Core process and competencies.
- d) Seven S analysis
- e) Organisation climate.

a) Financial review

If an analysis of the costs are done from 1989 to 1994, it can be seen that labour accounts for 45 % of working costs and stores 40 %, power is approximately 6 % (appendix 66). Overheads comprise approximately 9 % of costs (table 2.8.1).

b) Business Strategy

i). JCI's strategy may be described as:

- Promise what can be delivered.
- Deliver what was promised.
- Maintain 18-24 months proven ore reserve.
- Maintain economic feasibility.

ii). Mines strategy may be described as:

- Develop annual, two and five year plans.
- Achieve annual, two and five year plans.

iii). Issues which were not explicitly addressed:

- Proven ore reserves are dropping.
- Change in working costs.
- Change in throughput.

iv). Impact on:

COMPETITOR ANALYSIS (GOLD MINES)

L = LOW H = HIGH R = REGM

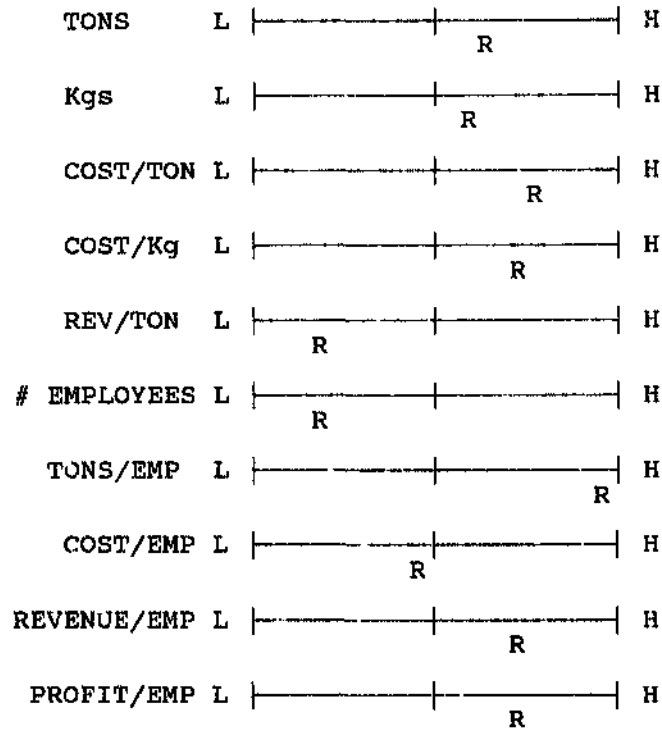


figure 2.8.2 COMPETITOR ANALYSIS

- Appropriate downscaling.
- Minimum standards (safety, wages, conditions of employment).
- Accommodation.

iv). Government GME

- No nationalisation.
- No change in reserve ownership for 2-3 years.
- Would like mining houses to issue competency certificates.
- Wanted standard 8 requirements for a blasting certificate to fall away, which it did.
- Could see the need to re-assess supervision specified by law.

v). Unions / Associations

- Progress on change must happen at a decentralised level.
- All see scope for change.
- Removal of closed shop agreement a concern.
- Requires disclosure to support changes.
- Safety standards must not change.

c) Competitor analysis

If we compare REGM to the rest of the Industry, we can see how well it has done in the past. However due to lower volume and grade we are fast losing our competitiveness (figure 2.8.2)

d) High level benchmarks

South Africa has the highest cost per oz in the World. REGM is worst than average (appendix 63), with the reduced tonnage this could seriously threaten REGM's continued existence if nothing is done to change the present situation. One of the main concerns is the decreasing productivity of REGM and all the mines in the Industry.

e) Industrial relations / Legal implications

There are various legislative obstacles for the effective use of labour.

- Sundays and public holidays.
- Work hours per fortnight.
- Frequency of blasting.

- d) High level benchmarks
- e) Industrial relations / Legal implications
- a) Market analysis

There are various factors influencing the Worlds markets.

- i). Loss of gold's perceived value as a currency reserve.
- ii). Reduction in central banks.

- iii). Discovery of new reserves.

- iv). Increasing fabrication demand.

- v). Impact of liberalisation and economic growth in China.

South Africa still produces 27 % of the Worlds gold. 64 % of the total supply comes from the mines (appendix 65). From this 71 % goes into jewelry.

The question most mining people are asking "What is the future gold price?". There is decreasing influence by political and emotional criteria, and an increasing influence by supply and demand.

b) External stakeholders

The external stakeholders are exerting increasing pressure on Mining Houses.

i). Supplies.

- Cost of service is high.
- Buying would like to work closer with the end user.
- Will be amenable to involvement in process.

ii). Stockbrokers select gold shares on basis of:

- Management structures.
- Control over costs.
- Capital expenditure profile.
- Life of mine.

iii). Government RDP

- Return of mineral rights to democratic Government.
- Downstream beneficiation.
- Employers to negotiate reorganisation of work.
- Anti trust legislation and monitoring.

APPROACH TO THE WORKSHOP:

STRUCTURE

EXTERNAL DRIVERS FOR CHANGE

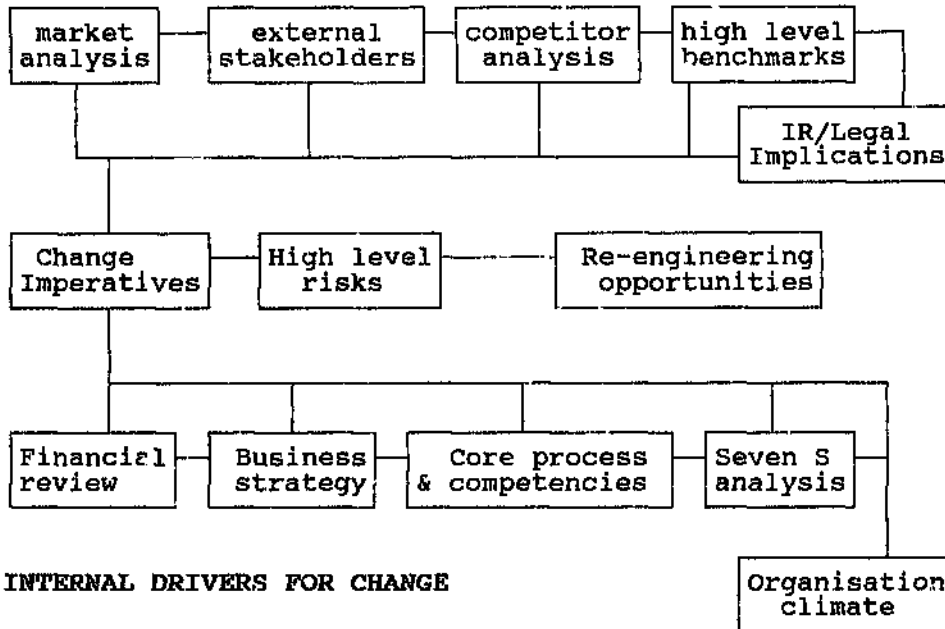


figure 2.8 1 Approach to workshop

Results: Automation expenditures have proven to be the least effective investment for improving productivity (Braxton Associates, strategy consulting division of Deloitte and Touche).

iv). Action: Numerous major corporations have made acquisitions to build synergies and scale economies.

Result: Only one of three acquisitions has generated a return greater than the Buyer's cost of capital (Internal publication of Deloitte and Touche).

v). Action: Numerous incremental schemes to improve productivity and customer service have been implemented.

Results: The majority of these fail before the benefits exceed the costs.

Despite the above track record it was still felt that BPR had merit and that it could work provided the correct groundwork was carried out prior to implementing any changes. It was then decided to conduct a pilot trial.

2.8.4 Selection of test site

In September 1994 key personnel from JCI gold mines were selected and formed the core team of BPR. Their function was to first learn the concepts of BPR and how to expose the rest of the now JCI Ltd. mines to these concepts.

Cooke 1 shaft has been selected as a starting point for JCI Ltd. After the successful implementation, the other mines in the group will begin. Although BPR has just begun, the different steps in the process will be discussed. These steps were first introduced to the senior management team ie. JCI Gold Division staff, then on mine to Section manager and above, to initially get their commitment as to why the company needs to change, and to get their input and ideas. Referring to appendix 64 it can be seen where all REGM's ore reserve is placed.

2.8.5 The Process :External drivers for change

Referring to figure 2.8.1 , there are five external drivers for change.

- a) Market analysis
- b) External stakeholders
- c) Competitor analysis

- a) Leader - a senior executive who authorises and motivates the overall reengineering effort.
- b) Process owner - a manager with responsibility for a specific process and the reengineering effort focused on it.
- c) Reengineering team - a group of individuals dedicated to the reengineering of a particular process, who diagnose the existing process and oversee its redesign and implementation.
- d) Steering Committee - a policy making body of senior managers who develop the organisation's overall reengineering strategy and monitor its progress.
- e) Reengineering czar - an individual responsible for developing reengineering techniques and tools within the company and for achieving synergy across the company's separate reengineering projects.

2.8.3 Past results of reengineering

Efforts in the past to adapt and respond have traditionally met with poor success.

Between 50% and 70% of reengineering efforts fail to achieve the goals set for them (Fortune magazine, 1993).

Results are lagging behind expectations for most reengineering projects (Deloitte and Touche, 1993). This is shown in the following five examples in terms of an Action and Result statement.

i). Action: 85 % of "Fortune" 1000 companies downsized white collar work forces in the past five years.

Result: Overhead rates remain significantly above the best global competitors (Chain Store Age Executive Edition, November 1991).

ii). Action: Scores of Organisations have undertaken major restructuring efforts.

Result: 52 % of surveyed executives say these efforts have not met original targets (K.S.Cameron, S.J.Freeman, A.K.Mishra 1991).

iii). Action: Hundreds of millions have been spent to automate inefficient business systems processes.

2.8 Re-engineering REGM's workforce

It is predicted that in two years time many South African gold mines will be in a loss making situation if they continue to operate using archaic methods, structures and systems.

With the reduced tonnage and grade at REGM and the projected cost per ton for conventional mining, REGM has been forced to embark on a re-engineering process. It is called Business Process Re-engineering or BPR for short.

2.8.1 Why re-engineer?

The definition of "Reengineering" is, according to M.Hammer and J.Champy,(1993) "the fundamental rethinking and radical redesign of business process to achieve dramatic improvements in critical contemporary measures of performance, such as cost, quality, service and speed".

The concept of business reengineering evolved in the early 1980's when it was noticed that a few companies had dramatically improved their performance in one or more areas of their business by radically changing the ways in which they worked. These companies had not changed the business they were in, rather they had significantly altered the processes they followed in those businesses or even replaced these old processes entirely (M.Hammer and J.Champy, 1993).

South African mines are high cost producers by world standards and JCI mines are in turn high cost producers by South African standards (appendix 63). We need to reverse real cost increase trends in order to enable competition with the rest of the world. The implications of not doing so might well result in the possible closure of our mines in 3-4 years due to increasing costs and diminishing payable ore reserves.

2.8.2 Who will reengineer?

If a company would like reengineering to be successful it has to have commitment from the Chief Executive Officer right down to the Workers on the shop floor.

According to M.Hammer and J.Champy, there were several emerging roles that were seen during the implementing of reengineering in the different companies. They were.

and it would allow us to mine a block of previously unpay ore-reserve.

2.7.3 Conclusion to section 2.7

One must take into account the footwall infrastructure required for a conventional block (table 1.3.4). However in the case of certain areas on REGM a combination of trackless hauling and conventional mining is done (appendix 14, 61), thus getting the benefits of both mining methods.

In my opinion the narrow trackless option is by far the best method of mining a block of ground that has a dip of no more than 10 degrees and a channel width of up to 2.0 metres in thickness. The ore reserve is opened up so much faster than, when using conventional means. However to ensure success, the gathering haulage in the footwall below the reef horizon must be kept up to the advancing reef development face.

This would be totally unpay at Cooke 1's 1995 paylimit of 3.87 g/t.

Narrow Trackless

Refer to appendix 61

$$(111m * 157m) - 1\ 695m^2 = 15\ 732$$

$$15\ 732m^2 * 1m * 2.75 = 43\ 263 \text{ reef tons}$$

$$1\ 475m^2 * 2m * 2.75 = 8\ 113 \text{ waste tons}$$

$$792m^2 * 0.8m * 2.75 = 1\ 742 \text{ waste tons}$$

$$13\ 465m^2 * 0.2m * 2.75 = 7\ 406 \text{ waste tons}$$

$$= 60\ 524 \text{ total tons}$$

Therefore % dilution is $17\ 261 / 60\ 524 = 28.5\%$

$$\text{Channel value } 7.2 \text{ g/t} * 43\ 263 = 311\ 494 \text{ g}$$

$$311\ 494 / 60\ 524 = 5.15 \text{ g/t}$$

This would be very profitable to mine.

Conventional

Refer to appendix 62

$$(111m * 157m) - 1\ 170m^2 = 16\ 257m^2$$

$$16\ 257m^2 * 1 * 2.75 = 44\ 707 \text{ reef tons}$$

$$16\ 257m^2 * 0.2 * 2.75 = 8\ 941 \text{ waste tons}$$

$$1\ 164m^2 \text{ (ASG)} * 0.8 * 2.75 = 5\ 122 \text{ waste tons}$$

$$= 58\ 770 \text{ total tons}$$

Therefore dilution is $14\ 063 / 58\ 770 = 23.9\%$

$$\text{Channel value } 7.2 \text{ g/t} * 44\ 707 = 321\ 890 \text{ g}$$

$$321\ 890 / 58\ 770 = 5.48 \text{ g/t}$$

This would be much better than our paylimit of 3.87 g/t

Fixed interfaces

- Data content, frequency and interface mechanisms formalised.

Considered inflexible

- The system structure prescribes how it is used, cannot easily adapt to changed business procedures.

Data considered inaccessible

- Computer literate or illiterate mine management must rely on IS professionals to provide mainframe based data.

Too much paper

- Inappropriate and unusable reports produced, with too few online enquiry features being used.

Slow to change

- System changes take a long time to implement and hence reduce the usefulness of the systems as they have difficulty in keeping up with mine changes.

There is appropriate technology available to sort out the problems with interfacing, integration, access to data, paper reduction, and user tools can be solved with what is available.

The systems have not been fully exploited for the benefit of the mines. Existing IS skills, networks and software tools have not yet been fully harnessed to overcome problems.

v). Staff and Skills

The present structure tends to trap a persons skills due to the bureaucracy and the complex communication structure of the mine.

There is a perception that staff utilisation could be significantly improved in the functional, services and core parts of the business.

There is a perceived opinion that the bonus schemes are all ineffective and there are a lack of skills in business, people and technical management.

Referring to figure 2.8.8 it can be seen that there are at present nine reporting levels in the mining structure at REGM. The proposal using the BPR concepts (figure 2.8.9) will only have five. This will allow better and faster communication both downwards and upwards.

iv). Systems

There are a vast number of computer software systems in place at the moment.

In the core processes the following are used

Capex	Mainpak	JMC
Cadsmine	MSS	Metal accts
CPM	Peqs	Laboratory
Datamine	Sh le	Sampling
Dalcys	GD..	Evaluation

In addition there are a large number of smaller systems. These consist of mainframe, PC and Unix based systems which address functional areas eg. survey, geology and metallurgy.

There have been many problems encountered due to informal interfaces. This means data transferred manually or electronically, but not all interface requirements are mapped out.

The systems are not integrated, meaning applications are installed at different times using different platforms and technologies.

There is poor end-user tools, meaning manual interfaces to spreadsheet tools.

In the infrastructural processes

Accpac	GMMS
Walker GL costing	Housing
Finance	Ventilation
Gold financial model	SAS
Gold hedging	PC support
Time and attendance	Mainframe support
Networks	IS Management
Adabas/Natural	

Small number of big systems

- The most important systems consist of integrated subsystems with much functionality.

The proposed structure for the test site at Cooke 1 is as follows. Refer to figure 2.8.9

PROPOSED STRUCTURE

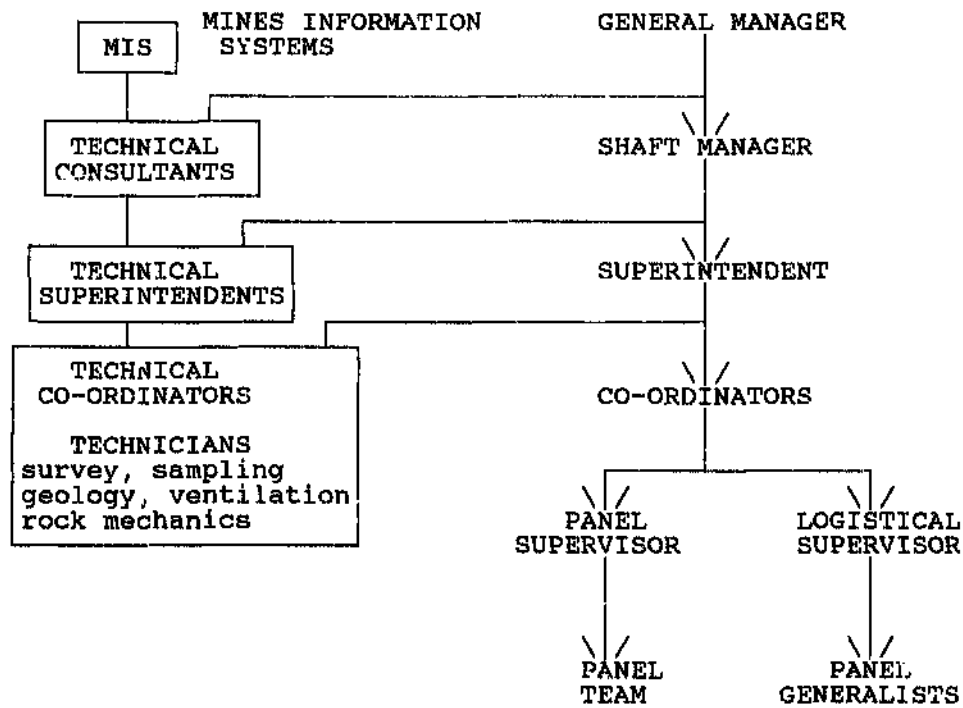


figure 2.8.9 Proposed structure

The present structure at Cooke 1 is as follows. Refer to figure 2.8.8

PRESENT STRUCTURE FOR MINING

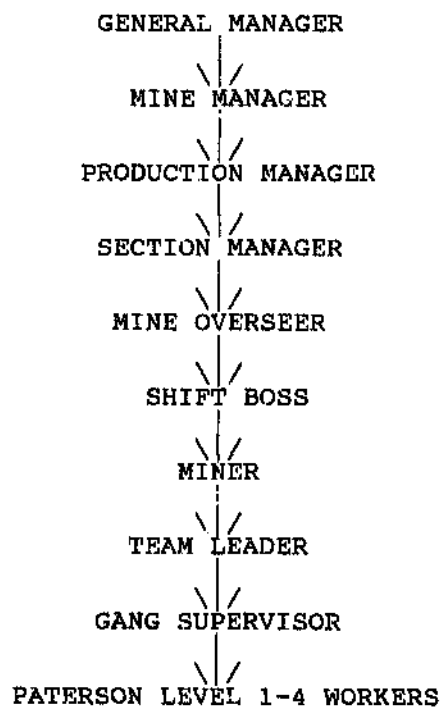


figure 2.8.8 Present structure

iii). Structure

There are at present thirteen levels in the organisation. Starting with the Board of Directors right down to the Labourer at the stope face (figure 2.8.7).

- An average of two levels per band
- Difficult to balance ARA (Authority, Responsibility, Accountability)
- Communication becomes an art form

Another problem encountered is dual reporting.

- Split reporting
- Impact on ARA
- Creates frustration, turf battles, handoffs, grade creep.

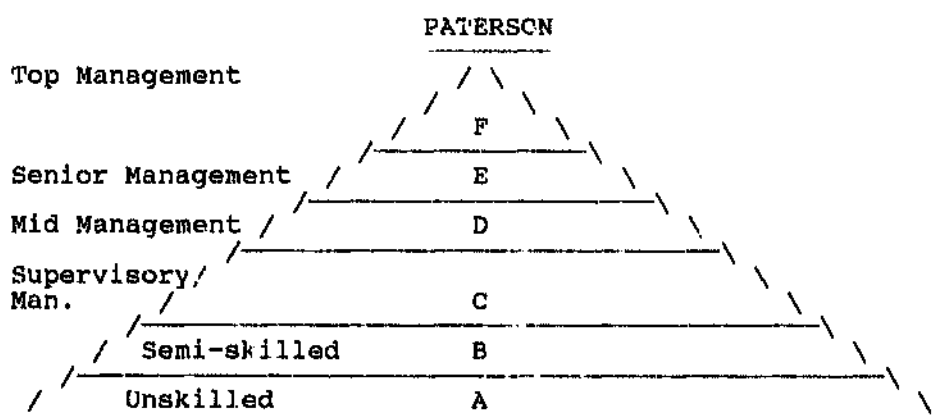
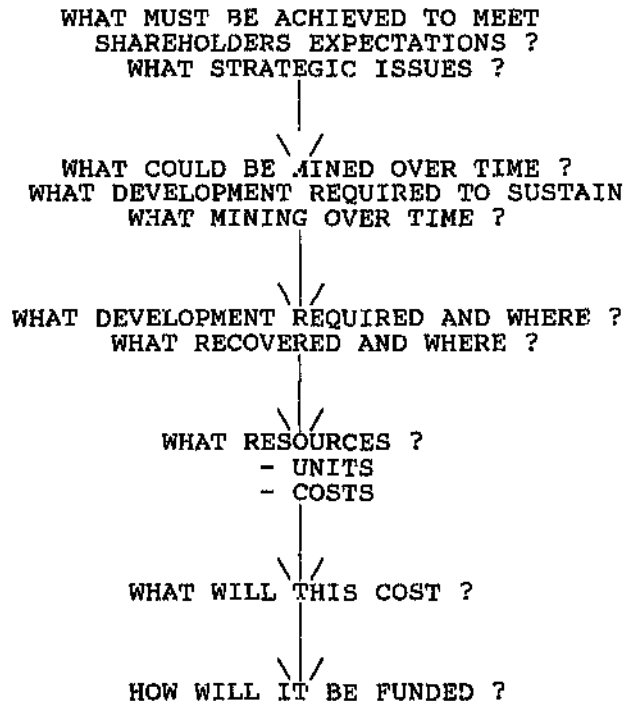


figure 2.8.7 Paterson banding

- 1 Board of Directors
- 2 Managing Director
- 3 Consulting Engineer
- 4 General Manager
- 5 Mine Manager
- 6 Production Manager
- 7 Section Manager
- 8 Mine Overseer
- 9 Shift Boss
- 10 Miner
- 11 Team Leader
- 12 Gang Supervisor
- 13 Generalist

OPPORTUNITY MODEL



CAUSE		EFFECT
- Attenuated process	_____	- Difficult to change
- Bottom up	_____	- Imbalanced ARA
- Numbers driven	_____	- No qualitative review of numbers
- History driven	_____	- No review of strategic leverage
	- Short term	
	- Reactive	
	- Sacrifice fundamentals (development)	

figure 2.8.6 Opportunity model

OPPORTUNITY MODEL

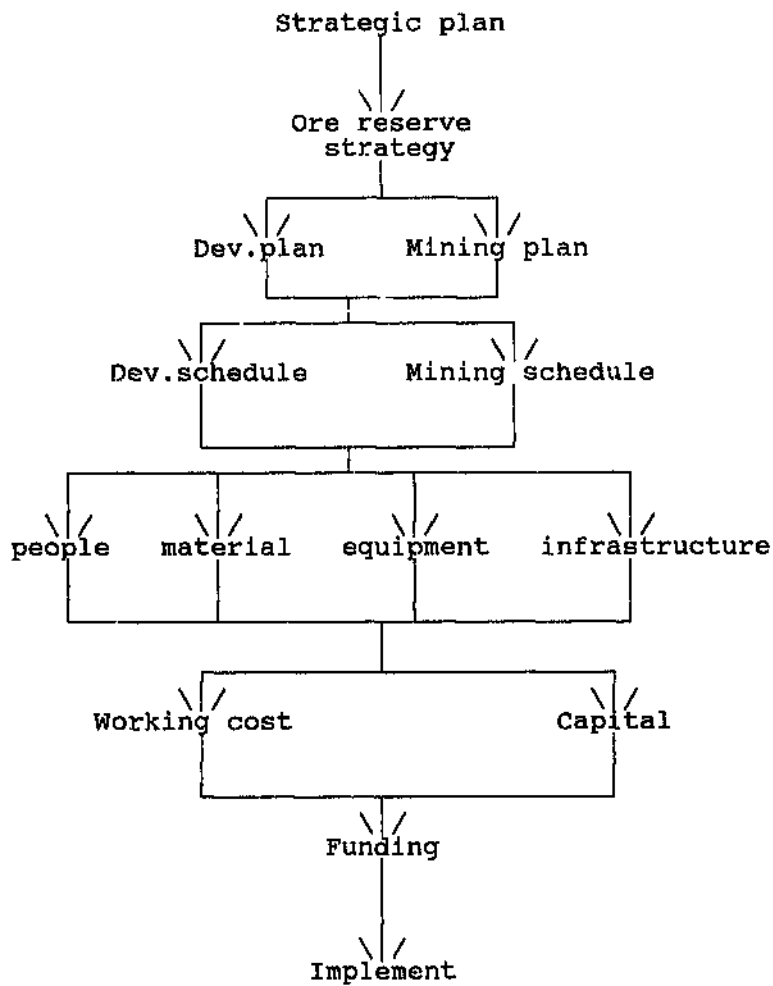


figure 2.8.5 Opportunity model

"AS IS " MODEL

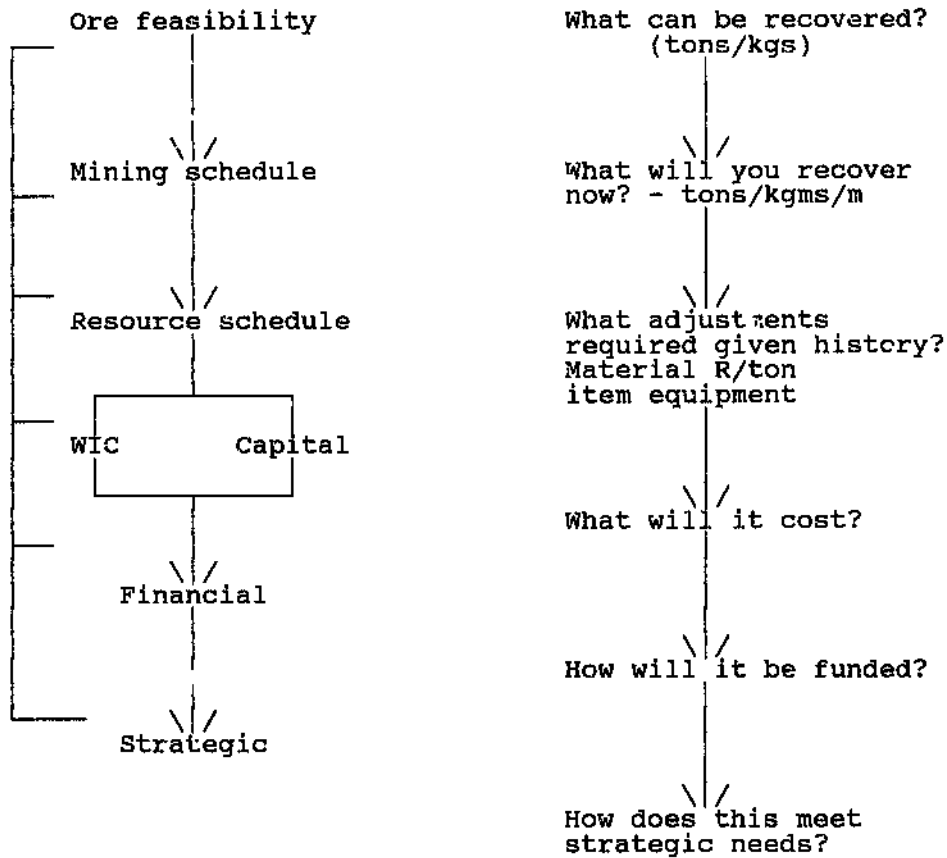


figure 2.8.4 As is model

Theoretical business model

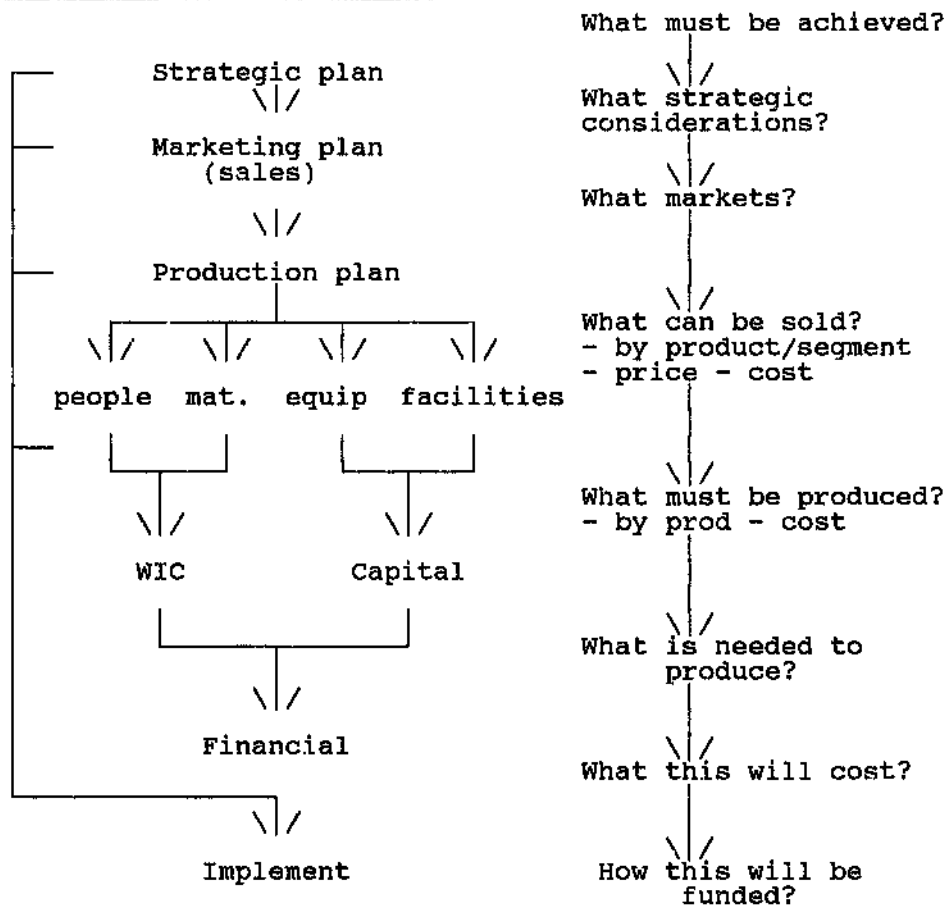


figure 2.8.3 Theoretical business model

- v). Staff & Skills
- vi). Style

i). Superordinate goal

The meaning of the superordinate goal is the combination of the above six topics into an objective. Although there is a reasonable understanding of the need to meet all stakeholders requirements there is:

- No direct translation of Head Office goals or criteria into business plans.

- Plans are process driven and adjusted to meet strategic criteria.

- While strategic goals translate into functionally specific policies these are not understood across functions.

- There is generally a tendency at the mine to focus on developing an achievable plan.

ii). Strategy

According to Professor P.H. van den Berg of the Graduate School of Business Leadership, University of South Africa, presented to the REGM Management team on the 4th October 1995, strategic planning involves a stream of decisions aimed at matching the organisations outputs (services and products) to the demands of its current and future environments. That is:

- Mission, profitability and growth objectives.

- Product or market scope

- Policies related to resources acquisition, allocation and deployment.

Referring to figure 2.8 3 it can be seen what the logical steps are in a theoretical business model.

Figure 2.8.4 shows an "as is" model from the ore feasibility stage to the strategic stage.

Figure 2.8.5 and 2.8.6 show slightly different models called an opportunity model.

- Major opportunities for improvement in training and development activities. (Multi-skilling, mentorship, on-the-job and supervisory training). Number of people needed to perform any specific task due to legal qualification requirements).

- Performance measurement - of overall human resources activities as well as linking of performance measurement to Management By Objectives system (MBO).

- Communication with the "masses" as well as representative bodies. There is great potential for improvements.

- Information systems should be more integrated, flexible and user friendly.

vi). Provide utilities

- Reclamation of used equipment in mined out areas. A structured programme is necessary.

- Inelasticity of utility cost curve makes incorporation of utility costs in volume change considerations essential (appendix 71).

vii). Provide information systems

- Plenty of technology and skills, but they are not exploited to the extent possible on the mines.

- Costing issues in use of technology (discourages use of mainframe and resources).

- disproportionate spending on infrastructural support systems, approximately 75 % of budget. (infrastructural systems are good, but utilizing obsolete technology. Core systems are not integrated, and have been installed in an ad-hoc manner).

d) Seven S analysis

The seven S's towards a superordinate goal are: (appendix 72)

- i). Superordinate goal
- ii). Strategy
- iii). Structure
- iv). Systems

materials management skills at the final place of inventory before use, could be having a major impact on costs.

- Moving of the place of supply closer to the area where required, to shorten the "logistics downtime" to the user.

- Lack of specific logistics planning and managing skills.

iii). Liberate and extract the gold

- Not a people process, people are rather just overseers.

- Double handling of data due to unintegrated information systems. Could eliminate duplicate handling and manual work.

- Current underutilization of qualifications. Could utilize skills more for environmental control.

- Little scope for reduction of costs, but large scope for better utilisation of available resources.

iv). Financial and administration services.

- Complex administrative requirements as a result of negotiated agreements ie. bonuses, allowances.

- Mismatch between reporting of physical performance measures and financial results could be clouding the decision making process.

- computer systems are not adequately integrated. This leads to inefficient duplication of effort in the production of various types and formats of reports.

- Administrative time required for the processing of documents. This is often complicated by numerous locations and departments involved.

v). Manage human resources

- Scope for improvements in recruitment, selection and assessment of skills levels. (Centralised, specialists, collate and share information).

in such a way so as to ensure the safe removal of these pillars at some later stage of the mines life.

It can be seen from section 2.7 that narrow trackless mining has the most benefits. The combination of conventional mining with trackless hauling should in most cases of narrow reef ore bodies be the most productive.

Most Employees at REGM recognize the need to change the way business is conducted in order to stay competitive. However very few are willing to change their particular area of work where they are comfortable. It is this attitude that management of companies need to change. The principles of Business Process Re-engineering are very simple and can easily be followed. The most difficult part of the exercise is to obtain, firstly, the understanding of BPR by the employees, and secondly, their commitment to the successful implementation of BPR.

There can be no doubt as to the problems REGM will face in the future to stay competitive. However REGM's main advantage is the huge ore reserve waiting to be mined from a reduced pay limit.

Referring to appendix 50 it can be seen what in fact actually happened to the pay limit read with the gold price. When tonnage started to drop in 1992 and 1993 the pay limit shot up from 3.9 g/t. to 4.7 g/t. What changed the trend was a sudden jump in the Rand per kilogram price. This dropped the paylimit back down to 4.1 g/t.

The current paylimit for 1995 is 4.4 g/t. With the proposed changes when BPR has been fully implemented we will see this pay limit going back below 4.0 g/t.

The section dealing with the sequence of events leading to the redeployment or retrenchment of a workforce can be considered as a thorough guide to any Industrial Relations practitioner.

When a business runs in to trouble and has to resort to such drastic actions as retrenchments, it is essential that a company has an effective communication structure in place that can conduct negotiations on their behalf. There have been many cases in the past where a company has not followed the basic principles (due to the sensitive nature, an example will not be given) and six months down the road in a Labour Court, are obliged to re-engage the retrenched Employees or pay out a package in excess of what they would have normally been entitled to.

There can be no doubt about the merits of conventional training underground on the face. REGM has successfully moved the training from surface to underground for relatively minor costs and disruptions to production. The training period has now been reduced because of better control.

As was stated in section 2.6, Management of companies would make negotiations with Unions less difficult if these companies embarked on a basic training programme in the financial running of the mines.

Looking at the list of machinery brought out from underground, it appears as if a large amount of capital is now tied up on surface. However most of the machinery has been worked to the end of its economical life. There are however, many small components which if required can be cannibalized. If these machines are not sold off REGM has a large supply of expensive trackless mining spares at its disposal which will assist in cost reduction.

As was stated in section 2.7 trackless roadways are generally large and require protection. However when a feasibility study of an ore reserve is conducted, protection pillars should be put in place. These must be

3 Conclusion and recommendations

Many statements have been made in the past about the implementation of trackless mining in narrow ore bodies and how cost effective it is. However in the case of Randfontein Estates Gold Mine, trackless mining has been successfully put in place and over a period of time was very cost effective. However with the lack of geological information on hand concerning the ore body, the consequences of stopping footwall haulages altogether was shown. One must accept that an ore body can change and that the type of mining, whether trackless or conventional, must adapt when this occurs, in order to stay profitable.

The experiences in trackless mining gained at REGM will be very beneficial to the mining method decided at JCI's new South Deep Project.

The Hazwise safety programme has been a huge success with many other mining houses requesting information on how to implement this programme on their mines. As is the case with any programme, after a length of time Employees attention is distracted with other issues and the need to do a follow-up programme is essential to achieve the maintenance of the current REGM safety trends.

As stated in section 2.1.3 the second section of the Hazwise programme will be implemented in 1996.

The reportable trends at most of the shafts have turned for the better as appendix 45 and 46 show. On 18th August 1995 REGM achieved another million fatality free shifts. This now gives a total of 2nd times. This is an industry record for Chamber of Mines Gold mines. On 12th September 1995, the surface section achieved the seventh consecutive million fatality free shifts.

As can be seen in section 2.6 the trackless machines have not been sold off. This also applied to the conventional equipment when REGM embarked on trackless mining. With hind sight it was the right decision. Relatively speaking very little capital had to be spent on conventional equipment.

Referring to section 2.3 on the implications of a reduced tonnage, it can be seen in appendix 49 what the effect is if the gold price had remained at R32 500 per kilogram. There would have been virtually no REGM left to mine profitably.

2.8.8 Conclusion to section 2.8

Although it is early days for the BPR process at REGM. The positive attitude that prevails on the mine will ensure success. In two years time it will be interesting to do a follow-up paper on the results obtained, and have a comparison done from the vision to what actually happened in reality.

To date, September 1995, the top management structure at the pilot site on Cooke 1 shaft has been completed and the role clarification of the various jobs discussed and finalised for implementation.

A potential stumbling block is the length of time taken to regrade the various occupations, and the intended change to the remuneration package. As a suggestion to any company who intends to embark on the BPR route, it would be wise to have these answers available when trying to sell the idea to the employees of a company. It has been the Cooke 1 management team's major problem area. The "what's in it for us" attitude as opposed to "how can we help the process change".

It was found that the various unions were very suspicious of REGM's true reason for wanting to re-engineer the mine. As was stated in section 2.4.6 the "apparent" lack of knowledge in the financial running and affairs of a mine is greatly hindering the progress of implementation of BPR.

PRESENT SYSTEM OF REPORTING

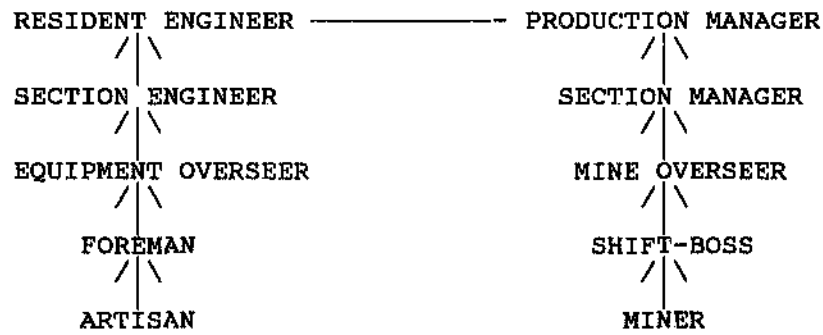


figure 2.8.10 Horse shoe effect

v). People

- Dual reporting and horse shoe communication (figure 2.8.10) can be eliminated by means of a fully integrated mining, engineering, stores construction and reclamation team.

- Multi-tasking will result in productivity increases. Training can be done in small modules varying between 3 and 15 days.

d) Infrastructure

i). To bring about an infrastructure conducive to the implementation of the logistics teams ideals and concepts.

- Reduce outlets.
- Ensure unidirectional flow of material cars.
- Mechanise material movement and off-loading of material.

e) Key benefits

i). Logistics

- Improved productivity.
- Efficient materials supply pipeline.
- Reduction of materials cost and wastage.

ii). Maintenance

- Faster reaction time to breakdowns.
- Improved reliability and availability of equipment.
- Reduced material wastage.

iii). Mining

- Focused supervision and quality mining.
- Better resource utilisation and efficiency.
- Clear authority, responsibility and accountability.

and training of the workforce.

iii). Optimise reliability and availability of equipment and machinery through pro-active maintenance programmes in ideally equipped workshops.

iv). Structured reclamation teams to assist with the expansion of salvage operation.

v). An efficient fully integrated materials supply chain, from supplier to end user, which will eliminate production downtime, reduce materials expenditure and optimise inventory holding.

This will be achieved through:

- An integrated materials planning capability.
- Expanded and integrated salvage and service exchange operations.
- Revised purchasing methods.
- Improved transport network.

c) The process

i). Move men

- Efficient schedules will ensure timeous transportation of core workforce to working area.
- Pedestrians affecting movement of ore and materials can be reduced.
- Time and attendance operation can be streamlined.

ii). Move Material

- Properly planned bills of material will ensure timeous delivery and availability of all materials to the user via a dedicated materials handling team utilising underground central loading points.

- Cars and workforce will be properly co-ordinated through implementation of selected technologies and control processes.

iii). Technology

- The implementation of selected technologies will enhance safety, productivity and generate financial savings on an ongoing basis.

e) Organisation climate

Main areas identified needing attention are:

- Job structure
- Management climate
- Change leadership
- Human resources function
- Industrial relations

2.8.7 BPR visions

The business Process Re-engineering principle and techniques as already described will be applied so as to achieve the following four visions that have been identified. They are, Mining, Logistics, The process and Infrastructure.

a) Mining

To introduce a two-shift stoping model (drill, blast and clean per shift), that will result in a dramatic increase in production and m2 per team member. This will be achieved through applying the following principles.

- i). Panels will be blasted and cleaned once every 24 hours.
- ii). Blasting will take place twice in 24 hours (if possible).
- iii). Team members must all be multi-skilled.
- iv). Miners mine only.
- v). Optimised effective shift lengths.

b) Logistics

- i). The efficient transportation of men, materials and ore.
- ii). Reduction of non-productive times through implementation of technologies, infrastructural changes

- There is the belief that there will be a reduction in staff due to multi-skilling and possible retrenchments.

- Long term view of REGM (only 10 years left) adds to scepticism about the initiative.

- Too many initiatives have already happened without success.

- Union involvement and agreement is crucial.

Industrial relations climate

- IR climate largely depends on current trends of the mine.

- In general, the Gold Division is seen as above average and the climate is fairly co-operative. This could however move to more pro-active initiatives.

- Frustrations occur due to:

- low levels of education
 - literacy and numeracy levels
 - political climate of empowerment
 - unrealistic demands

Why change ?

Currently there is a strong culture which:

- Is effective for managing crisis situations.
- Maintains overall control.
- Has effective measures when things go wrong.
- Is punitive when mistakes are made.
- Is reactive and focused on survival.

However !

- It does not focus on development and recognition of future potential which will ultimately grow the organisation.

- Non-performers are often either pressurised to resign or transferred.

- Survival is the key- mistakes are not easily tolerated however you will not be fired.

- Little credit is given for good performance.

Issues raised by Head Office

- Structure of JCI with the mining house having a dual role as major shareholder, as well as manager/consultant to the mine influences the relationship between mines and head office.

Advantage

- Provision of technical backup, expertise and general support by Head Office

Disadvantage

- All major decisions taken need to be feed through H/O
- Two power bases operating
- H/O seen as too involved in operational issues
- Perceived as "control" system rather than "consultative"

The challenge is to get the balance right with Head Office having a more strategic, policy setting, consulting role.

Leadership development

- An assessment centre has been set up for each mine.

- People in the past promoted because of technical skills rather than leadership skills. The intention is to change this.

- There is limited guidance on human resources areas eg. performance management, leading/ controlling, coaching/ counseling.

Change initiatives

- Clear objectives of reengineering project have not been communicated.

- People were confident and "believed" in the mine.
- Small turnover of managers, therefore a constant, stable environment.
- Time and money spent on the "people" side of business.

Decision making

- Limited joint decision making.
- Layers of decision making exist leading to information being filtered selectively.
- Over inspection and too much control occurring.
- Attitude of "I am responsible, I know the best way" prevails.
- Head office need to know everything. Too much detail needed at headoffice. Power should be decentralised down to the mines.
- Responsibility, accountability and authority not on applicable levels. We need to push down decision making.

Career advancement- what counts for promotion

- Personal interaction with other people.
- Credibility with other disciplines.
- "Academics"/ Theoreticians do not survive.
- Please the boss attitude, do not challenge too much.
- Success rate with regards to targets, specifically production targets.
- "Technical" rather than "leadership skills."

If you do not perform

- There is "harsh" and direct handling of non-performance.
- Person is attacked rather than the problem.

There is a perceived opinion on the mine that people are promoted for the wrong reasons. For example using promotion as a reward or a person who has had long service but not necessarily qualified in the skills required for that job.

vii).Style

This lengthy section on style will be covered in ten sections. This will result in a clear understanding of style. Every mine has a different style and to a large degree it is determined by senior management.

Performance management overall

- Performance appraisals do take place, but are perceived to be mechanistic.
- People who conduct performance appraisals are not adequately trained.
- The reward system in operation is not user friendly, it is difficult to understand, complex (multitude of bonuses) and not easily implemented.
- The remuneration system (specifically bonuses) is not flexible and is difficult to understand.
- Criteria are in place and measurement of the "harder" operational targets does take place. However "softer" criteria are harder to measure, not as clearly defined, and are therefore being overlooked.
- There is seldom credit given to good performance and people are not recognised for good achievement. There is however tough and harsh management of non-performance. It is also perceived as "inexcusable to make mistakes".

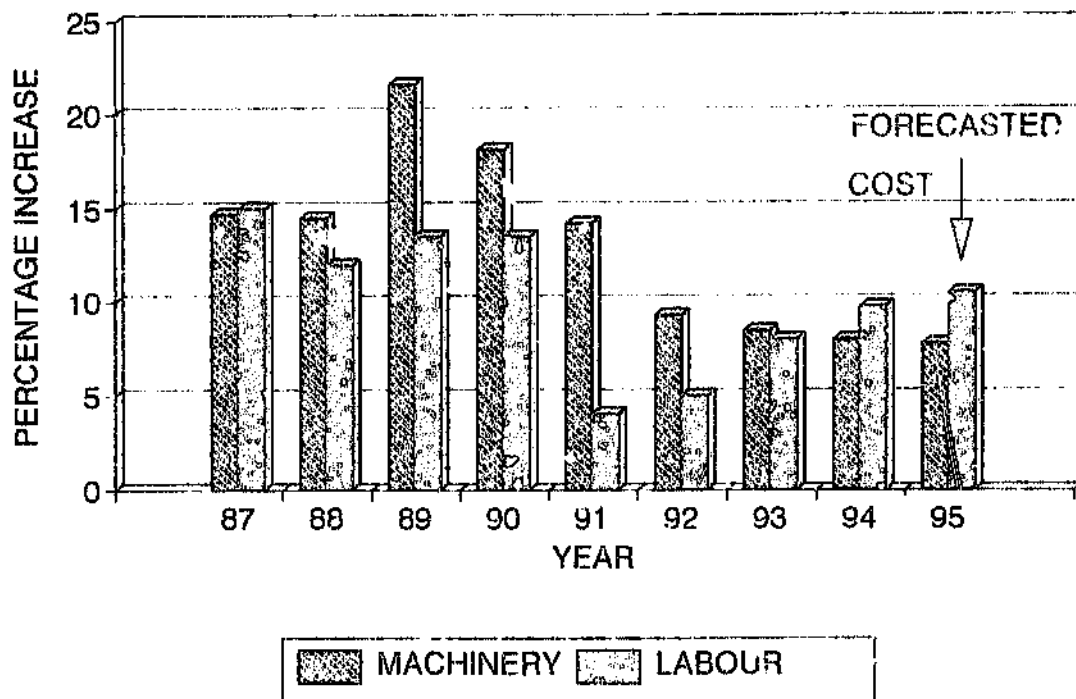
Background

The "culture" or "atmosphere, feeling" on a mine is closely aligned with the "financial" performance of the mine. Consequently changes in the performance directly affect the culture of that mine.

Historically REGM was the flagship of JCI's gold division. This meant that:

- Cost containment bonuses on monthly salaries were only paid by REGM.

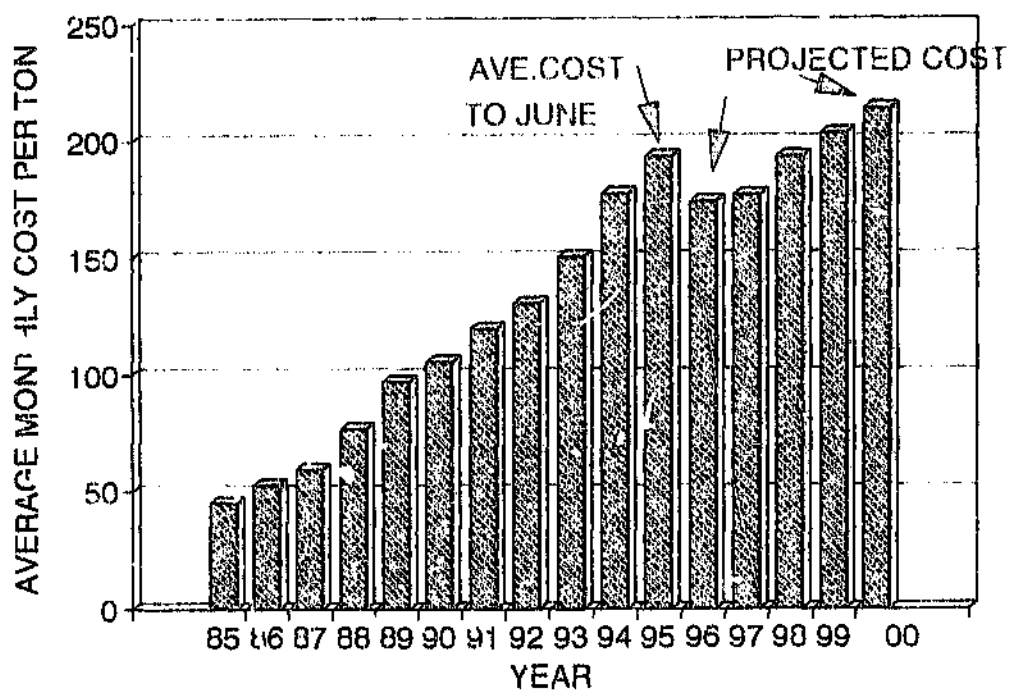
COST INCREASES IN LABOUR VERSES MACHINERY



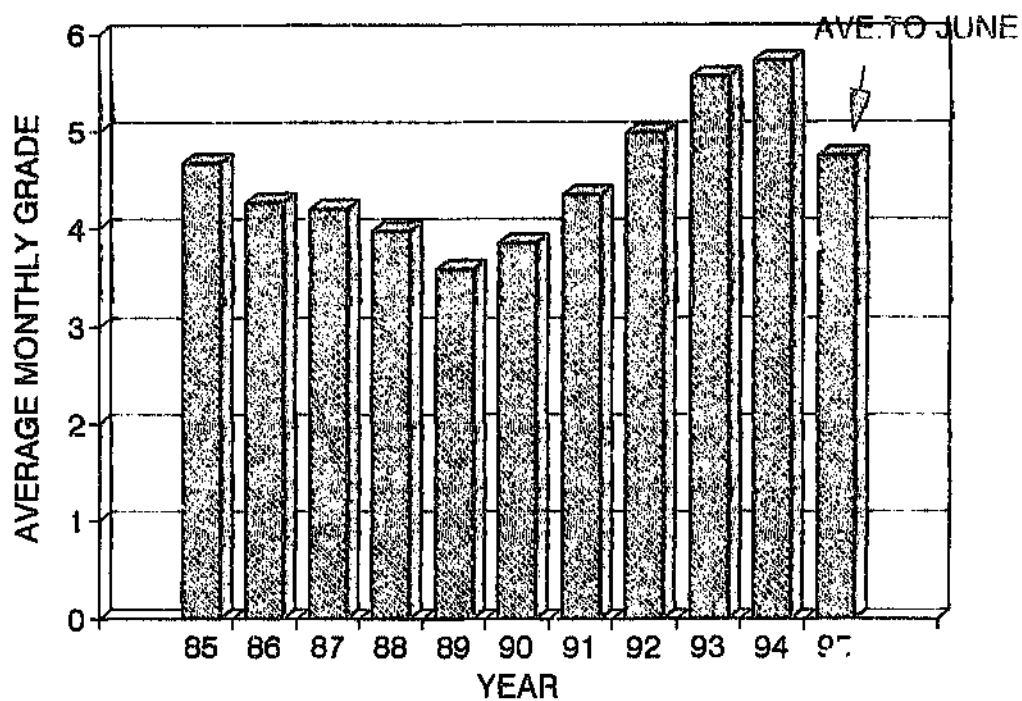
APPENDIX 12

RANDFONTEIN ESTATES GOLD MINE

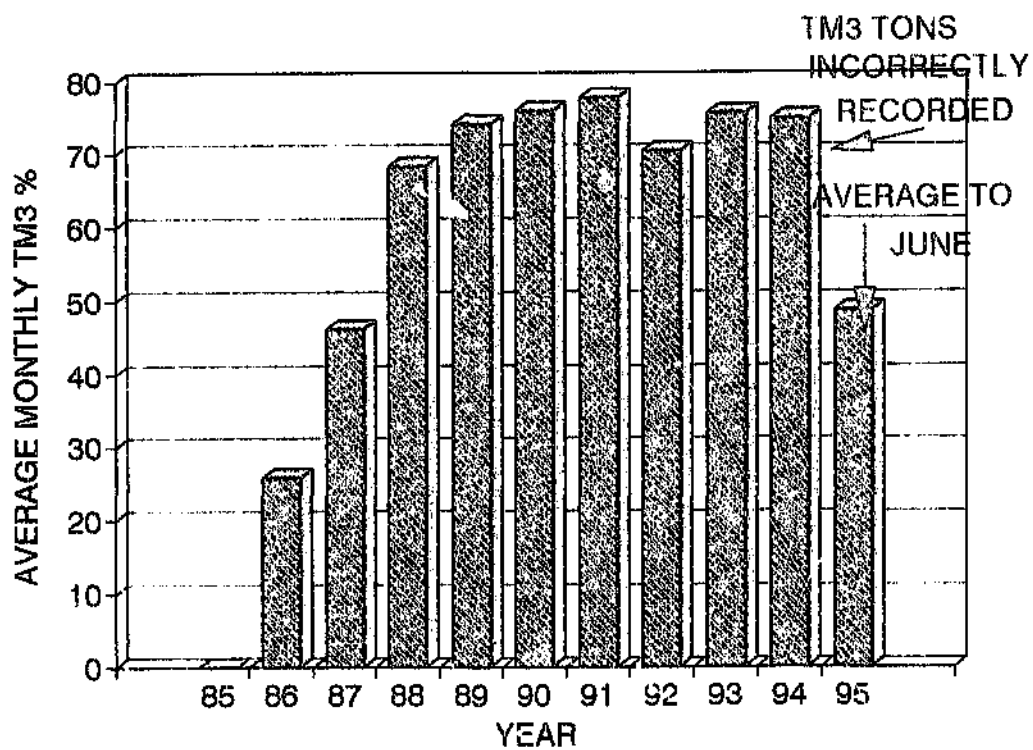
AVERAGE MONTHLY COST PER TON FROM U/G.



APPENDIX 11
RANDFONTEIN ESTATES GOLD MINE
AVERAGE MONTHLY GRADE FROM U/G.



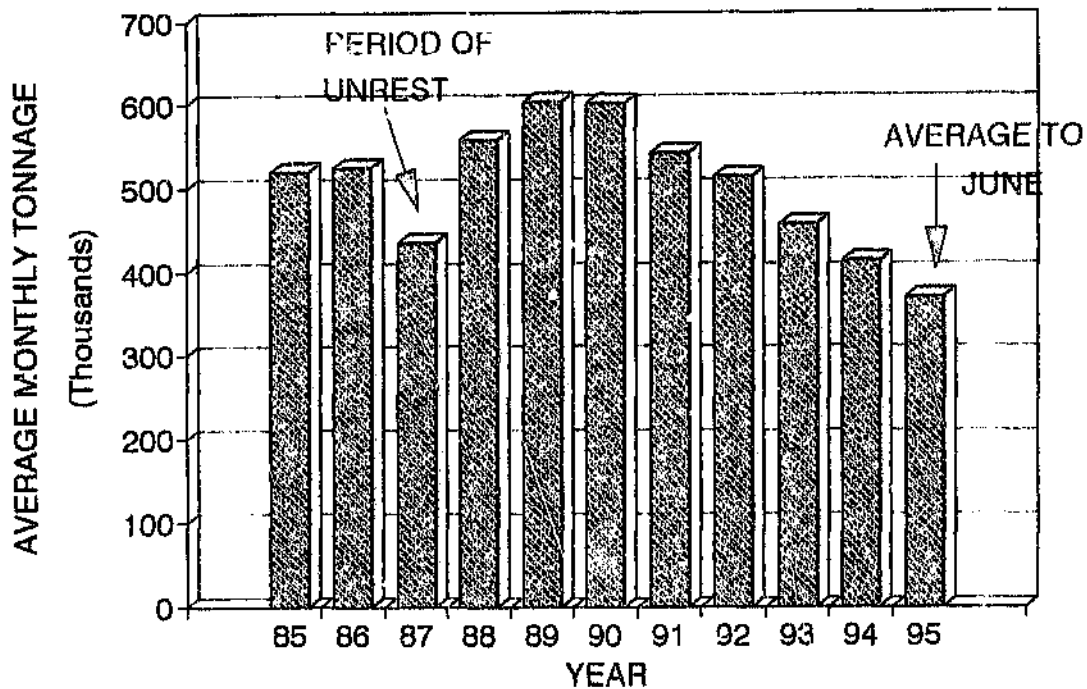
APPENDIX 10
RANDFONTEIN ESTATES GOLD MINE
AVERAGE MONTHLY TM3 % MINED

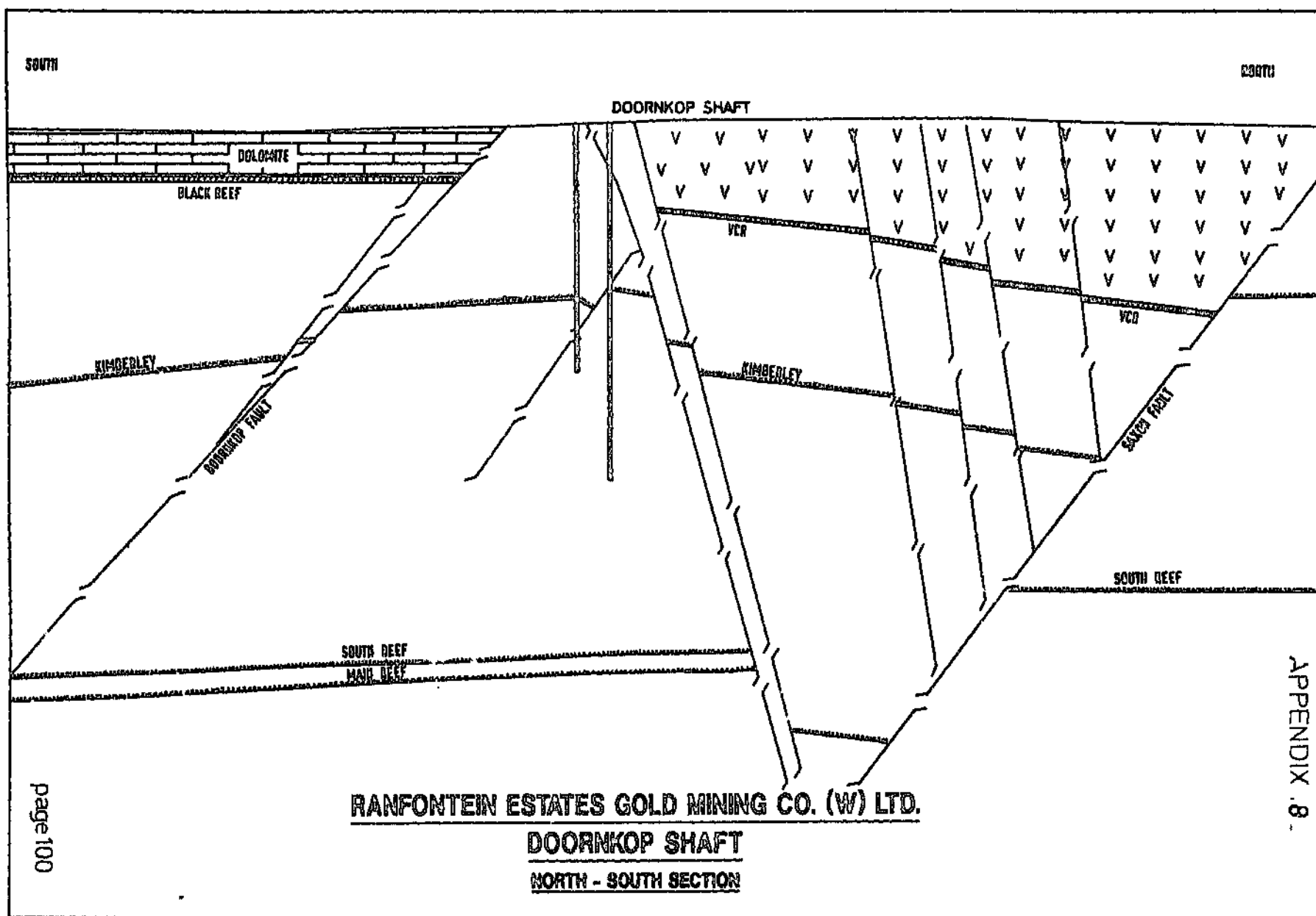


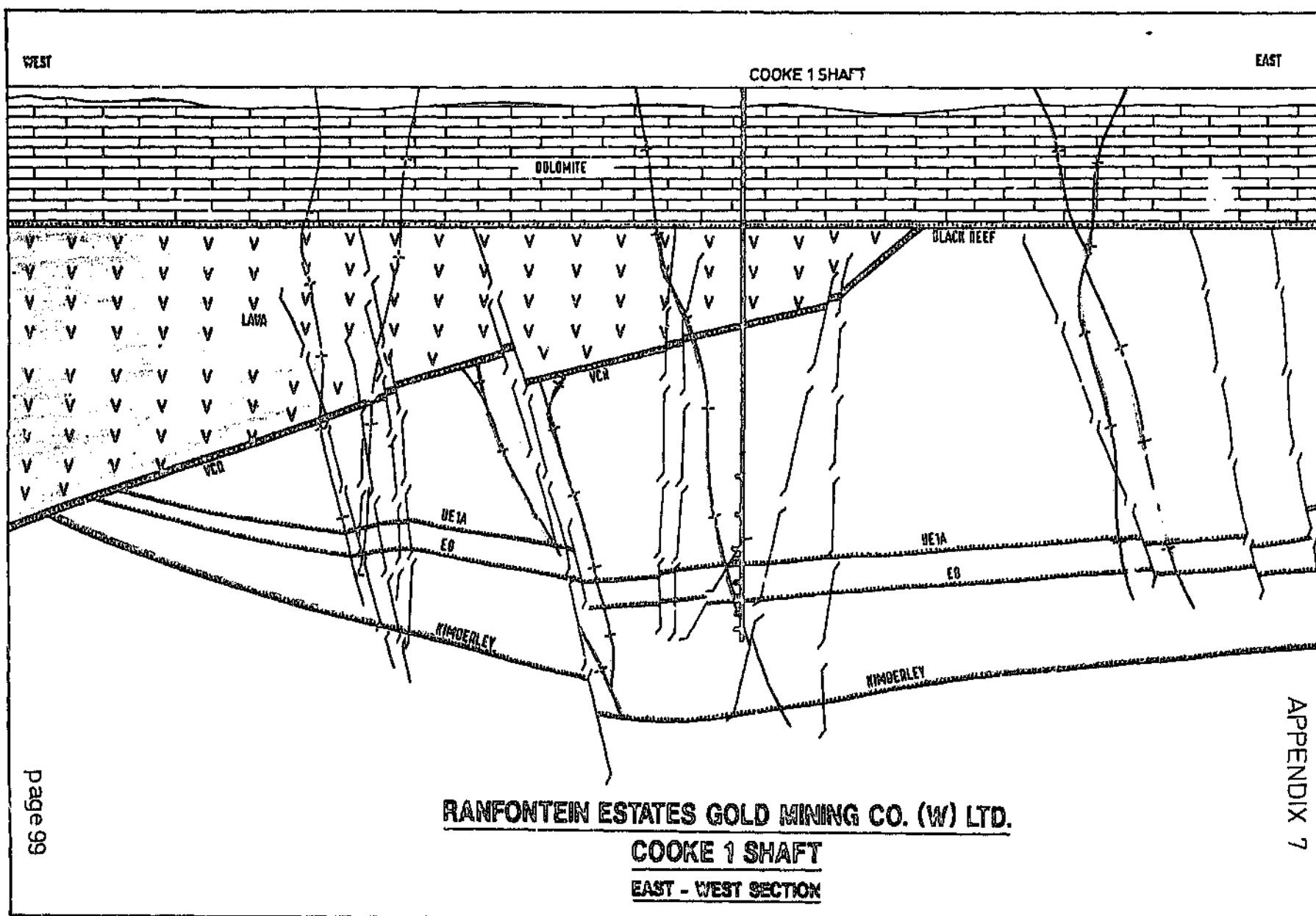
APPENDIX 9

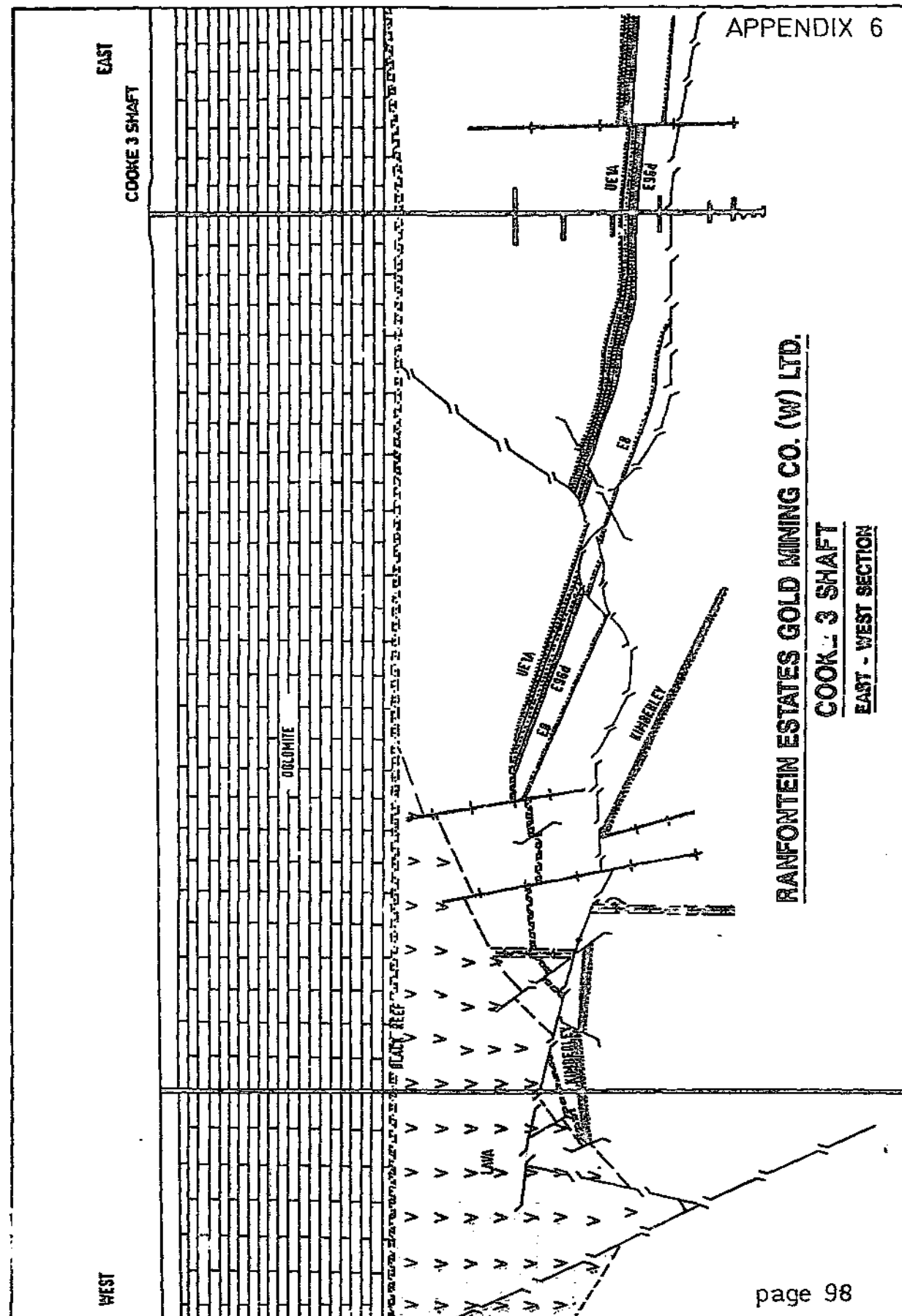
RANDFONTEIN ESTATES GOLD MINE

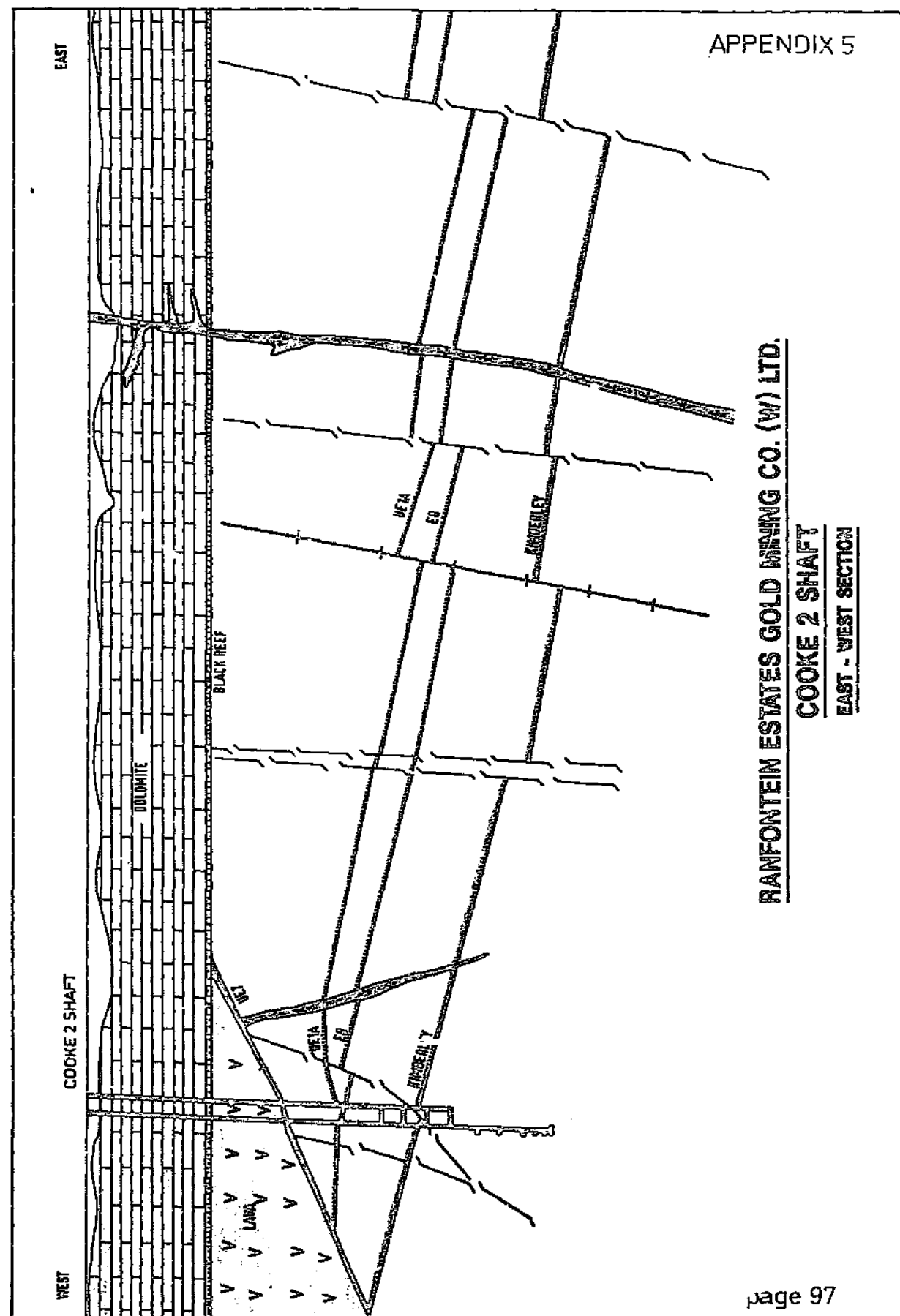
AVERAGE MONTHLY TONNAGE FROM U/G.



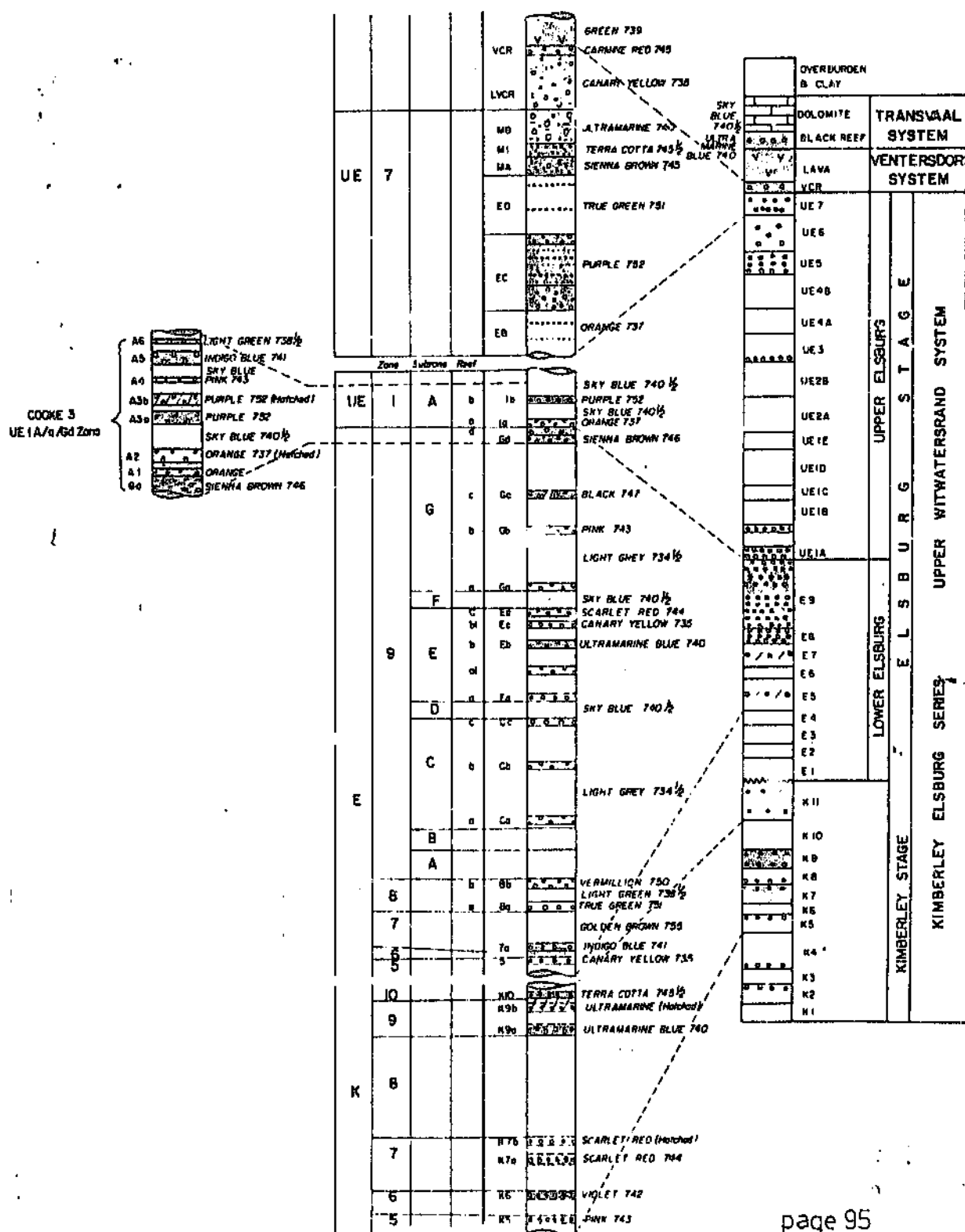


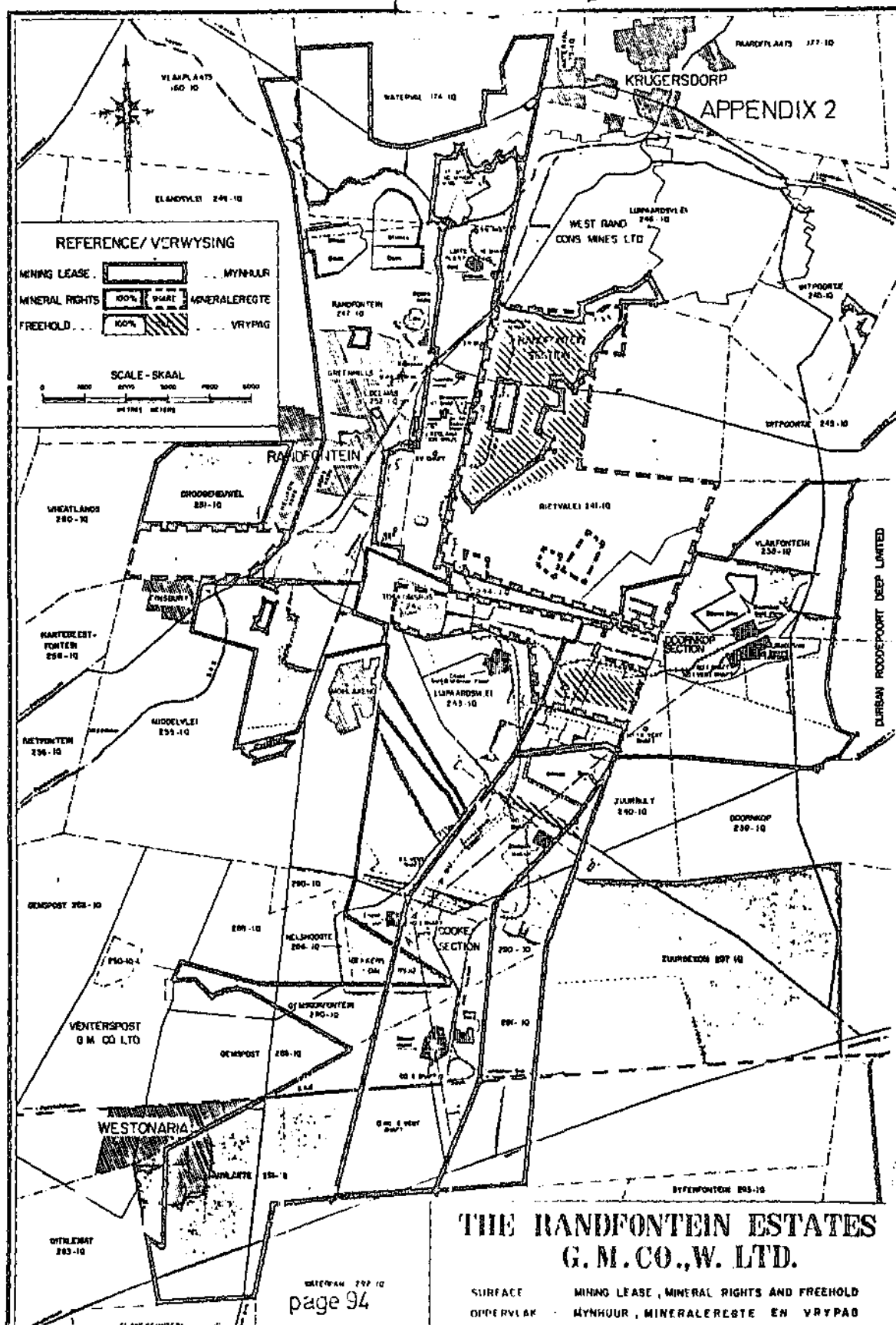


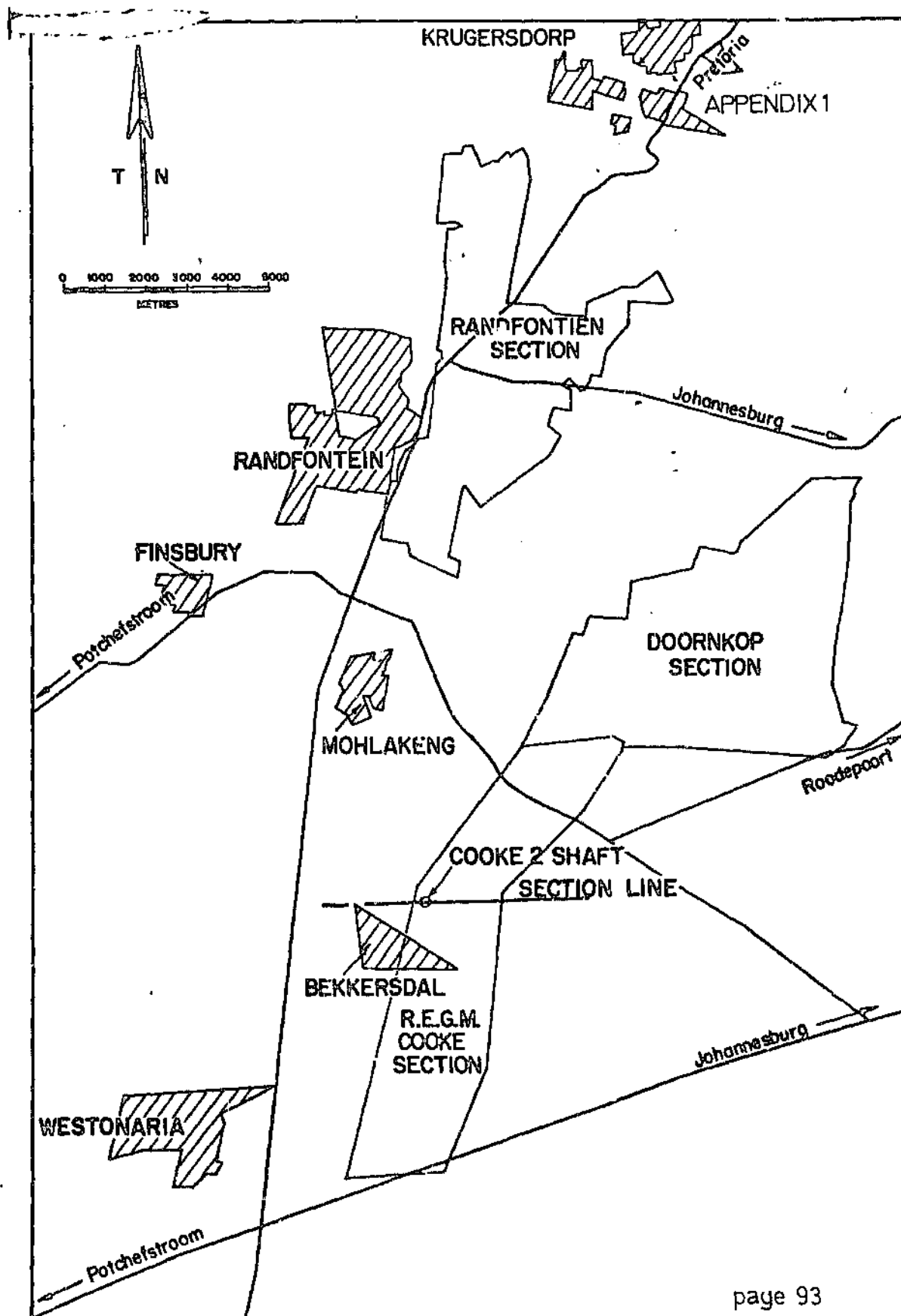




STRATIGRAPHICAL COLUMN OF COOKE SECTION

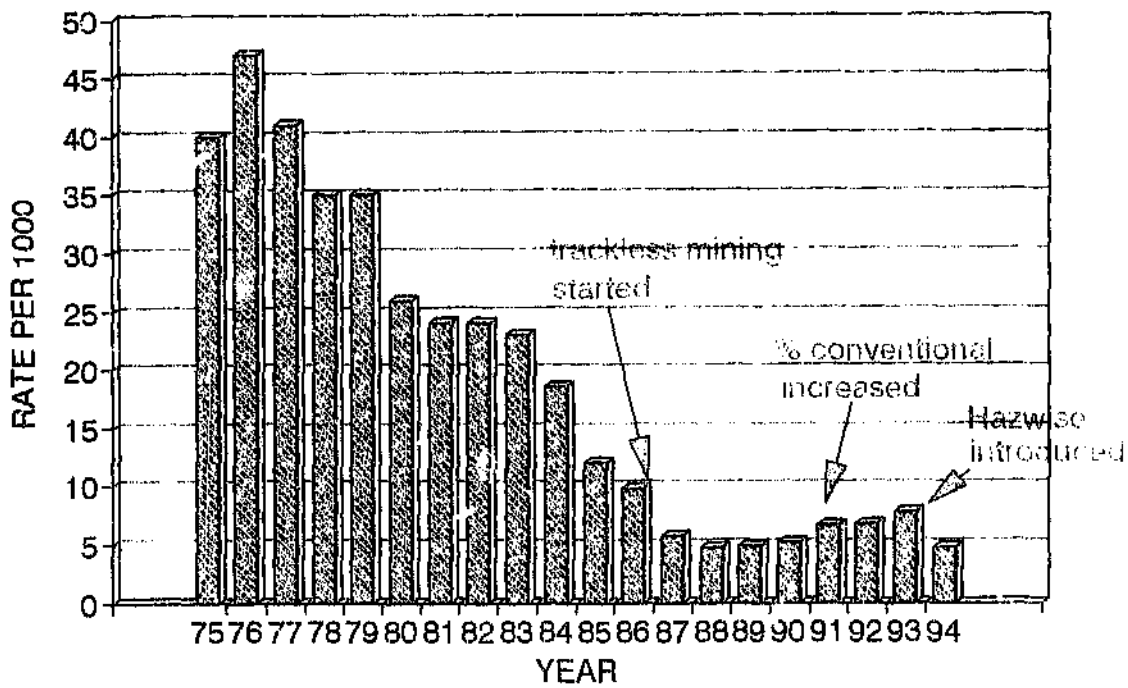






RANDFONTEIN ESTATES GOLD MINE

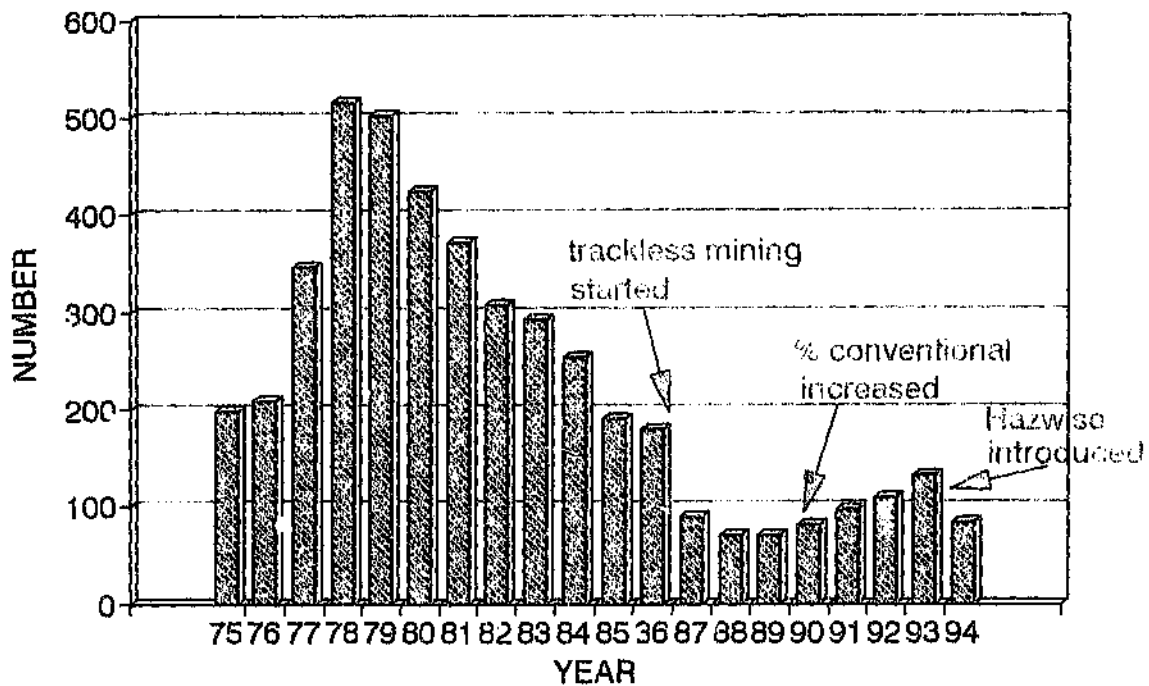
REPORTABLE RATE 1975 TO 1994



APPENDIX 24

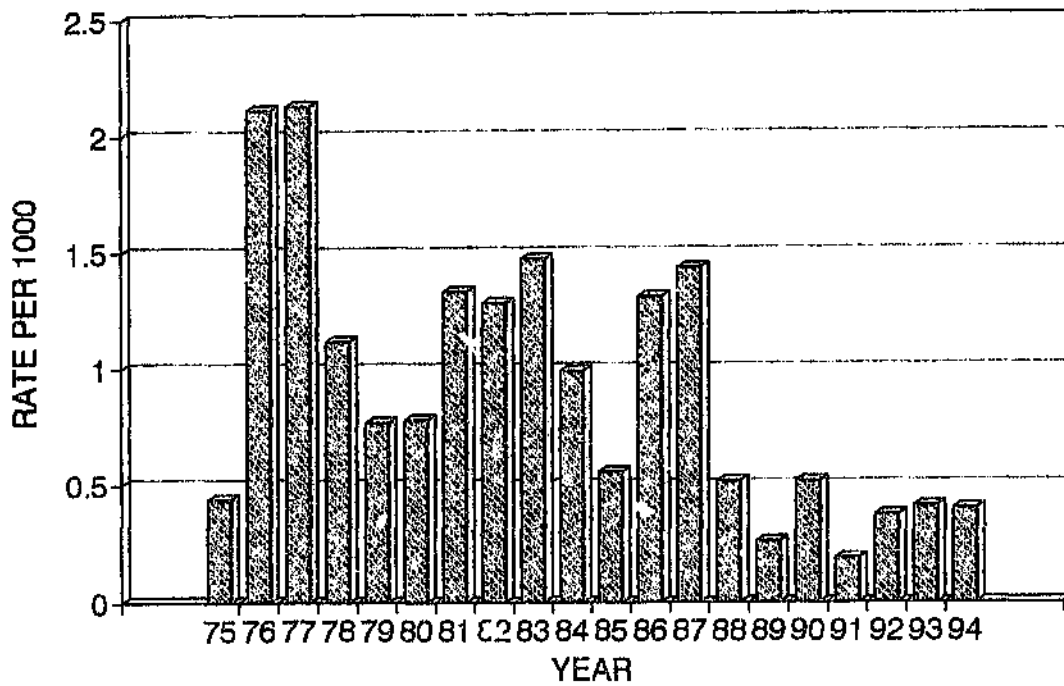
RANDFONTEIN ESTATES GOLD MINE

NUMBER OF REPORTABLES 1975 TO 1994



RANDFONTEIN ESTATES GOLD MINE

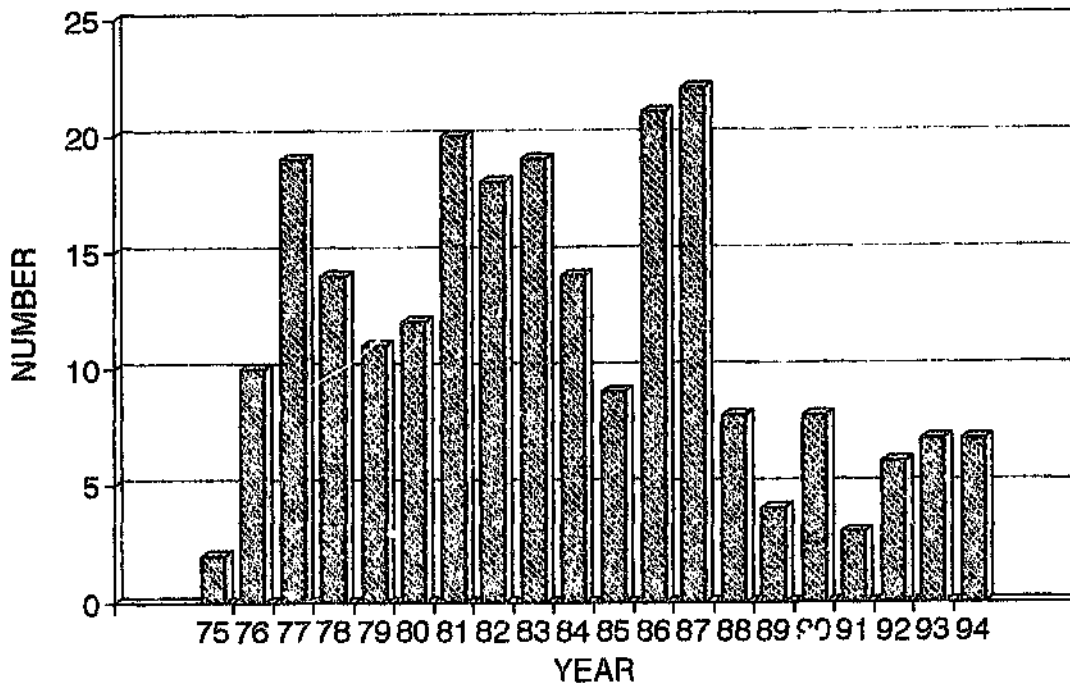
FATALITY RATE 1975 TO 1994



APPENDIX 22

RANDFONTEIN ESTATES GOLD MINE

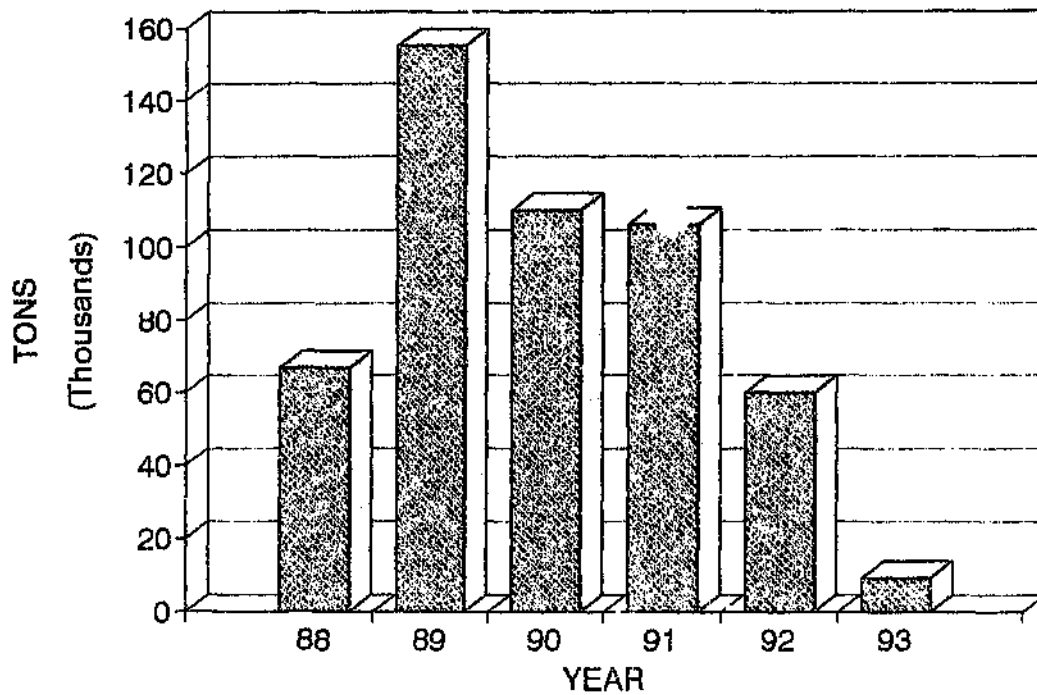
NUMBER OF FATALS 1975 TO 1994



APPENDIX 21

RANDFONTEIN ESTATE'S GOLD MINE

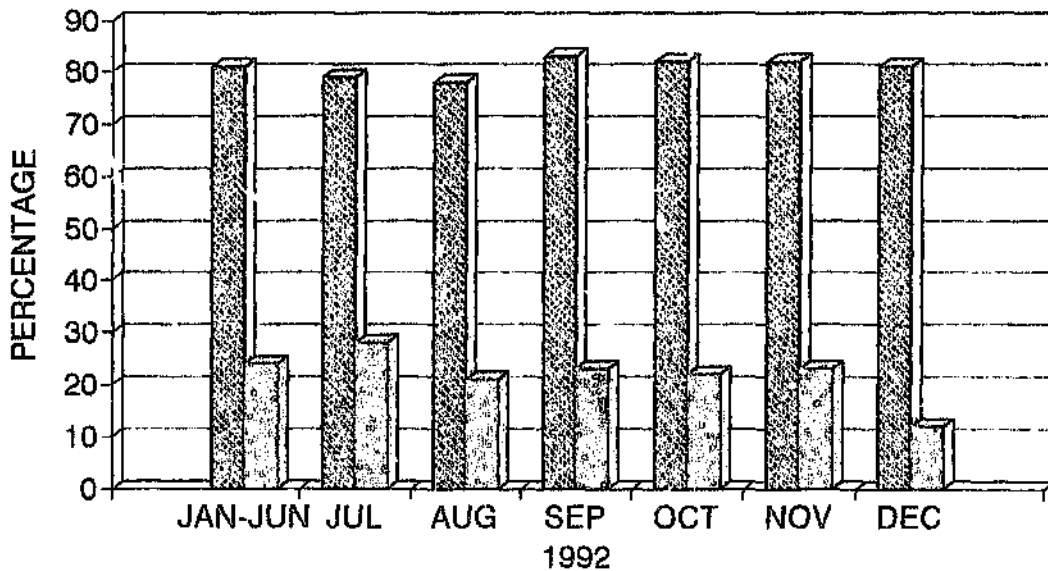
ANNUAL PRODUCTION BY STOMEC



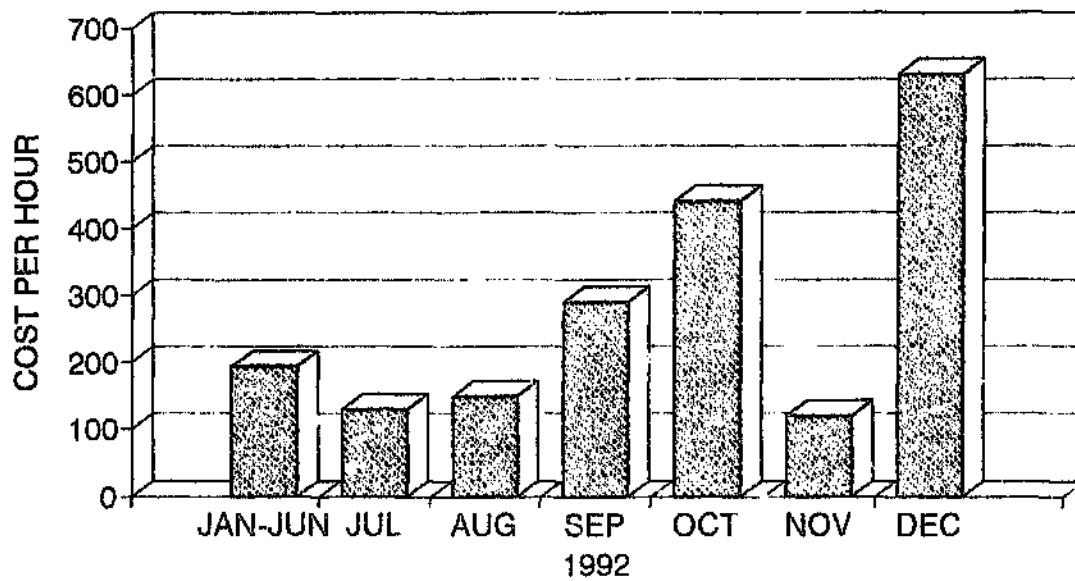
APPENDIX 20

RANDFONTEIN ESTATES GOLD MINE

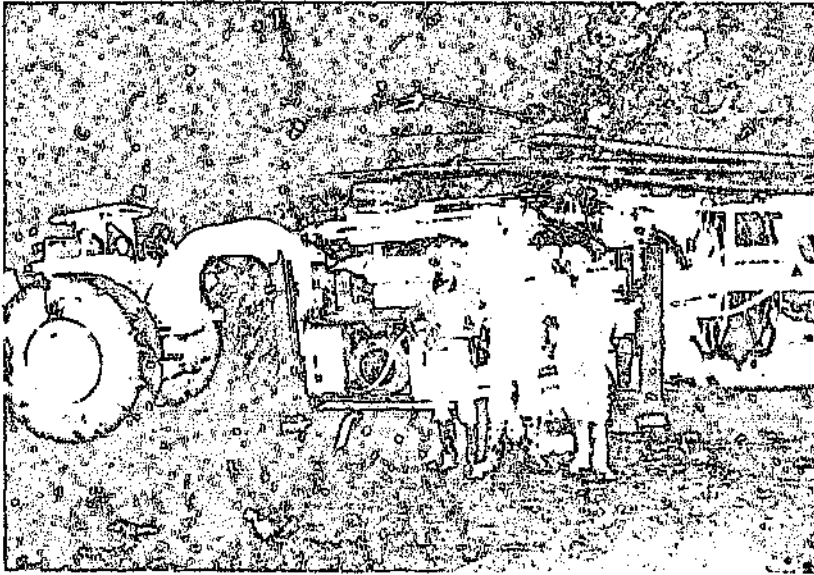
STOMECH AVAILABILITY VS. UTILIZATION



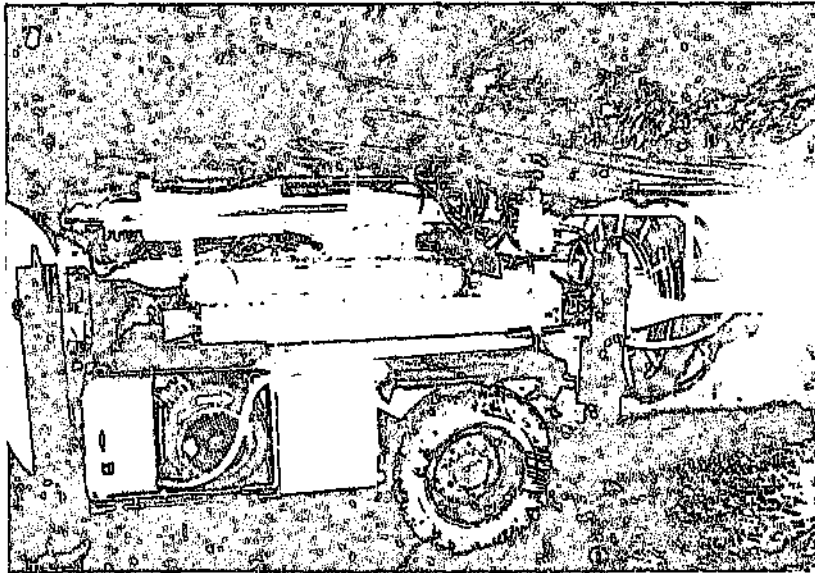
APPENDIX 19
RANDFONTEIN ESTATES GOLD MINE
STOMECH COST PER HOUR



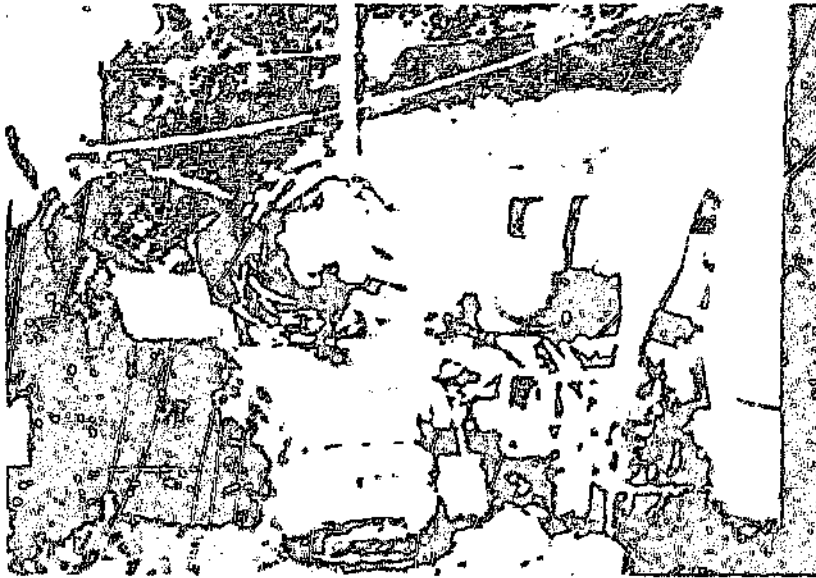
COST/HOUR



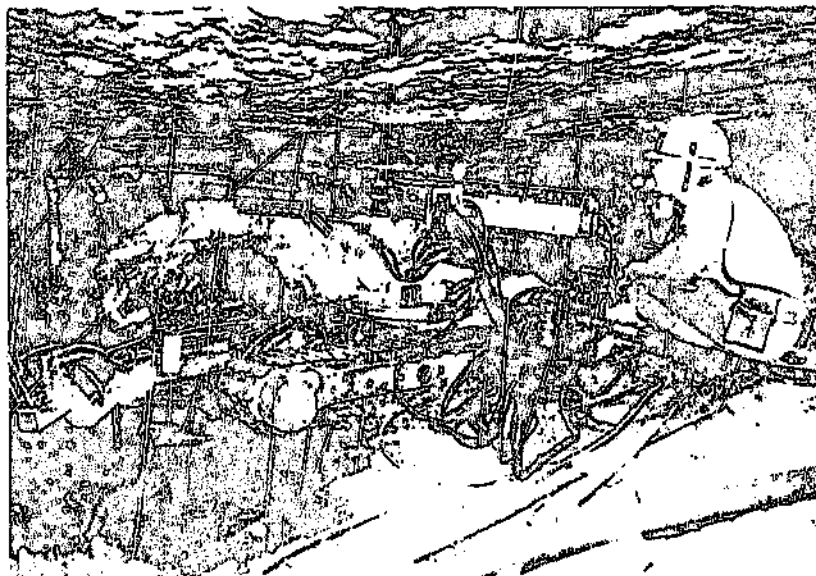
COMPLETE UNIT



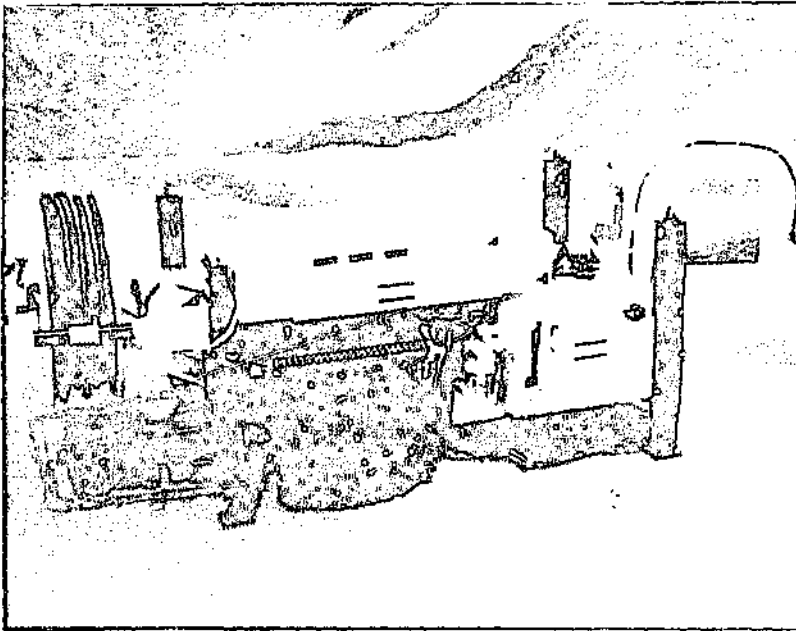
STOMEK CARRIER



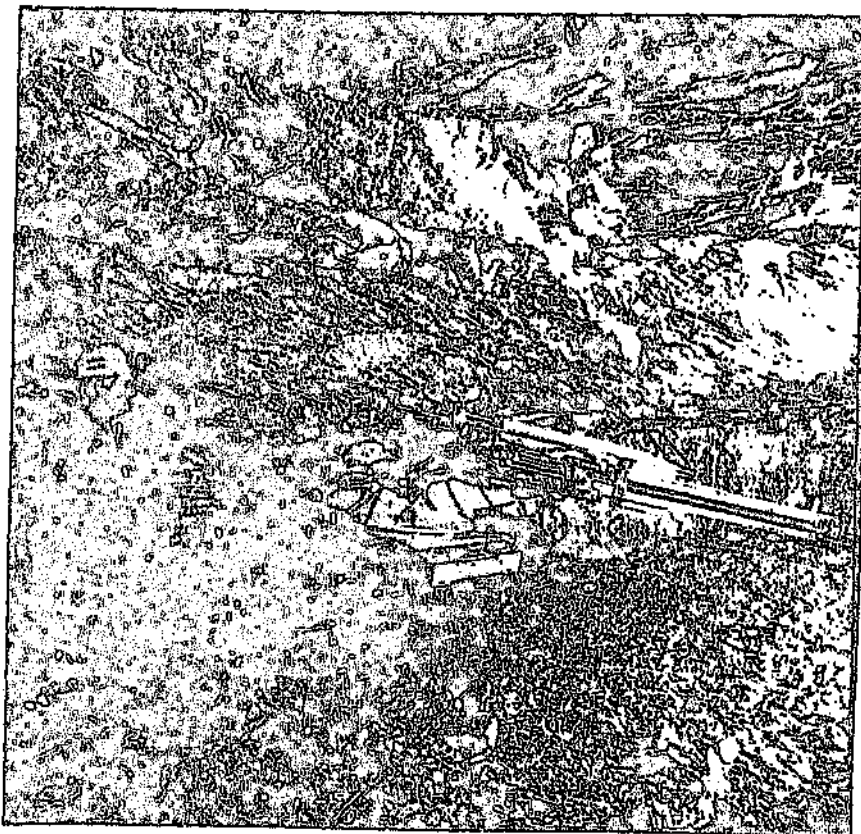
STOMEC, MOVING ONTO STOPE FACE



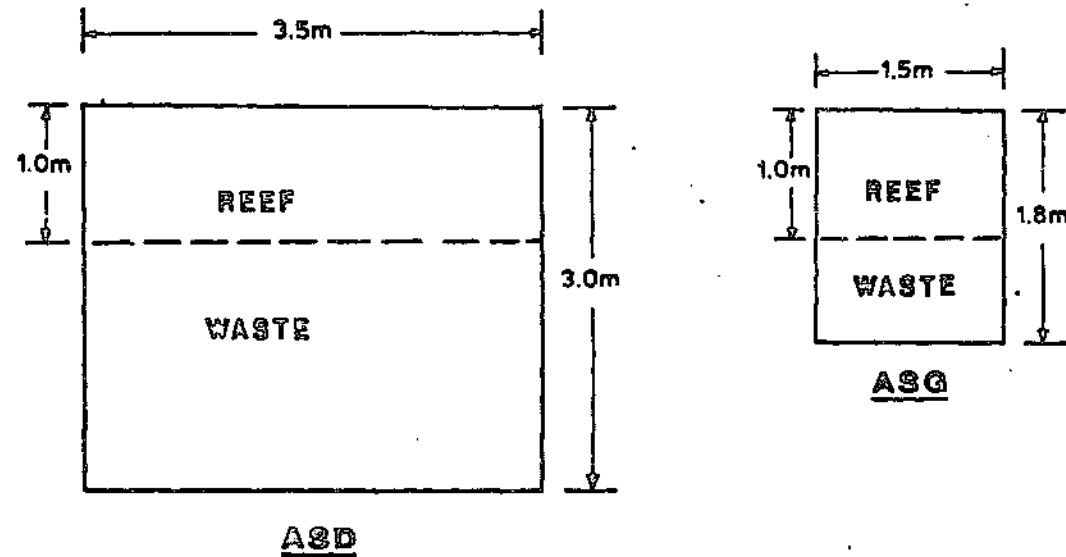
STOMEC DRILLING



STOME C CARRIER



STOME C DRILLING

SECTION SHOWING DILUTION OF ASD vs ASG

WASTE TONS ASD = 19.25 TONS/m ADVANCE

WASTE TONS ASG = 3.30 TONS/m

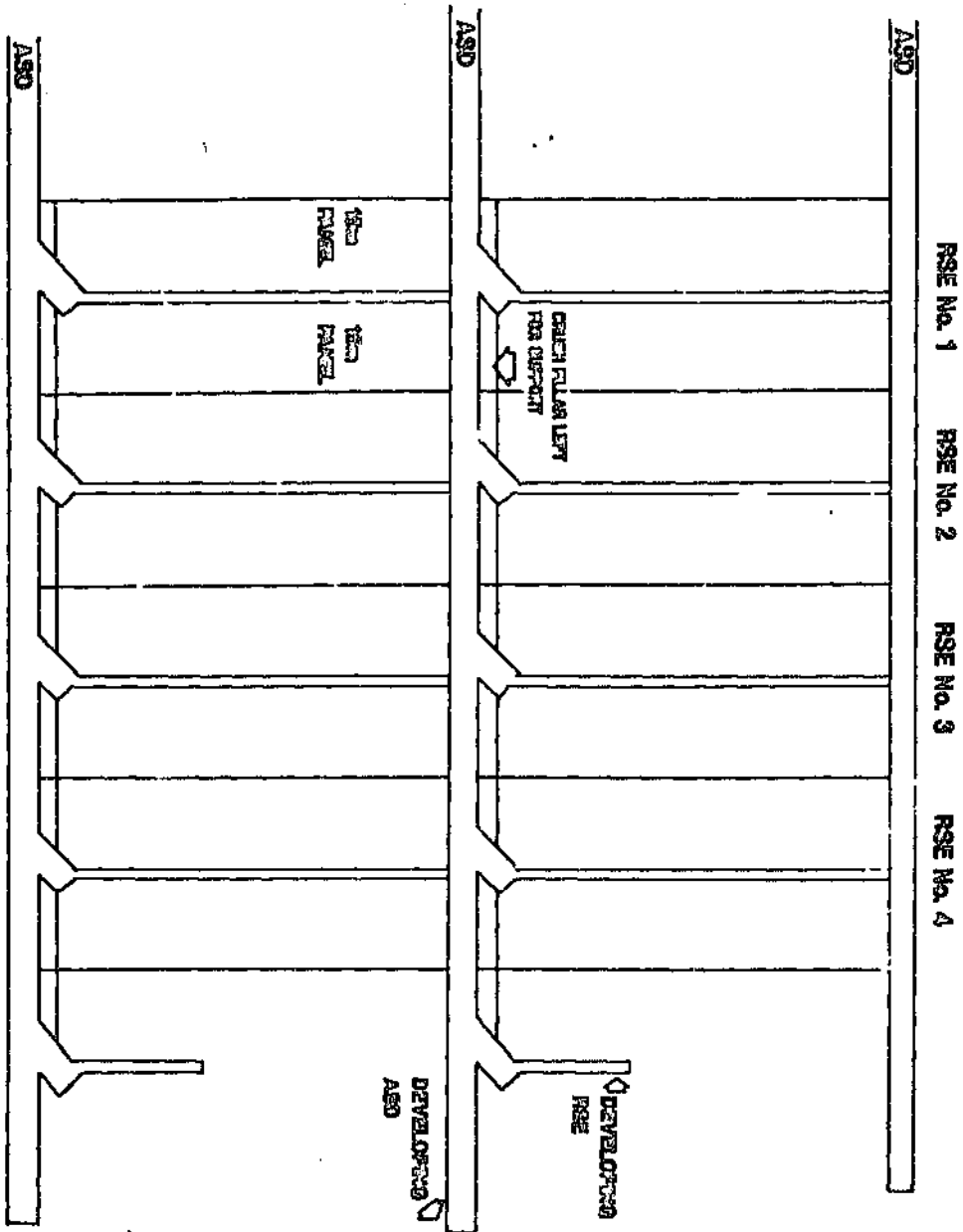
DIFFERENCE = 15.95 TONS/m

○ R165/TON MILLED .% 15.95 = 165

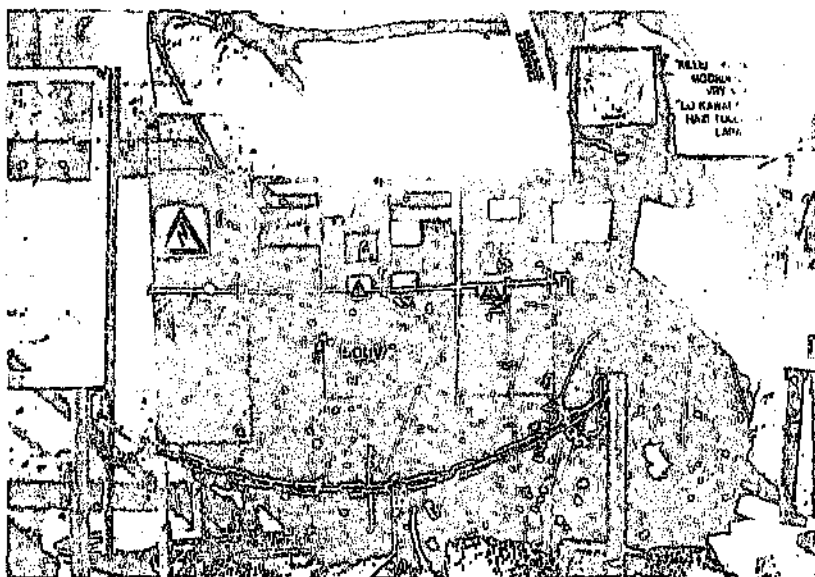
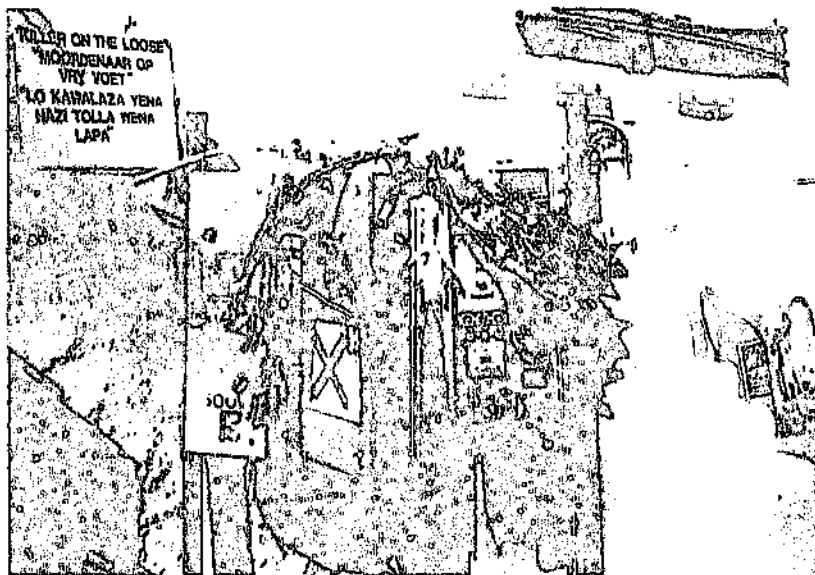
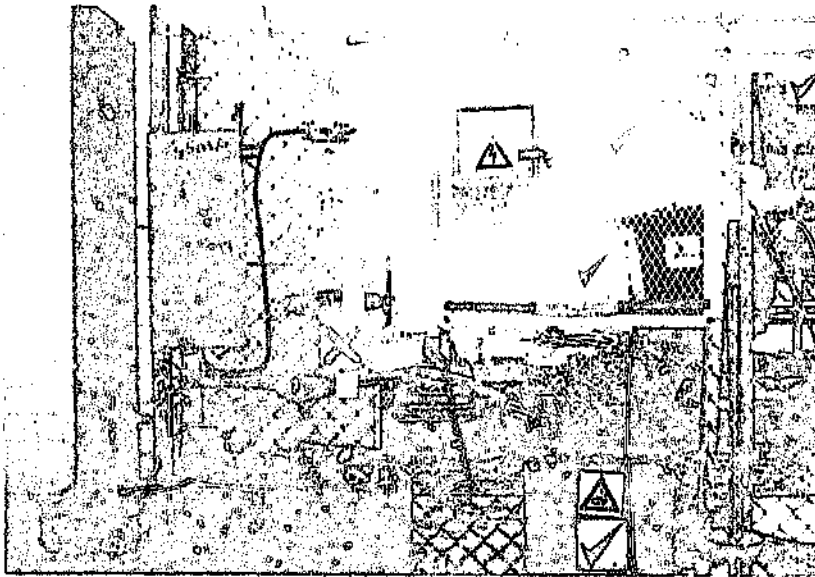
= R2 631.75/m ADVANCE

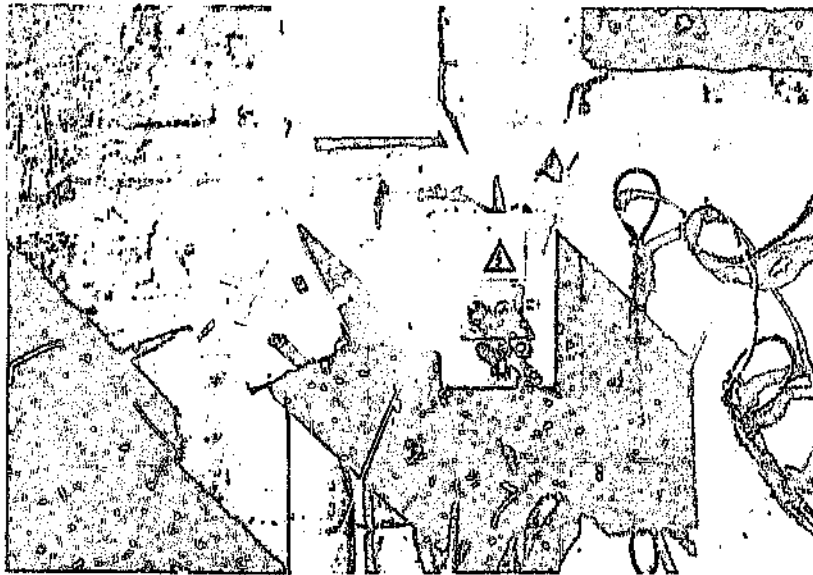
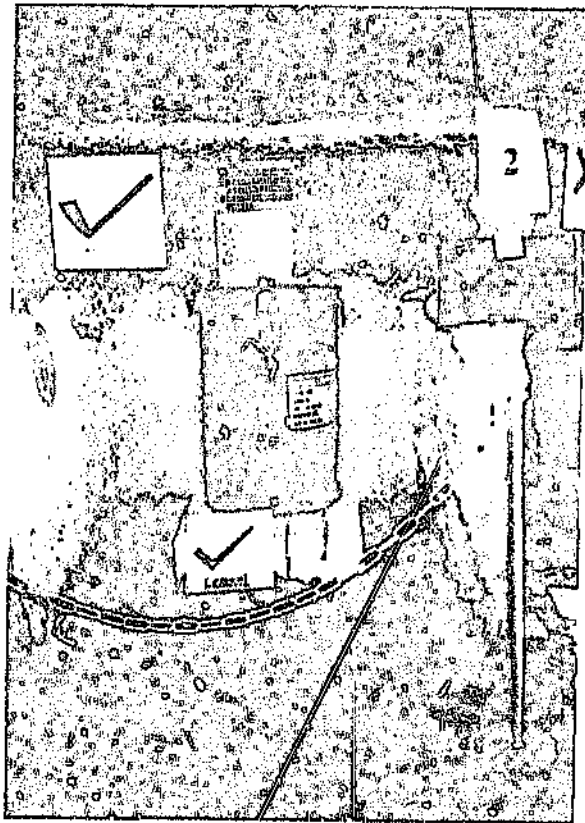
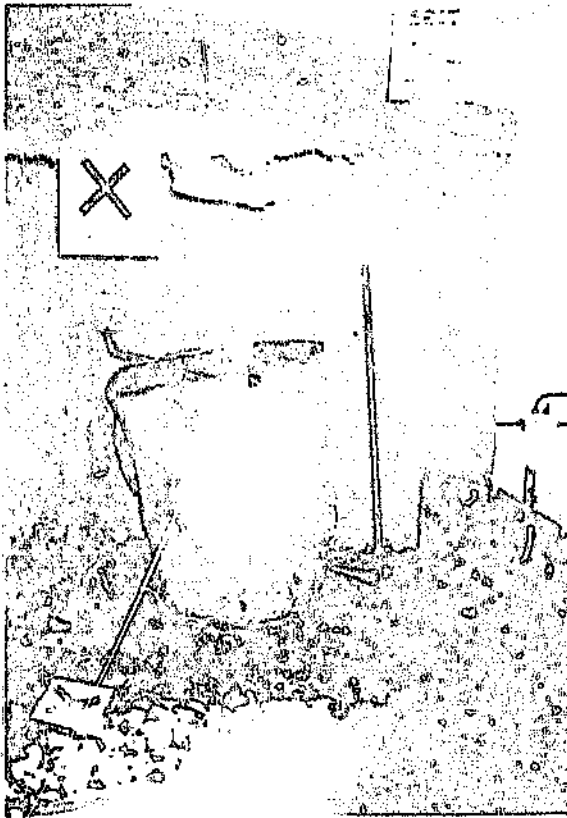
○ 15m FACE ADVANCE = R39 476/MONTH SAVING

NARROW TRACKLESS AT COOKE 3







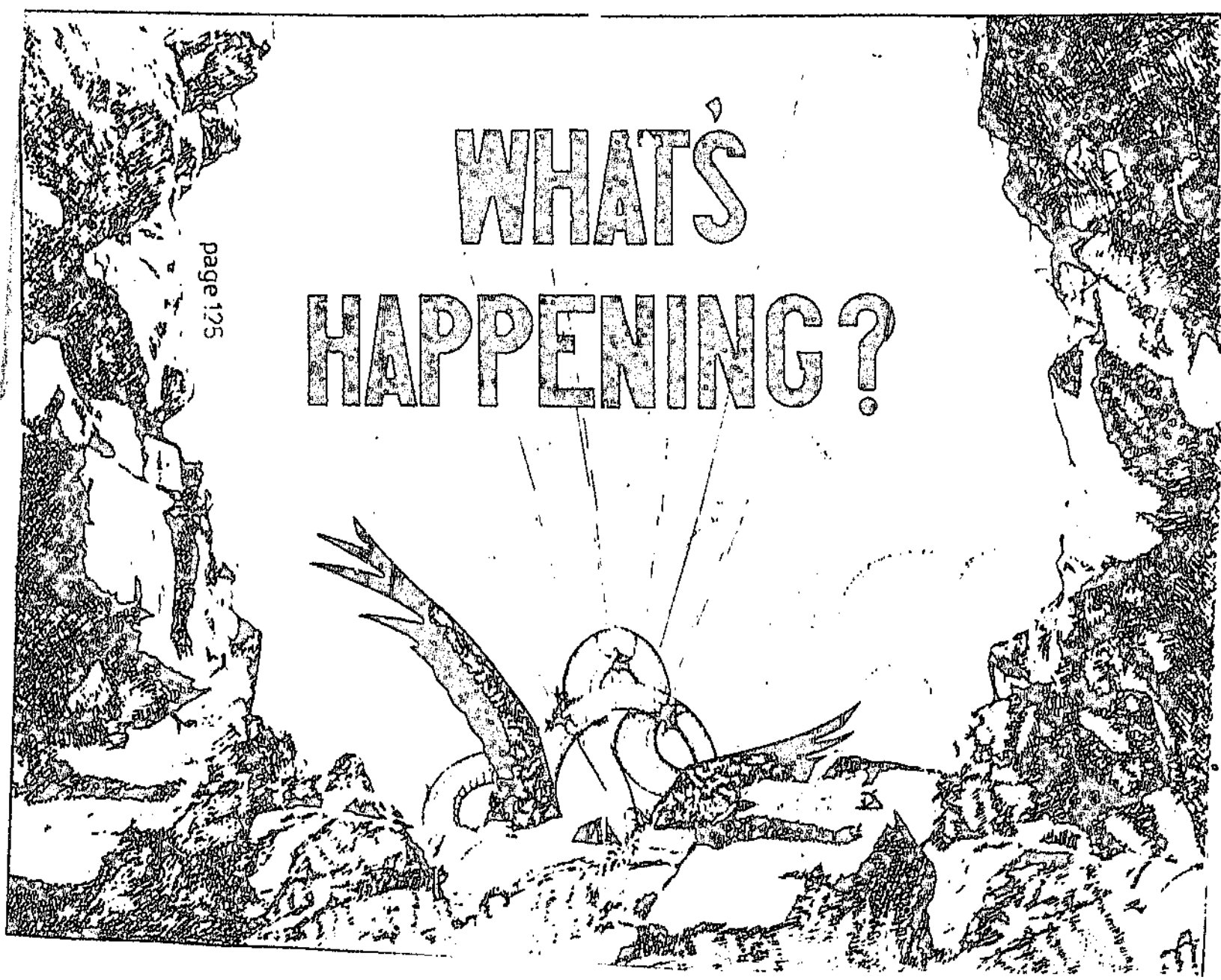




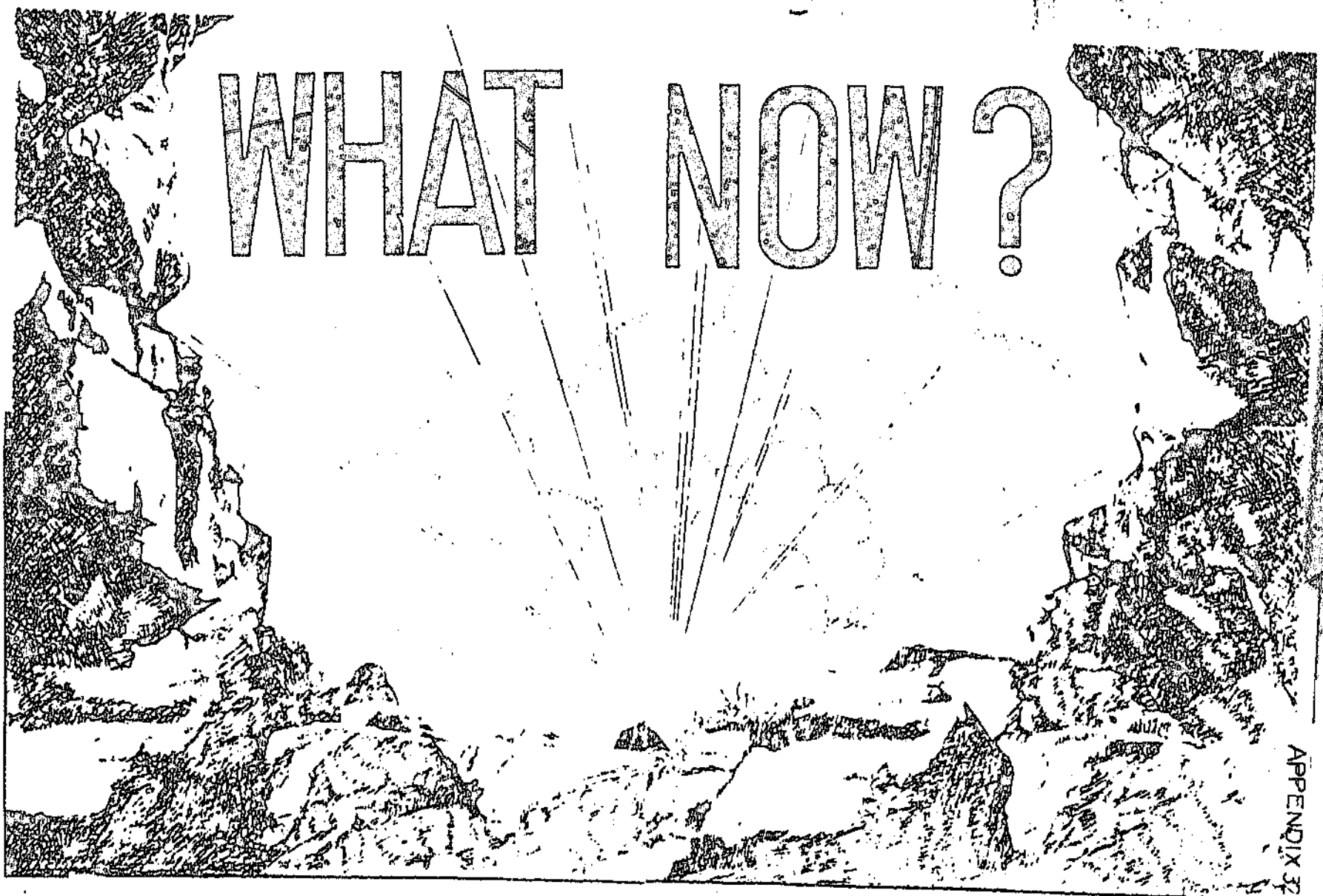
"HAZWISE"

WHAT'S HAPPENING?

page 125



WHAT NOW?

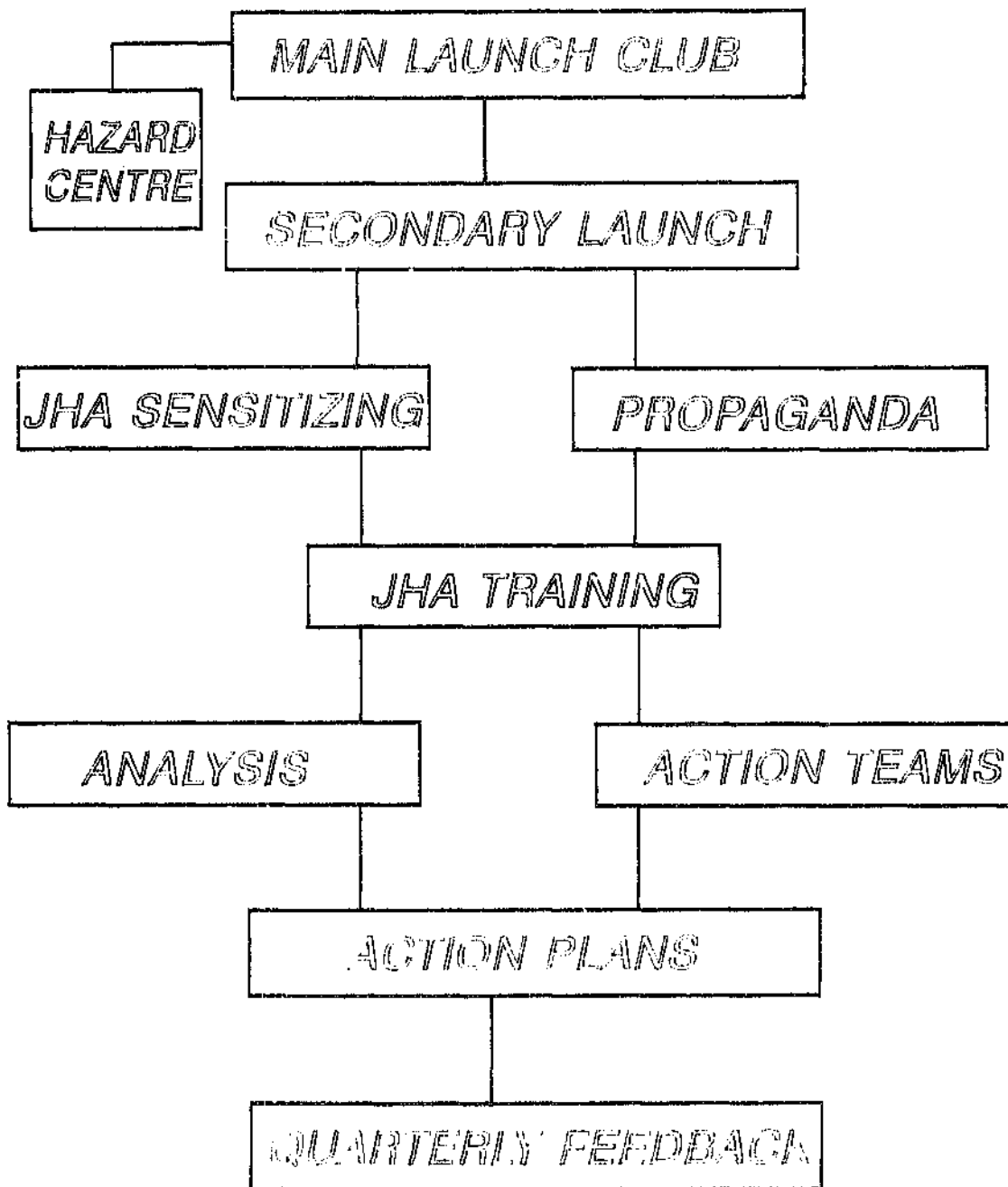


WIE GAAN WEN ?



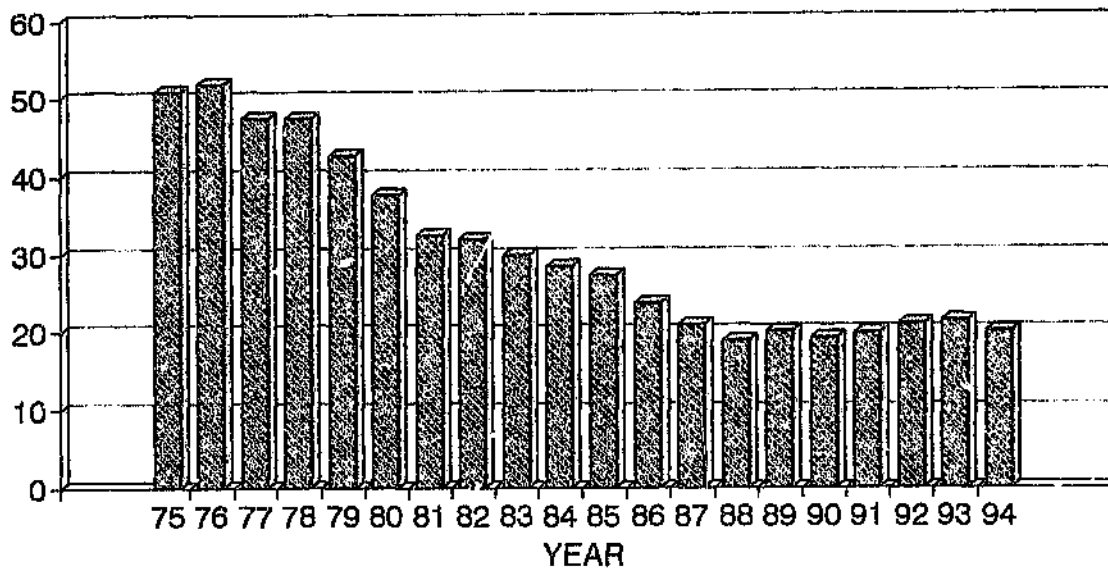
HAZWISE

HAZWISE - CAMPAIGN SEQUENCE



CHAMBER OF MINES GOLD

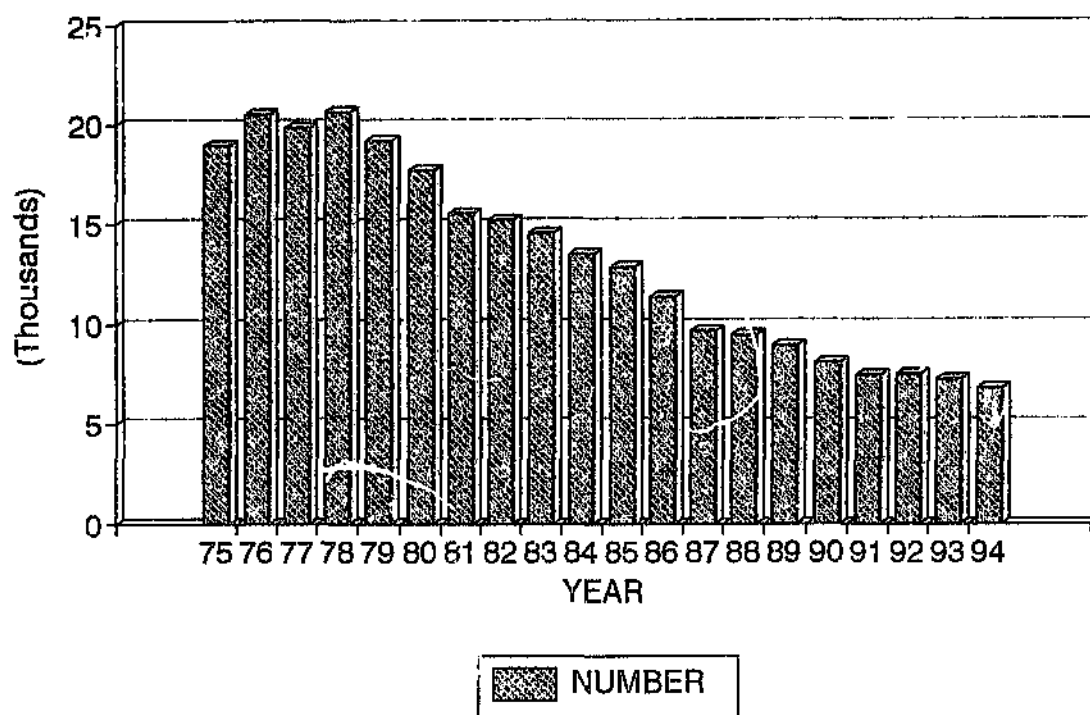
REPORTABLE RATE 1975 TO 1994



■ RATE PER 1000

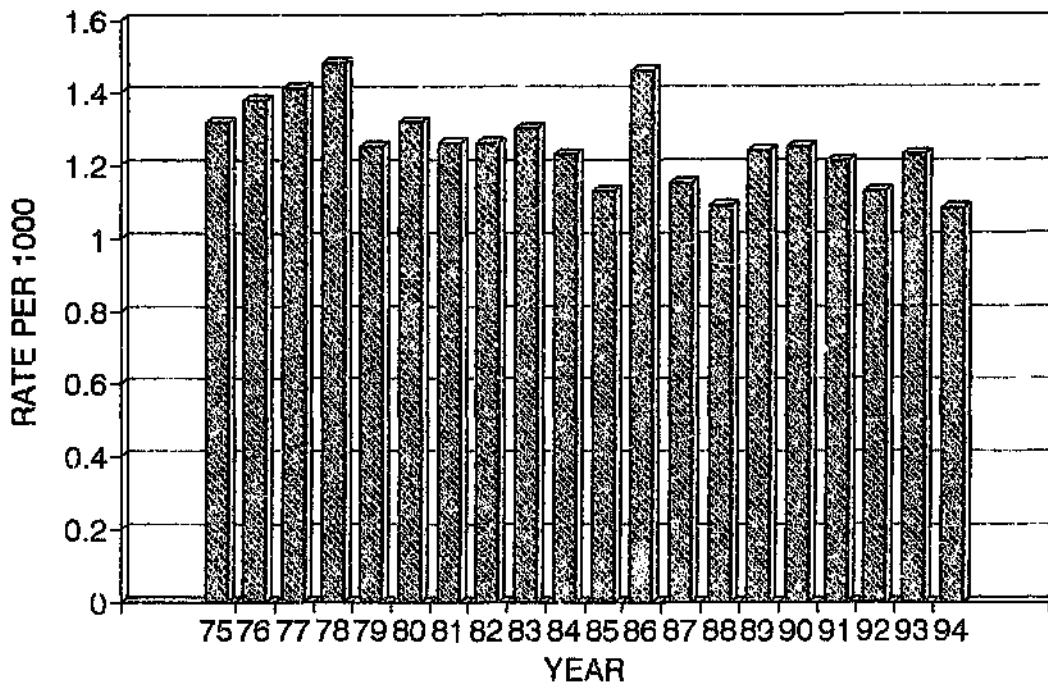
CHAMBER OF MINES GOLD

NUMBER OF REPORTABLES 1975 TO 1994



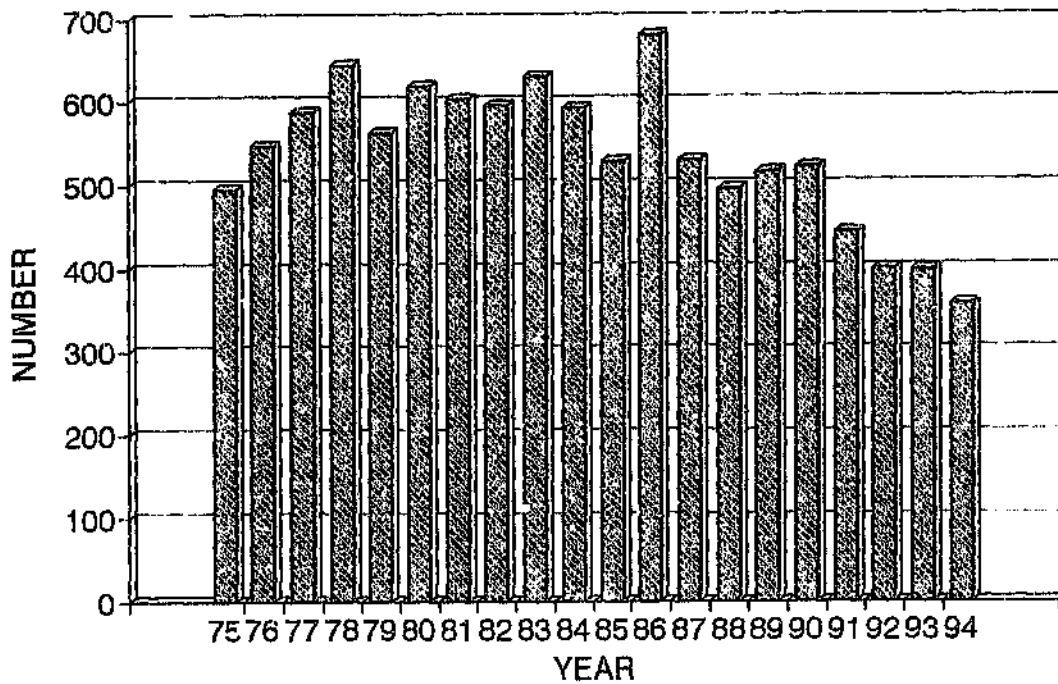
CHAMBER OF MINES GOLD

FATALITY RATE 1975 TO 1994



CHAMBER OF MINES GOLD

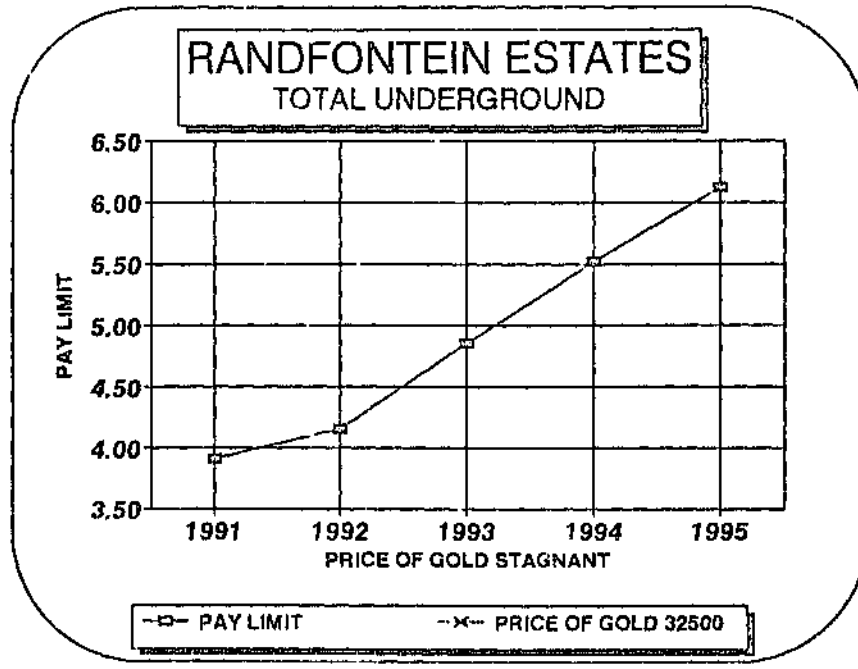
NUMBER OF FATALS 1975 TO 1994



RANDFONTEIN ESTATES GOLD MINING COMPANY PAY LIMITS FOR TOTAL UNDERGROUND

ASSUME STAGNANT GOLD PRICE

YEAR	PAY LIMIT	GOLD PRICE
1991	3.91	32500
1992	4.15	32500
1993	4.85	32500
1994	5.52	32500
1995	6.12	32500



	TOTAL MINE			
	PREV AVE ACT '90	YTD AVE ACT'94/95	INCREM COST	INCREASED TONS
TONS MILLED (000's)	802	389		481
GRADE	3.86	5.03		
KILOGRAMS RECOVERED	2324	1850		1058
ADDITIONAL TONS (000's)				92
TOTAL TONS (000's)				481
GOLD PRICE	43795	43795		
DIRECT INSTOPE COST	49952	32300	82.98	40558
SHARED SEMI - VAR	10280	9287	4.78	9719
ON SHAFT OVERHEADS	5084	4505	2.35	4708
	65297	46133	90.08	55042
TRANSPORT COST	1256	814	2.09	1001
TREATMENT COST	10429	8744	17.32	8384
TOTAL OVERHEAD COST	16779	14981	3.84	15318
TOTAL COST	92782	68051	113.34	79745
PAYLIMIT	3.80	4.40	2.88	4.14

ASSUMPTIONS:

- TOTAL INSTOPE COST DIRECT VARIABLE TO TONS.
- 20% OF SHARED SEMI-VARIABLE AND ON SHAFT OVERHEAD COST VARIABLE TO TONS.
- BUYING FEE, GOLD REALISATION CHANGE AND A PORTION OF CHAMBER COSTS ARE DIRECTLY RELATED TO TONS AND KG's.
- MINE CALL FACTOR AT 95% AND RESIDUE AT 0.18 G/T.
- ALLOCATED COMPANY OVERHEADS WILL INCREASE BY 10% OF INCREASE/DECREASE IN TONS MILLED
- TRANSPORT AND TREATMENT COSTS TO INCREASE/DECREASE IN RATIO TO TONS MILLED
- 1990 COST BASED ON CURRENT COSTS
- APPLIED BPR CONCEPTS INCREASE OF 30% IN TONS MILLED

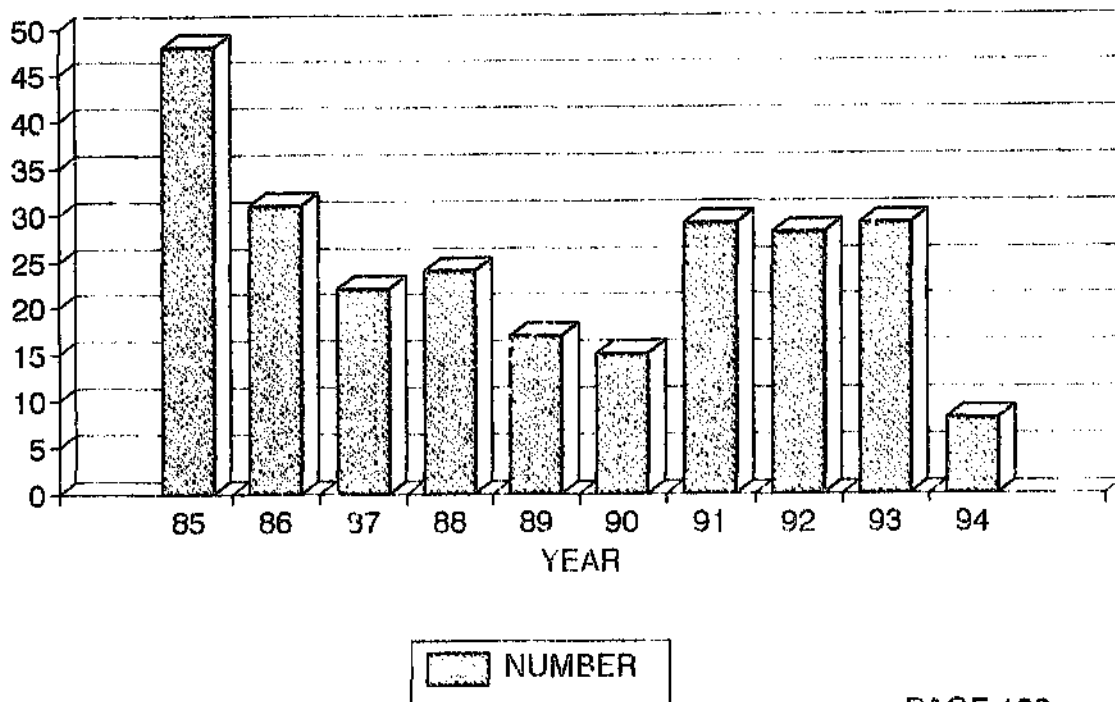
	TOTAL MINE			
	PREV AVE ACT '90	YTD AVE ACT'94/95	INCREM COST	REDUCED TONS
TONS MILLED (000's)	602	389		370
GRADE	3.86	5.03		
KILOGRAMS RECOVERED	2324	1958		1812
ADDITIONAL TONS (00' s)				-19
RECOVERY GRADE				5.03
ADDITIONAL KG's				-148
TOTAL TONS (000's)				370
GRADE				4.90
TOTAL KG's				1812
GOLD PRICE	43795	43795		
DIRECT INSTOPE COST	49952	32300	82.98	29954
SHARED SEMI - VAR	10260	2.37	4.78	9158
ON SHAFT OVERHEADS	5004	4585	2.35	4531
	65207	46133	90.08	43640
TRANSPORT COST	1258	814	2.08	779
TREATMENT COST	10428	6744	7.32	8356
TOTAL OVERHEAD COST	15779	14981	3.84	14884
TOTAL COST	92762	68651	113.34	65856
PAYLIMIT	3.88	4.40	2.88	4.42

ASSUMPTIONS :

- TOTAL INSTOPE COST DIRECT VARIABLE TO TONS.
- 20% OF SHARED SEMI-VARIABLE AND ON SHAFT OVERHEAD COST VARIABLE TO TONS.
- BUYING FEE, GOLD REALISATION CHARGE AND A PORTION OF CHAMBER COSTS ARE DIRECTLY RELATED TO TONS AND KG's.
- MINE CALL FACTOR AT 95% AND RESIDUE AT 0.16 G/T.
- ALLOCATED COMPANY OVERHEADS WILL INCREASE BY 10% OF INCREASE/DECREASE IN TONS MILLED
- TRANSPORT AND TREATMENT COSTS TO INCREASE/DECREASE IN RATIO TO TONS MILLED
- 1990 COST BASED ON CURRENT COSTS
- ALL COSTS EQUIVALENT TO 1994/1995 COST

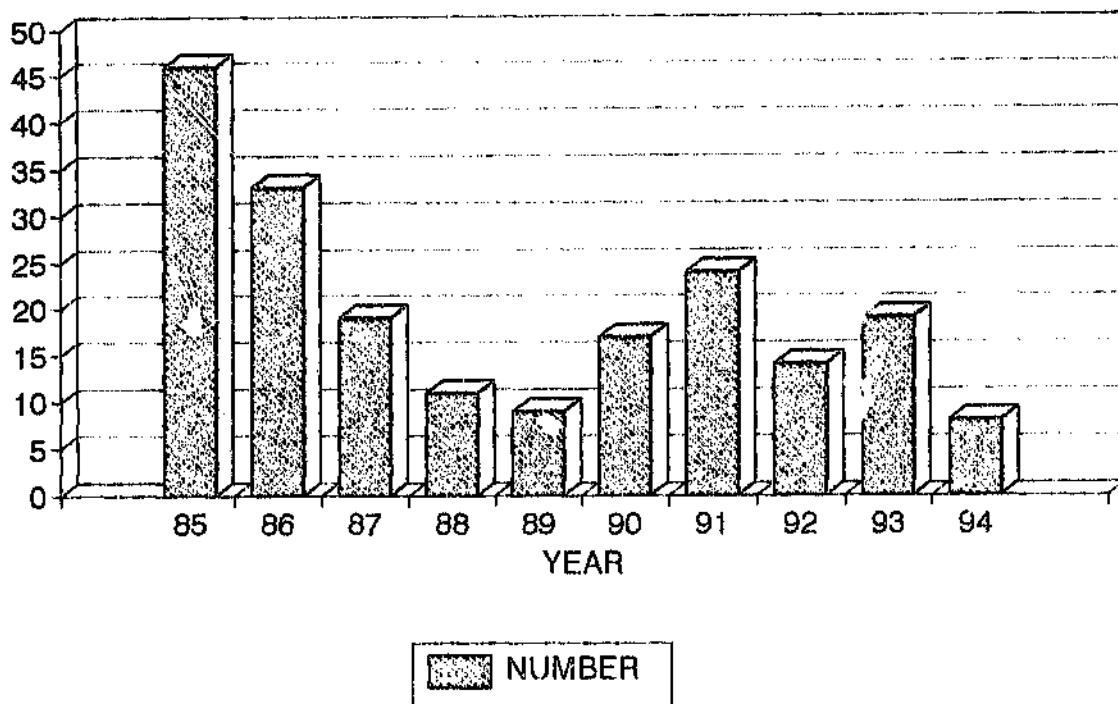
COOKE 2 SHAFT

NUMBER OF REPORTABLES 1985 TO 1994



COOKE 1 SHAFT

NUMBER OF REPORTABLES 1985 TO 1994

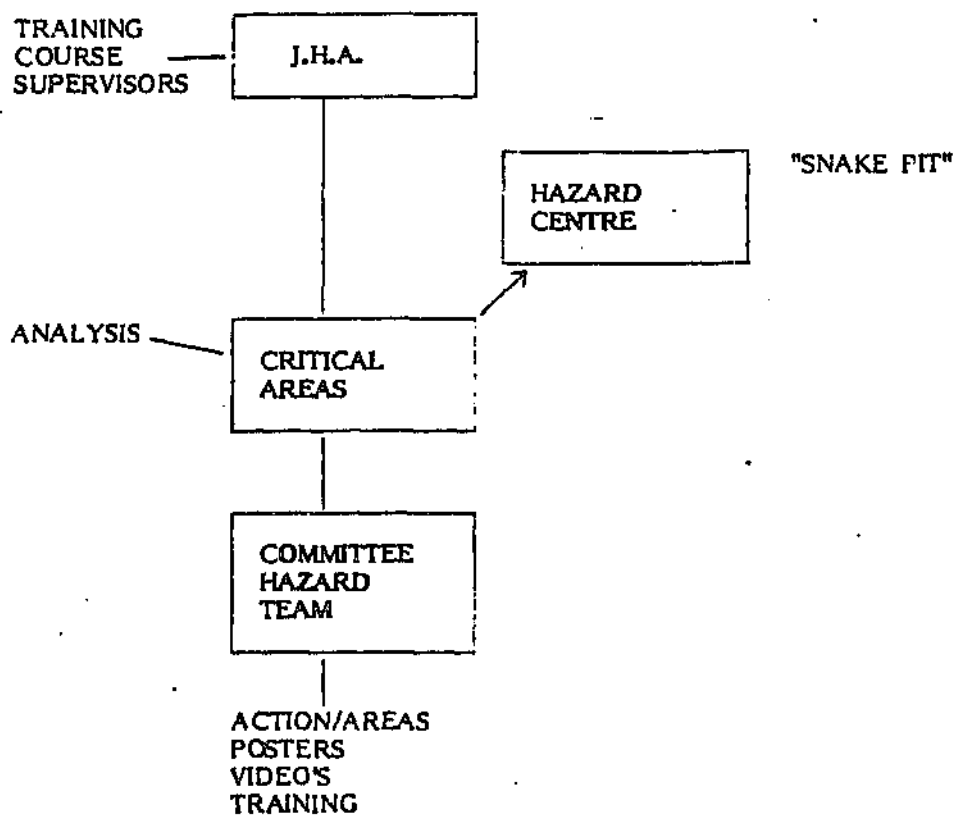


ANALYSIS MADE BY: _____ ANALYSIS APPROVED BY: _____

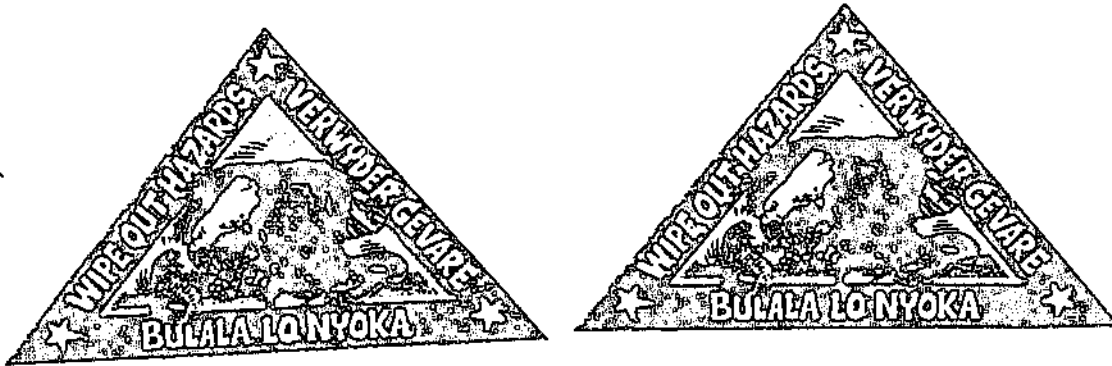
1. STRUCK BY (SB)	5. CAUGHT ON (CO)	9. FALL TO BELOW (FB)
2. STRUCK AGAINST (SA)	6. CAUGHT IN (CI)	10. OVER-EXERTION (OE)
3. CONTACTED BY (CB)	7. CAUGHT BETWEEN (CBT)	11. EXPOSURE (E)
4. CONTACT WITH (CW)	8. FALL - SAME LEVEL (F'S)	

ACTION PLAN

1. IDENTIFICATION OF JOB HAZARD ANALYSIS

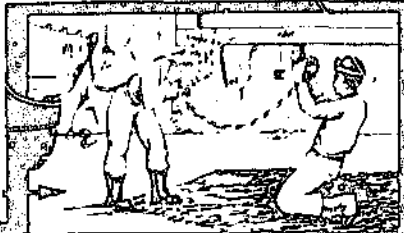


"HAZWISE" STICKERS AND RUBBER STAMPS



RANDFONTEIN STATES GOLD MINING CO. (W/LTD) HAZARD AWARENESS CAMPAIGN

CAN
YOU STRIKE
AGAINST SOME
THING WHILE
WORKING?



CAN YOU BE STRUCK BY
ANYTHING?



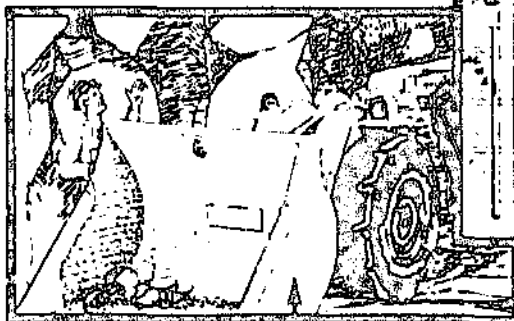
CAN YOU BE INJURED BY
OVEREXERCISE?

BE



CAN YOU FALL TO ANOTHER LEVEL

HAZARD AWARENESS



CAN YOU BE CAUGHT BETWEEN
ANYTHING?



CAN YOU
BE CONTACTED
BY HARMFUL
AGENTS?



CAN YOU BE CAUGHT IN OR
ON ANYTHING?

THE RANDFONTEIN ESTATES GOLD MINING COMPANY
Witwatersrand, Limited

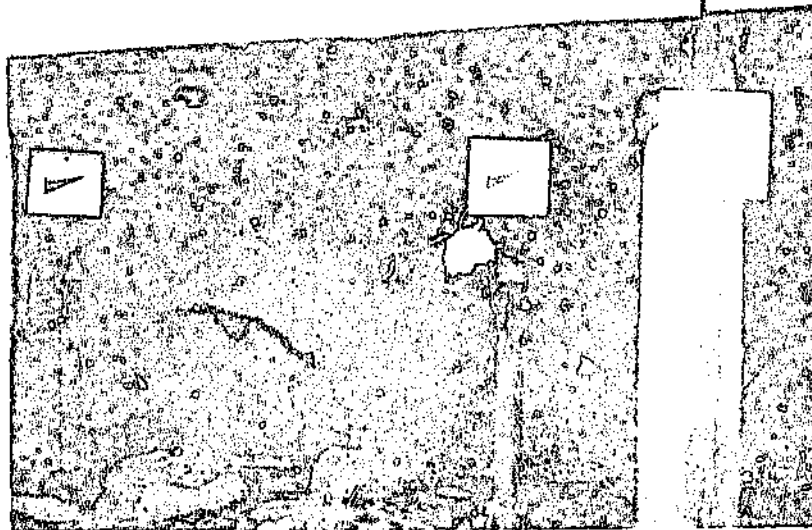
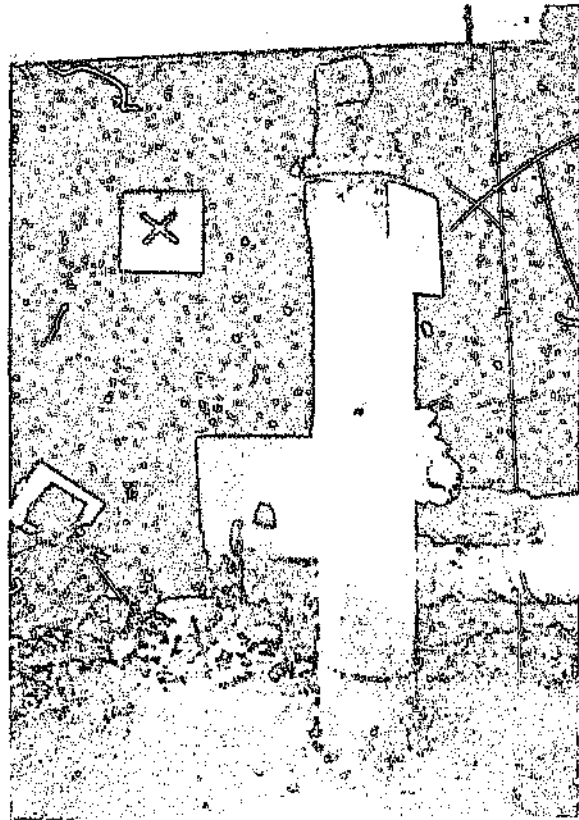
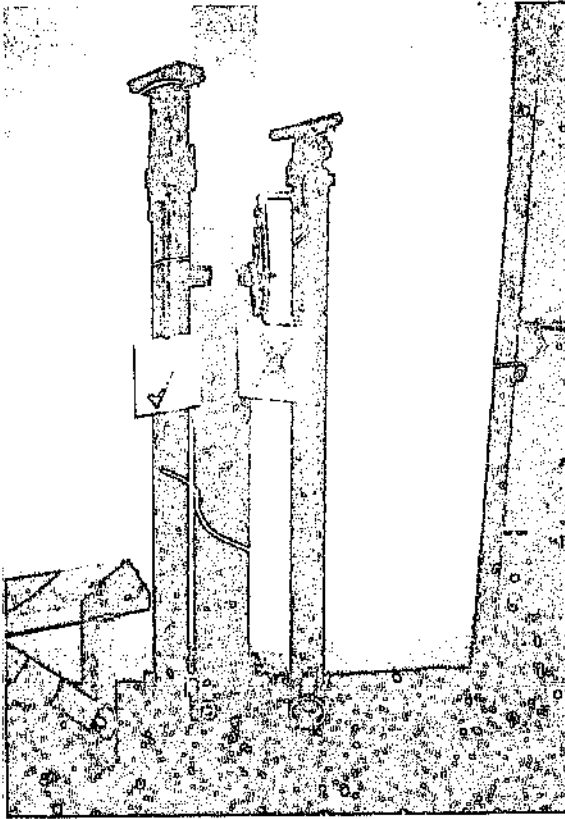


" HONOURS "

HAZWISE

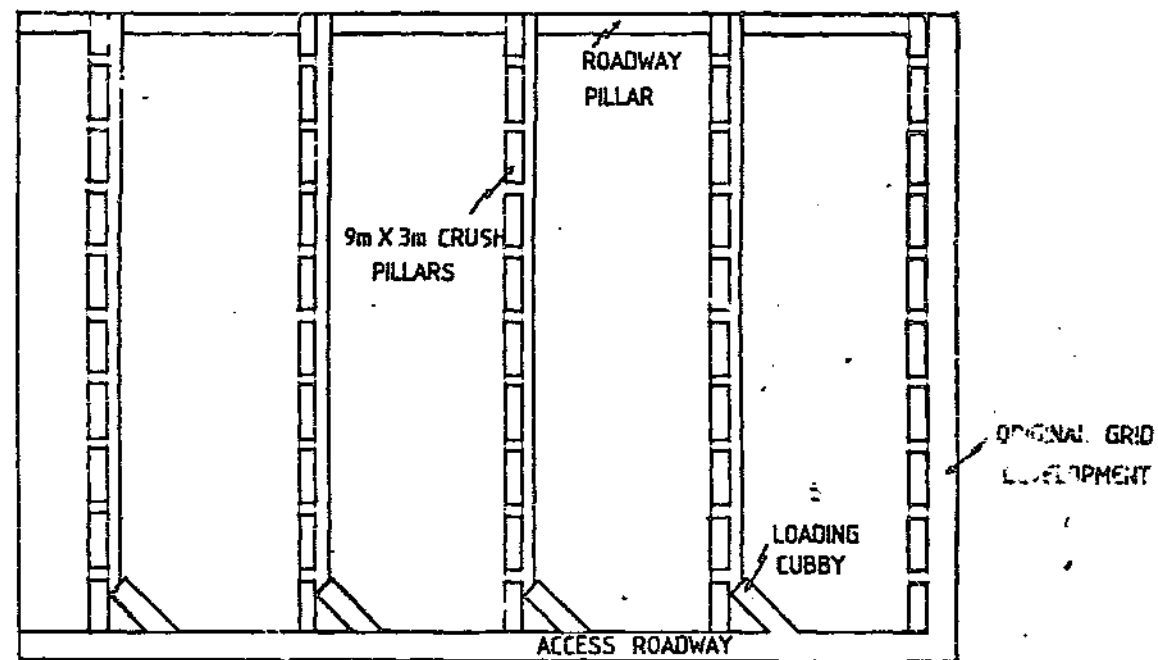


CAMPAIGN



TYPICAL NARROW TM³ LAYOUT

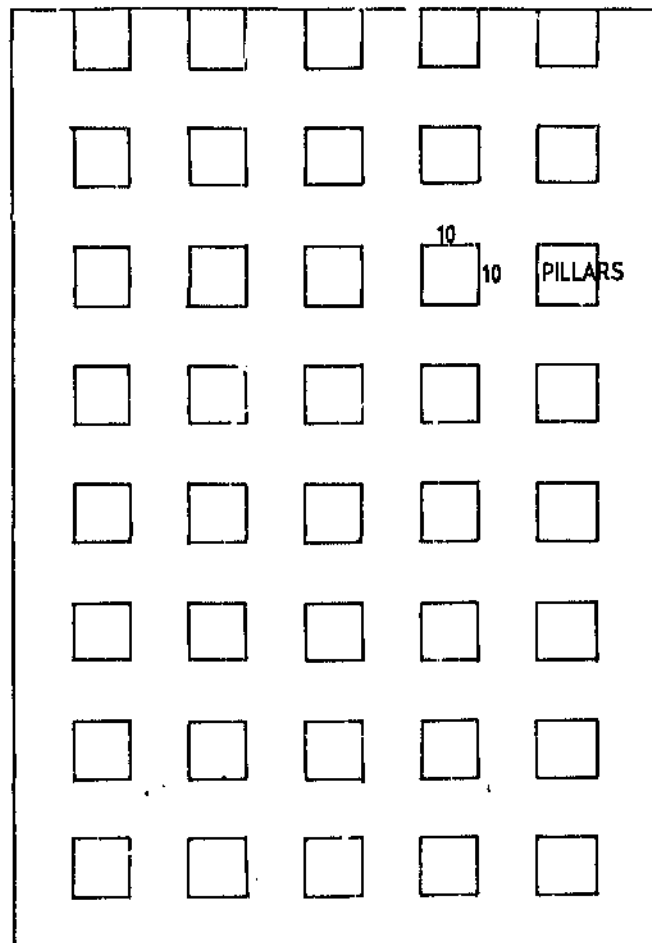
Scale. 1: 1000

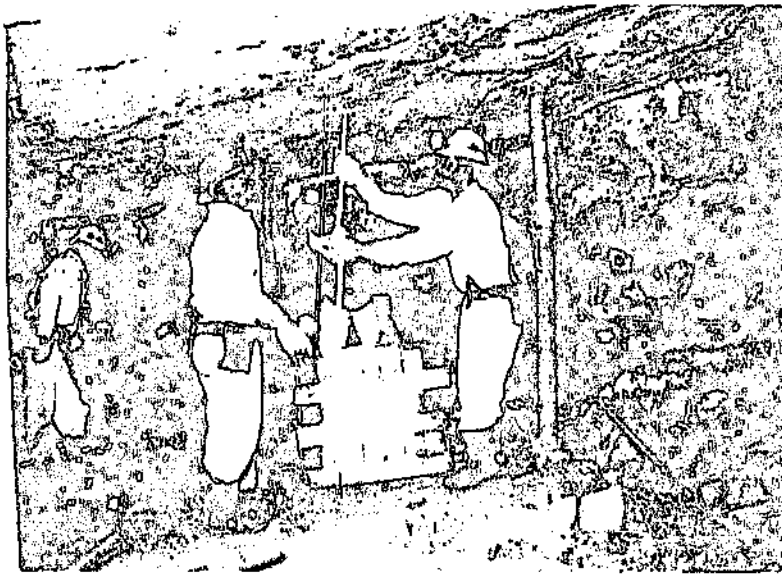


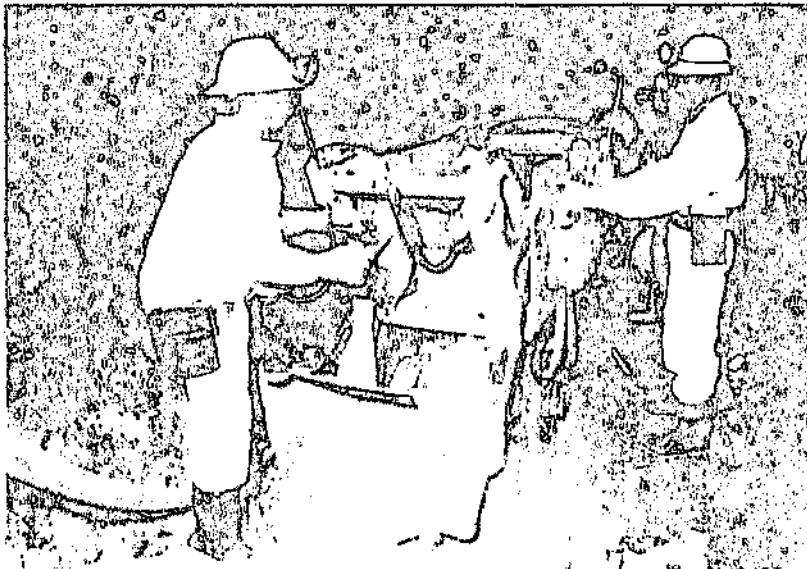
APPENDIX 61

TYPICAL TM³ LAYOUT

Scale 1:1000

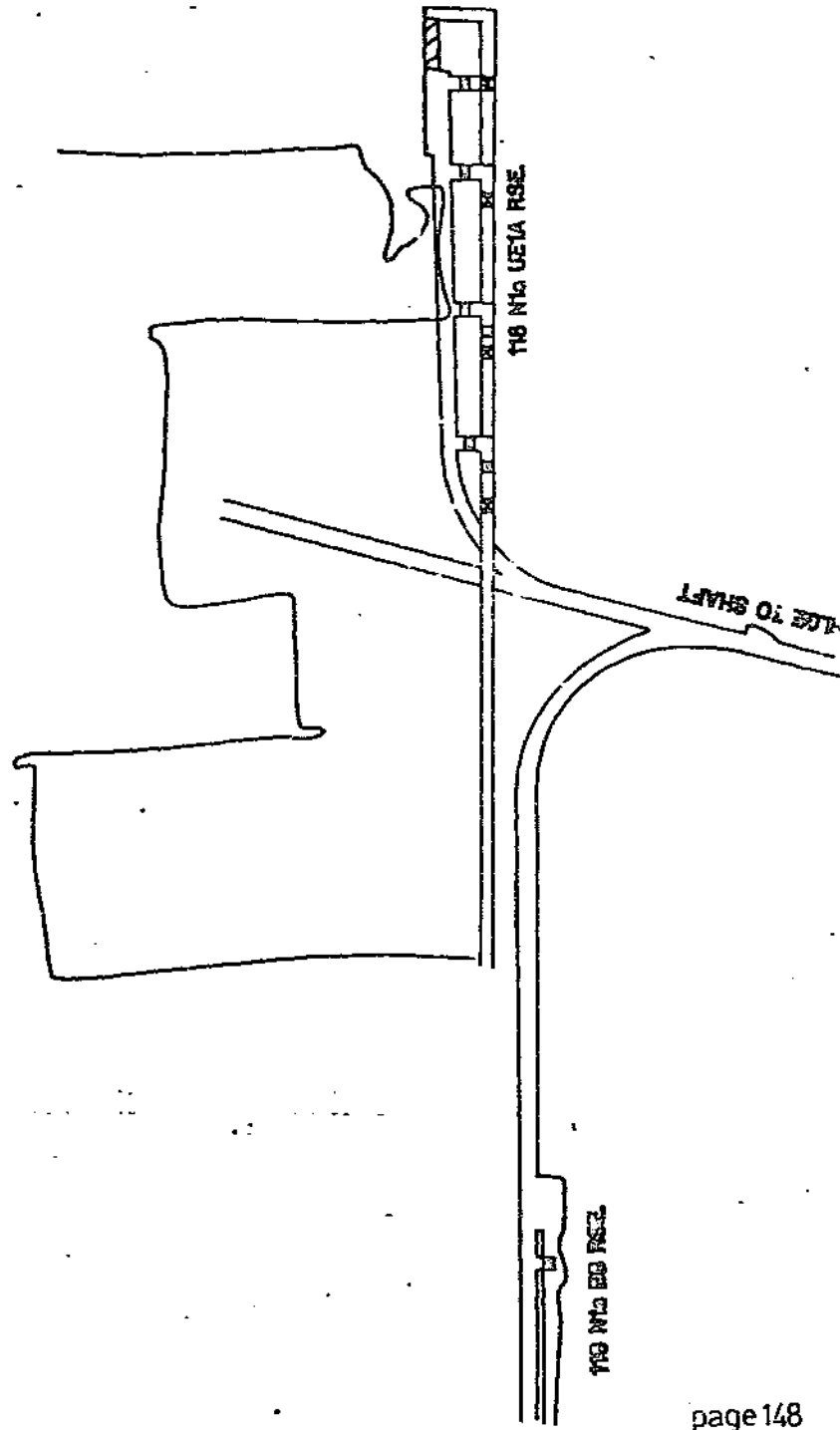






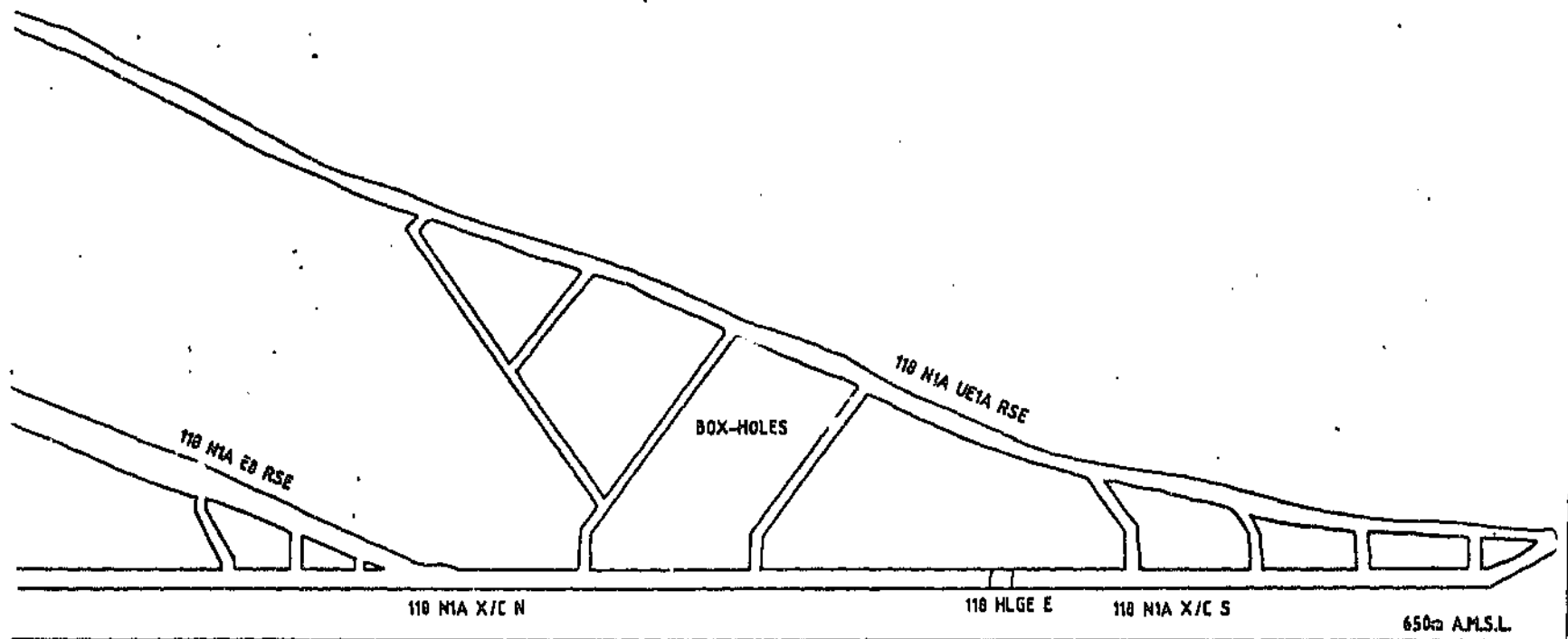


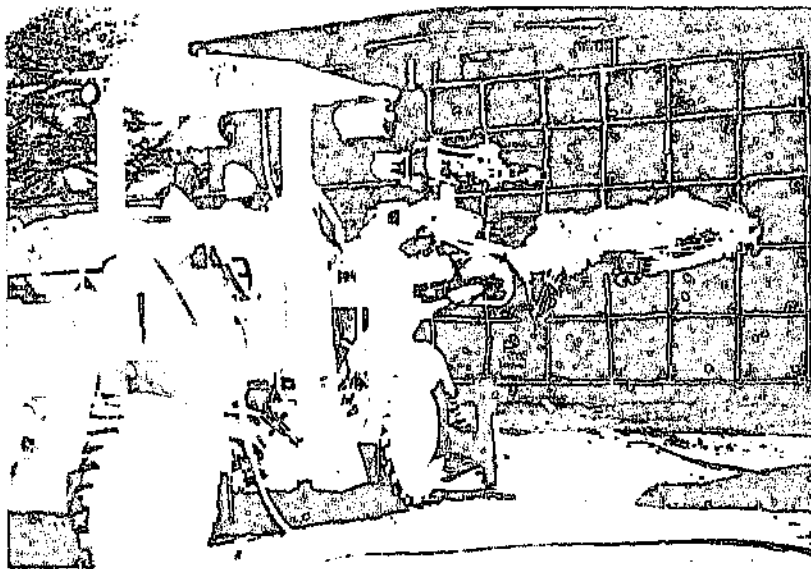
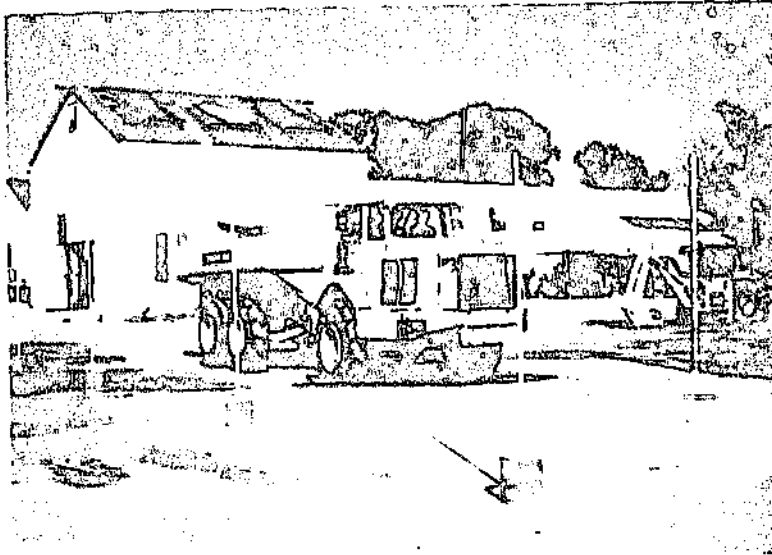
PLAN SHOWING TRAINING STOPES
SCALE 1:1000

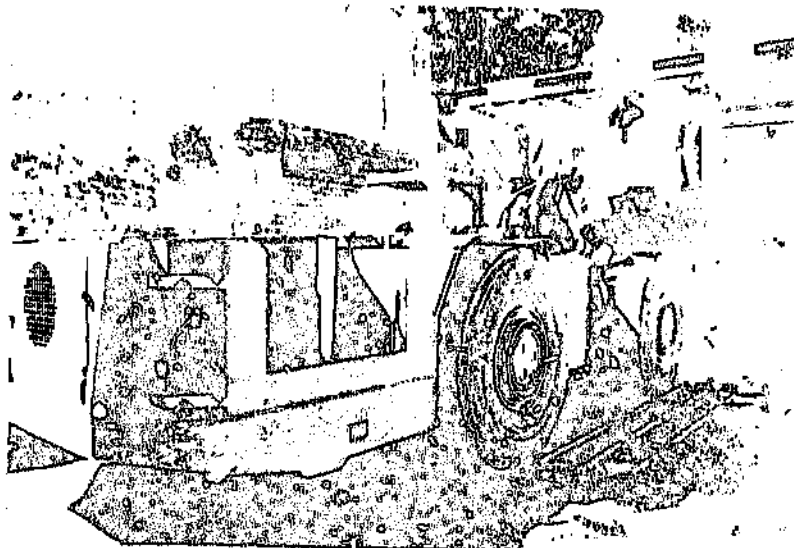
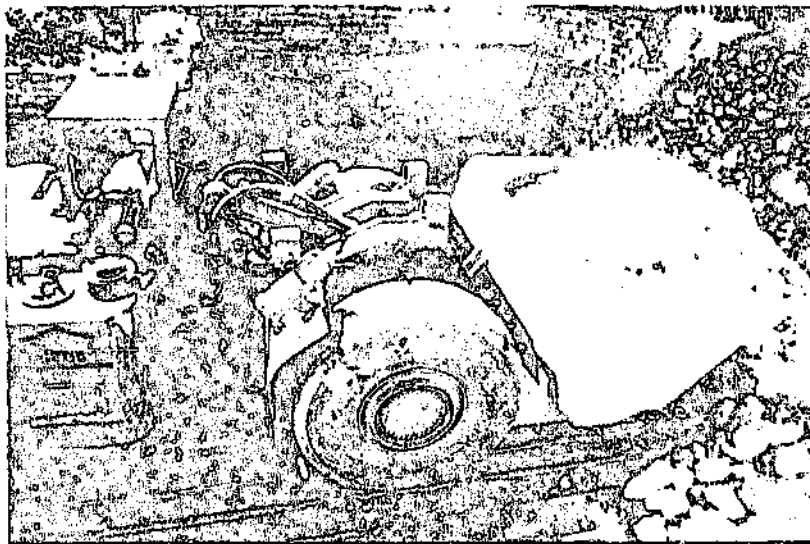
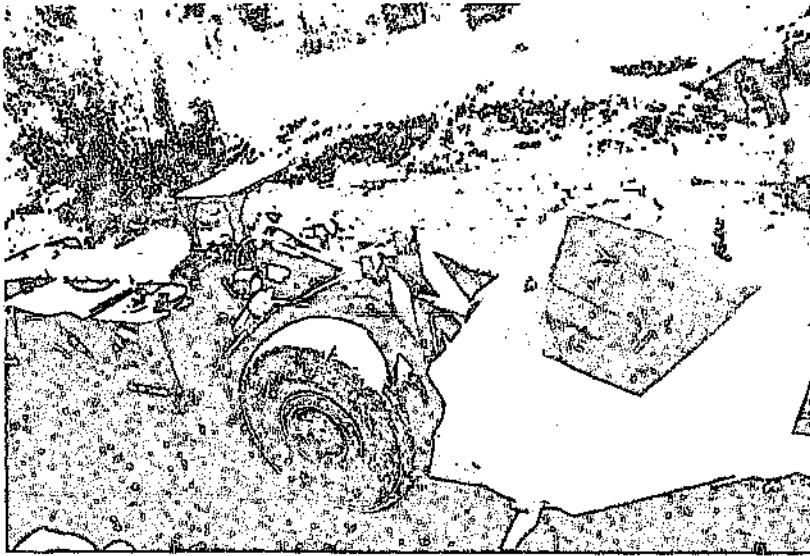


SECTION SHOWING TRAINING STOPES

SCALE 1:1000







THE RANDFONTEIN ESTATES GOLD MINING COMPANY,
WITWATERSRAND, LIMITED

MANAGEMENT BRIEF TO ALL EMPLOYEES

SAFETY FIRST; GO FOR GOLD

**OBJECTIVE: TO INFORM ALL EMPLOYEES OF A RATIONALISATION
PROGRAMME OF MANPOWER AND OCCUPATIONS AFFECTED.**

The Company wishes to advise you that it has downscaled mainly on the Trackless Mining operations and has increased conventional operations. The result is that the Company has a significant number of employees over complement.

The Company therefore has no option, but to regretfully embark on a rationalisation programme. Those affected, employees will be offered an opportunity to opt for a lower level job, voluntary retrenchment or early retirement (55 years of age and older).

We understand the difficulties and concern which the news brings us. We wish to advise that the preliminary estimates indicate that approximately 600 employees are involved. The details will be discussed with the relevant unions and your representative employee committees.

The employee organisations are being advised of this situation and will be invited to participate in further consultations which we aim to complete as soon as possible. Individuals affected will be advised accordingly.

Those employees who decide to be retrenched voluntarily or take early retirement will be paid a severance package.

In the case of voluntarily retrenchment or early retirement, employees will be required to make application to their relevant PO (Hostels) on or before February 25, 1994.

The Mine will render as much assistance to affected employees as is possible under the circumstances.

Yours faithfully in Safety and Peace

GENERAL MANAGER

THE RANDFONTEIN ESTATES GOLD MINING COMPANY,
WITWATERSRAND, LIMITED

JCI ANNOUNCES RATIONALISATION AT RANDFONTEIN ESTATES

The directors of Randfontein Estates Gold Mining Company Witwatersrand have come to the conclusion that labour rationalisation is inevitable at Randfontein.

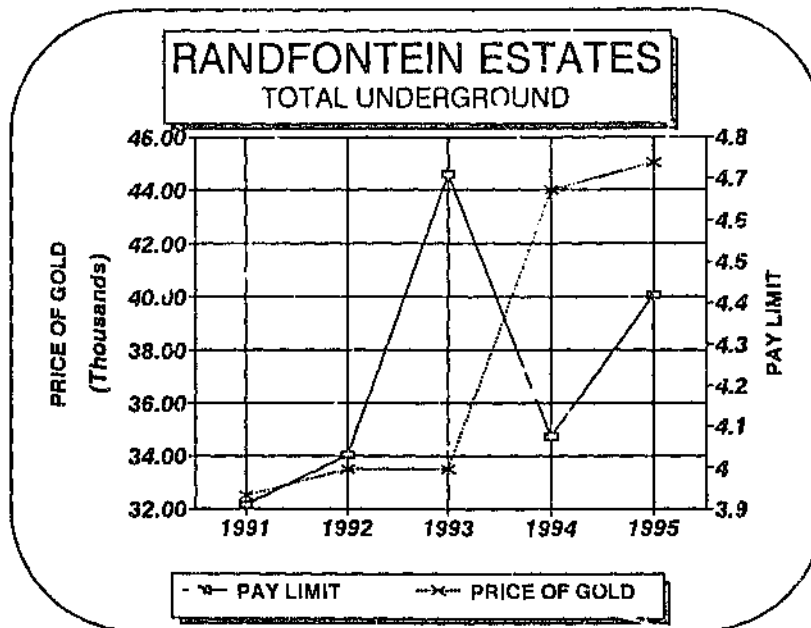
This is mainly due to the downscaling of Trackless Mining Operations and the increase of conventional mining in order to produce gold more cost effectively.

This very serious matter has been under consideration for a long period during which various other options were considered of which some were implemented. Those implemented options, however, did not have the required effect and accordingly leaves us no other alternative but to embark on a rationalisation exercise of approximately 700 employees and to enter into the required consultations with their respective union representatives.

The Chairman of JCI's Gold Division - Mr Kennedy Maxwell has stated that although the need for this course of action is regrettable, the current decline in cost effectiveness at Randfontein, the unstable gold price and the ongoing viability of its operations needs to be addresses, even if it calls for a serious intervention such as a downscaling of its manpower.

RANLFONTEIN ESTATES GOLD MINING COMPANY PAY LIMITS FOR TOTAL UNDERGROUND

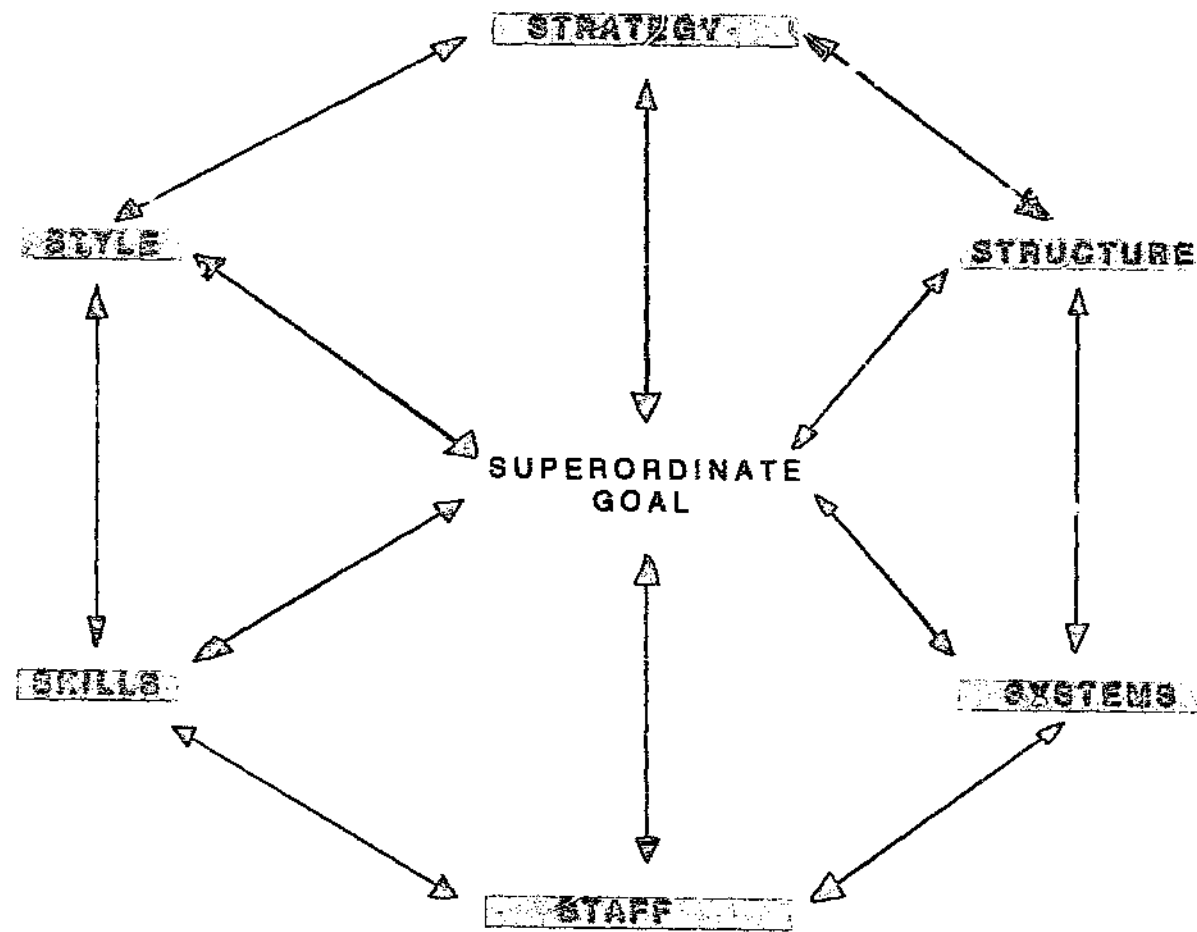
YEAR	PAY LIMIT	GOLD PRICE
1991	3.91	32500
1992	4.03	33500
1993	4.71	33500
1994	4.08	44000
1995	4.42	45000



REFERENCES

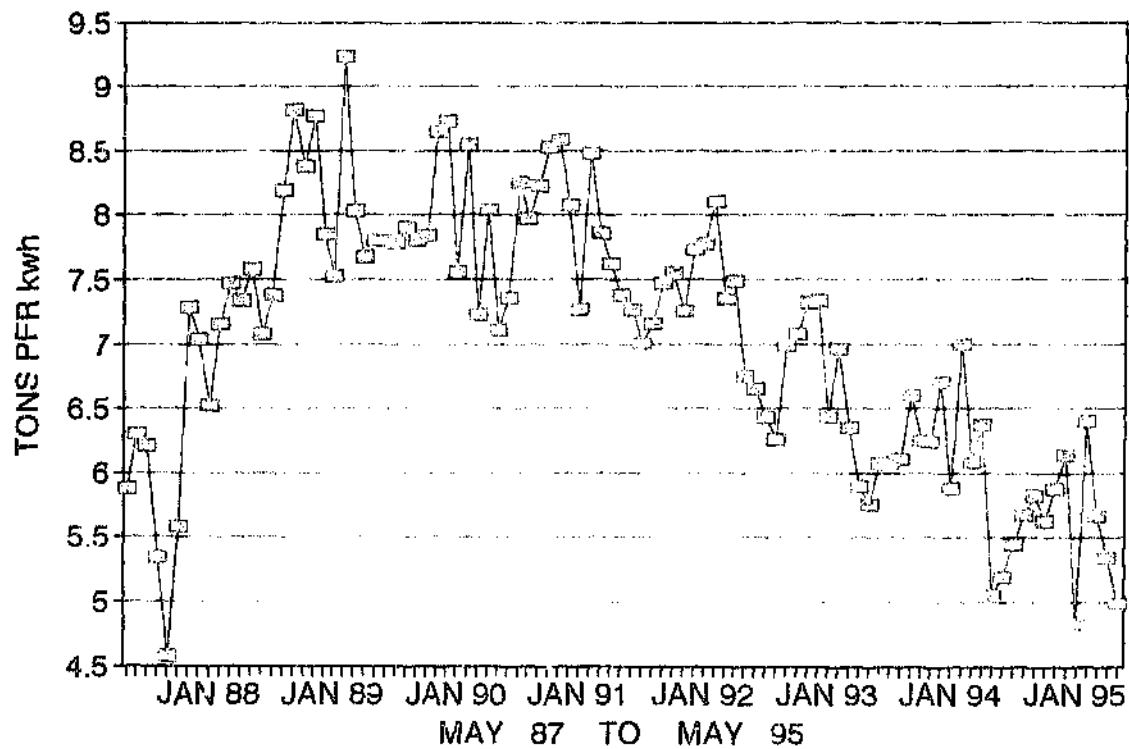
- 1) Randfontein Estates Gold Mine Geological reference book.
- 2) Hocking, A. (1986) Randfontein Estates. The First Hundred Years. Published by Holland, South Africa.
- 3) Association of Mine Managers of South Africa. Trackless Mining Symposium (1988). Published by The Chamber of Mines of South Africa.
- 4) Statistical summary of accidents reported to the GME for the period 1984-1994. Published by The Chamber of Mines of South Africa.
- 5) M.A.Smith and J.A.Cox (1988). The trackless mine of the 90's. Published by The Chamber of Mines of South Africa.
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- 9) J.C.Ball (June 1993). STOMEC H25-2 Electro hydraulic stope drill rig close-out report. Report for JCI, South Africa.
- 10) M.H.Rogers and A.L.E.Markus (1988). Surveying and evaluation procedures for trackless mechanised mining methods. Published by The Chamber of Mines of South Africa.
- 11) Chain Store Age Executive Edition (November 1991). The Downsize Dilemma, USA.
- 12) K.S.Cameron, S.J.Freeman, and A.K.Mishra (August 1991). Academy of Management Executive. Best practices in white-collar downsizing, Management contradictions, USA.
- 13) Braxton Associates, strategy consulting division of Deloitte and Touche. Survey of industry executives, A report done by Deloitte and Touche, South Africa.

INTERNAL DRIVERS FOR CHANGE



RANDFONTEIN ESTATES GOLD MINE

kwh POWER USED



APPENDIX 70
RANDFONTEIN ESTATES GOLD MINE
TIME UTILIZATION

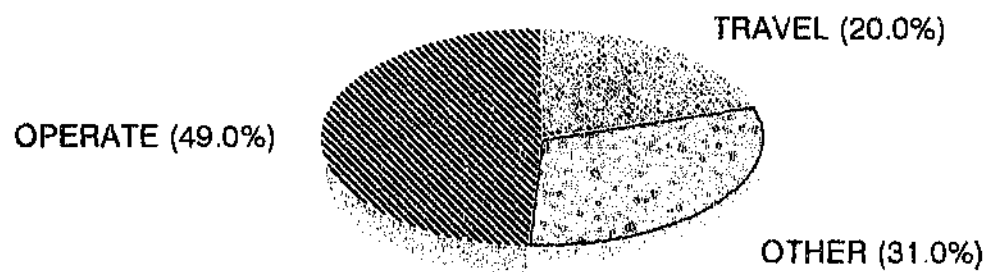
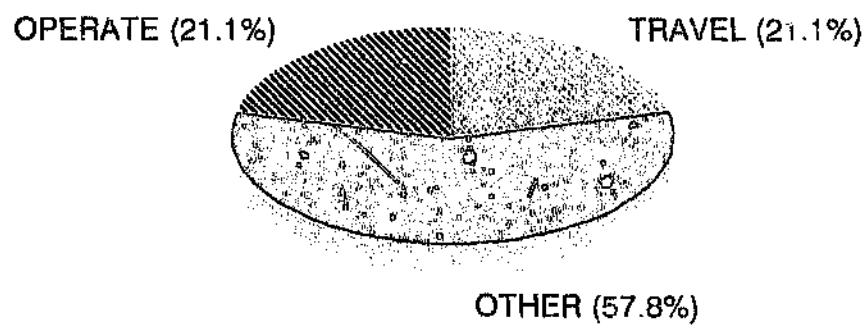


FIGURE 10-10: TIME UTILIZATION

APPENDIX 69
RANDFONTEIN ESTATES GOLD MINE
TIME UTILIZATION

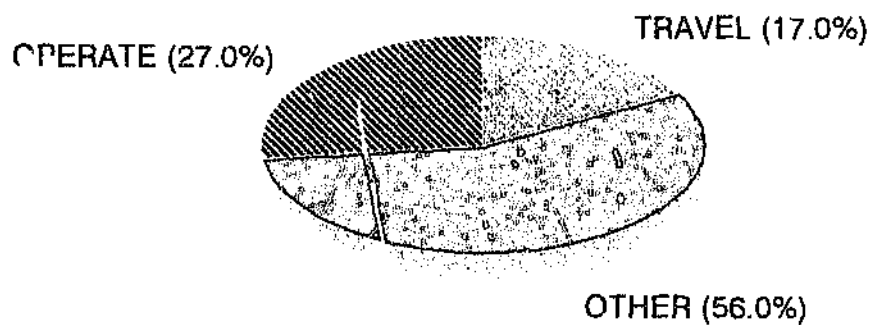


RANDFONTEIN ESTATES GOLD MINE

APPENDIX 68

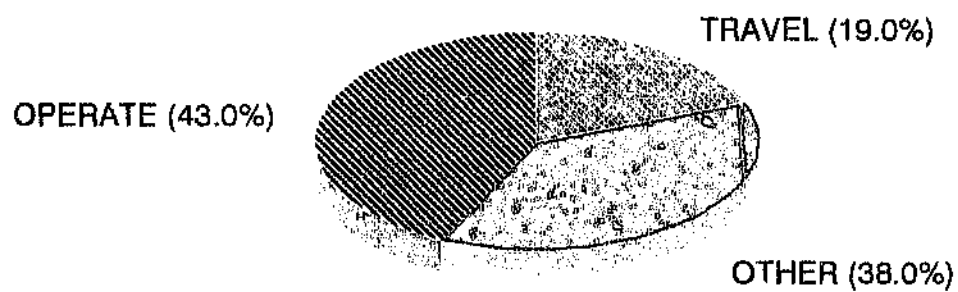
RANDFONTEIN ESTATES GOLD MINE

TIME UTILIZATION



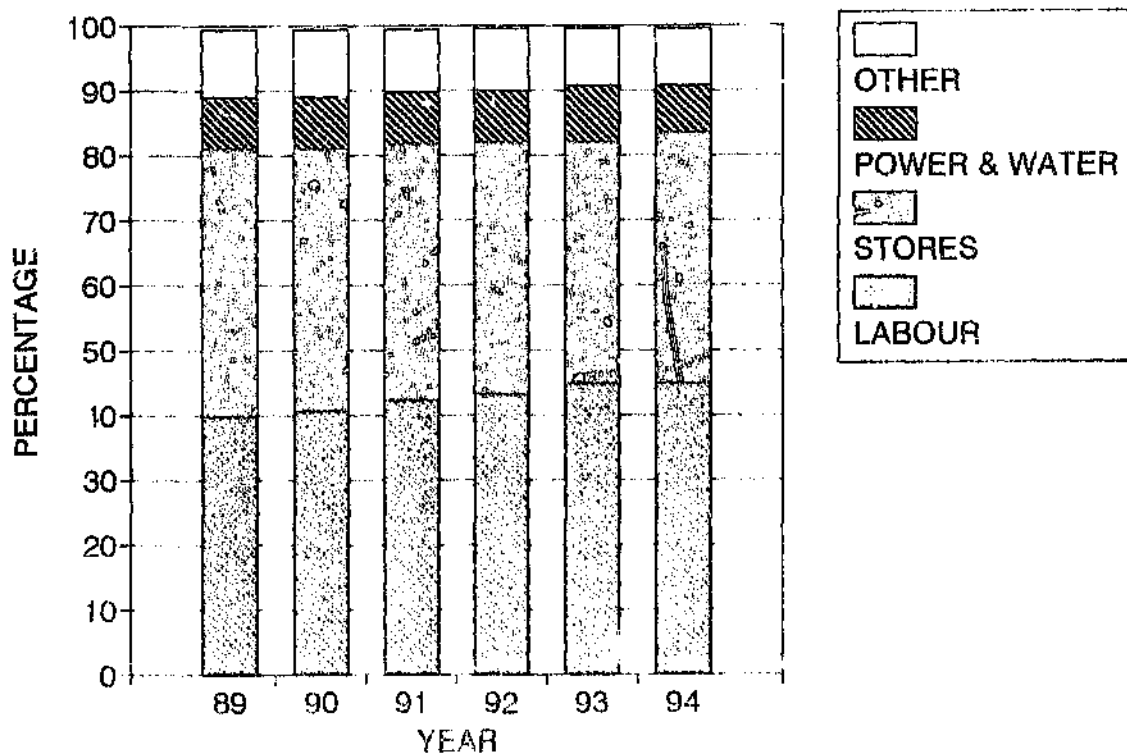
APPENDIX 68

APPENDIX 67
RANDFONTEIN ESTATES GOLD MINE
TIME UTILIZATION

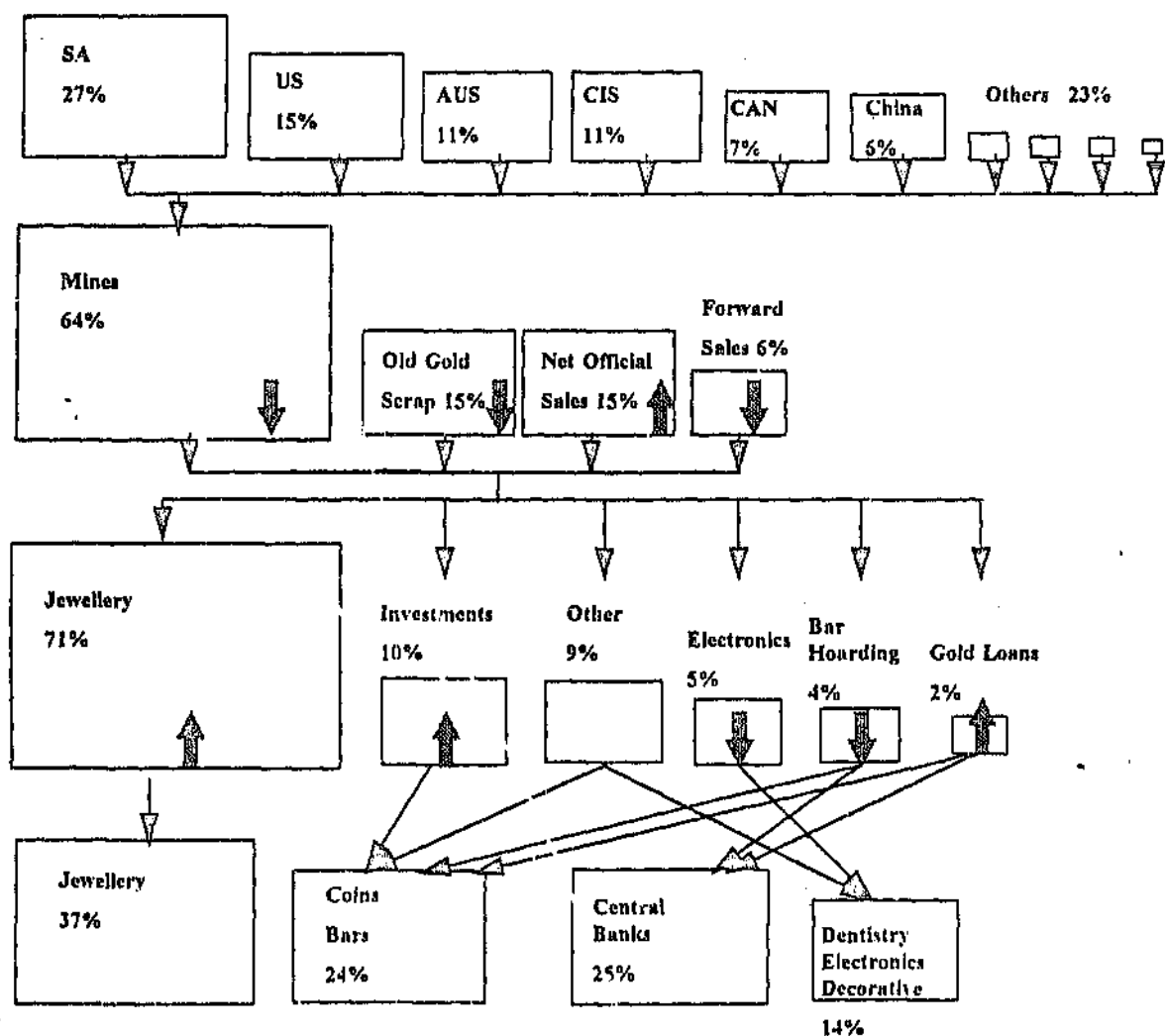


DAY SHIFT STOPPING

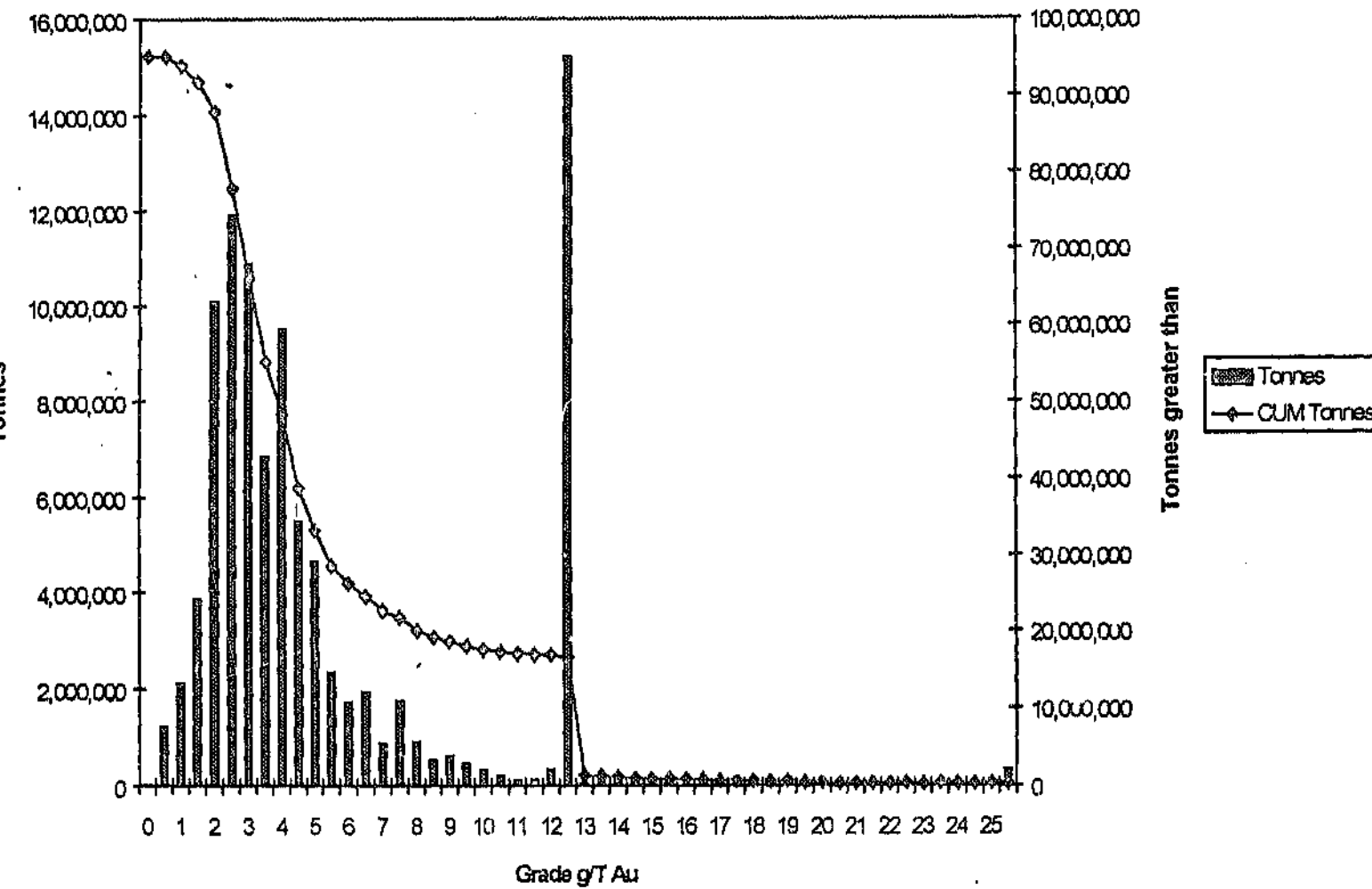
REGIONAL INVESTMENT IN HEALTH SERVICES



External Drivers For Change : Market Analysis

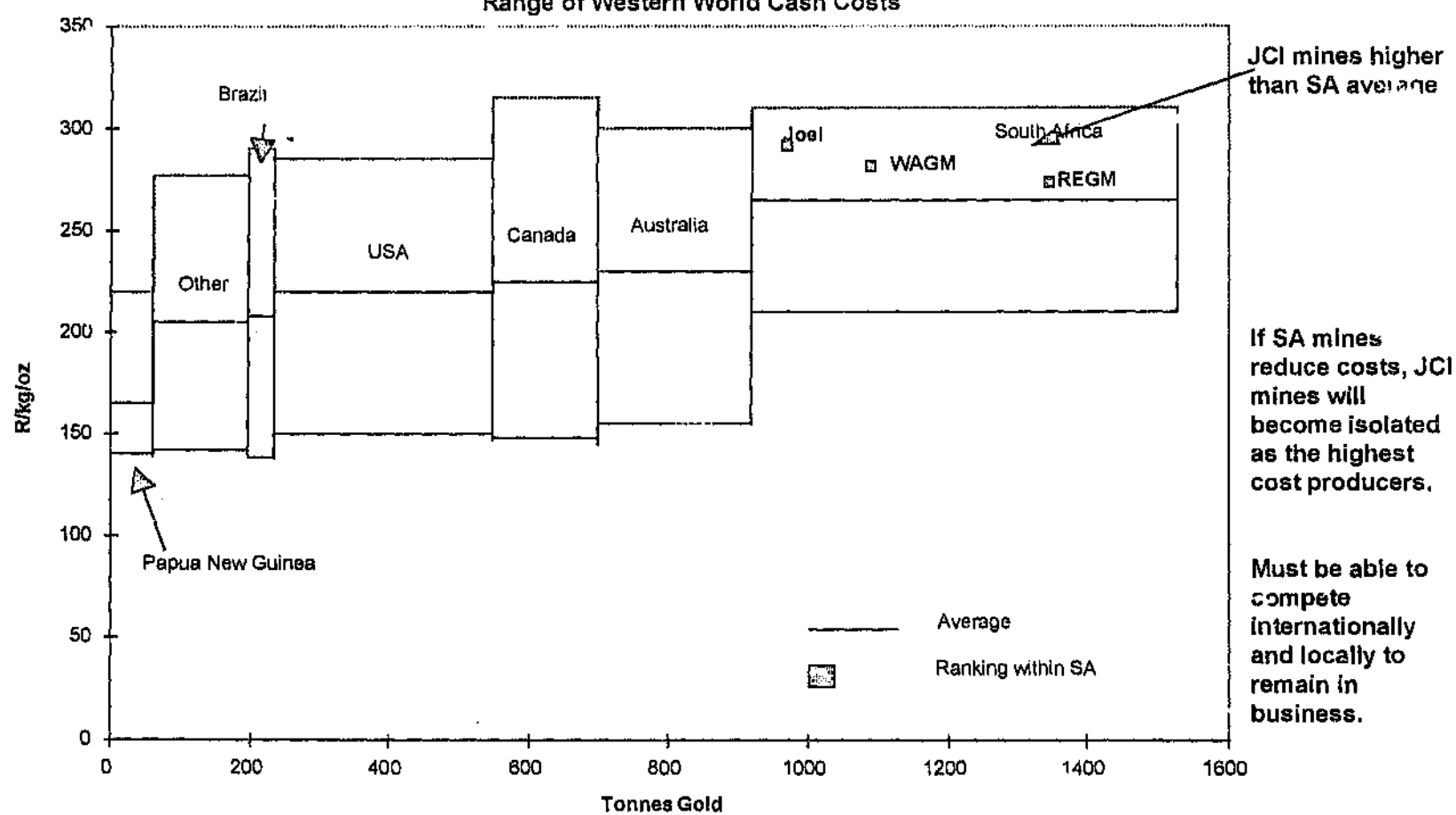


REGM Ore Distribution

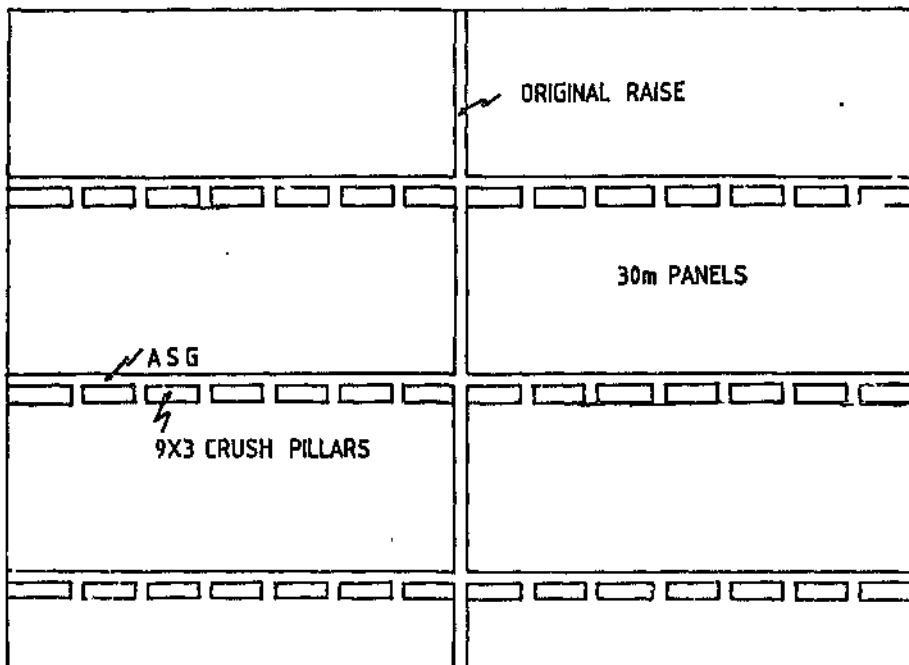


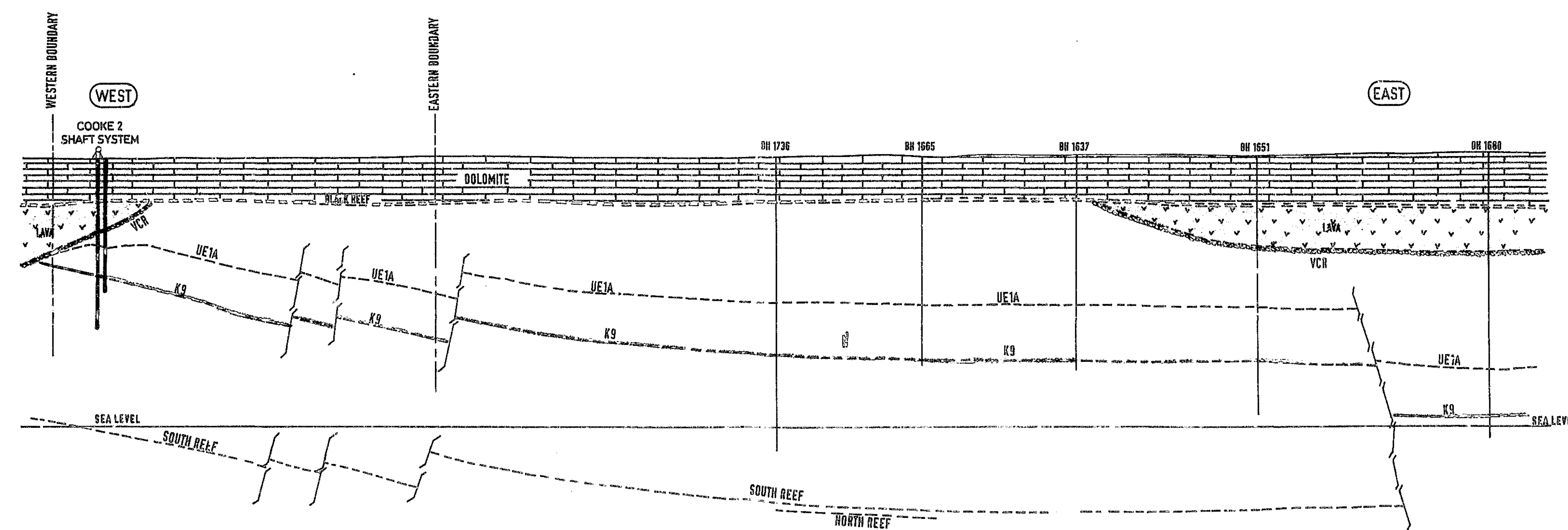
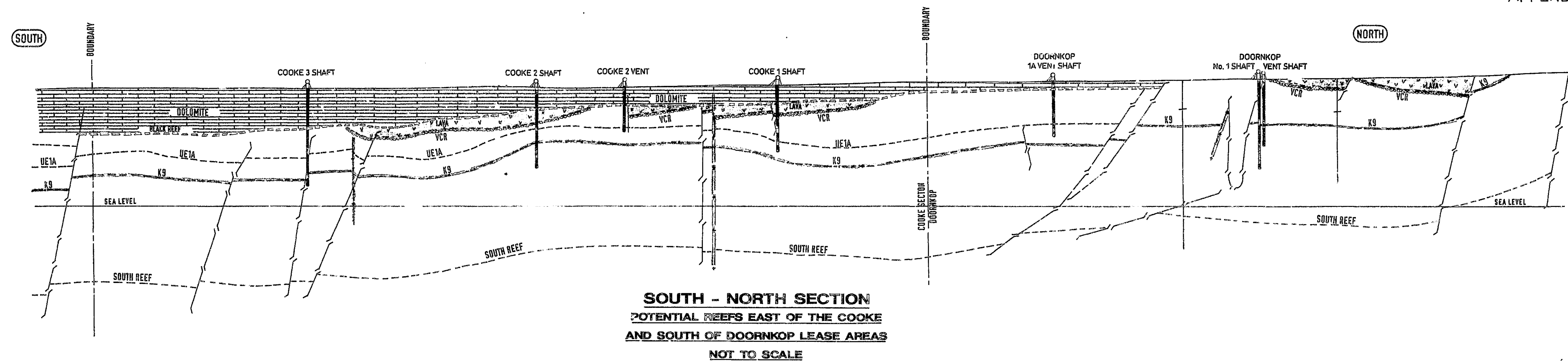
Re-engineering Our Mines

SA Costs highest and JCI mines amongst more expensive SA Mines
Range of Western World Cash Costs



c:\phase1\Blaer\Costs\Load\Bhow.ppt

TYPICAL CONVENTIONAL LAYOUTScale. 1: 1000



- 14) Internal publication of Deloitte and Touche. Strategic reengineering, South Africa.
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- 17) H.Scott-Russell (August, 1987). The introduction of mechanised cleaning methods into the South African Gold Mining Industry, submitted to the Faculty of Engineering, Department of Mining, University of the Witwatersrand, South Africa.

Author: Brown, Andrew.

Name of thesis: An analysis of the problems encountered when converting trackless mining back to a more conventional type of mining on Randfontein estates gold mine.

PUBLISHER:

University of the Witwatersrand, Johannesburg

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