ENVIRONMENTAL PLANNING OF THE BUFFELSFONTEIN TERTIARY SHAFT

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A project report submitted to the Faculty of Engineering, University of the Witwatersrand, Johannesburg, in partial fulfilment of the requirements for the degree of Master of Science in Engineering

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DECLARATION

I declare that this project report is my own, unaided work. It is being submitted for the Degree of Masters 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.

M. Dunda

(Signature of candidate)

FIRST day of JUNE 1990

ABSTRACT

A comprehensive study was undertaken to minimize the Technical Environmental risks in extending the current Buffelsfontein infrastructure, an additional 1 000 metres in depth, to a mean rockbreaking depth of 3 740 m below surface. A systematic approach was adopted in determining the environmental requirements, methods of ventilation and implications of physical limitations.

Maximum re-use of existing ventilation infrastructure and refrigeration machinery was pre-requisite. The determination of air reticulation capacities and chilling equipment operational limitations paramount.

The definitive estimate, within 10%, put quantifyable limits to the latitude of estimation and provided guidlines to the level and degree of environmental engineering required.

The extreme mining depths require extensive placement of backfill. Although advantageous for the reduction of heatflow, close fill to face distances result in high system pressure loss and a constraint in the mass flow of air. With nearly 40 MW of cooling to be transmitted at or near the mining horizon, analysis and prescription of a ventilation and cooling strategy was imperative.

Computer models were extensively used to predict and calculate sources of the mine heat and describe ventilation network changes with the deletion and addition of mining districts Energy balances were undertaken to ensure suitable psychrometric conditions prevailed and that thermal environments complied with a minimum Specific Cooling Power of 300 watts per square metre.

Despite the conservative methods of cooling and ventilation that were adopted in preference to more cost effective principles, such as the use of ice, hydrolift techniques and controlled reuse of air, there are no reservations that the exploitation of the Tertiary orebody can be adequately cooled and ventilated.

ACKNOWLEDGEMENTS

Assistance received in carrying out portions of the work in preparing this project report from the Buffelsfontein Tertiary Task Force is acknowledged.

To my wife Juliet and
Family for the support
in 1989

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Acceleration due to Gravity	g	m/s^2
Air Velocity	v	m/s
Atkinsons Friction Factor	k	Ns^2/m^4
Circumference	c	m
Cross Sectional Area	A	m^2
Density	w	kg/m^3
Diameter	D	m
Difference	Δ	
Dry-Bulb Temperature	db	°C
Effective Temperature	ET	°C
Enthalpy	H	J/kg
Face Advance	ℓa	m/month
Heat	q	J/kg
Heat Addition Rate	q'	kW
Length	L	m

Mass	m	kg
Mass Flow	M	kg/s
Moisture Content	r	kg/kg
Pressure Loss (or Gain)	p	Pa
Radius	r	m
Specific Cooling Power	SCP	W/m^2
Stope Span	ℓs	m
Stope Width	ℓw	m
Temperature	t	°C
Temperature Gradient	Δt	°C
Thermal Capacity	Cp	J/kgK
Thermal Conductivity	h	W/m°C
Velocity	v	m/s
Virgin Rock Temperature	VRT	°C
Volumetric Flow	Q	m^3/s
Wet-Bulb Temperature	wb	°C

1. INTRODUCTION

1.1 Buffelsfontein Ore Reserve Depletion

Depleting ore reserves within the existing Buffelsfontein mining districts have necessitated replacement ground to be located, proven and evaluated as economically and technically viable. Such an orebody exists at a mean rockbreaking depth of 3 740 m below surface at a mean Virgin Rock Temperature of 63°C.

Being in close proximity to the existing Strathmore Shaft, vertically displaced by 1 000 metres, access to the Tertiary orebody appears to be a natural extension of the existing infrastructure. The investigations and planning procedures adopted address in sequence major environmental concerns identifying constraints and limitations suggesting modifications, additions or alternatives.

Due to unprecedented depths and high rock temperatures, the classic environmental planning guidelines had to be extrapolated, augmented where necessary and supplemented by the most currently available prediction methods.

1.2 Background

1.2.1 Current mining area

The present mining area of Buffelsfontein Gold Mining Company Limited covers some 6 750 ha being 8 km on strike (east/west direction) and 6 km on dip. Figure 1.1 shows the Geographic location of Buffelsfontein in the Western Transvaal. The sub outcrop is approximately 1 000 metres below surface and the lowest working level is 3 100 metres below surface, beyond which the Vaal Reef is displaced some 1 200 metres down by the Jersey fault.

The Vaal Reef is heavily faulted in a southwesterly/northeasterly direction which complicates mining operations and design. The average reef dip is 27° but increases to the east, where bedding plane faulting causes inconsistency in the plane of the reef.

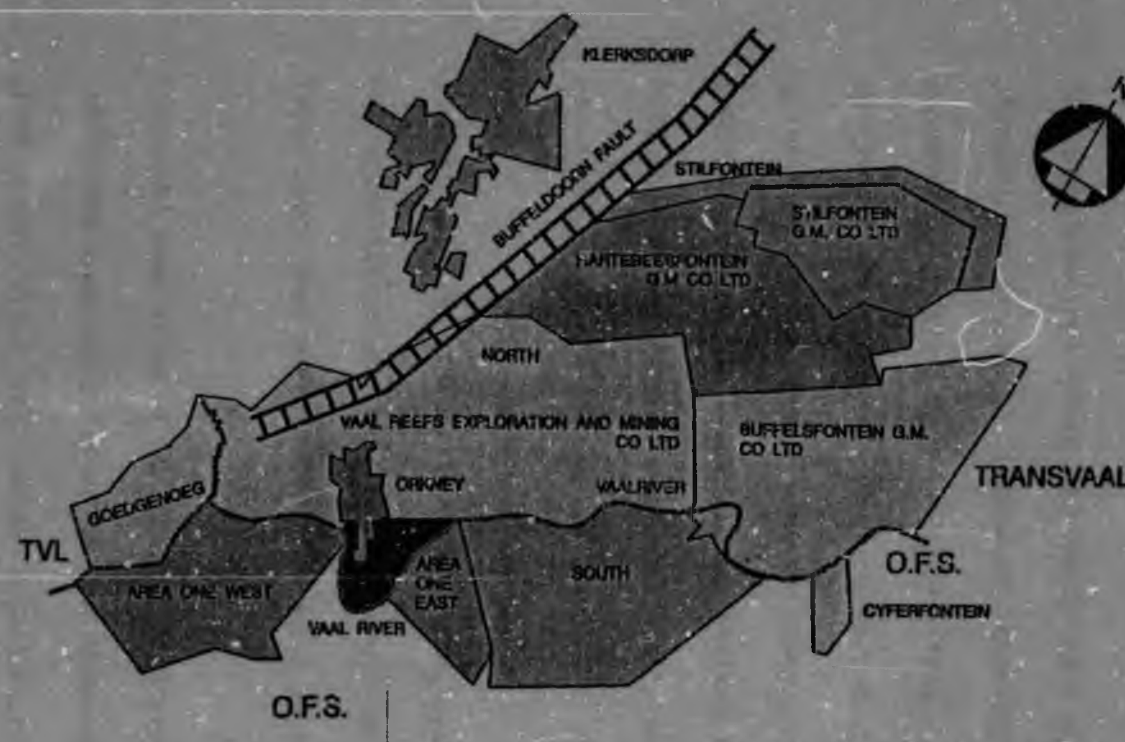


FIGURE 1.1

GOLD MINES OF THE KLERKSDORP AREA

1.2.2 Exploration

The existence of deep blocks of Vaal Reef in the area east of Eastern Shaft and northeast of Southern Shaft has been realised since the early 1970's, when the planning and design of Strathmore Secondary Shaft was commenced. This shaft was, in fact, originally designed to be part of a Tertiary system, intended to go as deep as 44 Level. The Tertiary part of this proposal (from 37 to 44 Level) was shelved in December 1979 when it was demonstrated that the Secondary Shaft was a viable proposition on its own.

It was indicated at that time that a re-evaluation of the viability of a Tertiary Shaft would be made at a later date, when more information would be available.

Since the completion of Strathmore Secondary Shaft (in 1984), the expected build-up of ore reserves for this shaft have fallen well below expectations, due to unexpectedly severe faulting in the Strathmore East area. It then became a matter of urgency, due to the consequent anticipated rapid depletion of the mine's ore reserves, to investigate the feasibility of a Tertiary Shaft System in the Strathmore area.

Early in 1985 a proposal was put forward detailing an exploration programme for the Tertiary Shaft area and the first borehole was commenced in July 1985.

The objective of the exploration programme was to obtain a minimum of 5 reef intersections on a grid spacing of approximately 750 m, within a radius of 3 000 m from the proposed Tertiary Shaft site. To achieve this, it was anticipated that, due to the expected highly complex structures, a minimum of 10 boreholes would be required. To date, 8 boreholes have been completed and another is in progress. Of the completed holes, 7 have intersected the Vaal Reef. This is an unexpectedly high success rate, considering the structural complexity of the area.

1.3 Technical Considerations

Confidence is high that with the co-operative support of all interactive disciplines, mining, rock mechanics and environmental, exploitation of the one reserves is possible.

Rock mechanics, employing new generation prediction techniques and current schools of thought, endorse that extraction at these depths is not prohibitive.

Environmentally, however, possible constraints of the present infrastructure are a concern particularly when adding the Tertiary block onto existing air reticulation capabilities.

1.4 Existing Mine

The Buffelsfontein Gold Mining complex is comprised of four semi-discrete shaft systems. Geographically located roughly 3 km apart, the shaft infrastructure covers a plan area shown in Figure 1.2 of approximately 60 km^2. The shafts are interactive with respect to ventilating air, chilled water reticulation and electrical power supply. Figure 1.3 is a diagrammatic section of the mine showing major interconnections on 13 Level, 23 Level, 26/28 Level and 31/34 Level.

FIGURE 1.2

RELATIVE PLAN POSITION OF EXISTING BUFFELSFONTEIN SHAFTS

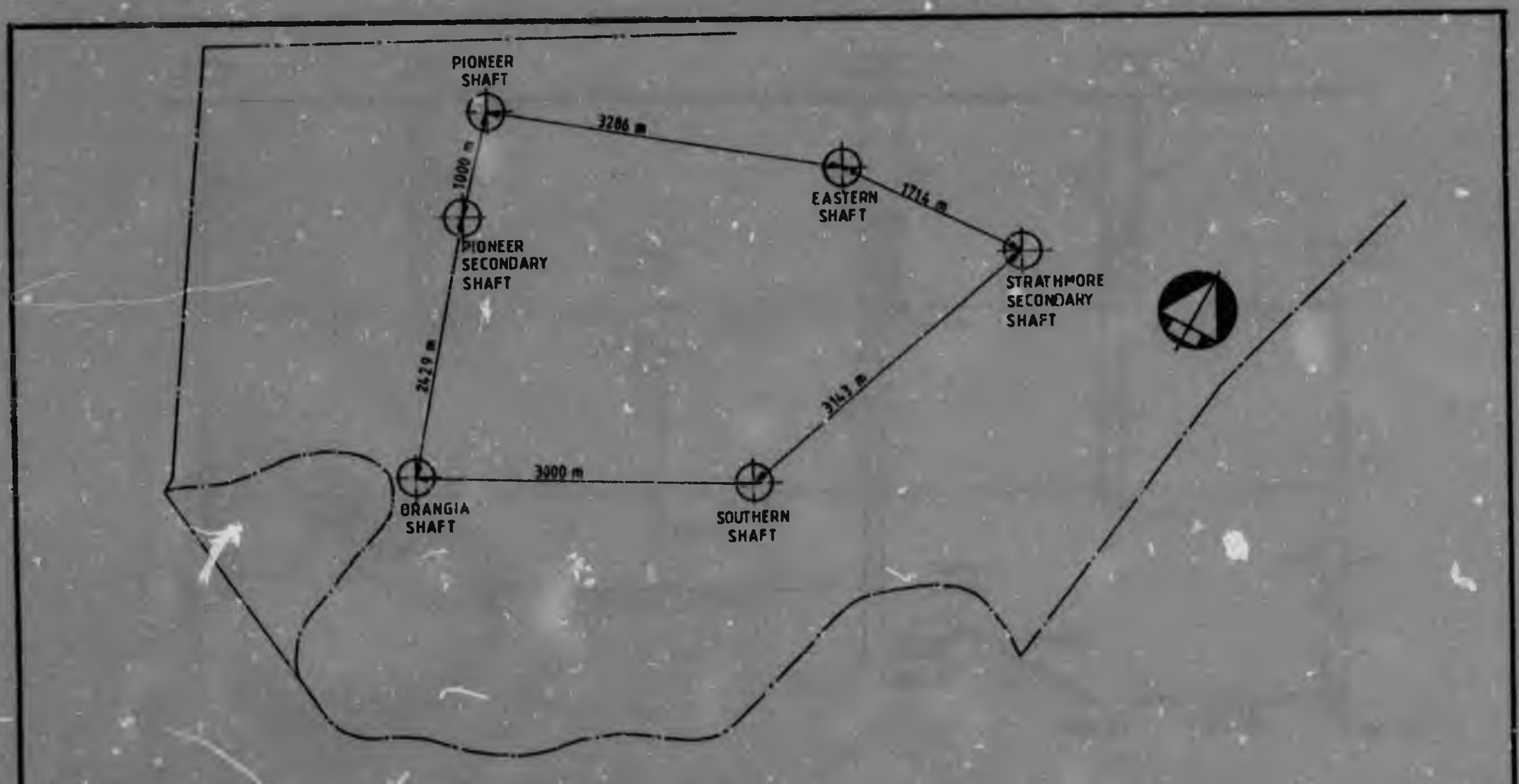


FIGURE 1.2

RELATIVE PLAN POSITION OF EXISTING BUFFELSFONTEIN SHAFTS

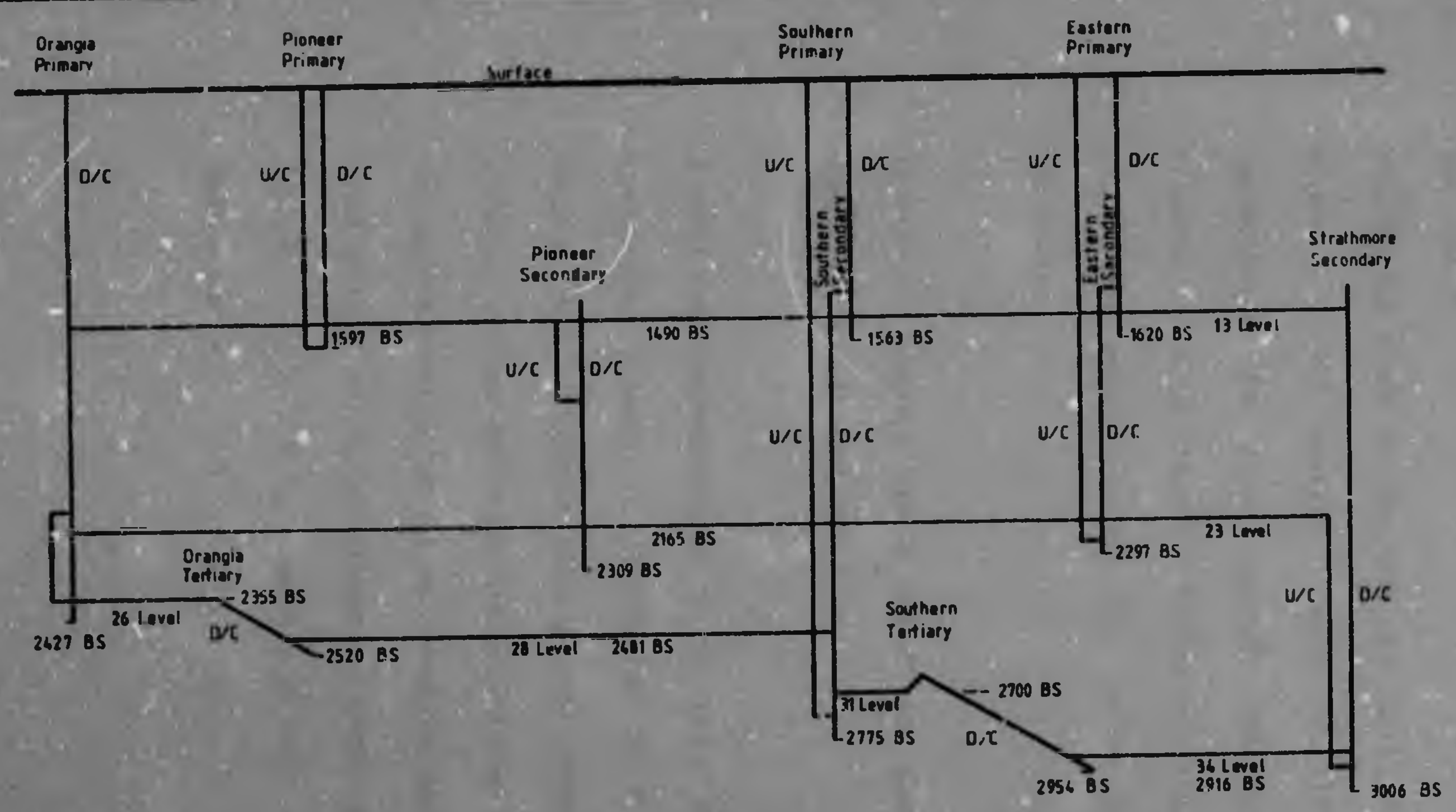


FIGURE 1.3

DIAGRAMATIC SECTION OF EXISTING BUFFELSFONTEIN SHAFTS

1.4.1 Ventilation

Ventilation flows in the mine are well defined yet complex. With dynamic changes in the network necessitated by fire control and containment procedures on a regular basis, a steady-state, accountable mass flow for the entire circuit is difficult.

Ventilation surveys tie-in upcast and downcast air quantities to within 5 %.

A plan view representation of the major interacting ventilation schemes is illustrated in Figure 1.4. Figure 1.5 is a section showing main interconnecting airways, airflow direction and upcast fan locations. The entire ventilation system circulates approximately 1 800 m^3/s of air at an average fan static pressure of 5,0 kPa. Table 1.1 is a summary of main surface fan design duties.

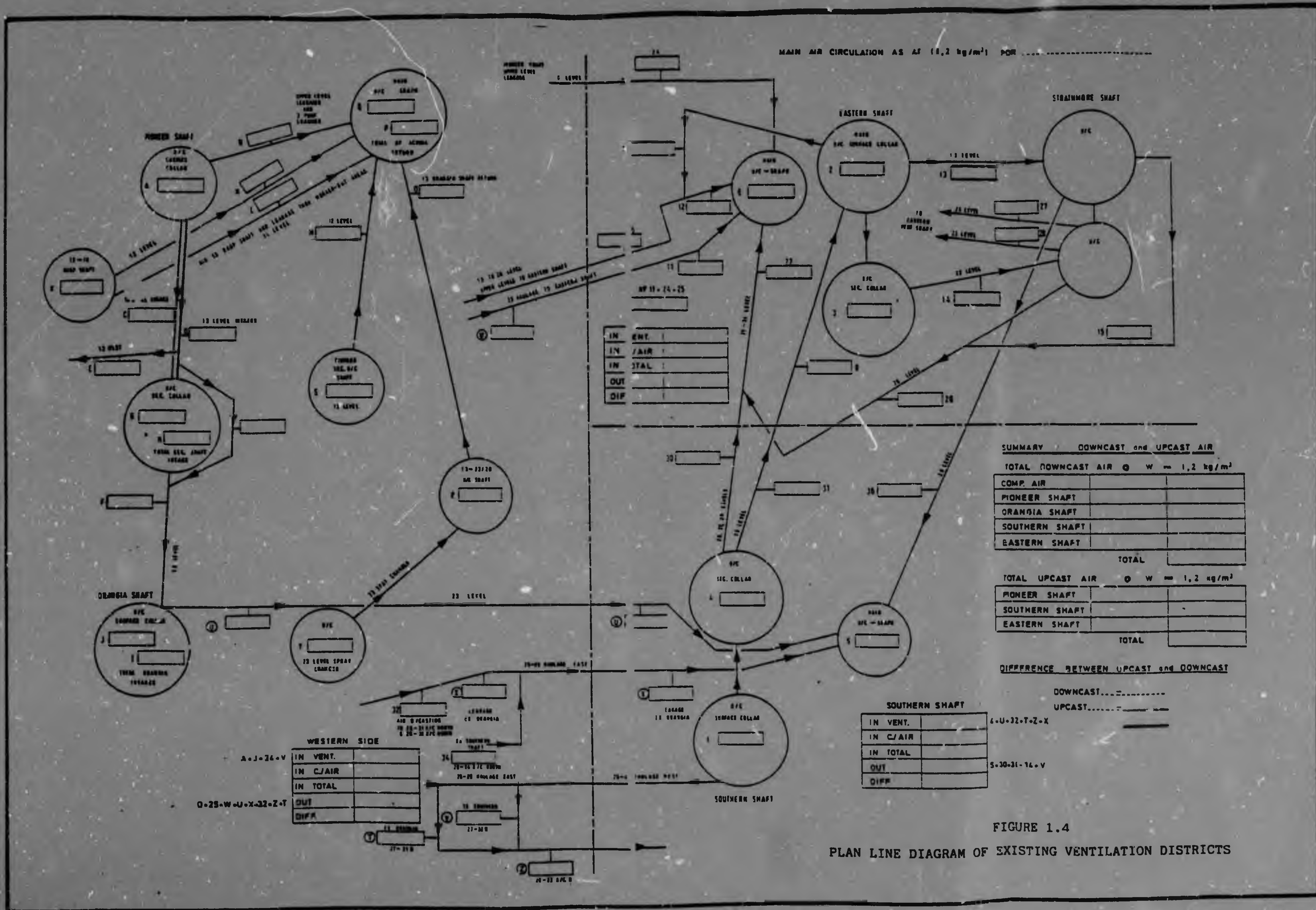


FIGURE 1.4

PLAN LINE DIAGRAM OF EXISTING VENTILATION DISTRICTS

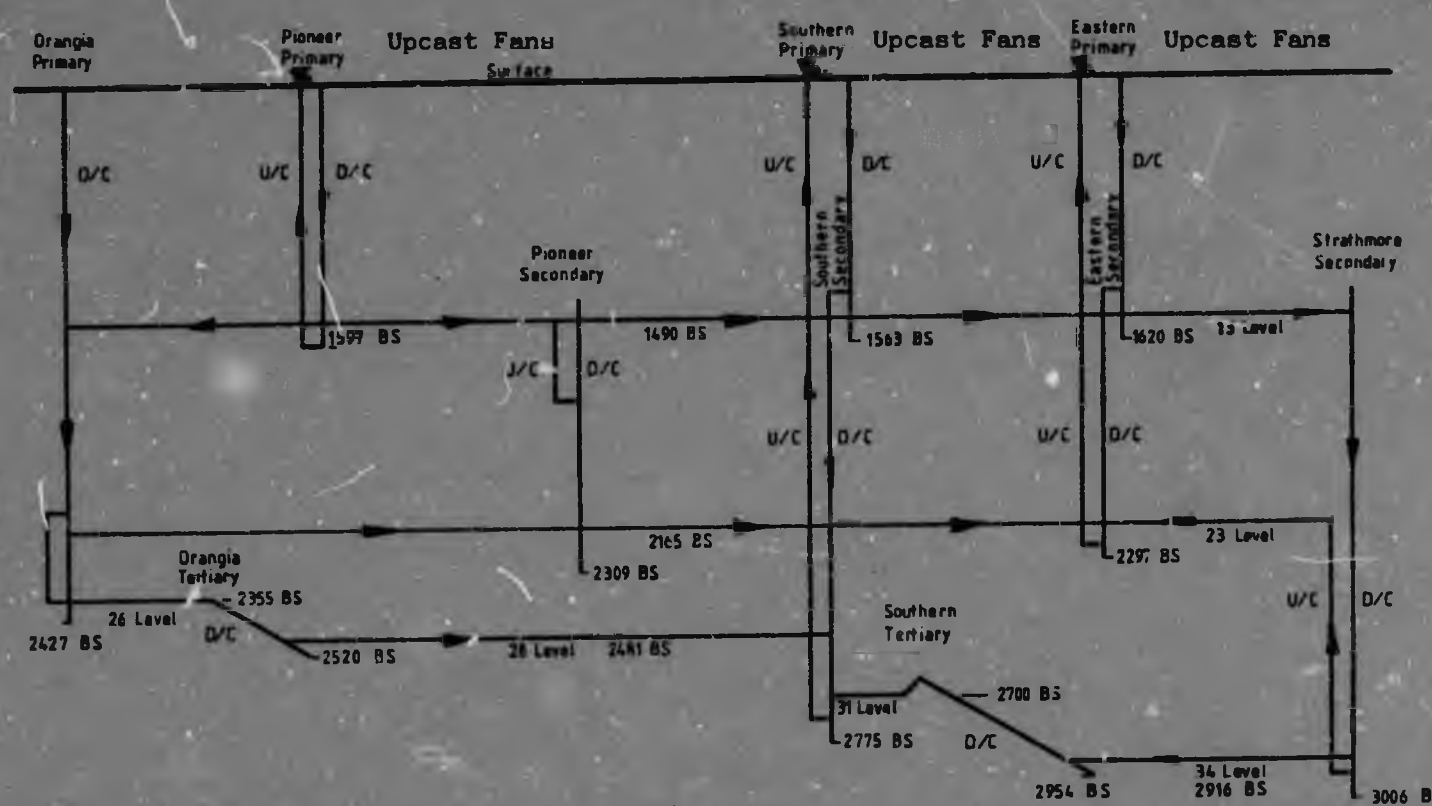


FIGURE 1.5

SECTION LINE DIAGRAM OF EXISTING VENTILATION DISTRICTS

SITE	DESIGN DUTY	
PIONEER SHAFT	2 x 2 425 kW:	Impeller diameter 3,86 m 250 m^3/sec. at 5,9 kPa speed 593 R.P.M.
	1 x 2 089 kW:	Impeller diameter 3,46 m 200 m^3/sec. at 5,9 kPa speed 750 R.P.M.
EASTERN SHAFT	3 x 4 250 kW:	Impeller diameter 3,86 m 380 m^3/sec. at 8,3 kPa speed 720 R.P.M.
SOUTHERN SHAFT	3 x 2 611 kW:	Impeller diameter 3,48 m 250 m^3/sec. at 7,4 kPa speed 750 R.P.M.

TABLE 1.1
MAIN SURFACE FAN DESIGN DUTIES

1.4.2 Refrigeration

Location of existing refrigeration plant is depicted in sectional drawing, Figure 1.6. Of the 115,3 MW of refrigeration capacity, 29 MW are located on surface at Orangia and Southern Shafts. The rest is distributed underground as follows:-

Orangia Shaft	11,5 MW
Pioneer Shaft	7,9 MW
Southern Secondary Shaft	26,8 MW
Strathmore Secondary Shaft	33,6 MW
Eastern Secondary Shaft	6,5 MW
TOTAL	86,3 MW

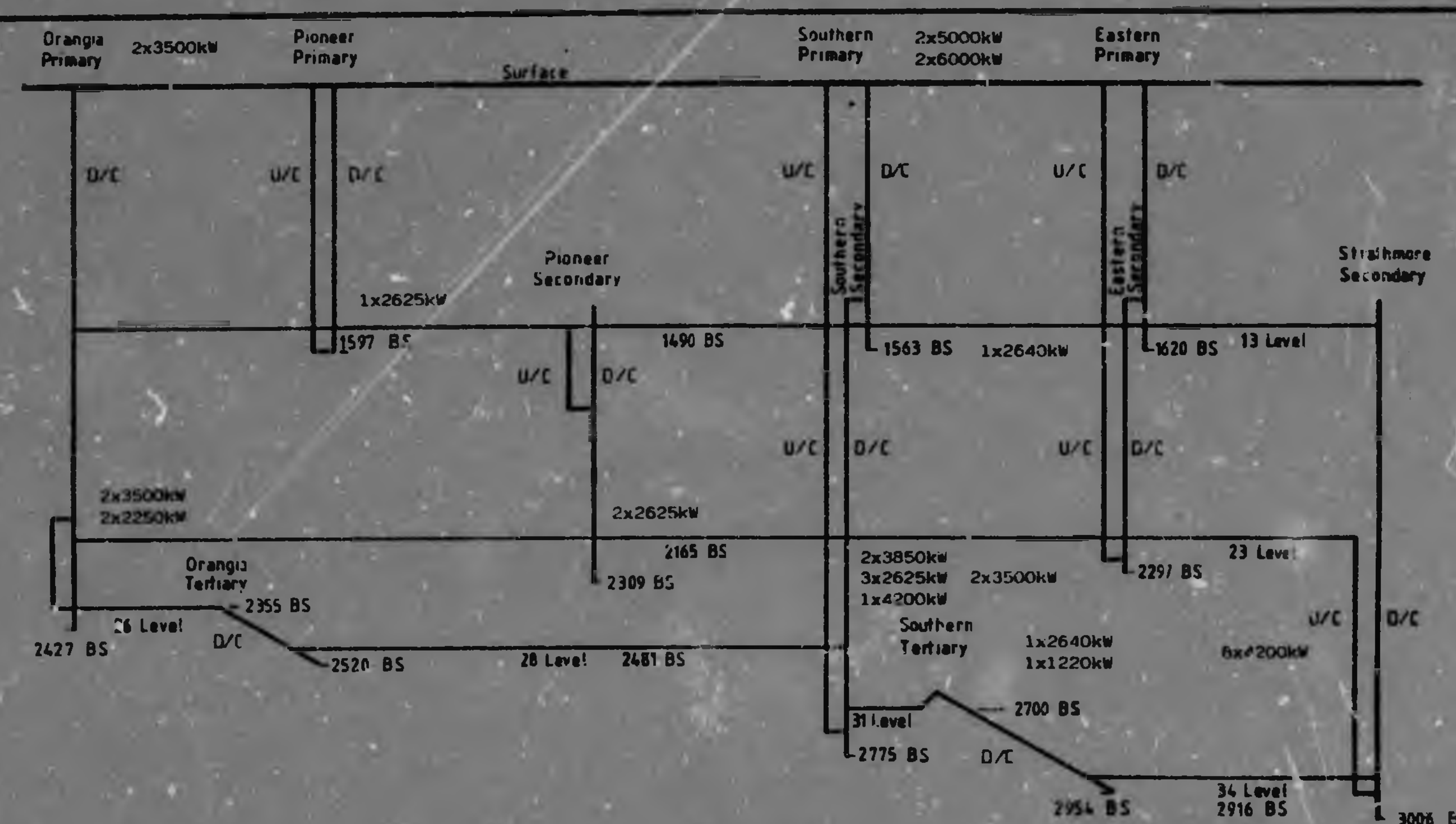


FIGURE 1.6

SECTION DIAGRAM OF EXISTING REFRIGERATION PLANT

An inventory of refrigeration machines and their specifications are shown in Table 1.2.

1.5 Global Depletion and Build-up Programme of ore reserves

Current ore reserves are rapidly being exhausted and mining of remnants and pillars in progress. Figure 1.7 shows the geographical plan position of the original Buffelsfontein ore reserves and the relative position of the Tertiary block.

The Strathmore Tertiary Shaft system has been designed to exploit a block of Vaal Reef covering an area of some 4,2 million square metres. It is situated within a wedge - shaped sector, eastwards from the Strathmore and Trough faults. The Vaal Reef ore body is located between 3 500 metres and 4 300 metres below surface.

This block of the ore body is part of the area designated the Lucas block and lies in the deepest part of the Klerksdorp Gold Field, where geological structures are complex.

SHAFT	CHAMBER	SIZE kW	NO	TIME AVAILABLE	TIME REQUIRED
PIONEER	13 LEVEL	2 625	P1	1989	1991
	19 LEVEL	2 625 2 625	P2 P3	1997 1997	1997 1997
EASTERN	14 LEVEL	2 640	E1	1991	1991
	24 LEVEL	2 640 1 220	E2 E3	1998 1991	1998 1991
ORANGIA		2 250 2 250 3 500 3 500	O1 O2 O3 O4	1996 1996 1996 1996	1996 1996 1996 1996
	SURFACE BAC	3 500 3 500	O5 O6	1998 1998	1998 1998
SOUTHERN	27 LEVEL	3 500 3 500 3 850 3 850	S1 S2 S3 S4	1989 1996 1989 1989	MID 1995 1996 MID 1995 MID 1995
		2 625 2 625 2 625 4 200	S5 S6 S7 S8	1989 1994 1998 1997	1991 1996 1998 1998
	SURFACE S.W.	5 000 5 000 6 000 6 000	S9 S10 S11 S12	1998 1989 1989 1989	1998 MID 1995 MID 1995 1996
STRATHMORE	28 LEVEL	4 200 4 200 4 200 4 200	ST.1 ST.2 ST.3 ST.4	1989 1989 1989 1989	1909 1989 1989 1989
		4 200 4 200 4 200 4 200	ST.5 ST.6 ST.7 ST.8	1989 1989 1989 1989	1989 1989 19[illegible] 19[illegible]

TABLE 1.2

REFRIGERATION PLANTS SPECIFICATIONS AND AVAILABILITY

FIGURE 1.7

GEOGRAPHICAL PLAN SHOWING THE TERIARY BLOCK LOCATION

Disclosure of the depletion programme, tonnages and grade are considered confidential and therefore not referred to.

1.6 Equipment Availability

As a result of the shift in mining location, most of the refrigeration plants and main fans will become available to be used in the Tertiary project. In an effort to economize the project, maximum re-use of existing equipment is planned.

2. PROPOSED MINING STRATEGY

2.1 Development

The principle factor to be considered in the mining of the Vaal Reef at the depths planned for the Tertiary Shaft, is to site excavations so that maintenance of these excavations is minimal and the level of seismicity is reduced.

The haulages and crosscuts are the essential arteries for the successful mining of the ground. Damage to these excavations will result in restrictions to air flow, rock flow, access to and from workings and possibly damage to services as well.

The importance of correctly siting the haulages specifically deep in the footwall is critical due to their required long life. Crosscuts are especially important and susceptible as they are close to the reef horizon. As a result, the strike haulages will be located approximately 160 m in the footwall from the reef. Crosscuts breaking away at 300 m intervals will be long, averaging 400 m.

Raise development will probably have to be done by cutting shoulders simultaneously with the development in the form of a wide raise. This will serve the purpose of stabilising the hangingwall during ledging operations.

2.2 Stoping

Stoping at these depths will result in the hangingwall being severely fractured. Convergence rates are expected to be higher than previously experienced and seismicity will be a cause for concern.

2.3 Planned Production Levels

Production levels are visualized starting on 42 level and decending to 49 level. Vertical level intervals will be 75 m, and will tie in with the Eastern/Strathmore system as shown in Figure 2.1.

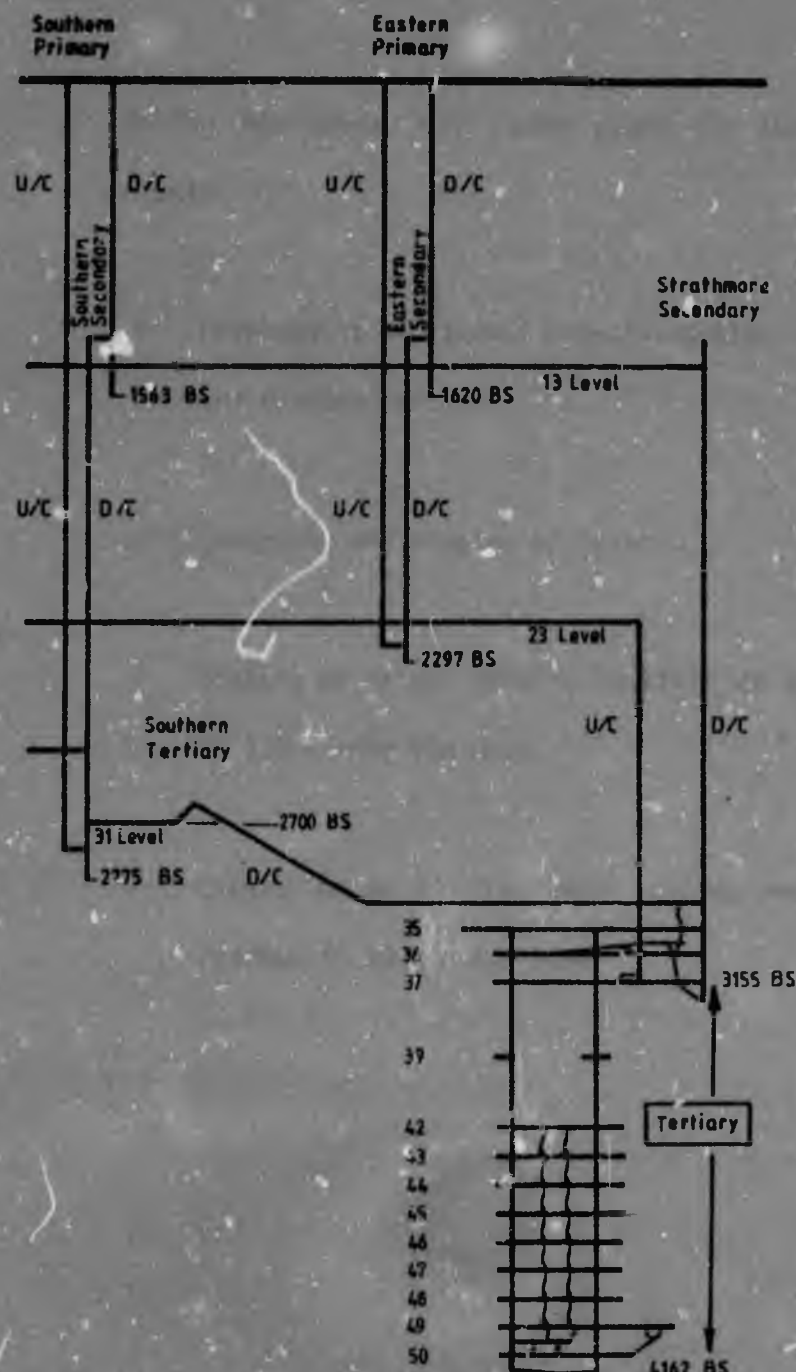


FIGURE 2.1

TERTIARY PRODUCTION LEVELS

2.4 Idealised Mining Layout

Mining operations will take place in four stages namely:

- Development of raises, travelling ways, box holes and slusher raises.
- Equipping and ledging of raises.
- Stoping of raises keeping backfill at an average of 3,5 m from the face.
- Completion of stoping until holing position is reached in next raise line.

3. TERTIARY VENTILATION DISTRIBUTION

3.1 Stope Ventilation

A diagramatic isometric drawing and details in section and plan, Figures 3.1 to 3.3, depict a stylized interpretation of the envisiaged mining method, incorporating backfill and continuous scraper ore handling techniques.

The air mass flow into the stope is primarily governed by the cross-sectional face area and maximum velocity.

Maximum stoping width	:	1,4m
Average fill to face distance	:	3,5 m
Maximum face velocity	:	3,0 m/s
Face quantity	:	15 m^3/s
Face mass flow	:	21 kg/s

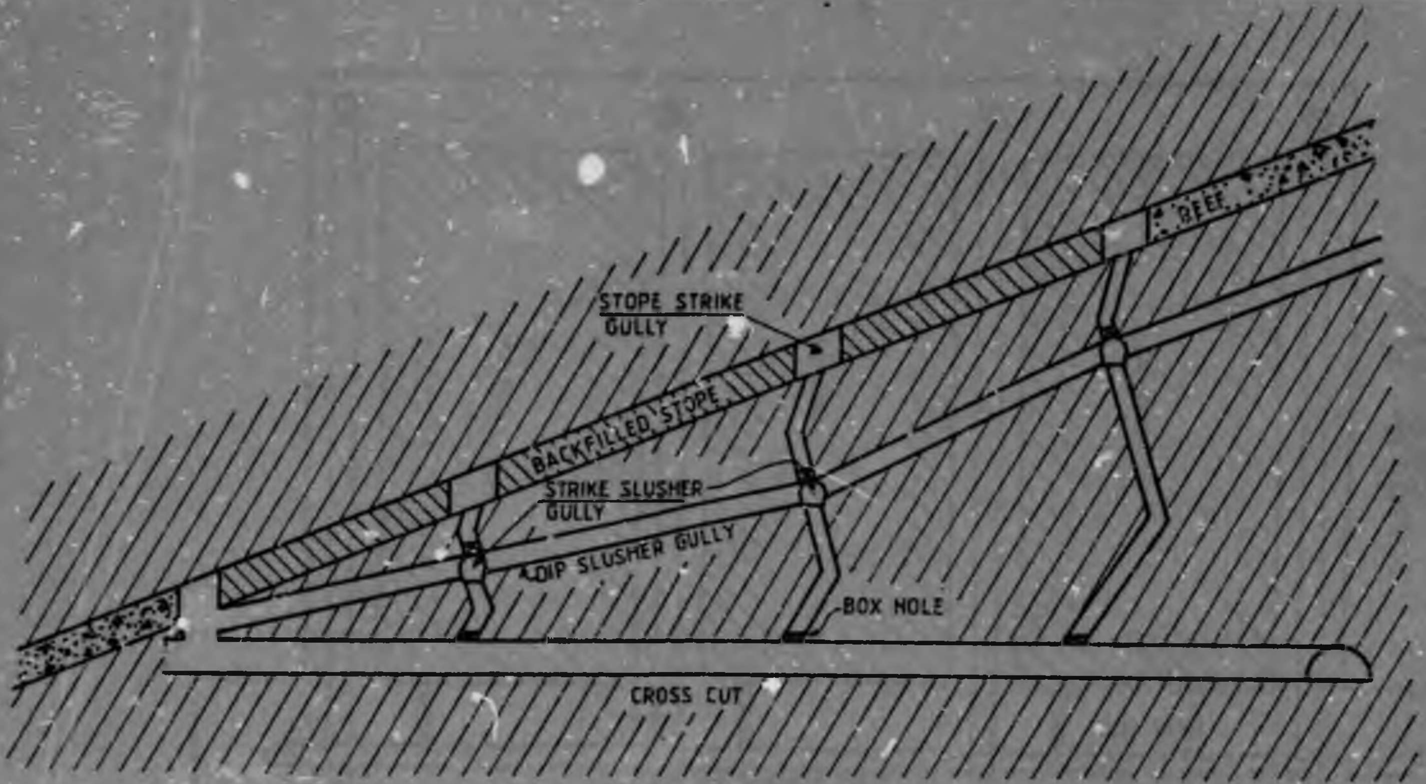


FIGURE 3.1

SECTIONAL VIEW OF AN IDEAL STOPING CONFIGURATION

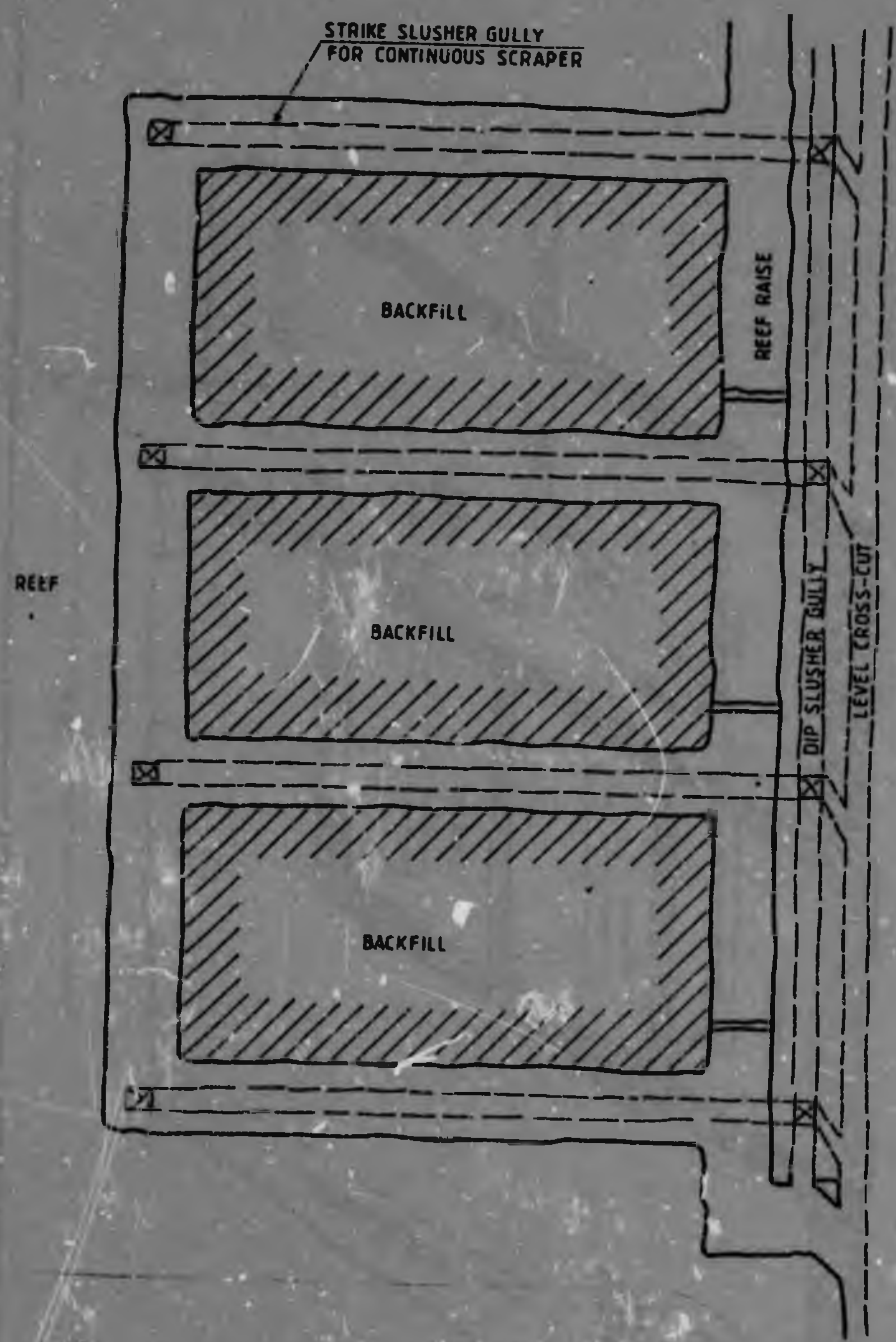


FIGURE 3.2

PLAN VIEW OF PROPOSED STOPING

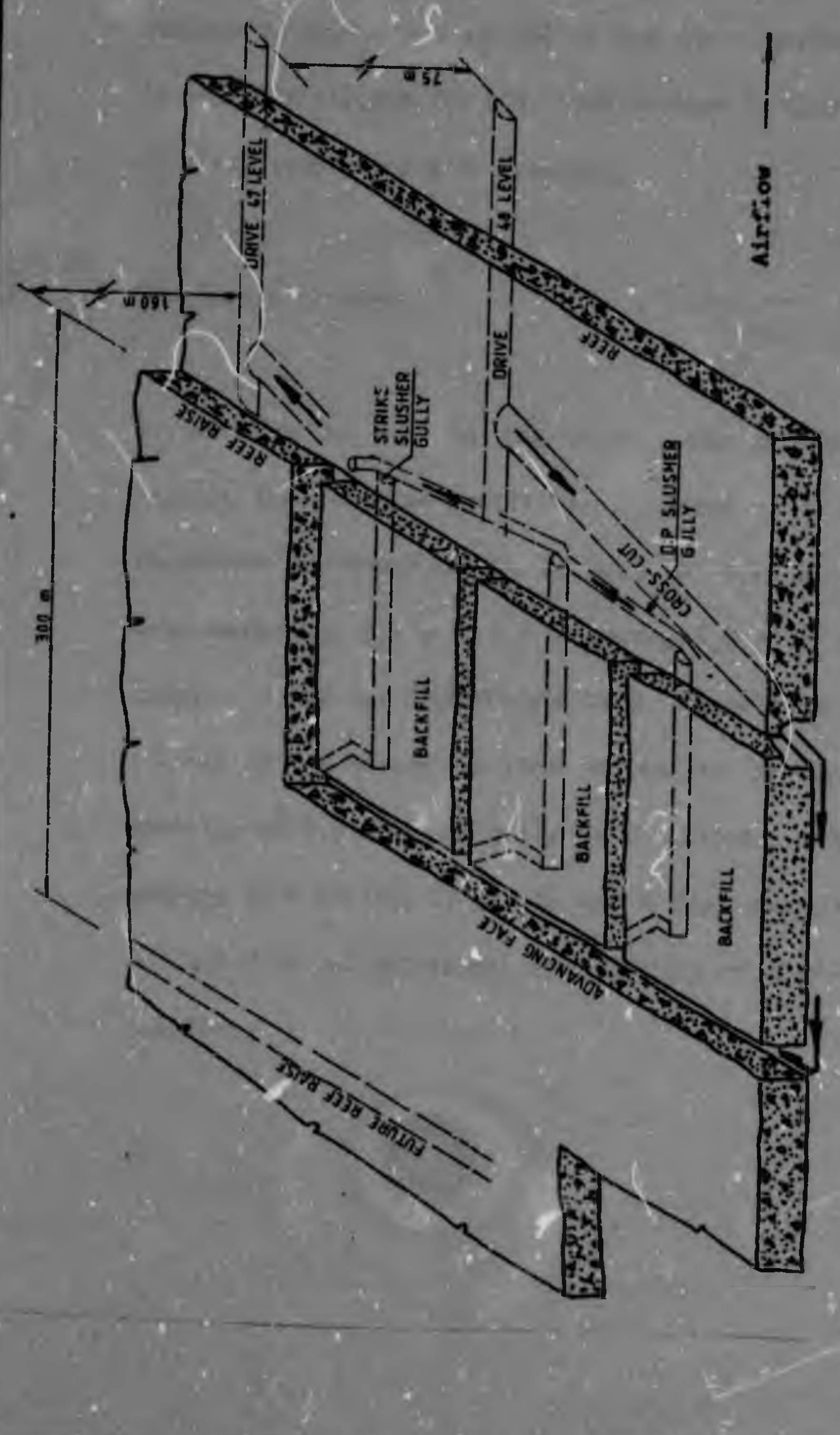


FIGURE 3.3
ISOMETRIC VIEW OF IDEALIZED STOPING

Due to the high degree of backfill, rapid closures and rigid dip gully controls, leakage is expected to be reduced to approximately 20% of the air entering the stope. Air allowed for the stope horizon is therefore 19 m^3/s (stope face plus leakage).

3.2 Continuous Scrapers

In addition to air to the stope, slusher gullies require force ducted ventilation. Based on a legal requirements (Mines and Works Act) of 0,15 $m^3/s/m^2$, ends measuring 2,3 m x 2,8 m having 1,0 m^3/s would comply. Due to high Virgin Rock Temperatures and slusher drive motors located in intake airways, a quantity of 1,5 m^3/s per gully is allocated. With an average of 4 gullies (3 strike and 1 dip) operational in each line an additional air quantity of 6 m^3/s is required.

3.3 Air Per Active Stope Line

Air to face	15 m^3/s	21 kg/s
Air leakage in stope	4 m^3/s	6 kg/s
Air to slusher gullies	6 m^3/s	9 kg/s
Crosscut air quantity	25 m^3/s	36 kg/s

at $W = 1{,}4\ kg/m^3$

3.4 District Air Requirements

A group of 5 lines will comprise an ideal mining district, of which 3 will be producing (stoping), one vamping and one developing. Air requirements for the various functions are shown in Table 3.1.

LINE FUNCTION	AIR QUANTITY
STOPING	25 m^3/s
VAMPING	15 m^3/s
DEVELOPING	10 m^3/s

TABLE 3.1
AIR QUANTITIES FOR INTER-LEVEL AIRWAYS

3.5 Tertiary Air Requirements

Exploitation of the ore reserves within the Tertiary block will commence in the upper block, levels 42 to 45, progress to the lower horizon, levels 46 to 49 and terminate with extraction between levels 44 and 46.

Appendix A shows plans of levels 42 to 49 which depict the structural geology of the reef to which drives and crosscuts have been added.

From an overlay analysis, level to level the most probable connecting raise lines have been established. This preliminary exploitation programme represented as sectional sketches is detailed in 3 Phases in Appendix B.

Maximum air commitments for the various phases and mining horizons are summarized in Table 3.2.

	UPPER BLOCK 42 TO 45 LEVEL	CENTRAL BLOCK 44 TO 46 LEVEL	LOWER BLOCK 46 TO 49 LEVEL
PHASE 1	175 m^3/s	-	100 m^3/s
PHASE 2	100 m^3/s	-	125 m^3/s
PHASE 3	-	200 m^3/s	-

TABLE 3.2

TERTIARY DISTRICT AIR REQUIREMENTS WHICH INCLUDE STOPING, RAISE LINE DEVELOPMENT AND VAMPING ONLY

The largest theoretical air demand, based on simultaneous district mining, occurs during Phase 1. The expected 275 m^3/s is escalated by additional air required for primary development, shaft station requirements and leakage. This amounts to an additional 40 %.

Approximately 60 % of the circulated air is expected to be dedicated to the mining districts.

The total air requirements are shown in Table 3.3 and results in a total circulated air quantity of 460 m^3/s for the Tertiary.

ALLOCATION	%	m^3/s
* AIR TO MINING DISTRICTS	60	275
* COMMITMENTS TO LEVELS (PRIMARILY DEVELOPMENT)	20	93
* SHAFT STATION (TIPS, HOISTS)	10	46
* LEAKAGE THROUGH SEALED X-CUTS	10	46
TOTAL	100	460
		640 kg/s @ = 1,4 kg/m^3

TABLE 3.3
TOTAL TERIARY AIR REQUIREMENTS

This Airflow equates to an air factor of 3,56 kg/s/kiloton/month or 2,56 m^3/s/kiloton/month based on a mining rate of 180 000 tons/month.

3.6 Industry Norms

3.6.1 COMRO annual ventilation returns

As verification of the air factor, 3,6 kg/s/kiloton/month, industry norms published by the Chamber of Mines were referred to. (COMRO Ventilation Return). An extract of the 1987 return is shown in Appendix C.

Based on total tonnage and industry average air circulation, the following air factors for various groups of mines are derived and summarized in Table 3.4.

	kg/s/kton/month
PROGRESSIVE INDUSTRY AVERAGE 1974 - 1987 For Quarters Ending March	4,4
ENTIRE INDUSTRY AVERAGE	5,08
RANGE	1,05 - 11,60
INDUSTRY 7 Deepest Mines	6,03
RANGE	3,46 - 8,75
INDUSTRY Weighted To Rockbreaking	6,50
INDUSTRY Mines deeper than 2 000 m	4,7
RANGE	2,99 - 6,77

TABLE 3.4
INDUSTRY AIR CIRCULATION STATISTICS

Using industry trends as a guideline, a probable mass flow for the Tertiary lies between 4,4 kg/s/k ton/month and 6,5 kg/s/k ton/month, an average of 5,5 kg/s/k ton/month. At 180 000 tons per month this results in a required mass flow range of 792 to 1 170 kg/s.

Based on a conservative reject wet-bulb temperature of 26°C the cooling capacity of air deminishes to nil at a depth below surface of approximately 2 000 m (Environmental Engineering in South African Mines: 956). Using this depth as a limit for deeper conventional mines, the average air factor equates to 4,7 kg/s/kton/month for mines in this catagory.

For partially backfilled mines, an air factor of 3,9 kg/s/kton/month is representative this is based on two mines using backfill on a large scale: East Driefontein, 300 000 tons placed and Randfontein Estates, 200 000 tons placed. Air factors for these mines are 3,4 and 4,4 kg/s/kton/month respectively.

Published guidelines give further substantiation to reduced air factors at depth (Environmental Engineering in South African Mines: 960).

"Despite the experience to date it is not easy to specify an overall air quantity. Air quantities on South African gold mines range from 3 to 6 kg/s per 1 000 ton of rock mined per month. An amount of 4 kg/s per kiloton per month is a typical value for deep mines in the Orange Free State. This allows some degree of flexibility."

The guidelines quoted apply to conventional mining WITHOUT the use of backfill.

3.6.2 Auto-compressive effect

Air decending vertically is subject to the natural heating effect of auto-compression due to increase in potential energy. The potential energy increase results in a temperature rise in the air. In the case of a 3 800 m elevation change, the auto-compressive heat loads are shown as a function of mass flow in Table 3.5. Calculations are based on the air heat gain relationship:

$$0{,}979 \text{ kJ/kg per } 100 \text{ m}$$

MASS FLOW (kg/s)	AUTO-COMPRESSIVE Heat Load (MW)
400	14,8
450	16,6
500	18,5
550	20,3
600	22,2
650	24,0
700	25,9

TABLE 3.5
AUTOCOMPRESSIVE HEAD LOAD OF VARIOUS AIR MASS FLOWS TO THE TERTIARY MEAN ROCKBREAKING DEPTH

Generally, in deep mines auto-compression accounts for approximately 20 to 30% of mine heat loads. In the Tertiary case, auto-compression is 33 % as shown in Section 15.

Notably, all auto-compression heat must be counteracted by refrigeration in the circuit, thus clearly the lower mass flow in this respect the better.

For heat load considerations, the minimum non-recirculated mass flow of air from surface is a primary criterion. This mass flow must however be sufficient to :

- Be able to dissipate the tertiary heat load with the refrigeration provided (ie. be adequate in mass as a heat transfer medium).

- Be able to dilute gases and dust in the normal course of mining to acceptable concentrations.

- Cater for production, development, services and leakage.

3.6.3 COMRO environmental engineering verification

As a result of an independent study by the Chamber of Mines Research Organization a statement on air quantity from the summary reads :

"The present study has confirmed that the optimum downcast airflow, as suggested by GENMIN, is about 2,2 m^3/s per kton/month OR 3,08 kg/s/kton/month. This is lower than that generally used for shallower mines (the average for S A Gold Mines is close to 4 m^3/s per kton/month or 5,5 kg/s/kton/month). However, for the extreme depths being envisaged, the selection of a lower value is entirely in keeping with current thinking." (COMRO December 1988)

3.6.4 Conclusion

A proposed mass flow of 600 kg/s, considering depth and degree of backfill, is adequate. This relates to an air factor of 2,4 m^3/s/kton per month (at a density of 1,4 kg/m^3).

3.7 Controlled Re-use of Air

As part of the primary design philosophy controlled re-circulation of air was <u>not engineered into the solution</u>. Implementing this concept however has great potential.

The advantages and disadvantages of re-circulation were considered at project inception and the major features are listed below:

3.7.1 Advantages

1. Re-circulation would reduce considerably the mass flow of air in the circuits particularly the Eastern and Strathmore Shafts supplying the Tertiary, whilst maintaining design mass flows in the working horizon. The implications are :

(a) a reduction of upper system pressure losses
(b) a reduction in main fan power
(c) a reduction in auto-compressive heat, subsequently total heat load and therefore a saving in installed refrigeration, particularly at the 27L mid shaft cooling installation
(d) a better control over the distribution of air within the Tertiary mining horizon.

2. The Tertiary configuration lends itself almost ideally to large scale re-circulation with the intake and return airway infrastructure on 35, 36 and 37 levels.

3. The successful application of South Africa's largest controlled air re-circulation scheme at Winkelhaak Gold Mine, has given GENMIN a high degree of confidence in the practical implications of the controlled re-use of air.

3.7.2 Disadvantages

1. The major constraint in the re-use of air is the uncontrolled contamination produced by multi-blasting twin drives in a concerted effort to access reef in the shortest possible time.

   Blasting fumes and dust would unpredictably be entrained in the airstream rendering the airflow highly contaminated and unusable. The scrubbing of such large quantities of air containing nitrous fumes is impractical.

2. An additional large air cooling infrastructure in the order of 20 MW would have to be established on 36L. This would compensate the reduced duty of the 27L mid shaft installation which under this condition would be under-utilized.

3.7.3 Possible future application

Agreement in principle was reached with the mining department that once development to reef was concluded, all other development blasting would be timed. Contaminated air could be predicted and the re-use of air made possible.

Uncertainty of the reef location and the subsequent total air quantity and pressure demands may require this technology to be applied. If so, a great deal of flexibility exists in using (a) regional or (b) total controlled air re-circulation.

Regional re-circulation is envisaged applicable section to section or level by level as problems arises. A conceptual re-circulation scheme is depicted in Figure 3.4.

Alternatively additional mining levels or a major change in the mining scope could be catered for by a total re-circulation scheme. Figure 3.5 shows the proposed airways for a non-recirculating scheme. Modifications illustrated in Figure 3.6 would allow a controlled mass flow of air to be re-entrained into the Tertiary circuit after being sensed and cooled. Automatic conversion to a one-pass system would be effected during blasting or emergency procedures.

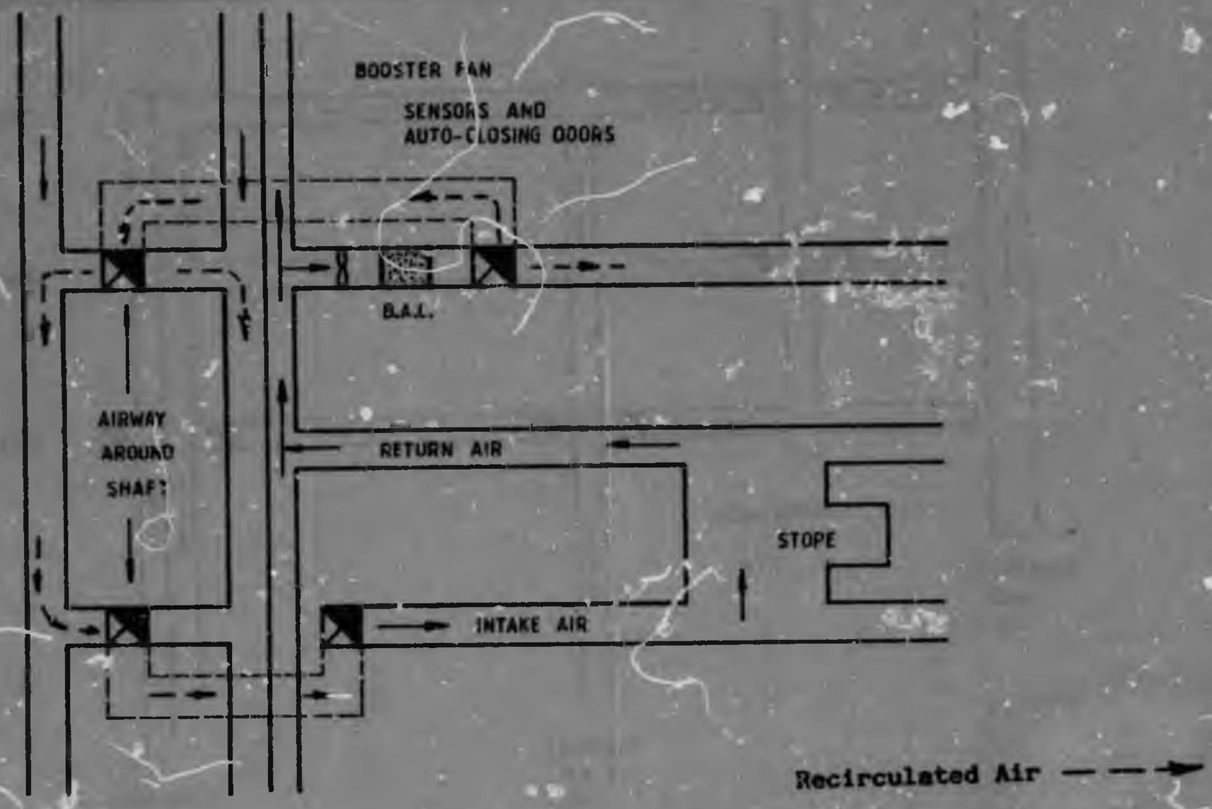


FIGURE 3.4

A POSSIBLE REGIONAL RECIRCULATION APPLICATION

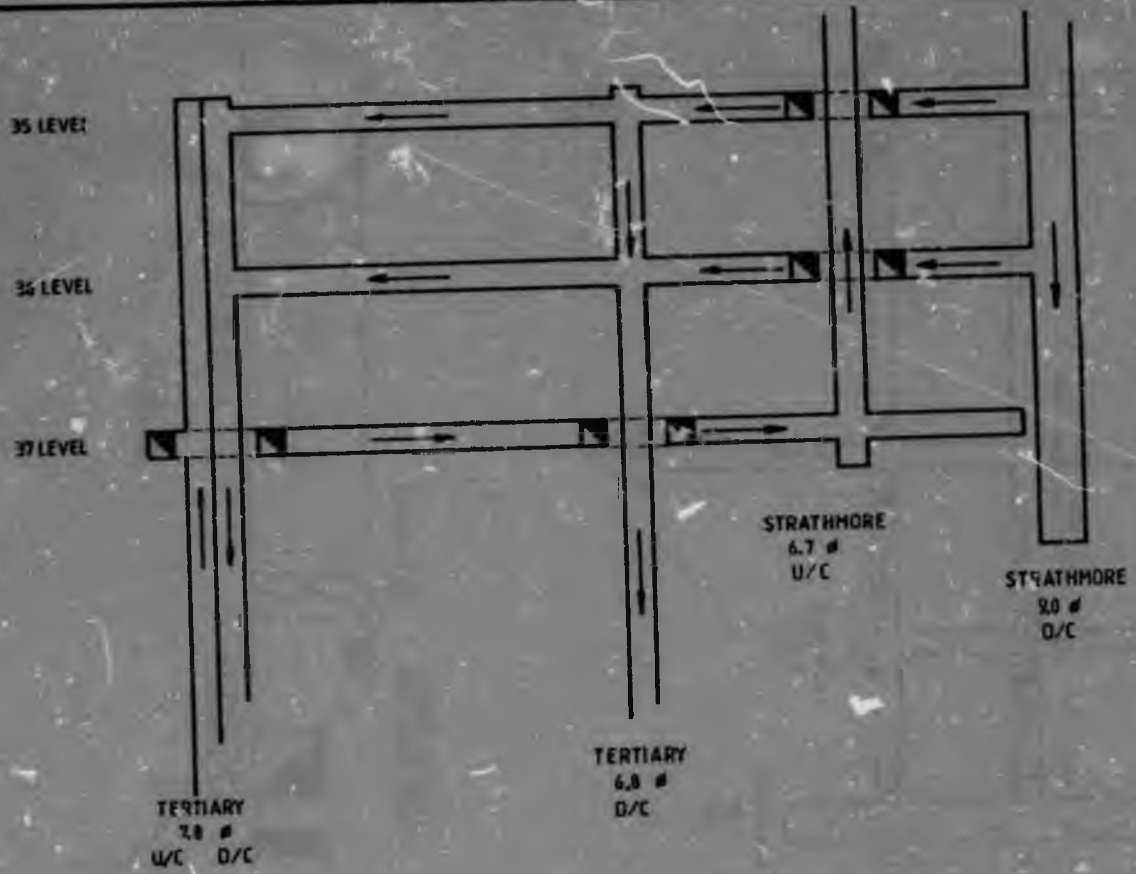


FIGURE 3.5

EXPECTED 35, 36 AND 37 LEVEL AIRWAY DETAIL

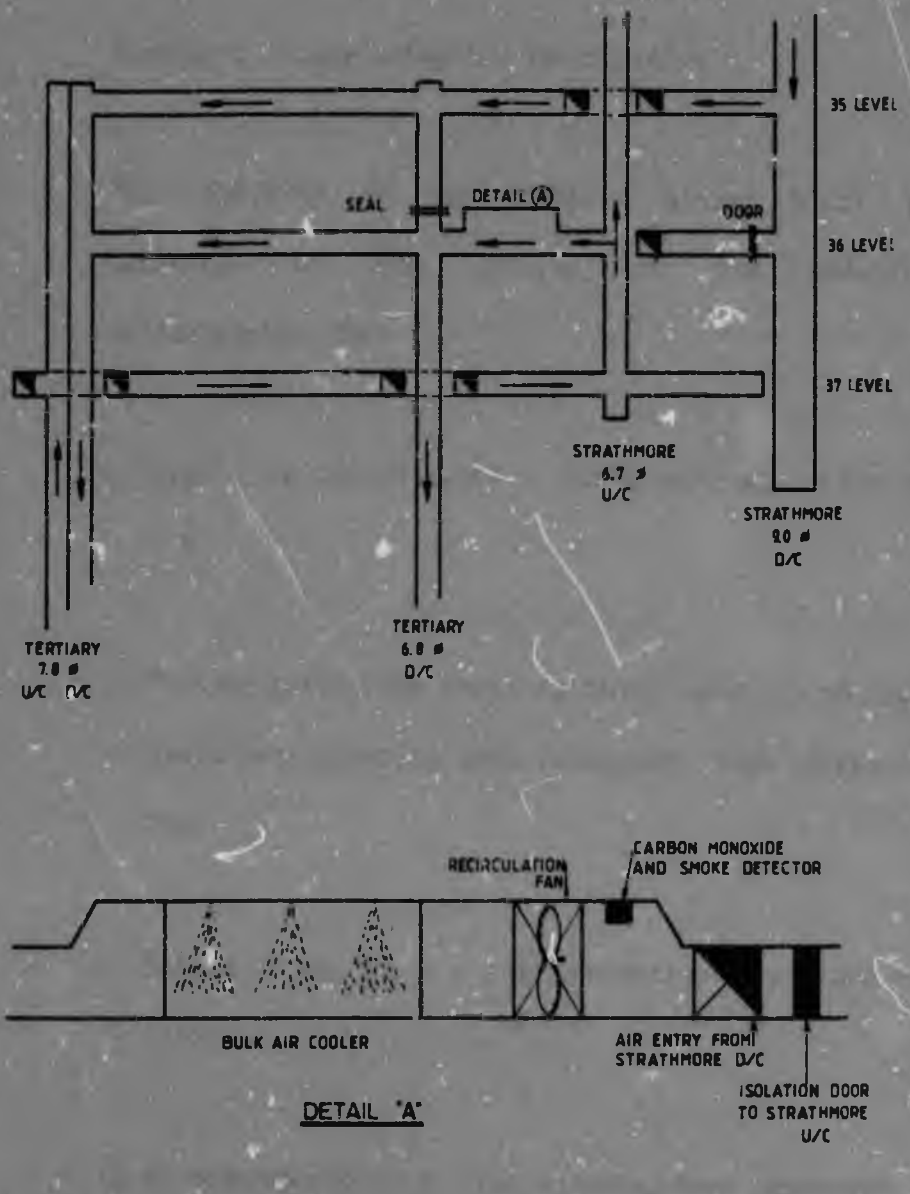


FIGURE 3.6

A POSSIBLE TOTAL CONTROLLED AIR RECIRCULATION APPLICATION

4. STRATEGIC AIRWAYS

4.1 Strategic Airway Integrity Verification

The viability of the Tertiary mining block is dependent on the ability of the existing infrastructure to:

- Supply to the district a sufficient mass flow of air.
- Provide sufficient cross sectional area to obviate excessive velocity and subsequent high pressure loss.
- Ensure airway integrity by prevention of closure or collapse.

In an ordered sequence, the following were addressed:

- Major existing air circuits.
- Airway capacities as a function of limiting velocity.

- Quantification of airway resistances.

- Identification of potential hazards and alternatives for circumvention.

4.1.1 Major air circuits

Primary air circuits for the ease of understanding are reduced to upcast and downcast shafts, main interconnecting haulages intake and return levels servicing ventilation districts. The main reticulation features are listed in Table 4.1 and can be read in conjunction with Figure 4.1.

ITEM	DESCRIPTION
A	Eastern Primary Downcast Shaft
B	13 Level Intake Airways
C	Strathmore Secondary Downcast Shaft
D	Strathmore Workings
E	Eastern Secondary Workings
F	Main Return Airways 20 to 26 Levels
G	Air ex Pioneer
H	Air ex Pioneer
I	Eastern Secondary Downcast
J	Eastern Workings
K	12 Level to Eastern Workings
L	Eastern Upcast Shaft
M	Eastern Main Upcast Fan
N	Southern Primary Downcast Shaft
O	Regulated Air from South to East
P	Southern Secondary Downcast Shaft
Q	Southern Workings
R	Air ex Orangia
S	Southern Upcast Shaft
T	Southern Main Upcast Fans
U	Strathmore Upcast Shaft

TABLE 4.1
PRIMARY EXISTING AIR CIRCUITS

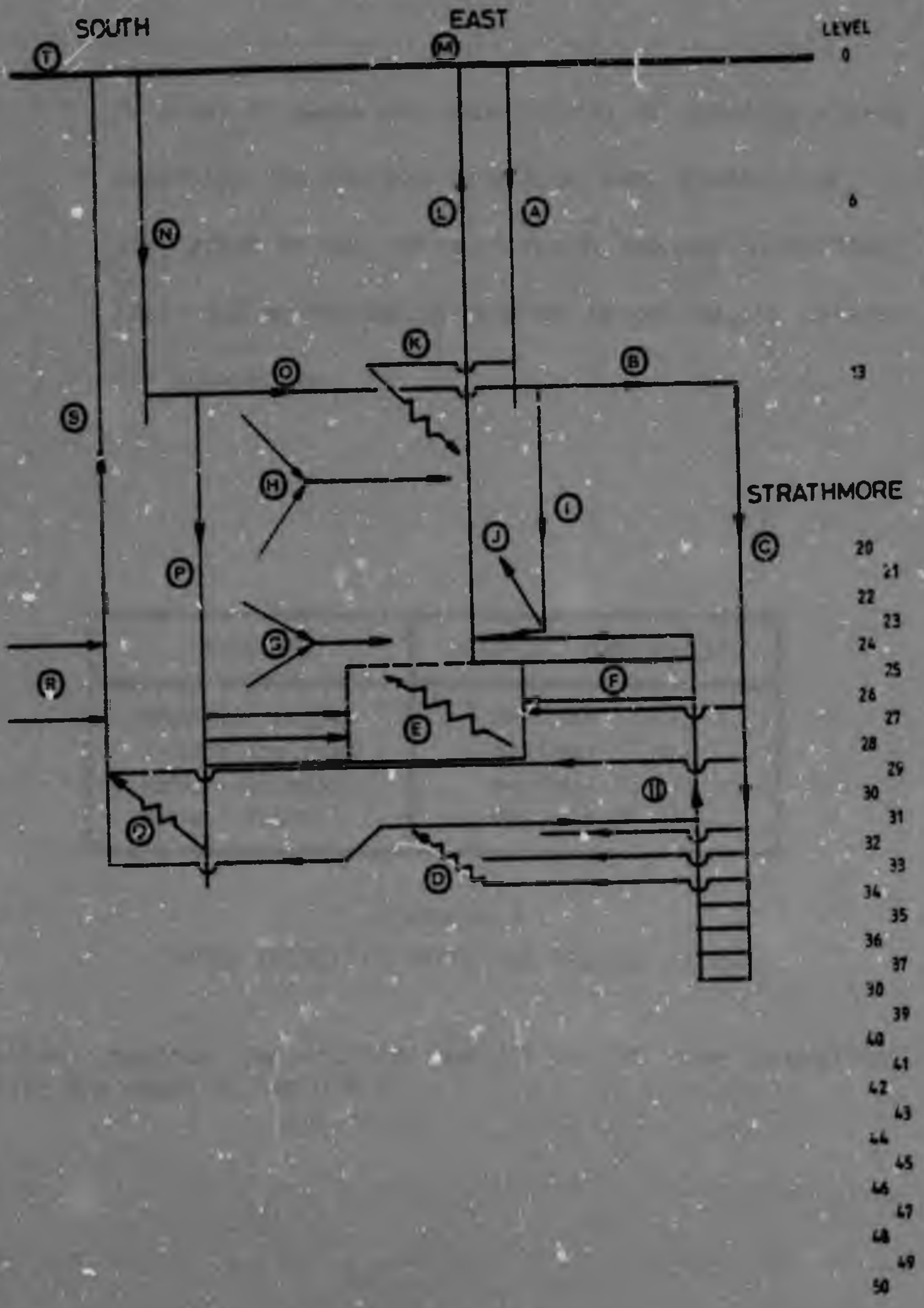


FIGURE 4.1

PRIMARY EXISTING AIR CIRCUITS

4.1.2 Airway capacity

In order to gauge the upper limits of existing airway capacity, and thereby determine what constraints, if any, exist in the infrastructure, maximum velocities, Table 4.2 were used as a guide to estimating maximum air quantities.

AIRWAYS	VELOCITY CONSTRAINTS
Downcast Shafts	Maximum 12 m/s
Intake Haulages	Maximum 7 m/s
Return Airways	Maximum 10 m/s
Upcast Shafts	Maximum 20 m/s

TABLE 4.2
UPPER VELOCITY LIMITS FOR VARIOUS AIRWAYS

Resultant maximum permissible quantities at the prevailing density are shown in Table 4.3.

ITEM	AIRWAY	VELOCITY LIMIT (m/s)	DIMENSION (m)	NUMBER OFF	AREA (m^2)	QUANTITY (m^3/s)
A	EASTERN PRIMARY DOWNCAST	12	8,5 dia	1	56,7	680
B	13L STRATHMORE INTAKES	7	6 x 5	4	120	840
I	EASTERN SECONDARY DOWNCAST	12	8,5 dia	1	56,7	680
C	STRATHMORE SECONDARY DOWNCAST	12	9,0 dia	1	63,6	763
U	STRATHMORE UPCAST	20+	6,7 dia	1	35,2	705
F	23L STRATHMORE RETURNS	10+	VARIABLE	MULTIPLE	36	[redacted]
L	EASTERN UPCAST	20+	6,7 dia	1	35,2	705
N	SOUTHERN PRIMARY DOWNCAST	12	8,5 dia	1	56,7	680
O	14L INTERCONNECTION	7	6 x 5	4	120	840
P	SOUTHERN SECONDARY DOWNCAST	12	8,5 dia	1	56,7	680
S	SOUTHERN UPCAST	20+	5,5 dia	1	23,7	475

TABLE 4.3
AIR QUANTITY CONSTRAINTS BASED ON EXISTING AIRWAYS
AND VELOCITY LIMITS

The Multiple return airways between Strathmore and Eastern shaft at present are estimated to be 72 m^2. These airways from 20 to 26 level are SUSPECT WITH RESPECT TO THEIR INTEGRITY and must be discounted as future stable returns. Two airways however on 23L are sound, meshed and laced with a cross sectional area of 36 m^2.

4.1.3 Tertiary air requirements

A detailed examination of air requirements for ventilation and cooling of the Tertiary is shown in Sections 3.3 and 14. A design mass flow of 600 kg/s has been found to be adequate.

4.1.4 Conclusion

From the maximum permissible quantities shown in Table 4.3, the only constraint appears to be the Strathmore return airways.

4.2 Strathmore Return Airways

4.2.1 Quantification of 23L resistance

An attempt has been made to quantify the resistance of the two main components in the return configuration viz;

- Dedicated and Reinforced Airways on 23L, 23 - 52A Airways.
- Remaining return airways between 20 - 26L.

A list of the prime components between 20 and 26L are shown in Table 4.4.

LEVEL	AIRWAY	LENGTH m
20	20 HAULAGE WEST	800
21	21 HAULAGE EAST	2 000
	21 HAULAGE WEST	750
22	22 HAULAGE WEST	900
	22 - 55 HAULAGE	650
23	23 EX PIONEER	500
	23 - 59 HAULAGE WEST	550
	23 - 52A HAULAGES	2 300
	23 - 59 HAULAGE	3 000
	23 - 59 HAULAGE	800
24	24 - 60 AIRWAY	600
	24 - 50 HAULAGE EAST	2 100
	24 - 59 X/CUT	400
	24 - 52 TWIN HAULAGE	1 700
	24 - 52 HAULAGE WAST	2 400
	24 - 53 X/CUT	500
25	25 - 31 HAULAGE EAST	1 700
26	26 - 53 HAULAGE	750
	26 - 53 TWIN HAULAGE	1 500
	26 - 53 HAULAGE EAST	1 500

TABLE 4 4

RETURN AIRWAY CHARACTERISTICS FROM 20 TO 26 LEVEL BETWEEN THE STRATHMORE AND EASTERN SHAFTS

4.2.2 Pressure survey

In order to assess the component pressure losses, a pressure survey was undertaken. Details are shown in Appendix D.

As a result of the measurements taken during the survey in March 1989, the following conditions were found to be indicative of 23 - 52A haulage:

- measured pressure loss 880Pa
- air density 1,18kg/m³
- air quantity 108 m³/s

The measured R value is therefore:

$$\frac{P}{Q^2} = \frac{880}{(108)^2} = 0{,}0754 \ Ns^2/m^8 \text{ at a density of } 1{,}18 \ kg/m^3$$

A calculated, theoretical value of R is:

$$\frac{KCL}{A^3} \frac{W}{1{,}2} = \frac{0{,}0167 \times 18 \times 2\ 300}{(20)^3} \times \frac{1{,}18}{1{,}2} = 0{,}0850 \ Ns^2/m^8$$

where airway dimensions are:

4 m high, 5 m wide

2 300 m long

and $K = 0{,}0167\ Ns^2/m^4$

Assuming the differential of 880 Pa to be effective over the remaining airways, the present equivalent resistance of all these remaining returns is approximately:

$$R = \frac{880}{(250)^2} = 0{,}0141\ Ns^2/m^8$$

At present only one of the twin haulages of the 23 - 52A pair are being used as a return. When refurbishment in the form of lacing and grouting is complete both will be run in parallel.

The future air handling capacity of the return circuits is planned to be catered for primarily by 23 Level where two high integrity airways exist. The airways are simplified in Figure 4.2 and an air split based on a total mass flow of 600 kg/s shown.

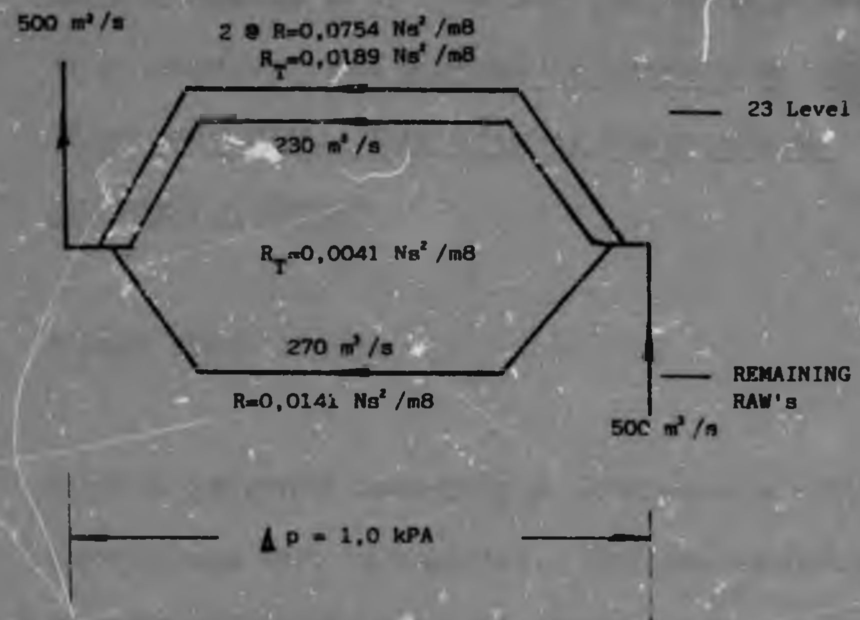


FIGURE 4.2

RESISTANCE LINE DIAGRAM OF RETURN AIRWAYS BETWEEN STRATHMORE AND EASTERN SHAFT

The total return system resistance is calculated to be:

$$R_T = 0{,}0041\ Ns^2/m^8 \text{ at } w = 1{,}18\ kg/m^3$$

At a design of 500 m^3/s the resultant pressure loss would be expected to be 1 kPa over the returns. This is an acceptable value provided the returns from 23 to 26 are maintained. This integrity however cannot be guaranteed in future.

4.2.3 Present status of returns

Prior to operations commencing at Strathmore, all the major airways were re-supported. This was necessary as these haulages are over and understoped except for the Twin Haulages 23 - 52A on 23 Level which are in virgin ground for most of the distance.

Over a period of five years, large portions of these airways have collapsed, causing severe restrictions. Of these portions, some have been refurbished where air temperatures allowed, and an ongoing programme has been instituted to maintain a competent ventilation circuit.

At present, a portion of these return airways are utilised to handle return air from Pioneer, Orangia and Southern Shafts. Frequent occurrences of fires on these shafts cause certain airways to be temporarily abandoned until they are clear of heat and smoke.

At best, the return airway resistance may be maintained at current levels, however, logistics and cost may preclude this strategy.

4.2.4 Return airway options

4.2.4.1 Alternative 1 MULTIPLE RETURNS IN SOLID GROUND ON 23L

Multiplicity of airways on 23L is expected to relieve the impending restrictions. Variations are summarised in Table 4.5 and shown diagrammatically in Figure 4.3, item (1). The effect multiple of parallel, high integrity airways is evident.

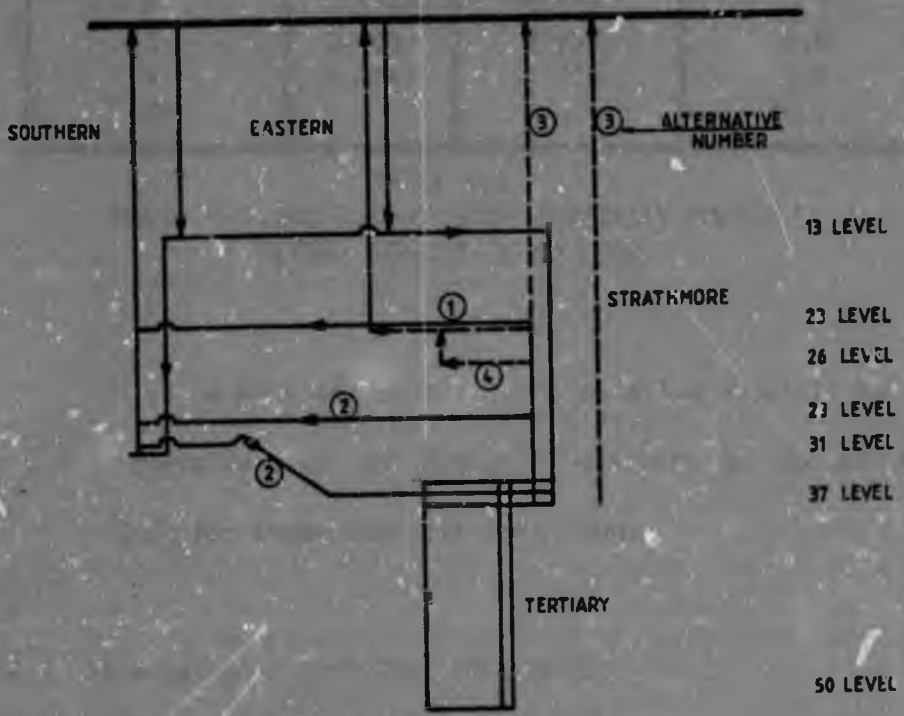


FIGURE 4.3

ALTERNATIVES TO RETURN AIR
FROM THE TERTIARY CIRCUIT

NUMBER OF MULTIPLE RETURNS	RESISTANCE $Ns^2/m8$	Δ P @ 500 m^3/s (Pa)	CAPITAL COST R MILLION
1	0,0754	-	-
2	0,0190	4 750	-
3	0,0084	2 100	3,5
4	0,0047	1 175	7,0
5*	-	-	10,5

TABLE 4.5
THE EFFECT AND COST OF HIGH INTEGRITY MULTIPLE RETURN AIRWAYS ON 23 LEVEL

Five returns may be advisable to allow flexibility in "changing over" an airway to maintain an SCP of 300 W/m^2 for inspection and refurbishing.

4.2.4.2 Alternative 2 SOUTHERN SHAFT UPCAST

The possibility exists to use 28/29L and 31/36L between Southern and Strathmore as a bypass to divert approximately 100 m^3/s to alleviate the load on 23L.

Depending on the rest of the Southern Ventilation commitment, the additional 100 m^3/s may or may not be handled by the Southern Shaft fans.

4.2.4.3 Alternative 3 SEPARATE UPCAST SHAFTS

Based on 6,7 m diameter shafts, two scenarios were considered. One, a dedicated upcast from 13L and the other from 37L. In both cases:

- Capital costs were prohibitive (R26 to 36 million).
- Time to completion was too long (3,8 to 5,2 years).
- Additional surface fans were required.

4.2.4.4 Alternative 4 RETURNS IN SOLID GROUND IN ADDITION TO 23L

Additional multiple airways on 23L constitude a problem of cross-overs and approach to the shaft. Portions of haulages have already been developed from the Eastern upcast on 23L with haulages starting at the Strathmore upcast on 24 and 26L in solid ground. Vertical interconnections can be made between levels through 2 m diameter raise bored holes. Diagramatically the network is shown in Figure 4.4 and a circuit resistance is estimated to be 0,0045 Ns^2/m^8. This will handle the total design mass flow of 600 kg/s (500 m^3/s) at a pressure loss of 1 125 Pa.

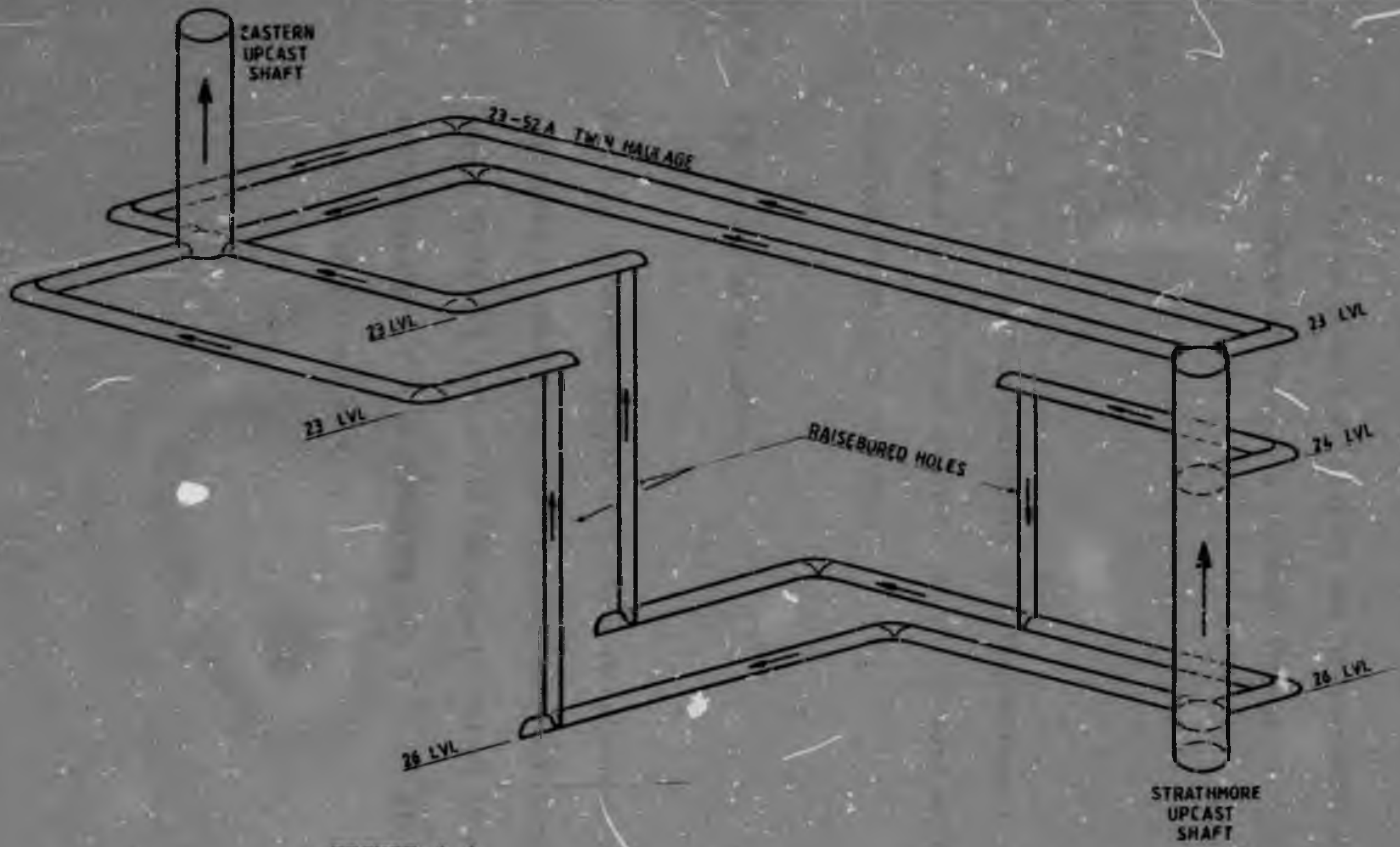


FIGURE 4.4

ISOMETRIC VIEW OF USING EXISTING AIRWAYS IN SOLID GROUND AND RAISEBORE HOLES

Cost of development is R7,6 million depending on the:

- Logistics of airway location and interconnection.
- The amount of re-usable existing airways.

4.2.4.5 Conclusion

Of all alternatives, elements of Alternatives 1 and 4 are most likely to be adopted. In either case extensive development will be required.

4.2.4.6 23L Airway Optimization

Based on an airway length of 2 300 m an optimization was performed to verify the most economical cross sectional area. Table 4.6 summarizes the results and an airway of approximately 4 x 4 m conveying 125 m^3/s of air at a velocity of 7,8 m/s is best.

AIRWAY DIMENSION	AIRWAY AREA	R Ns²/m8	Q (m³/s)	V (m/s)	Δp (kPa)	POWER COST (Rm)	DIST (m)	DEV COST (Rm)	TOTAL COST (Rm)
3 x 3	9,0	0,530	125	13,9	5,3	4,15	2300	2,25	6,40
3,3 x 3,3	11,1	0,337	125	11,3	3,4	2,43	2300	2,78	5,21
3 x 4	12,0	0,261	125	10,4	2,6	2,05	2300	3,00	5,05
4 x 4	16,0	0,126	125	7,8	1,3	0,98	2300	4,00	[redacted]
5 x 4	20,0	0,075	125	6,3	0,7	0,79	2300	5,00	5,79
5 x 5	25,0	0,041	125	5,0	0,4	0,65	2300	6,90	5,90

TABLE 4.6
OPTIMUM RETURN AIRWAY DIMENSIONS

5. SYSTEM RESISTANCE

5.1 Existing Mine System Resistance

In view of the critical dependence of the Tertiary air supply via the existing infrastructure and modifications thereof, an understanding of the physical interaction of the ventilation districs and the aerodynamic characteristics of the major airways was necessary. Manual calculations of airway resistance were verified by computer simulation and a wide variety of dynamic ventilation scenarios predicted.

5.1.1 Quantifying the existing mine system resistance

5.1.1.1 Calculations using approximate physical conditions

A simplified mine circuit is shown in Figure 4.1, and serves as framework to approximate the mine's air reticulation system and as a guide to quantify pressure drops in the network.

Physical characteristics of the major airways are listed in Table 5.1 which allows a theoretical estimate of pressure losses to be made. With well approximated flow rates of air in the circuit, the results in Table 5.1 give a mine resistance of 5,0 kPa.

Static pressures at the Eastern upcast fans have been known to vary from 4,5 to 5,0 kPa. Ventilation changeover's and seasonal density variations are primarily responsible for this spread. This first approximation of a simplified model appears therefore to be representative.

AIRWAY	CALCULATED R $(Ns^2/m8)$	AIR QUANTITY (m^3/s)	ESTIMATED PRESSURE LOSS (Pa)
EASTERN PRIMARY DOWNCAST	0,0020	645	830
13 LEVEL MAIN INTAKES	0,0032	412	540
STRATHMORE DOWNCAST	0,0013	412	220
STRATHMORE WORKINGS	0,0074	366	990
STRATHMORE UPCAST	0,0014	366	200
STRATHMORE RETURNS	0,0041	413	700
EASTERN UPCAST	0,0031	700	1 500
			5 000

TABLE 5.1

PHYSICAL CHARACTERISTICS AND CALCULATED PRESSURE DROPS FOR EXISTING MAIN AIRWAYS

5.1.1.2 Main fan performance measurements

Arising from the pressure survey conducted, an average fan drift to atmosphere pressure differential at the Eastern fans of 4,3 kPa was measured. The two pressure measuring instruments used were an aneroid Fuess Mico Barometer and an Electronic Pressure Transducer, SURVBAR (Dumka, 1987).

On a separate occasion, suppliers of the fans were requested to verify their performance characteristics. Some doubt had arisen to the operating point of the fans and confirmation of the actual duties was imperative.

A complete set of measurements were taken and the fan characteristics corroborated. Results of the investigation are detailed in Appendix E, with a summary in Table 5.2.

	FAN NO. 1	FAN NO.2
FAN AIR QUANTITY m^3/s	471	471
FAN STATIC PRESSURE Pa	4 392	4 490

TABLE 5.2

EASTERN SHAFT FAN STATIC PRESSURE AND AIR QUANTITY RESULTS ARISING FROM A PERFORMANCE TEST

Operating points of both fans although on a very low mine characteristic do lie exactly on the manufacturers curve at an inlet guidevane setting of 0°. (APPENDIX E5)

5.1.2 Computer modelling

In order to further, more accurately approximate the Ventilation flow system, and ultimately the complex network, use was made of a Chamber of Mines Computer simulation: "VENTFLOW" (COMRO Application No. 10)

"VENTFLOW" is an interactive programme which allows the originator to:

(a) Create a realistic mine flowsheet of nodes and airways using the best information available. Physical characteristics of airways can be specified, supplemented by best estimated K-factors. Actual pressure drops and mass flows can be introduced which arise from pressure and quantity survey measurements on the mine.

(b) Solve the network or model thereby establishing self-balancing mass flows dependent on airway geometry.

(c) Enter fan data, for main surface fans or boosters and the operating point of each fan installation will be determined.

The model, once proven reliable, can be used to predict airflow pattern changes when:

- Extensions to the network are made.
- Elimination of redundant workings is undertaken.
- Additional upcast or downcast shafts are added.
- Booster fans in the circuit are supplemented.

Development of such a model enables the ventilation phasing-out flowrate changes to be monitored within the as well as the effects in the tertiary mining block.

5.1.2.1 Model creation and reliability test

The "CURRENT" mine model began with an over simplification of the mine described in Section 4.1.1. By incorporating more mining districts and circuit interactions with other primary networks, a realistic airflow configuration evolved.

The present Eastern, Strathmore and Southern mine complex is emulated by a network called "CURRENT" and comprises of:

- 38 branches;
- 34 nodes;
- 2 fan installations, and
- 8 fixed flow airways

The model is based on the mines airflow accounting schematic Figure 5.1 and interacts the existing workings and associated ventilation districts.

The model "CURRENT" incorporates best known physical characteristics for primary arteries which are listed in Tables 5.3 and 5.4 and a minimum number of fixed flow or regulated branches are used. This allows air networks to self-balance more realistically. Assumptions were made to estimate pressure drops in the South, East and Strathmore workings, which are representative of current mining procedures. An equivalent R value for conventional workings is assumed to be 0,0071 Ns^2/m^8.

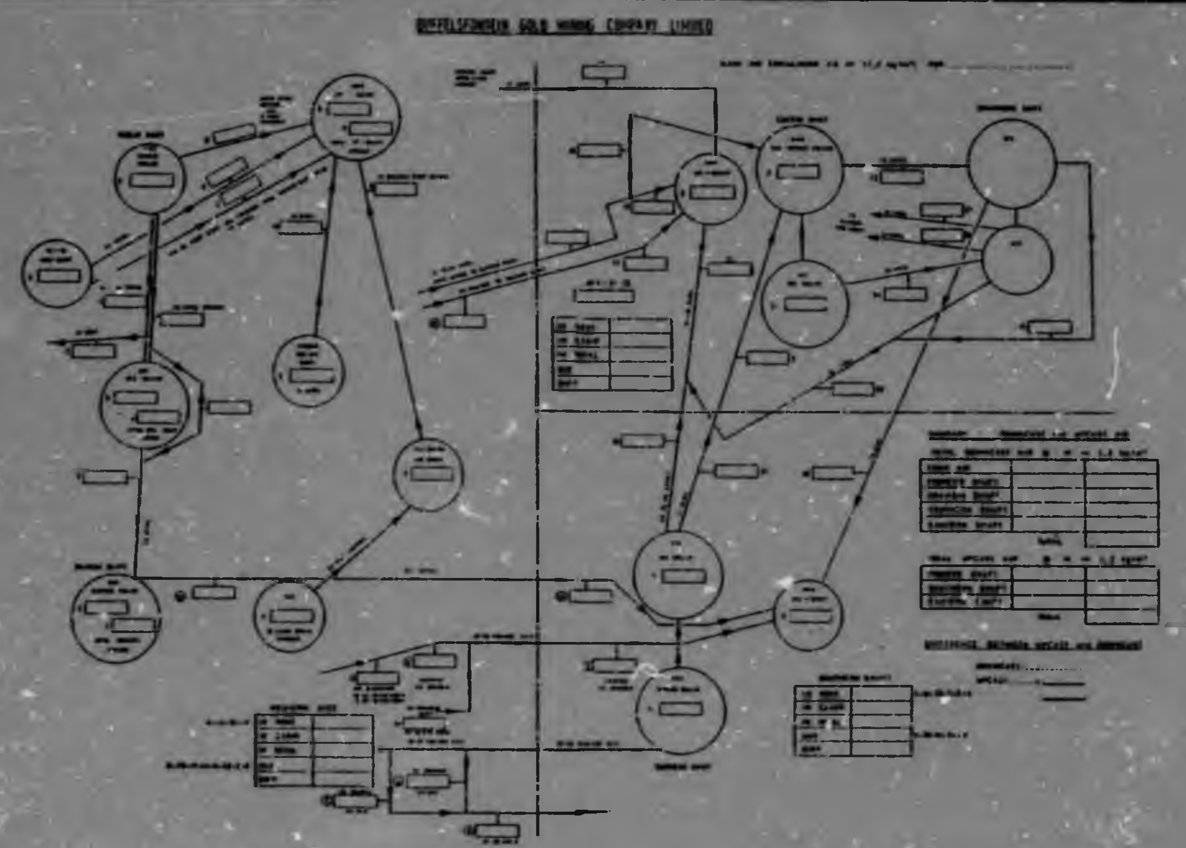


FIGURE 5.1

BUFFELSFONTEIN UPCAST AND DOWNCAST AIR SUMMARY

SHAFT	DIAMETER (m)	LENGTH (m)	ATKINSON K FACTOR (Ns^2/m^4)
EASTERN:			
Primary Downcast	8,5	1 619	0,0065
Primary Upcast	6,7	2 187	0,0030
Secondary Downcast	8,5	866	0,0065
STRATHMORE:			
Secondary Downcast	9,0	1 751	0,0065
Secondary Upcast	6,7	1 000	0,0030
SOUTHERN:			
Primary Downcast	8,5	1 565	0,0065
Primary Upcast	8,5	2 744	0,0030
Secondary Downcast	8,5	1 361	0,0065

TABLE 5.3

EXISTING SHAFT PHYSICAL CHARACTERISTICS

AIRWAY	DIAMENSION (m)	LENGTH (m)	ATKINSON K FACTOR (Ns^2/m^4)
EASTERN TO STRATHMORE INTAKE 13L	4 off 6 x 5	1 900	0,0167
STRATHMORE TO EASTERN RAWS 20 - 26L	multiple 4 x 4	500 to 2000m	0,0167

TABLE 5.4

EXISTING AIRWAY PHYSICAL CHARACTERISTICS

Figure 5.2 is a cross sectional line diagram of the major interactive airways used in the VENTFLOW analysis. Free flow airways, regulated airways, nodes and fans are numbered and relate to the computer print out's in Appendix F.

The mass flow predictions from "CURRENT" are shown in Figure 5.3 and air gains into the Southern, Eastern and Strathmore geometry appear as input arrows, i.e. "ex Orangia".

The predicted fan operating points of the model "CURRENT" can be compared to actual fan operating characteristics shown in Table 5.5.

OUTPUT	EASTERN UPCAST FANS	SOUTHERN UPCAST FANS
AIRFLOW (m^3/s)		
- ACTUAL	942	657
- PREDICTED "CURRENT"	932	620
FAN STATIC PRESSURE (kPa)		
- ACTUAL	4,4	5,1
- PREDICTED "CURRENT"	4,4	5,5

TABLE 5.5

A COMPARISON OF ACTUAL AND PREDICTED FAN OPERATING POINTS

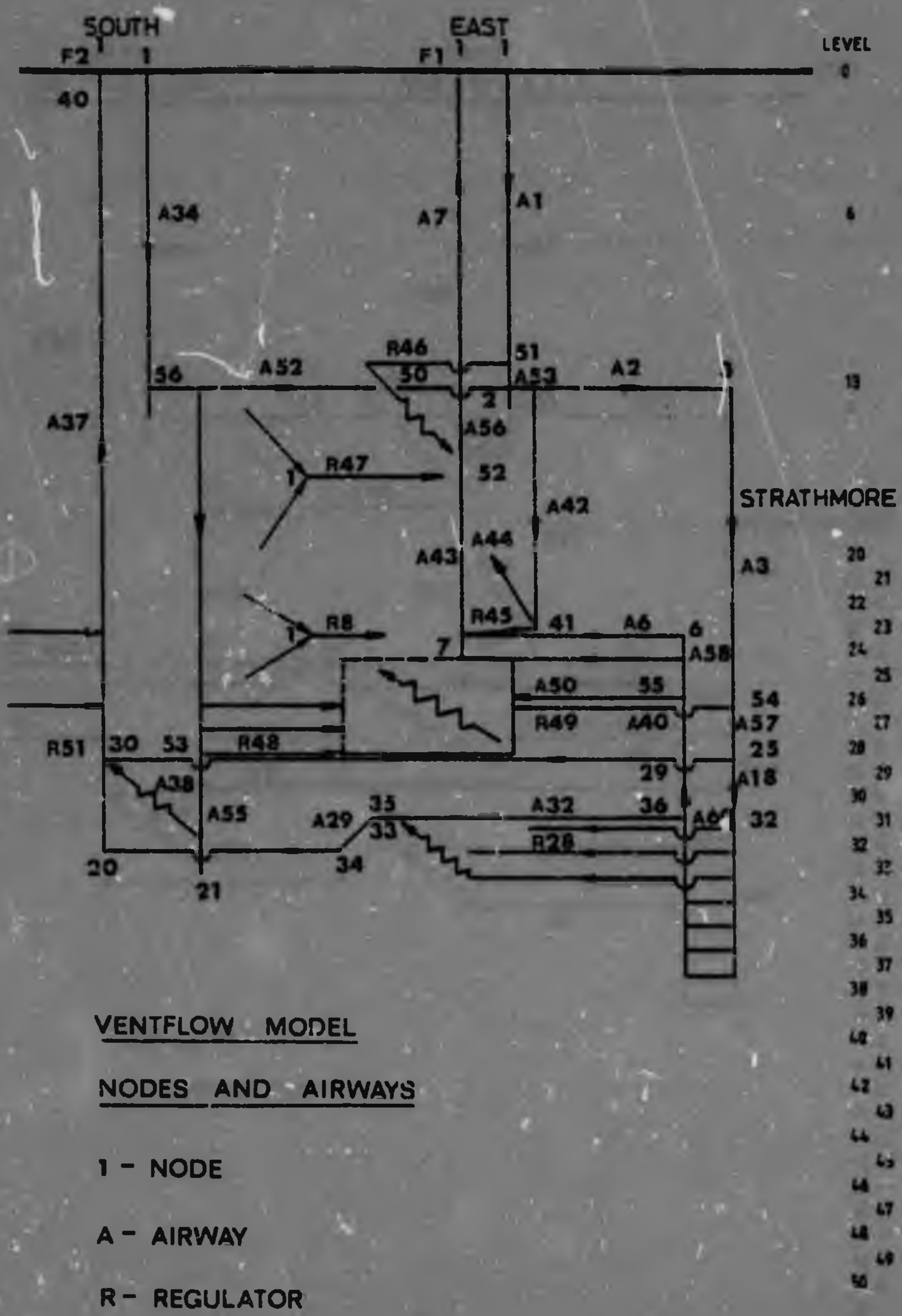


FIGURE 5.2

VENTFLOW MODEL NODES AND AIRWAYS

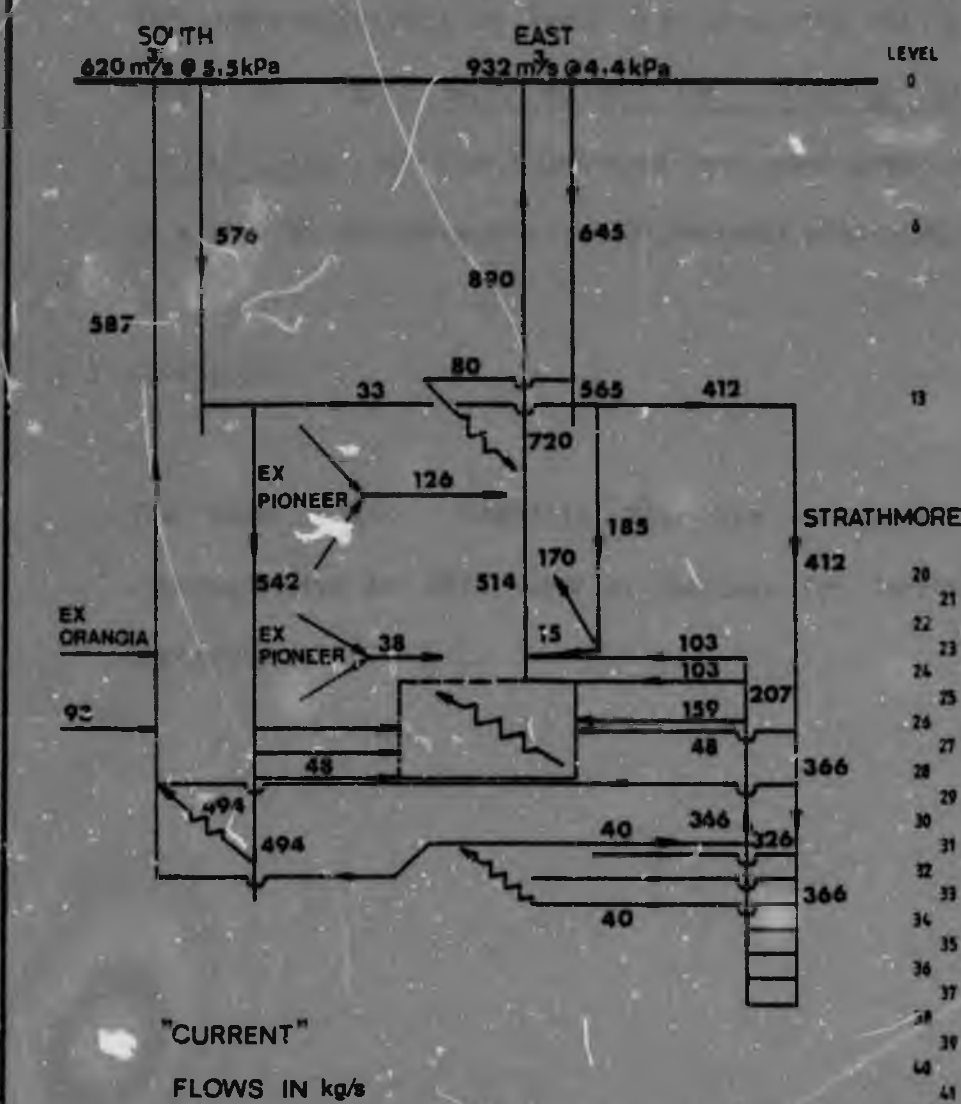


FIGURE 5.3

PREDICTED AIRFLOWS IN THE EXISTING BUFFELSFONTEIN VENTILATION DISTRICTS

The comparative data in Table 5.5 shows the VENTFLOW model to have approximated the existing mine realistically. Airflow quantities have been predicted to within 5% and operating system pressures within 8%.

5.1.2.2 Conclusion

The base model "CURRENT" is therefore accepted as representative and will serve as the base for Tertiary prediction.

6. TERTIARY SHAFTS

With the validity of the existing Buffelsfontein air circulation model having been verified, the primary Tertiary components which require qualification and quantification are:

- downcast and upcast shafts;
- main intake and return airways;
- developing and stoping programmes, and
- computer modelling of Tertiary coupling.

6.1 Selection of the Shaft System

The rate of ore reserve development is largely influenced by the fastest method possible of establishing these reserves. For this reason various options were investigated to determine the most efficient, cost effective and timely method of exploiting the ore reserves:

(a) Lengthening of Strathmore Secondary Downcast Shaft: This proposition was discounted for the following reasons. The Strathmore Secondary shaft has already 10 stations and is 1 800 m long. The winder's design capacity is rated for this length and a deepened shaft would result in an additional 1 000 m of shaft and an additional 10 stations. A 10 station shaft of 2 800 m length is considered impractical. Production at Strathmore Secondary shaft would be affected considerably.

(b) Tertiary Incline Shaft: Due to the vertical depth of approximately 1 000 m, and an incline gradient of 15°, the total length of incline will be 3 480 m. This length of shaft is considerable and the large number of stations would also restrict the capacity . In addition to this, ventilation requirements due to the heat and depth are excessive and at least six inclines would be necessary. Four intake inclines and two return.

The reason for this large number of inclines is due to the reduced air velocity requirement compared to vertical shafts. This system was discarded due to the technical difficulties and the excessive development that would be required.

(c) Vertical Shaft System: Access to the Tertiary shaft position is available on the 35, 36 and 37 levels of Strathmore Secondary Shaft enabling development of the hoist rooms, rope raises, airways, belt levels and loading boxes/headgear to be done.

The estimated shaft wind would be 1 200 m, being approximately 4 200 m below surface.

A vertical shaft system was favoured for the following reasons:

- Proximity of the Tertiary Shaft to the secondary shaft eliminates the necessity for large scale development of airways between the two systems.

- A maximum travelling distance of 3 000 m is estimated from the new shaft to the reef.

- Travelling and haulage time between the Secondary and Tertiary shaft is minimal due to its proximity.

6.2 Vertical Shaft Systems

The vertical shaft system would have to be planned to comply with the following requirements:

- The necessity to complete the project timeously so as to replace existing mining areas.

- Provide all services namely downcast ventilation, refrigeration and electrical supply.

- Providing men/material hoisting.

- Allow for a second escape way.

- Be sized to suit both logistical (engineering) and ventilation requirements.

- Be sited in known geological rock formations.

- Be equipped so that all services be compatible with the capabilities of the Eastern shaft and the Strathmore Secondary shafts.

[illegible]3 Tertiary Shaft Position

Various positions for the shaft were considered. These options were:

1. The shaft centrally positioned in the block to mine;

2. The shaft positioned to the north in the Strathmore Eastern block, and

3. The shaft positioned to the west close to the Strathmore secondary shafts.

6.3.1 Option 1

With the shaft in this position an unacceptably large shaft pillar results which ties up ± 1,4 million m² of ore reserve. This pillar size is required in order to keep induced stresses and strains within the shaft to within acceptable limits.

6.3.2 Option 2

With the shaft positioned to the north, the Strathmore Eastern block will be tied up as a shaft pillar. This is approximately 500 000 m² of ore reserves.

6.3.3 Option 3

The preferred site for the Tertiary shafts is west of the block to be mined, in close proximity to the present Strathmore Secondary Shaft system. The main advantages of this location are:

1. No shaft pillar is required due to its position in a large fault loss.

2. A full length hole has been drilled close to the proposed site and the geology of the area is known.

3. Existing development on 35, 36 and 37 level the site can be quickly accessed and accellerates the start date.

4. Since Tertiary ventilation and refrigeration services will be intimately linked to Strathmore, the shortest distance between is preferred.

5. Centralization of maintenance and logistics of ore handling is an important consideration.

6.3.4 Conclusion

The proposal therefore is to sink Tertiary shafts close to the existing Strathmore Secondary shaft system.

Figure 6.1 shows the relative postion of the Tertiary Shafts.

FIGURE 6.1

RELATIVE POSITION OF TERTIARY SHAFTS

6.4 Number and Size of Tertiary Shafts

6.4.1 Minimum number of shafts

As a result of the Tertiary shaft positions they require to be sunk to slightly below 50 Level to a depth 1 162 m below 35 level collar. Since a minimum of two downcasts are required for second escape, the following scenarios were considered.:

1. Two discrete downcast shafts plus a dedicated upcast shaft, and

2. One dedicated downcast shaft plus a split upcast/downcast shaft.

6.4.2 Shaft optimization

NOTE:

Calculations shown, optimize upcast and downcast shafts exclusively in the ventilation context and are strictly therefore an economic optimum.

This implys the capital cost of the shaft and fan running costs are optimized in isolation of any other discipline.

6.4.2.1 Design criteria

The parameters shown in Table 6.1 were used to calculate optimum shaft diameters.

	UPCAST	DOWNCAST
SINKING A EQUIPPING COST	AS PER TABULATION - SOURCED O V.D. WALT AND B BLUHM	
K FACTOR	0,003 Ns^2/m^4	0,0065 Ns^2/m^4
SHAFT DEPTH	1 000 m	1 000 m
AIR QUANTITY	414 m^3/s	414 m^3/s AND 207 m^3/s
LIMITING VELOCITY	>17 m/s	≤12 m/s
MEAN AIR DENSITY	1,45 kg/m^3	1,45 kg/m^3

TABLE 6.1

PARAMETERS FOR SHAFT OPTIMIZATION CALCULATIONS

SHAFT Ø (METRES)	COST TO SINK & EQUIP (R MILLION)	Δ P (PASCALS)	kW POWER (kW)	POWER COST OVER 10 YRS (R MILLION)	OWING COST (R MILLION)
4,0	8	26 090	14 402	50,4	58,4
4,5	12	14 530	8 021	28,1	40,1
5,0	15	8 635	4 767	16,7	31,7
5,5	18	5 315	2 934	10,3	[redacted]
6,0	22	3 435	1 896	6,6	28,6
6,5	25	2 310	1 275	4,5	29,5
7,0	28	1 600	883	3,1	31,1

TABLE 6.2

OPTIMUM SHAFT DIAMETER FOR A SINGLE EQUIPPED DOWNCAST SHAFT

DOWNCAST SHAFTS (2 OFF) - EQUIPPED

SHAFT Ø (METRES)	COST TO SINK AND EQUIPMENT (R MILLION)	Δ P (PASCALS)	kW POWER (kW)	POWER COST OVER 10 YRS (R MILLION)	OWING COST (R MILLION)
4,0	8	6 523	3 600	12,6	20,6
4,5	12	3 632	2 004	7,0	[redacted]
5,0	15	2 159	1 192	4,2	19,2
5,5	18	1 329	734	2,6	20,6
6,0	22	859	474	1,7	23,7
6,5	25	577	318	1,1	26,1
7,0	28	399	220	0,8	28,8

TABLE 6.3

OPTIMUM SHAFT DIAMETERS FOR TWIN EQUIPPED DOWNCAST SHAFTS

UPCAST SHAFT

SHAFT Ø (METRES)	COST TO SINK (R MILLION)	Δ P (PASCALS)	KW POWER (kW)	POWER COST FOR 10 YRS (R MILLION)	OWING COST (R MILLION)
4,0	5,4	5 218	2 880	10	15,4
4,5	6,4	2 906	1 604	5,6	12,0
5,0	7,4	1 727	953	3,3	[redacted]
5,5	8,6	1 063	587	2,1	[redacted]
6,0	10,0	687	379	1,3	11,3
6,5	11,7	462	255	0,9	12,6
7,0	13,3	320	177	0,6	13,9

TABLE 6.4

OPTIMUM SHAFT DIAMETER FOR A SINGLE UPCAST SHAFT

6.4.3 Alternatives based on optimized calculations

6.4.3.1 Case 1

The optimization shows each of the two downcasts to be 4,5 m diameter and an upcast of between 5,0 m and 5,5 m diameter is required.

6.4.3.2 Case 2

The optimum dedicated downcast remains 4,5 m diameter. The split shaft will require an ecivalent hydraulic diameter of 4,5 m or 16 m^2 for the downcast and 5,0 m diameter, 20 m^2 for the upcast. The final theoretical overall diameter of the split shaft is expected to be 6,8 m to accomodate a brattice wall, piping and conveyances.

6.4.3.3 Cost implications

Estimates based on Cost Engineering, show the two (case 1) and three (case 2) shaft options are within 5 %. The two shaft system requires less station development and less development time.

6.4.3.4 Other considerations

Based entirely on Ventilation requirements the optimum shaft diameters would be:

- One equipped dedicated downcast 4,5 m diameter.
- One bratticed shaft 6,8 m diameter overall.

Hoisting capacities and rolling stock standardization however, require shafts of a slightly larger dimension.

6.4.3.5 Conclusion

Due to other factors therefore, the final shaft diameters are:

- One equipped downcast shaft 6,8 m diameter, and
- one bratticed shaft 7,8 m diameter with:
- equipped downcast of 27,8 m^2 area
- upcast compartment of 20 m^2

6.5 Shaft Air Velocities

Assuming the air split to be evenly regulated to both downcast shafts, worst case air velocites based on maximum air quantity and minimum free area were calculated. Results are shown in Table 6.5 and the hignest velocity expected is:

$$= \frac{215\ m^3/s \text{ (maximum quantity)}}{14,8\ m^2 \text{ (minimum free area with both conveyances at the same elevation)}}$$

$$= 14,5\ m/s$$

This occurrance is of short duration and "Average" downcast conditions are well within the maximum design velocity of 7 m/s.

SHAFT	DIAMETER m	AREA m^2	FREE AREA m^2	HOISTING OBSTRUCTION FREE AREA m^2	MAXIMUM AIR QUANTITY m^3/s	AVERAGE AIR QUANTITY m^3/s	MAXIMUM VELOCITY m/s	AVERAGE VELOCITY m/s
DEDICATED DOWNCAST	6,8	36,3	32,7	CAGE 27,5 SKIP 30,5 BOTH 25,3	215	107	8,5	3,3
SPLIT DOWNCAST	7,8	27,8	24,3	CAGE1 18,8 CAGE2 20,3 BOTH 14,8	215	107	14,5	4,4
SPLIT UPCAST	7,8	20	20	20	430	215	21,5	NA

TABLE 6.5

TERTIARY SHAFT AIR VELOCITIES

6.6 Shaft Sinking

Ventilation for shaft sinking and station devlopment will incorporate both force and exhaust overlap principles. The four-gate ventilation system due to its complexity was not considered. In most cases force ventilation is adopted due to the larger usefull air quantity on the face. Multiblast plugs will be permitted.

By complying to the Mines and Works Act regulation 10.10.1, multiblast sinking would be subject to certain conditions which include but may not be restricted to:

- The intake air to this development cannot be contaminated by blasting from other working places.

- The fumes and dust from blasting operations are led directly into return airways to the upcast shaft in such a way that they cannot contaminate the air of other working places.

- The quantity of air supplied to each end being developed shall not be less than 5 m^3/s.

- Standby diesel units, which shall be tested daily and kept in working order, shall be available to operate the fans required to force air down the shaft and circulate air over the bank area, in case of a failure to the power supply.

- A scheduled person whose duty it shall be to ensure that the fans operate properly, shall be appointed in writing and be in attendance at the fans during the re-entry interval.

- Respirators that are effective against nitrous fumes shall be supplied to and used by persons travelling through the plug of smoke or fumes after each blast.

- Monthly returns of the nitrous fumes and the carbon monoxide content of the air in addition to dust counts at the movable stage 30 minutes after the blast and the distance of the ventilation column from the advancing face, shall be furnished.

6.6.1 Sinking ventilation and cooling requirements

Sinking ventilation requirements are based on:

Re entry	:	10 minutes
Air changes	:	5
Stage height	:	30 m

which results in:

6,8 diameter = 9,1 m^3/s

7,8 diameter = 11,9 m^3/s

A recommended air quantity (Environmental Engineering: 282) based on 0,25 m/s velocity per square metre results in:

6,8 diameter = 9,1 m^3/s

7,8 diameter = 12,0 m^3/s

and compliance will be maintained if 10 m^3/s and 15 m^3/s are allowed for shaft bottom ventilation.

Cooling capacity of the spray chamber of 1 500 kW is sized to counteract the heat input by the sinking fans. Air quality entering the cooler on 37 Level will be approximately 21,7°/23,5°C.

The spray chamber will condition air to arrive at shaft bottom at a maximum of 26°/27°C and up to 400 m into developing ends at 29°/32°C. If further penetration is necessary during development either more cooling will be needed at the spray chamber or secondary cooling coils may be used.

Station cutting operations and development will be allocated 7 m^3/s per end. An air quantity of 50 m^3/s per shaft will be entrained on 37 Level a leakage of 20% results in 40 m^3/s for ventilation of shaft bottom and level development.

In the both cases, 4 ends can concurrently be developed in conjunction with shaft sinking.

6.6.2 Shaft sinking ventilation configuration

Combined force fans and sinking sprays shown in Figure 6.2 will be established on 37 Level. The arrangement will include:

SINKING FANS: 4 x 25 m^3/s, 11kPa @ 1,4 kg/m^3;

DIESEL STANDBY: 1 x 25 m^3/s, 11 kPa @ 1,4 kg/m^3, and

SINKING SPRAYS: 1 x Spray chamber, 1 500 kW.

SINKING COLUMNS: 1067 mm diameter; 7,8 shaft 3 off; 6,8 shaft 2 off

6.6.3 Upper level development

In order to access the orebody with minimum delay an exhaust change-over will be effected. A main booster fan will be connected and run at partial load to ventilate twin developments on 42 to 46 level. A proposed configuration is shown in Figure 6.3.

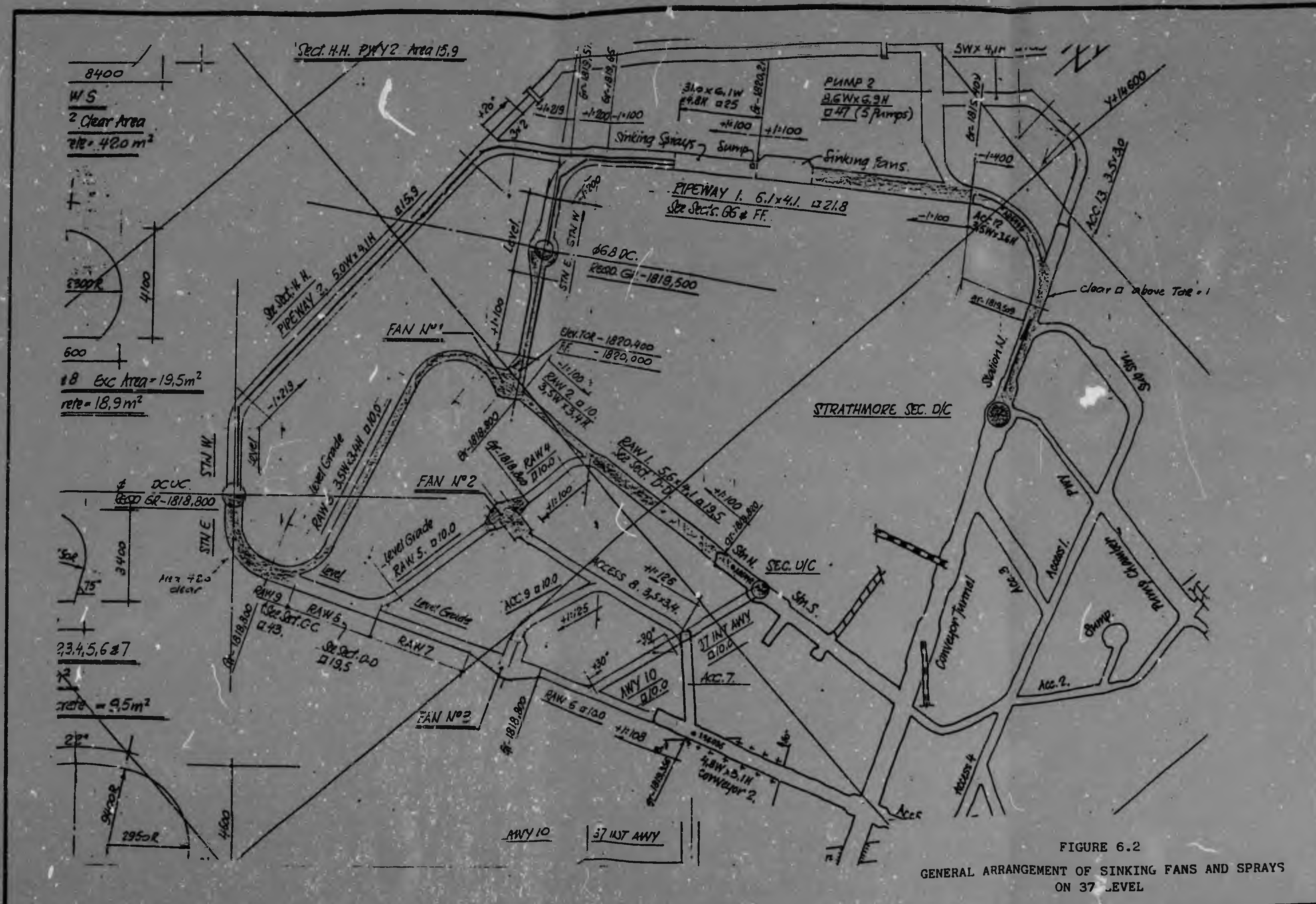


FIGURE 6.2
GENERAL ARRANGEMENT OF SINKING FANS AND SPRAYS ON 37 LEVEL

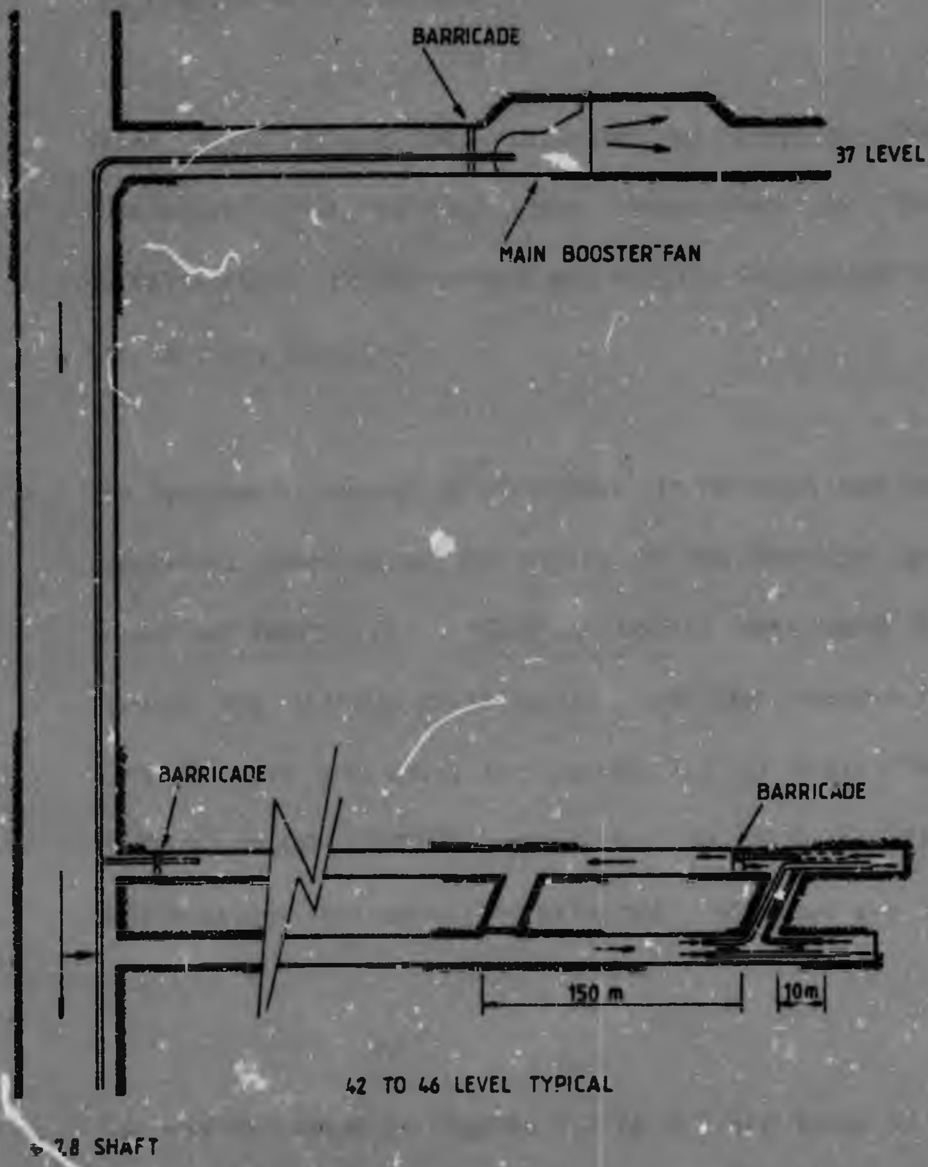


FIGURE 6.3

EXHAUST VENTILATION FOR TWIN DEVELOPMENTS

7. TERTIARY COUPLING TO THE EXISTING BUFFELSFONTEIN NETWORK

7.1 Ventilation Infrastructure

Three major phases are expected to complete the changeover from turning down operations in the Buffelsfontein infrastructure and merging the mining of the Tertiary block.

The systematic removal of redundant air networks and the concurrent development and mining in the Tertiary are shown in Table 7.1. VENTFLOW models were used to predict the airflow distributions and the changes in each scenario are shown in Figures 7.1 to 7.3. The VENTFLOW model "CURRENT" was used as a base with modifications systematically effected. Airflows are in kg/s.

The airflows shown in Figures 7.1 to 7.3 are based on a Tertiary resistance of $R = 0{,}0076\ Ns^2/m^8$ (very little backfill) and illustrate the relative shift of air mass flows in phases 1 to 3.

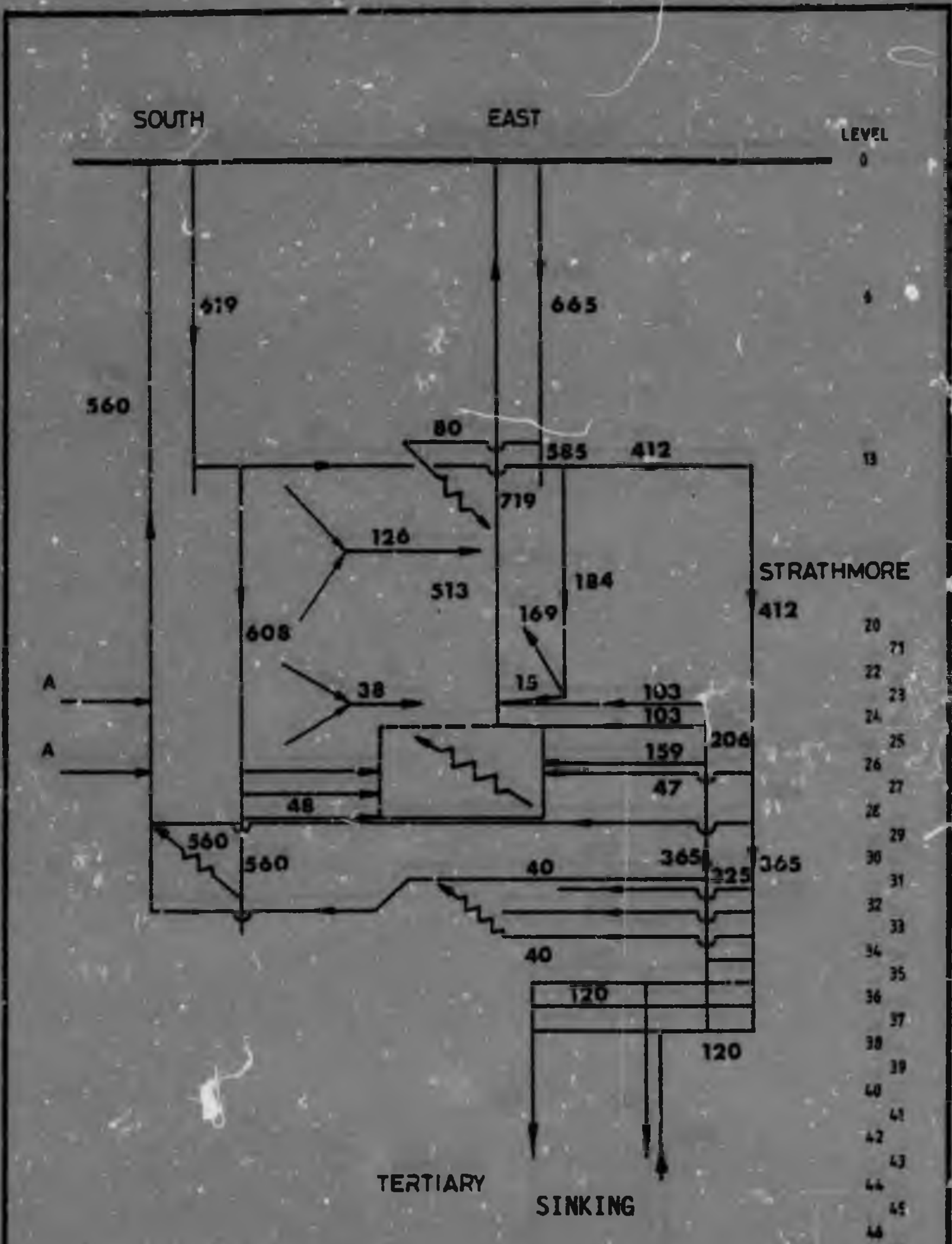


FIGURE 7.1
TERTIARY AIRFLOW, PHASE 1

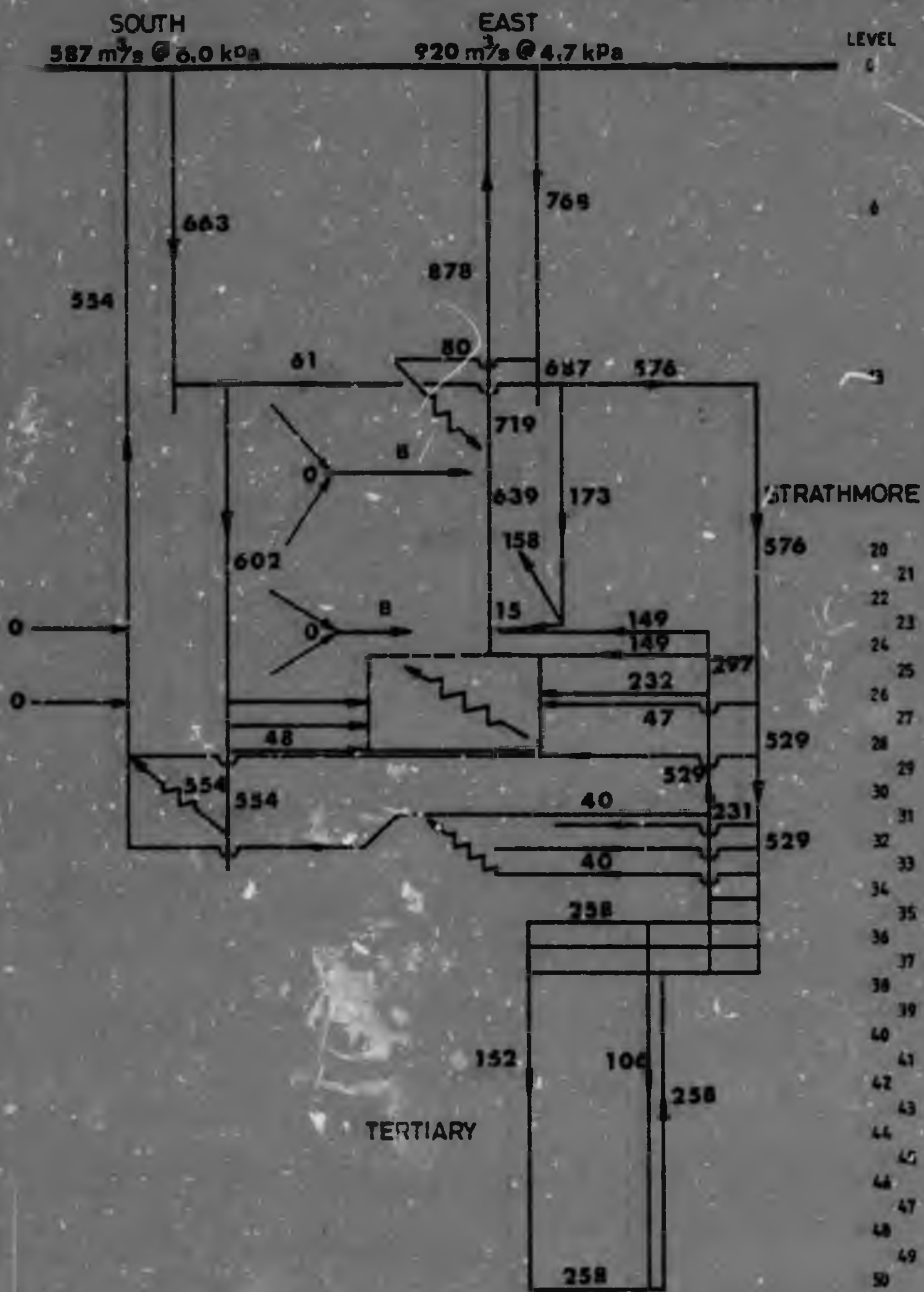


FIGURE 7.2

TERTIARY AIRFLOW, PHASE 2

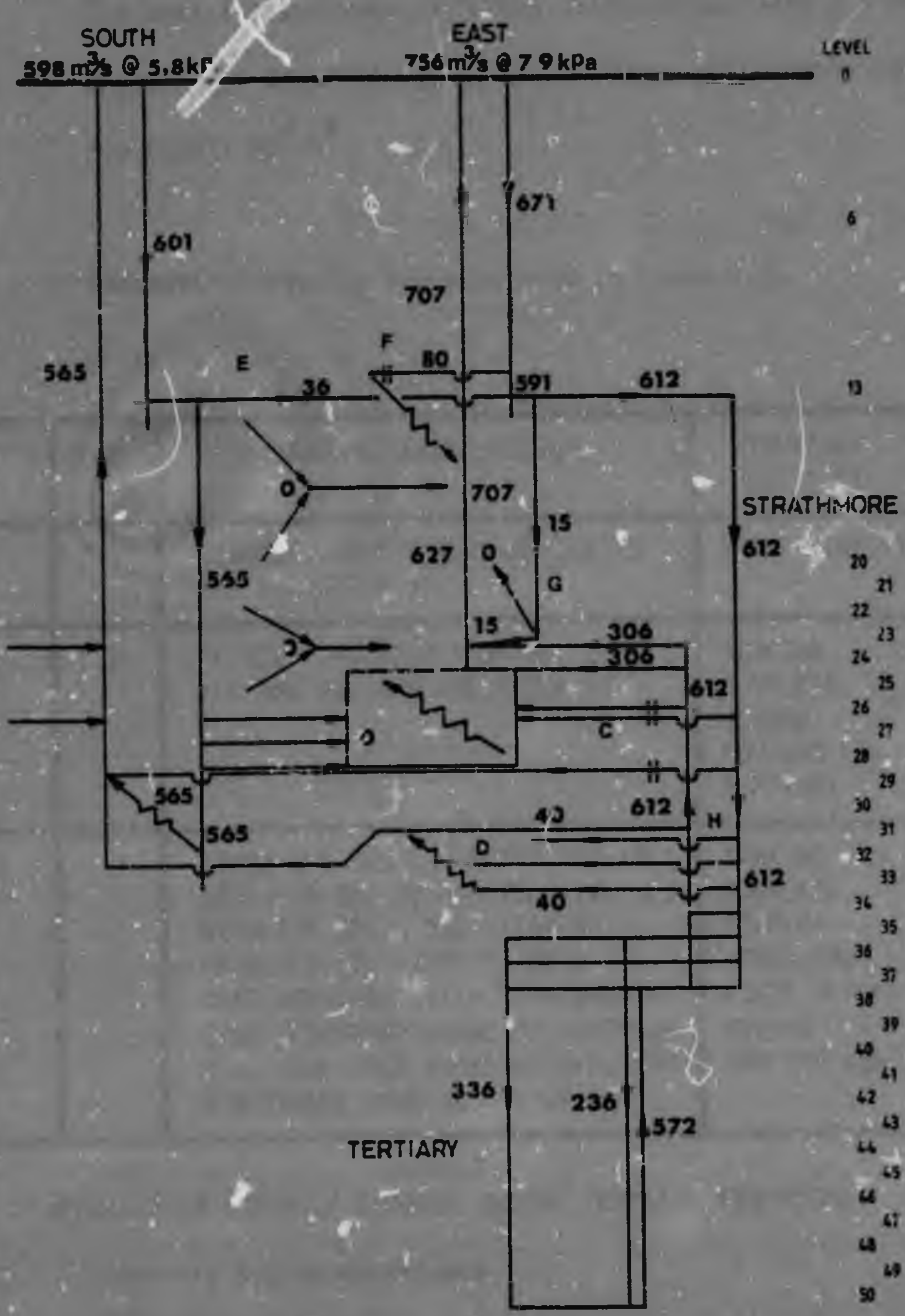


FIGURE 7.3

TERTIARY AIRFLOW, PHASE 3

The same analysis was performed using stopes with a high degree of backfill giving Tertiary resistances of $R = 0{,}0517\ Ns^2/m^8$.

Comparative results are tabulated in Table 7.3.

PHASE	FIGURE	AIRFLOW TERMINATIONS	TERTIARY ACTIVITY
1.	7.1	ORANGIA SHAFT SEALED OFF ON 23 AND 26L (ITEM B)	START SINKING
2.	7.2	PIONEER SEALED OFF ON 6L, 13 - 20L AND 23L HAULAGE (ITEM B)	SINKING COMPLETE MINING 100 000 tpm TERTIARY
3.	7.3	SOUTH AND EASTERN SHAFT ISOLATED: SEALS ON 26, 27 AND 28L (ITEM C) REGULATE 32L + 33L (ITEM D) REGULATE 13L (ITEM E) SEAL 12L EAST WORKINGS (ITEM F) REGULATE EAST SECONDARY DOWNCAST (ITEM G) i.e. SEAL EAST WORKINGS SEAL OFF STRATHMORE WORKINGS (ITEM H)	SINKING COMPLETE DEVELOP UPPER AND LOWER BLOCKS MINING 180 000 tpm

TABLE 7.1
VENTILATION DISTRICT CHANGES DURING TERTIARY EVOLUTION

7.2 Tertiary System Resistance

Due to the unusual mining method and the implications of backfill a better estimate of system resistance is required to assess the implications on air circulation and system pressure.

A simplified stope air circuit is shown in Figure 7.4. Based on an Atkinson resistance factor K face = 0,035 $Ns^2/m4$ due to high face irregularity and K strike = 0,020 $Ns^2/m4$ the resistance values are calculated to be:

R strike = 0,0556 Ns^2/m^8

R face = 0,7327 Ns^2/m^8

R circuit = 0,8439 Ns^2/m^8

At a design face air quantity of 15 m^3/s the expected pressure drop is 190 Pa for a single stoping line extending one level.

Published data from other backfilled stopes studies (solid line), Figure 7.5 shows pressure drops of a similar magnitude (Matthews, SAIMM 1988).

The stope resistance and subsequent pressure drop at close fill to face distances for the Tertiary (dotted line) can be expected to be high, and an approximation is two times higher than conventional unfilled stopes.

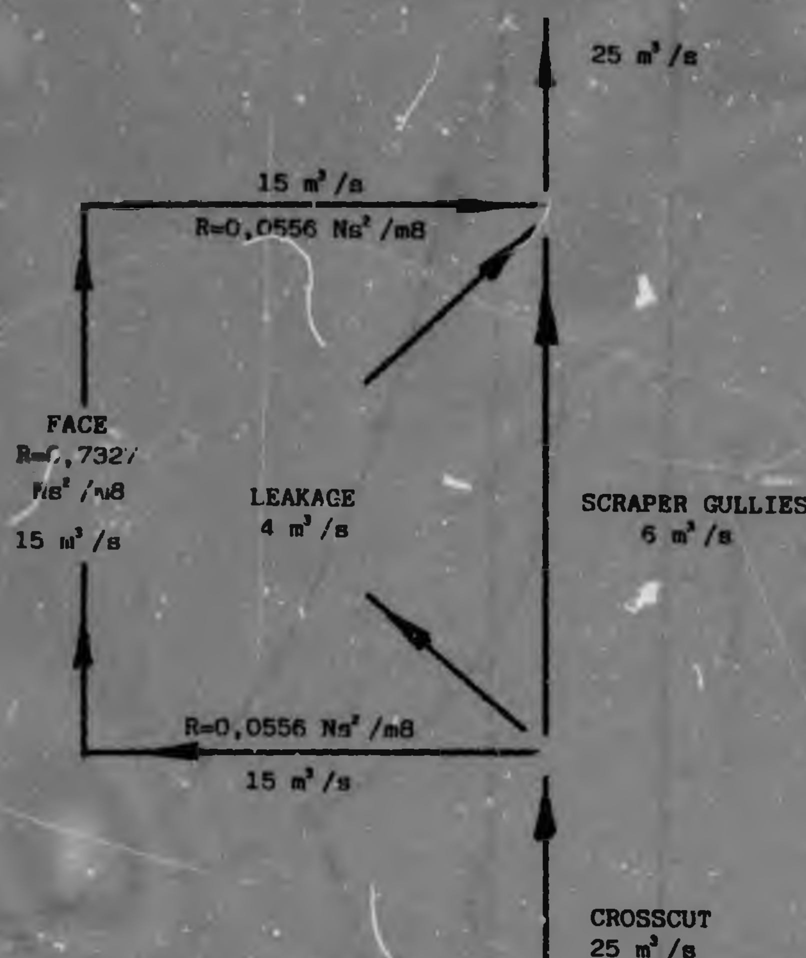


FIGURE 7.4

SIMPLIFIED TERTIARY [illegible] AIR CIRCUIT AND RESISTANCE PATHS

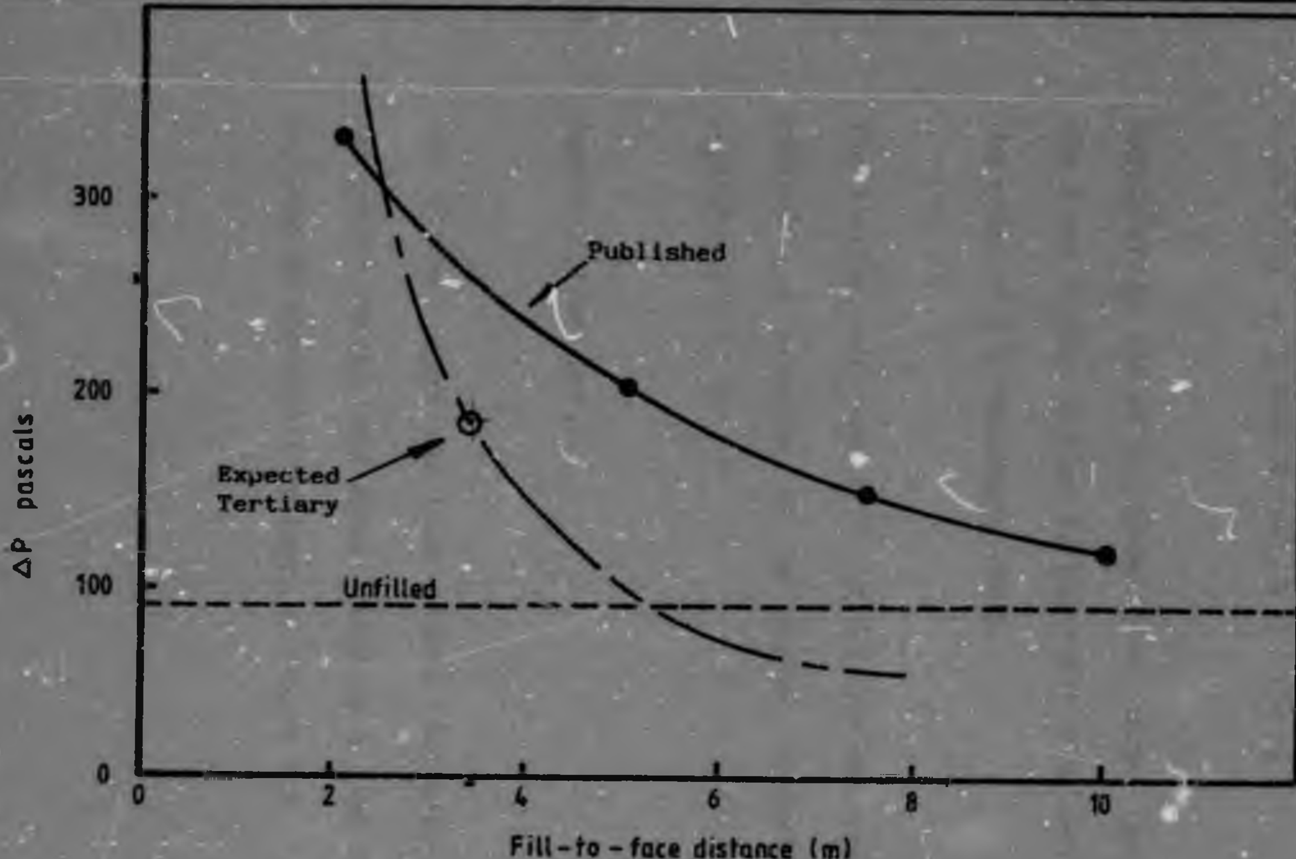


FIGURE 7.5

STOPE PRESSURE DROP AS A FUNCTION OF FILL-TO-FACE DISTANCE

At estimated stope resistances the pressure losses are:

R filled = 0,8439 Ns^2/m^8; ΔP = 190 Pa at Q = $15m^3/s$ & W = 1,4 kg/m^3

R unfilled = 0,4889 Ns^2/m^8; ΔP = 110 Pa

Depending on the mine, conventional stope resistances are a small contribution to the total mine orifice, in the order of 6 % (Matthews, 1988).

The Tertiary with backfill, strict air management and controls is expected to be higher accounting for approximately 11 %.

7.3 Tertiary Airway Detail

The Tertiary upper and lower blocks, including intake and return airways are approximated as a line diagram, dictated by structural geology, in Figure 7.6.

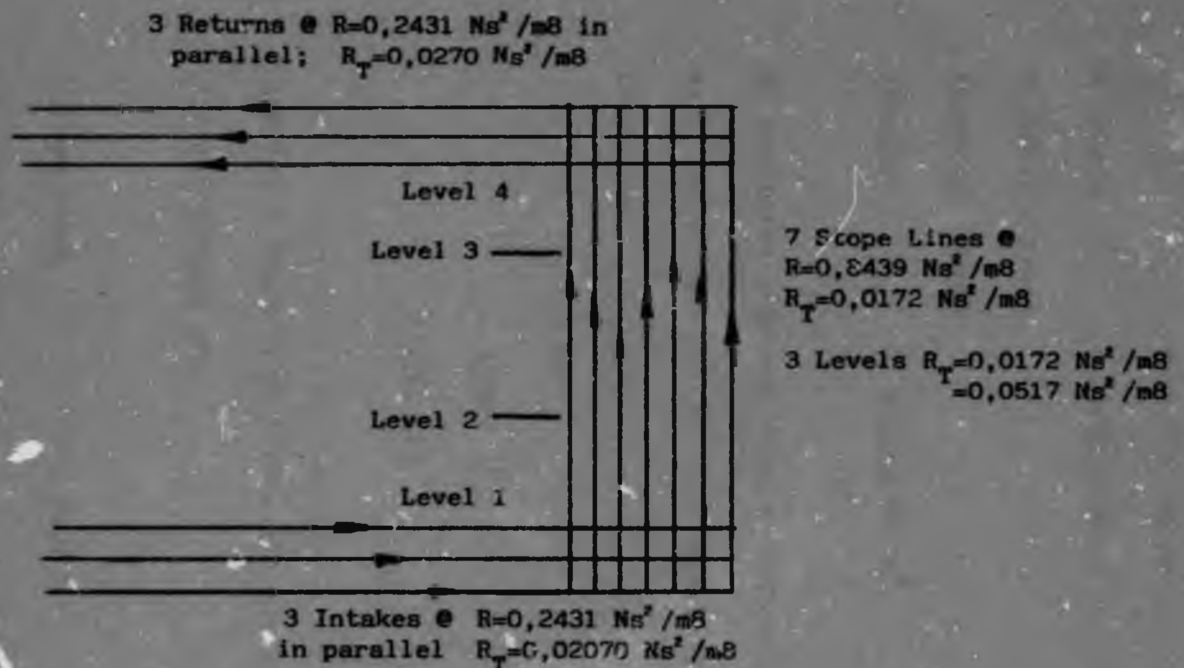


FIGURE 7.6

DIAGRAMATIC SECTION OF TERTIARY UPPER (45 TO 42 LEVEL) OR TERTIARY LOWER BLOCK (49 TO 46 LEVEL) WITH EXPECTED RESISTANCE VALUES

R intake = $0{,}0270\ Ns^2/m^8$

R stopes = $0{,}0172\ Ns^2/m^8$ x 3 in series maximum

= $0{,}0517\ Ns^2/m^8$

R return = $0{,}0270\ Ns^2/m^8$

R tertiary block

(upper or lower)

circuit = $0{,}1057\ Ns^2/m^8$

An estimated pressure loss therefore at a design air mass flow of 600/2 = 300 kg/s per block or 200 m^3/s is therefore:

$$P = RQ^2$$
$$= 0{,}1057\ (200)^2$$
$$= 4{,}22\ kPa$$

Assuming 7 inter-level lines, air quantities through the vertical airways are:

5 stope lines	5 x 25 m^3/s	= 125 m^3/s
1 developing line		= 10 m^3/s
1 vamping line		= 15 m^3/s
		150 m^3/s

The calculated stoping resistance component, R = 0,0517 Ns^2/m^8 gives a component system pressure of 1,36 kPa at prevailing density and an air flow of 150 m^3/s. In addition to this, the intake and return airways carrying 210 m^3/s will contribute at a total R = 0,0540, a compcntent pressure of 2,78 kPa. The Tertiary in this case contributes a total of 4,14 kPa pressure loss to the system.

As a percentage, Tertiary stoping is expected to be (1,36/12) = 11 % of the system pressure drop.

7.4 Residual Pressure

Based on predicted mass flows during the Phase 3 VENTFLOW scenario, and estimated R values of primary airflow arteries, a residual pressure as a result of the Eastern shaft fans of approximately 2 kPa exists at the top of the Tertiary complex, 37 Level.

The calculated system pressure losses and the expected Eastern fan operating point are shown in Table 7.2.

AIRWAY	k Ns^2/m8	MASS FLOW kg/s	PRESSSURE DROP PASCALS
EASTERN D/C	0,0020	671	900
13L INTAKES	0,0032	612	1 123
STRATHMORE D/C	0,0013	612	486
STRATHMORE U/C	0,0014	612	524
23L RETURNS	0,0041	612	1 535
EASTERN U/C	0,0031	675	1 412
			TOTAL 5 980

EASTERN FAN PRESSURE	7 900
RESIDUAL PRESSURE (APPROX)	2 000

TABLE 7.2
RESIDUAL PRESSURE ESTIMATION ON THE TERTIARY AS A RESULT OF THE EASTERN FANS

By calculation, this pressure is estimated to induce a sub-standard mass flow of 386 kg/s (276 m^3/s) into the Tertiary.

PHASE	DESIGN MASS FLOW FOR TERTIARY (kg/s)	MASS FLOW INTO TERTIARY (kg/s) *	MASS FLOW INTO TERTIARY (kg/s) **
1	120	120	120
2	400	258	232
3	600	572	405

TABLE 7.3

PREDICTED AIR MASS FLOWS IN THE TERTIARY WITHOUT THE USE OF BOOSTER FANS

* without booster fans and a conventional stope resistance $R = 0{,}0076\ Ns^2/m^8$ without backfill.

** without booster fans and a highly backfilled stope resistance $R = 0{,}0517\ Ns^2/m^8$

<u>The VENTFLOW models corroborate the initial estimates and the need for booster fans in the circuit to circulate and control a design mass flow of 600 kg/s in the Tertiary</u>. Regulation of the mine above the Tertiary is possible to force air further down. However, in doing so the main Eastern upcast fans will be forced dangerously close to stall. A complete analysis and justification for booster fans appears in Section 9.

7.4.1 Example:

With Eastern upcast fans operating unassisted, a differental pressure assumed to be 2,0 kPa across Tertiary intakes and returns exists.

The Tertiary R = 0,1057 Ns^2/m^8

Air quantities induced to the upper and lower Tertiary blocks at prevailing density $= \sqrt{\frac{2\ 000}{0,1057}}$

$= 138\ m^3/s$

The resultant mass flow at density = 193 kg/s (per block).

The resultant mass flow for total Tertiary = 386 kg/s

(The required mass flow is 600 kg/s).

7.5 Tertiary Development and Mining Implications

A summary of VENTFLOW scenarios, at Phases 1 to 3, without booster fans, is shown in Table 7.3. Mass flow of air into the Tertiary is due to residual pressure from the Eastern shaft fans and varies considerably with backfill.

7.6 Tertiary with Booster Fans

With the addition of booster fans on 37 Level a fully developed Tertiary network is shown in Figure 7.7, described in Table 7.4.

ITEM	DESCRIPTION
A	EASTERN PRIMARY DOWNCAST SHAFT
B	13 LEVEL INTAKE AIRWAYS
C	STRATHMORE (SECONDARY) DOWNCAST SHAFT
D	STRATHMORE WORKINGS
E	EASTERN SECONDARY WORKINGS
F	MAIN RETURN AIRWAYS 23 TO 26 LEVEL
G	AIR EX PIONEER
H	AIR EX PIONEER
I	EASTERN SECONDARY DOWNCAST
J	EASTERN WORKINGS
K	12 LEVEL TO EASTERN WORKINGS
L	EASTERN UPCAST SHAFT
M	EASTERN MAIN UPCAST FANS
N	SOUTHERN PRIMARY DOWNCAST SHAFT
O	REGULATED AIR FROM SOUTH TO EAST
P	SOUTHERN SECONDARY DOWNCAST SHAFT
Q	SOUTHERN WORKINGS
R	AIR EX ORANGIA
S	SOUTHERN UPCAST SHAFT
T	SOUTHERN MAIN UPCAST FANS
U	35 LEVEL AND 36 LEVEL MAIN INTAKES
V	6,8m DIAMETER DEDICATED DOWNCAST SHAFT
W	DOWNCAST PORTION OF 7,8m BRATTICED SHAFT
X	UPCAST PORTION OF 7,8m BRATTICED SHAFT
Y	37 LEVEL MAIN RETURN
Z	TERTIARY BOOSTER FANS

TABLE 7.4
EASTERN STRATHMORE AND TERTIARY MAIN AIRWAY DESCRIPTION

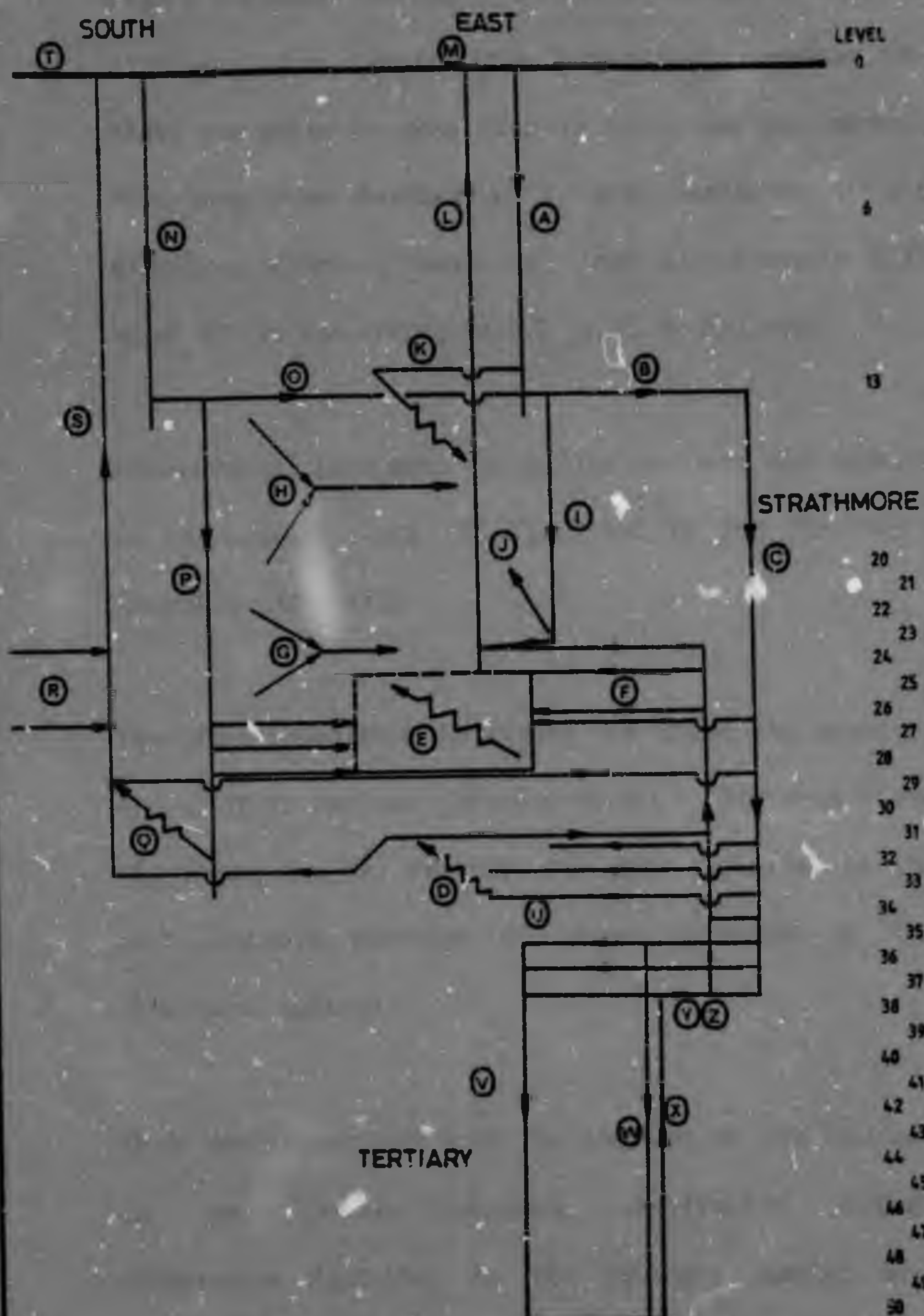


FIGURE 7.7

ELEMENTS OF THE EASTERN AND SOUTHERN AIRWAYS CONNECTED TO THE TERTIARY MINING BLOCK

Figure 7.8 shows the magnitude and direction of expected airflows. For "ventilation accounting" purposes the flows are given as mass flow in kilograms per second. This simplifies matters as a large variation of air densities occurs. These vary from approximately 0,97 kg/m^3 in the fan drifts to 1,5 kg/m^3 on 50 Level.

Allowance has been made for pillar recovery and leakage in the Eastern block (80 kg/s) and in the Strathmore Secondary (40 kg/s).

The inset tabulation in Figure 7.8 shows the expected air split to various Tertiary levels. Air mass flows (I) intake and (R) returns have been estimated on the most probable position of stopes governed by the structural geology.

It is envisiaged that with the coupling of the Tertiary to the Eastern/Strathmore ventilation scheme, progressive isolation to the Southern complex will occurr. In principle both ventilation networks can circulate independently however interconnections between the two may be desirable to facilitate the following:

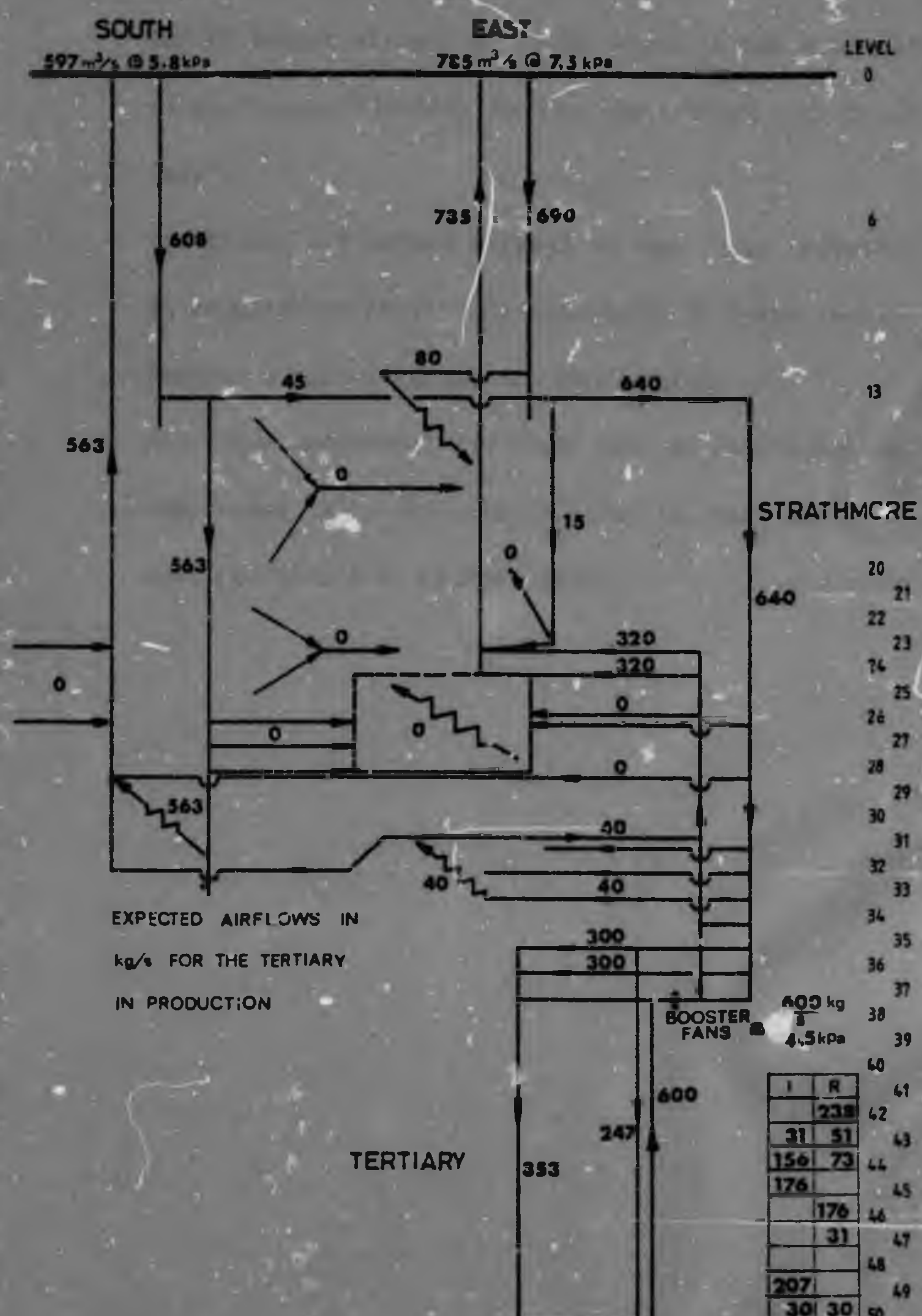


FIGURE 7.8

DESIGN AIR MASS FLOWS IN THE TERTIARY AND CIRCULATED AIR WITHIN THE REST OF THE MINE

- Relief return airways 31 to 33 level in the event of excessive restrictions in the upper block, 23 to 26 level.
- Additional hot return airways if more heat rejection is required at Strathmore mid-shaft, 27 inter level.
- Provide a second escape for the Tertiary.
- Allow for additional services such as refrigeration, compressed air and electricity in the event of service failure in Eastern shaft.

8. MAIN UPCAST FANS

8.1 General

The two upcast fan systems that are most likely to influence the Tertiary shaft are the Eastern and Southern shaft fans. As the Tertiary develops to maturity the Southern fans contribution will have been severely regulated and the interaction between the Eastern and Southern ventilation districts will be negligible.

8.2 Eastern Shaft Main Upcast Fans

8.2.1 Design characteristics

The Eastern upcast fan installation consists of three Airtec Davidson, single inlet backward Aerofoil bladed fans, trifurcation-mounted over a 6,7 metre diameter upcast shaft.

Technical specifications are shown in Table 8.1.

DESIGN OPERATING DUTY	AIR QUANTITY : 370 m^3/s STATIC PRESSURE : 8,2 kPa INLET AIR DENSITY: 0,98 kg/m^3
FAN SPECIFICATIONS	TYPE : W6F 3 850 IMPELLER DIAMETER: 3 820 mm FAN SPEED : 12,5 rps
MOTOR SPECIFICATIONS	TYPE : SIEMENS POWER : 4 250 kW SPEED : 12,5 rps

TABLE 8.1

EASTERN SHAFT MAIN FAN TECHNICAL SPECIFICATIONS

Manufacturers performance curves are shown in Figure 8.1.

Two of the three fans are operated in rotation with routine maintenance and cleaning performed on the third. Critical components such as:

- Motor.
- Impeller.
- Shaft.
- Coupling.
- Isolation doors.
- Electrical supply.

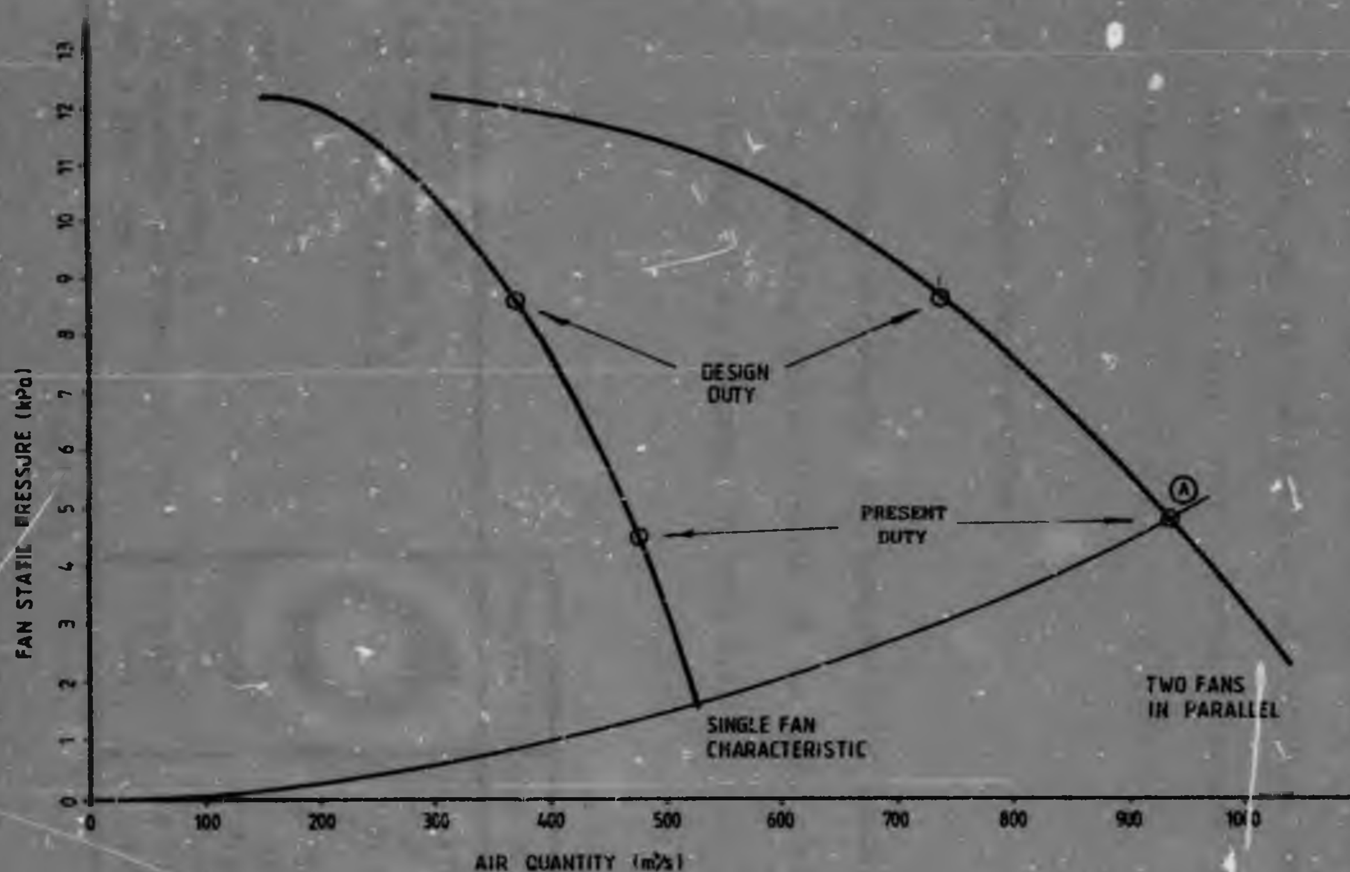


FIGURE 8.1
EASTERN SHAFT MAIN FAN PERFORMANCE CHARACTERISTICS

are individually scrutinized and gauged against standards. The condition of the fan installation appears to be satisfactory as verified by an Airtec/Howden site visit during February 1989. (Appendix E)

8.2.2 Performance characteristics

The recent performance test Appendix E has established the operating characteristics to be very close to theoretical and are shown in Table 8.2.

	FAN NO. 1	FAN NO. 3
FAN QUANTITY m^3/s	471	471
FAN STATIC PRESSURE Pa.	4 392	4 290
FAN STATIC EFFICIENCY %	68.6	61.6
VOLTS	6 750	6 750
AMPS	360	339
POWER FACTOR	0.76	0.92
MOTOR EFFICIENCY %	0.96	0.96
FAN ABSORBED POWER kW	3 071	3 500
MOTOR SPEED rpm	749	749

TABLE 8.2

EASTERN SHAFT ACTUAL PERFORMANCE CHARACTERISTICS

8.2.3 Conclusion

As ultimately the primary air mover for the Tertiary, the reconciliation of the performance characteristics was imperative. Any major deviation between actual and theoretical curves due to wear or guide-vane misalignment would have serious implications.

The performance check reported that two fans operating in parallel, point A, Figure 8.1 lies on the manufacturers published (theoretical) curve. Although the system resistance at the time of the test was considerably lower than design, a degree of confidence is placed on the accuracy of the curve representing 0° inlet guidevane setting. For the purposes of the exercise the curve throughout its range is assumed to be equally correct, however, a cross-check would be advisable at an induced higher resistance and a different guidevane setting.

8.3 Tertiary Resistance and Effect on Fan Performance

A number of uncertainties exist within the Tertiary mining zone which will have a marked effect on the mine system resistance. Items of greatest concern are:

8.3.1 Influence of backfill

The use of backfill is expected to increase the stope airway resistances considerably. Stope resistance factors are estimated to escalate from 0,0076 Ns^2/m^8 for conventional unfilled stopes to 0,0517 Ns^2/m^8 in the case of backfilled stopes envisaged for the Tertiary.

8.3.2 Downcast shafts

Two downcast shafts are planned to supply the Tertiary:

- A vertical 6,8 m diameter shaft (36 m^2).
- A vertical 7,8 m diameter split shaft (of which 28 m^2 is dedicated to the downcast).

Natural air flows have a 59 % bias to the 6,8 m diameter shaft, and 41 % by mass flow to the split shaft.

Due to the uncertainty of the final brattice material in the split shaft and its method of suspension, the resultant thermal heat transfer through the barrier may become a large source of heat.

In the event of high heat transfers prevailing, and to prevent excessive heat pick-up, it may be necessary to divert most of the downcast air through the 6,8 m diameter dedicated downcast.

The result is an increase in pressure drop due to the elimination of a primary parallel downcast.

8.3.3 Mining method and cooling strategy

Although the orebody geometry has been roughly estimated, no guarantee can be given to its continuity. The integrity of the reef plane has a pronounced effect on the ventilation and cooling strategy in the mining block. With the possibility of longer intake and return airways to access the ore, in addition to more mining panels, the mining horizon system resistance will increase further.

8.4 Tertiary Resistance Prediction

8.4.1 Calculations using main airways

Based on the verified fan performance, an equivalent mine orifice R, is established to be 0,0050 Ns^2/m^8.

This value is considerably lower than a series addition of resistances of the main airways; R = 0,010 Ns^2/m^8. This value of R does not account for the numerous and intricate parallel airways which presently interact.

VENTFLOW models with balanced meshes account for these interactive networks and by calculation give an existing system R of 0,0051 Ns^2/m^8.

As the Tertiary evolves and the rest of Buffelsfontein is sealed off, a number of the major parallel loops and meshes will fall away. Subsequently the airway geometry will approach true series and at worst result in a resistance of 0,010 $Ns^2/m8$. Coupled to this will be the Tertiary mining horizon in total, (upper and lower blocks each at an R value of 0,1057 Ns^2/m^8) giving an R0,0264 Ns^2/m^8. The parallel downcast shafts will account for an R of 0,001 Ns^2/m^8 and the upcast shaft R equals 0,004 Ns^2/m^8. Little change is expected in the return air system resistance on 23 Level. Summarily the <u>expected system resistance</u> for the Tertiary coupled to the existing infrastructure is:

Eastern and Strathmore infrastructure	:	0.0100
Tertiary Downcasts	:	0,0010
Tertiary intakes, working and returns	:	0,0264
Tertiary Upcast	:	0,0040
TOTAL		0,0414 Ns^2/m^8

8.4.2 Eastern main fan operating points as a function of system resistance

Figure 8.2 shows a family of system resistance curves indicative of the Tertiary ventilation circuit. The theoretical design operating point, item A, 600 m^3/s at 10,7 kPa relates to an $R = 0,0297$ Ns^2/m^8.

The expected system resistance however, is $R = 0,0414$ Ns^2/m^8 (section 8.4.1) which gives operating point B, 525 m^3/s at 11,4 kPa.

In the configuration shown, a narrow margin of 1,[illegible] kPa exists between the theoretical operating point and the fans' stall pressure. This represents approximately 12 % of the operating range and is felt to be too close to stall.

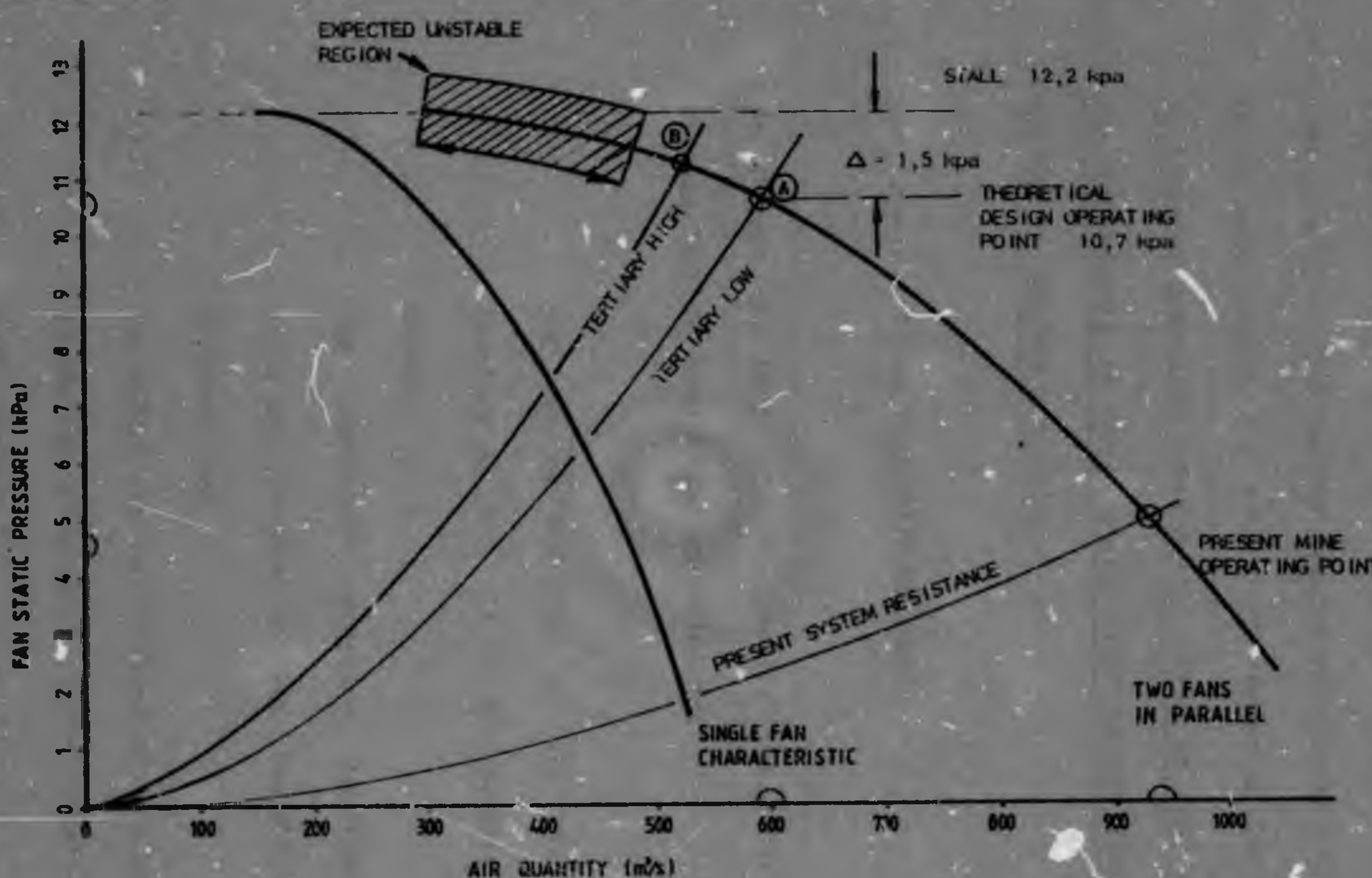


FIGURE 8.2
EXPECTED TERTIARY SYSTEM RESISTANCE AND EASTERN FAN OPERATING POINTS

An additional problem arises in the possibility of an unstable zone or ECK lines being encountered prior to stall (Rabia). This further reduces the pressure buffer to an unacceptable 0,8 kPa. The expected Tertiary resistance, operating point B, is likely to compromise the performance of the Eastern shaft upcast fans if allowed to operate unassisted.

8.4.3 Tertiary resistance from VENTFLOW models

An attempt has been made to quantify the effect of various Tertiary resistance scenarios on:

(1) the induced quantity of air into the Tertiary as a result of the Eastern fans alone;

(2) the additional pressure required to circulate the designed mass flow within the Tertiary, and

(3) the effect and implications of (1) and (2) on the operating point of the Eastern shaft main surface fans.

VENTFLOW models were used in all cases and results are shown in Table 8.3.

SCE-NARIO	BACKFILL MAJOR/ MINOR	TERTIARY DOWNCAST TWIN/ SINGLE	TERTIARY MASS FLOW kg/s	BOOSTING PRESSURE REQUIRED kPa	EASTERN SHAFT FANS OPERATING POINT m^3/s @ kPa	MINE CIRCUIT RESISTANCE $Ns^2/m8$
1	MINOR	TWIN	527	0	756 @ 7,9	0,0138
2	MAJOR	TWIN	[illegible]50	0	636 @ 9,4	0,0232
3	MINOR	SINGLE	546	0	681 @ 8,3	0,0179
4	MAJOR	SINGLE	408	0	550 @ 11,0	0,0364
5	MINOR	TWIN	600	1,8	735 @ 7,5	0,0172
6	MAJOR	TWIN	600	6,0	738 @ 7.5	0,0248
7	MINOR	SINGLE	600	3,5	740 @ 7,4	0,0199
8	MAJOR	SINGLE	600	9,5	735 @ 7,5	0,0315

TABLE 8.3
VENTFLOW PREDICTIONS OF TERTIARY AIRFLOW
AT VARIOUS RESISTANCE CONFIGURATIONS

8.4.4 Conclusion

<u>The design, mass flow of 600 kg/s, can be achieved only by installing booster fans.</u>

The range of <u>boosting pressures</u> required are summarized as:

- high degree of backfill and single downcast 9,5 kPa scenario 8.
- high degree of backfill and twin downcast 6,0 kPa scenario 6.

- low degree of backfill and single downcast 3,5 kPa scenario 7.
- low degree of backfill and twin downcast 1,8 kPa scenario 5.

The most probable boosting pressure range appears to be 3 to 6 kPa. It will therefore be imperative to minimise the Tertiary system resistance to reduce the boosting energy required.

Careful consideration will be required to possibly increasing the number and size of main intake and return airways and the possibility of increasing face to backfill distances, rock engineering permitting.

9. BOOSTER FANS

9.1 Performance Characteristics

Three booster fans are proposed to be integrated into the circuit located on 37 level. These boosters will be in parallel with each other and in series with the main Eastern shaft fans. The interactive curves are shown in Figure 9.1 and individual fan operating points identified.

The actual mine network is not a true series and parallel air configuration due to leakage and regulated air entering the system. Therefore, operating points in Figure 9.1 are indicative of what is expected and are used to show trends and approximate duty points.

Figure 9.2 is an illustration of the extended pressure capacity provided by the booster fans. The various resistance curves are derived from VENTFLOW analyses and manual calculations. Relevant data associated with the operating characteristics are shown in Table 9.1.

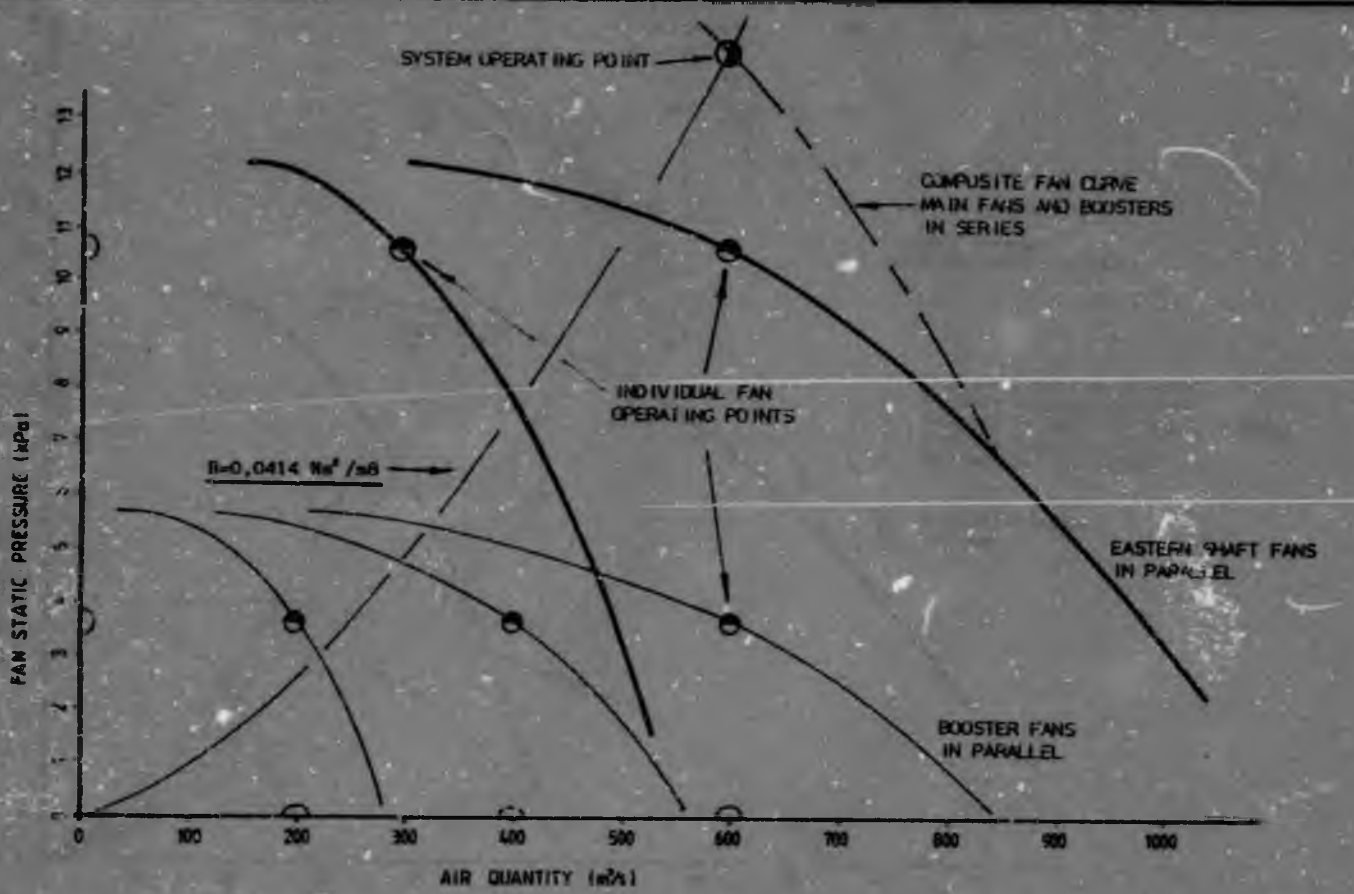


FIGURE 9.1
RESULTANT COMPOSITE CURVES WITH BOOSTER FANS IN THE CIRCUIT

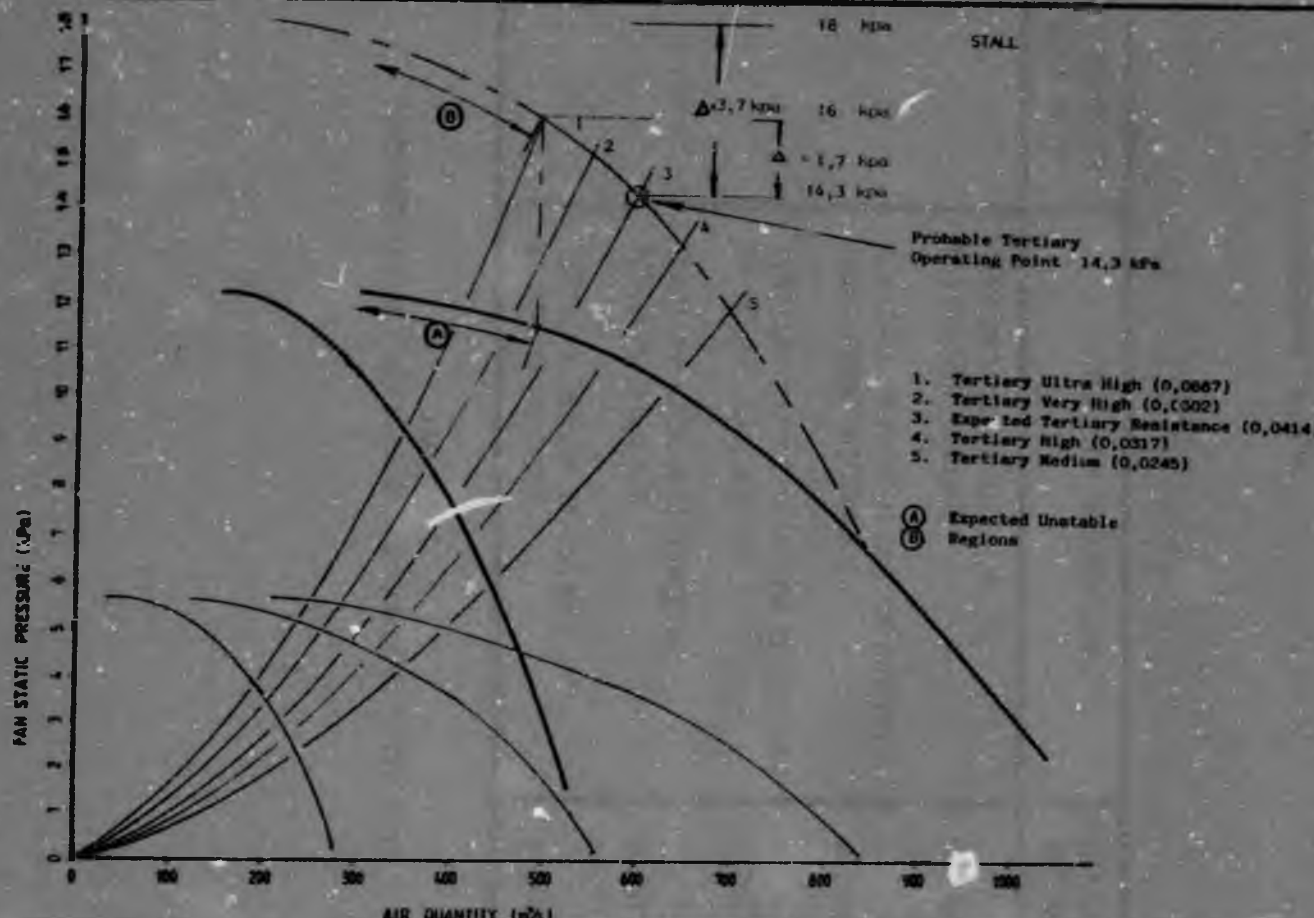


FIGURE 9.2
EFFECT OF VARIABLE TERTIARY RESISTANCE ON THE EASTERN FAN - BOOSTER FAN COMBINATION

The stall buffer with boosters in the system is extended to 3,7 kPa, now 20 % of the operating span, and 1,7 kPa to the unstable region.

OPERATING POINT NUMBER	CONDITION DESCRIPTION	RESISTANCE $Ns^2/m8$
1	TERTIARY ULTRA HIGH DUE TO RESTRICTED AIRWAYS	0,0667
2	TERTIARY VERY HIGH DUE TO EXTENDED DEVELOPMENT DISTANCE	0,0502
3	TERTIARY EXPECTED SYSTEM RESISTANCE BY MANUAL CALCULATION	0,0414
4	TERTIARY HIGH RESISTANCE BY VENTFLOW SOLUTION	0,0317
5	TERTIARY MEDIUM RESISTANCE BY VENTFLOW SOLUTION	0,0245

TABLE 9.1
EXPECTED TERIARY RESISTANCES AT VARIOUS CONDITIONS

9.2 Specification

A three fan booster installation is proposed which will have the capacity to work in whole (3 of 3) or in part (2 of 3) to handle 600 kg/s air at a design pressure of 4,5 kPa with an air inlet density of 1,4 kg/m^3. Operating control is achieved with inlet guide vanes.

The over-specification will cater for:

- A breakdown of one fan.
- A higher operating pressure in the event of a worst case scenario.

A fan suitable for the installation is shown in Appendix G.

9.3 Location

The most suitable position of the fans is 37 level. Rock engineering, assure the ground for the fan chamber excavations and airways is stable. This is imperative as only one airway joins the Tertiary and Strathmore upcast shafts. Figure 9.3 shows the relative position of the three fan excavations with specially developed fresh air access tunnels to the motor and drives.

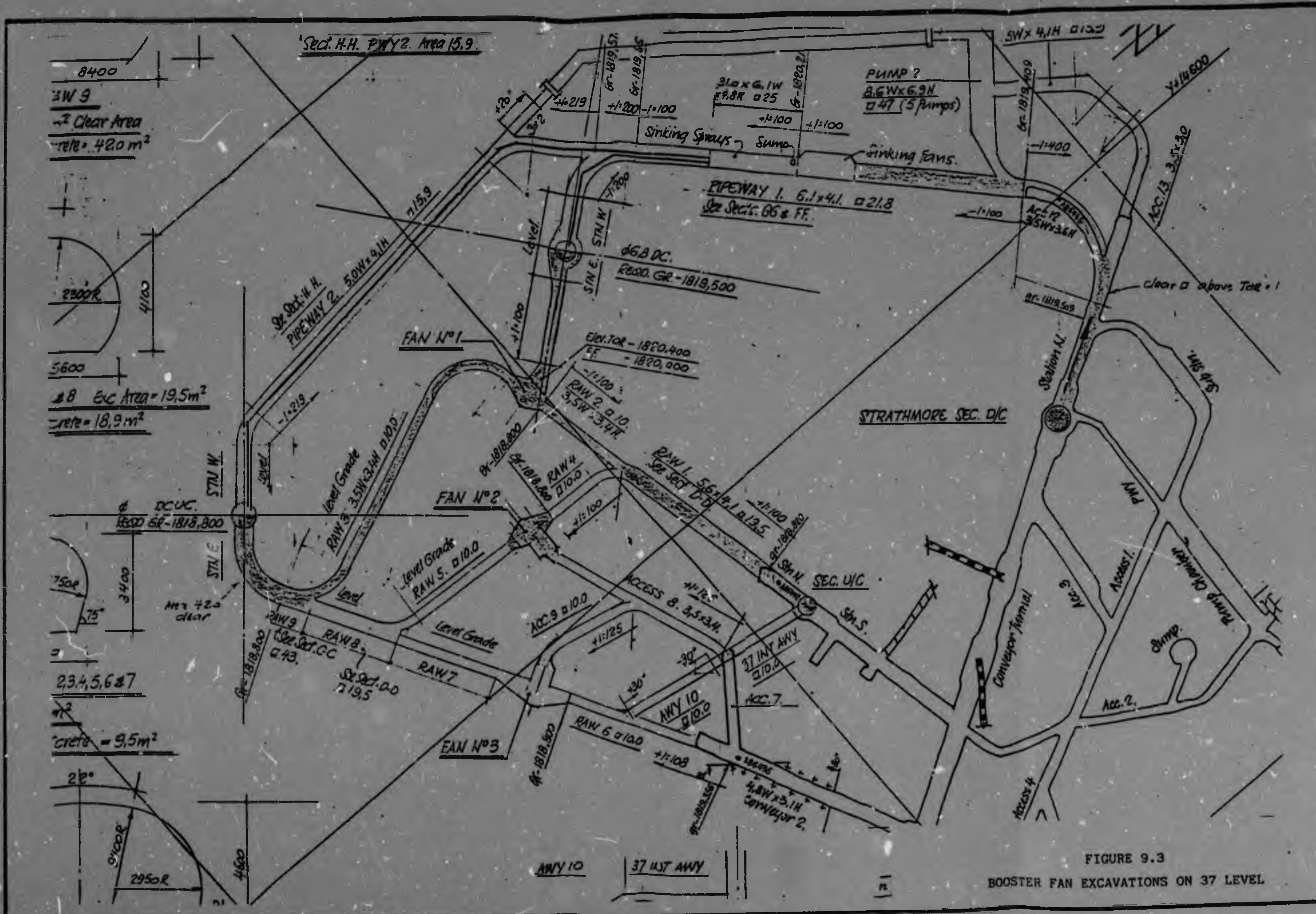


FIGURE 9.3
BOOSTER FAN EXCAVATIONS ON 37 LEVEL

9.4 Summary

Although the premium of introducing boosters to a system which could "almost cope without" is high, the following considerations justify their installation:

- Tertiary resistance is an uncertainty.
- Tertiary air regulation will contribute to additional energy losses not accounted for in the VENTFLOW models.
- Implementation of large scale controlled re-use of air is more easily effected.
- Fewer ventilation restrictions would be imposed if an increase in production were necessitated or the depth or distance of the mining horizon extended.

10. TERTIARY HEAT LOAD PREDICTION

10.1 Virgin Rock Temperature

10.1.1 Data extrapolation

The accurate estimation of the virgin rock temperature (V.R.T) in any heat load estimate is imperative. The behaviour of the geothermal gradient at depth in the Klerksdorp area has been predicted using historical data. Published graphical information from three sources correlates very well and the estimate of VRT for the mining horizon is felt to be representative.

Table 10.1 summarises the relevant information and the form of the equations obtained by linear regression.

SOURCE	DATE	LINEAR REGRESSION
ENVIRONMENTAL ENGINEERING IN S.A. MINES (266)	1982	y = 0,011 x + 21
ENVIRONMENTAL ENGINEERING IN S.A. MINES (572)	1982	y = 0,010 x + 22
COMPILATION OF GEOTHERMAL DATA FROM THE WITWATERSRAND BASIN C.O.M. GUIDE NO. 13	1988	y = 0,011 x + 21

WHERE y = VRT (°C) x = DEPTH BELOW SURFACE (m)

TABLE 10.1
VIRGIN ROCK TEMPERATURE GRADIENT EQUATIONS

From the three equations a best fit expression is expected to be:

$$y = 0{,}011\ x + 21{,}3$$

10.1.2 Predicted vrt's

Using this relationship the following virgin rock temperatures are expected on the main operating levels. Table 10.2 lists the relevant details.

LEVEL	DEPTH BELOW SURFACE (m)	V.R.T. (°C)
35	3 001	54,3
36	3 073	55,1
37	3 155	56,0
42	3 529	60,1
43	3 604	60,9
44	3 679	61,8
45	3 754	62,6
46	3 829	63,4
47	3 004	64,2
48	3 979	65,1
49	4 054	65,8
50	4 124	66,7
SHAFT BOTTOM	4 162	67,1
WEIGHTED MEAN ROCKBREAKING DEPTH	3 842	63,6

TABLE 10.2
EXPECTED VIRGIN ROCK TEMPERATURES IN THE TERTIARY MINING BLOCK

Figure 10.1 shows the actual measured V.R.T's of mines in the Klerksdorp area with an extrapolated line to the mean rockbreaking depth of 3 842 m (COMRO Guide No. 13).

10.2 Comparison of Heat Load Prediction Techniques

Various heat load estimation techniques were used to predict the Tertiary shaft requirements. Results are summarized in Table 10.3. The comparison begins with graphical techniques and progresses from the more elementary to the most complex analyses.

10.2.1 Summary

The average predicted total mine heat load, including autocompression, of the ten techniques is 76,6 MW.

The variance is - 4 MW and + 5 MW or at worst + 7 %.

It is felt that of all techniques, the results of the Chamber of Mines analysis are the most representative. Confidence is highest because:

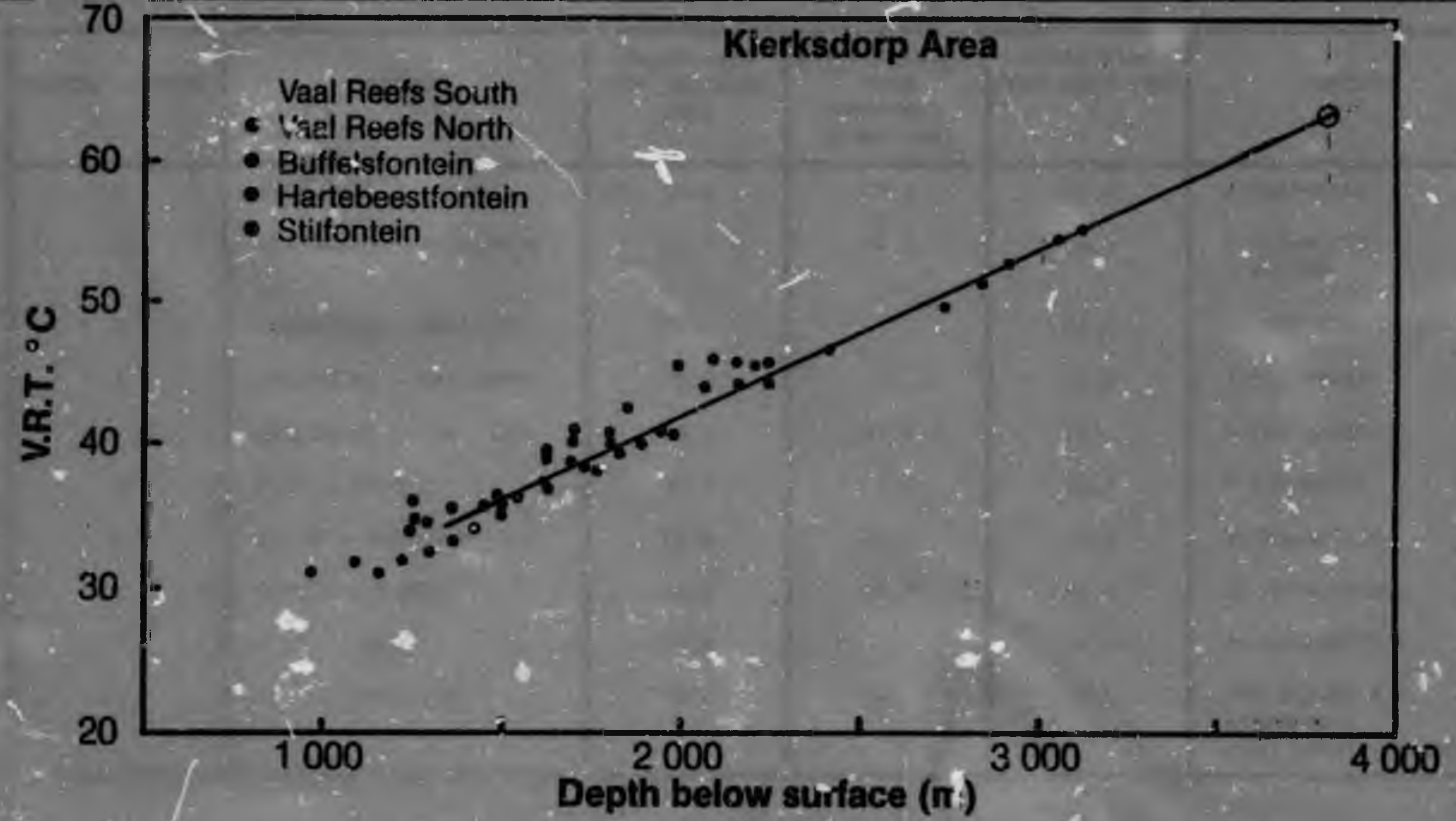


FIGURE 10.1

MEASURED VIRGIN ROCK TEMPERATURES AT DEPTH IN THE KLERKSDORP AREA

IDENTIFICATION	METHOD	CALCULATED MINE HEATLOAD (MW)	AUTO-COMPRESSION COMPONENT ADDED (MW)	TOTAL MINE HEAT LOAD (MW)	COMMENTS
1	GRAPHICAL - STROH	59,4	22,2	81,5	2 PARAMETER
2	GRAPHICAL - BARENBRUG	50,4	22,2	72,6	2 PARAMETER 3 REGION
3	GRAPHICAL - WHILLIER	81,0	-	81,0	3 PARAMETER
4	GRAPHICAL - BARENBRUG	75,6	-	75,6	4 PARAMETER
5	GRAPHICAL - ENV. ENG.	55,9	22,2	78,1	4 PARAMETER
6	P.C. - FRIDGE	74,6	-	74,6	3 PARAMETER
7	HP 97 - FRIDGE 5	51,6	22,2	73,8	15 PARAMETER
8	P.C. - OPTIMIZE	51,1	22,2	73,3	26 PARAMETER
9	P.C. - GREF	76,8	-	76,8	28 PARAMETER
10	P.C. - HEATFLOW	78,0	-	78,0	300 BLOCKS @ 5 PARAMETERS 1500 INPUTS (APPROX)

TABLE 10.3

SUMMARY OF HEAT LOADS FOR THE TERTIARY USING VARIOUS TECHNIQUES

1. The most current, up-to-date heat flow algorithms are used for tunnels and stopes.

2. Effect of backfill is incorporated.

3. The mining block has been divided into numerous heat source modules.

10.2.2 Examples

A worked example using Tertiary data is added for completeness.

10.2.2.1 Heat load estimation - STROH

Stroh suggests an approach, using a single graph to show how heat production depends upon virgin rock temperature. The proportion of this heat which can be removed by the ventilation air is then calculated, the difference giving the amount of cooling. The heat production graph is shown in Figure 10.2. This graph applies to a defined mining configuration and to temperatures commonly encountered in gold mines in the Orange Free State and Western Transvaal during summer. (Environmental Engineering: 607)

Based on Tertiary criteria the expected mine heat load is:

330 watts per ton per month x 180000 tons per month

= 58,4 MW

Since the air provides no cooling and is in fact a heat load due to auto-compression, the predicted total heat load is:

59,4 MW + 22,2 MW

= 81,6 MW

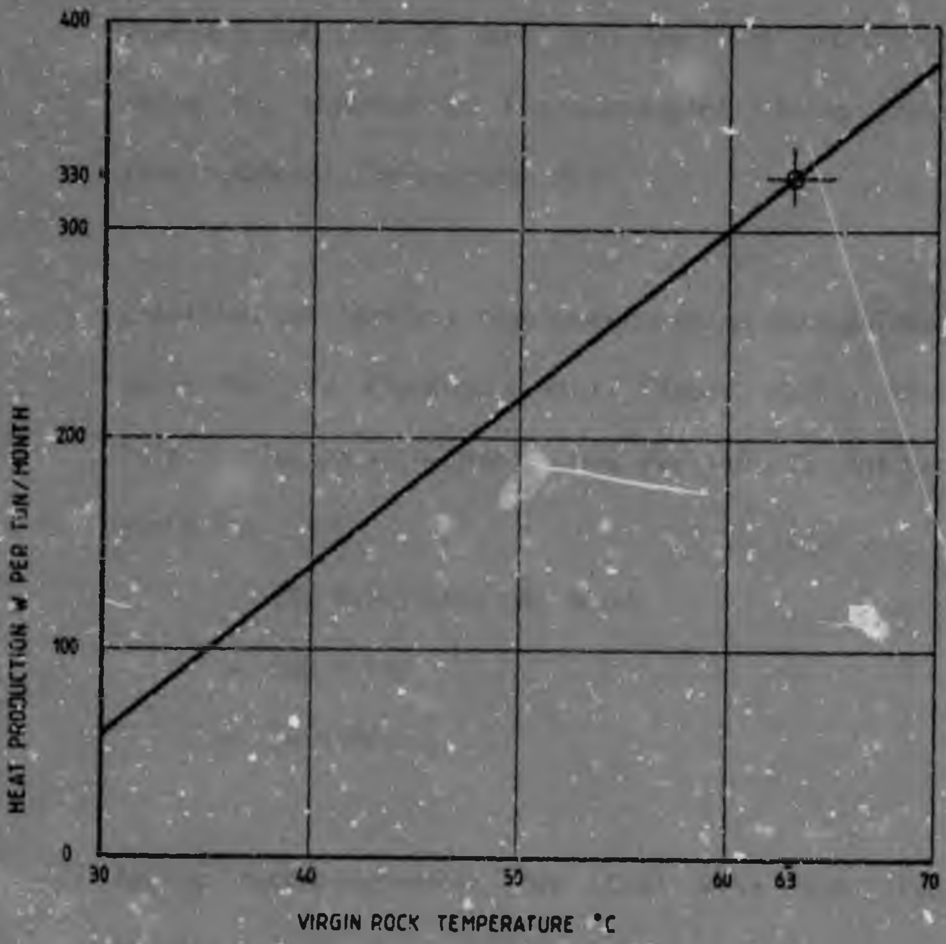


FIGURE 10.2

HEAT PRODUCTION HEAT PRODUCTION VS. VRT, STROH METHOD

10.2.2.2 Heat load estimation - BARENBRUG METHOD (1)

Information contained in this graphical representation is based on industry averages and more concise planning can only be done when the heat production rates are adjusted to the particular mining areas (Environmental Engineering: 270).

Examining the Tertiary requirements on an extrapolated curve for the Klerksdorp area, Figure 10.3 a heat load, excluding auto-compression, for 180 000 tons per month is:

280 kW/kiloton per month

= 280 x 180

= 50,4 MW

Adding auto-compression the total mine heat load becomes:

50,4 MW + 22,2 MW

= 72,5 MW

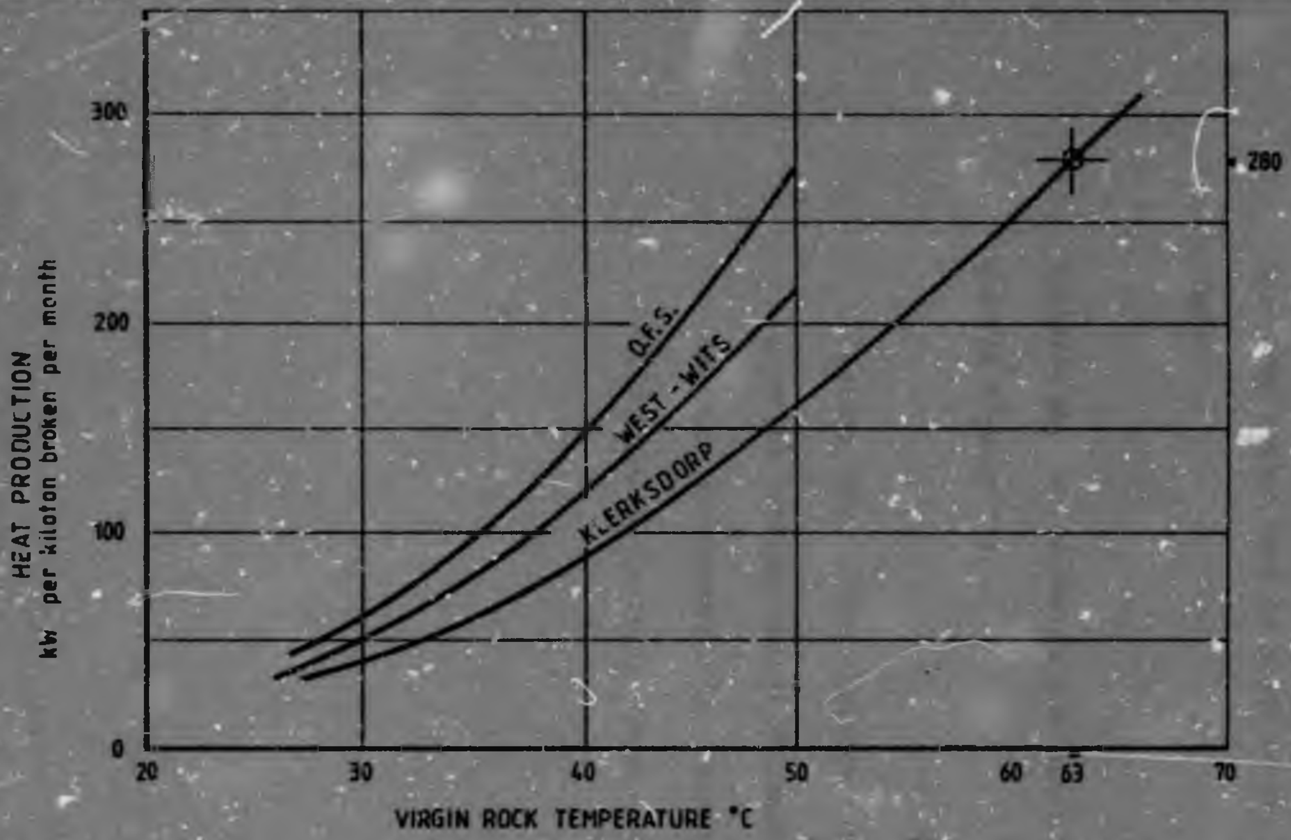


FIGURE 10.3

HEAT PRODUCTION VS. VRT, BARENBRUG METHOD 1

NOTE:

1) Auto-compression must be added to the graphical figure.

2) Estimate may be low due to the use of industry average figures i.e.

- Average stope face advance 7m/month; expected Tertiary actual 20m/month.
- Average development to stoping ratio. 20%; expected Tertiary actual: 40%.

10.2.2.3 Heat load estimation - WHILLIER

Whillier has also developed a graphical method. This is not strictly speaking an empirical method, for it is based upon theoretical calculations.

Whillier uses a graph with axis of virgin rock temperature and depth relative to sea level. It is then possible to plot on this graph the virgin rock temperature/depth relationship for the actual mine being considered. Superimposed on this graph are diagonal lines giving the amount of cooling required, again expressed watts per ton broken per month. This graph is shown in Figure 10.4. (Environmental Engineering in South African Mines: 608).

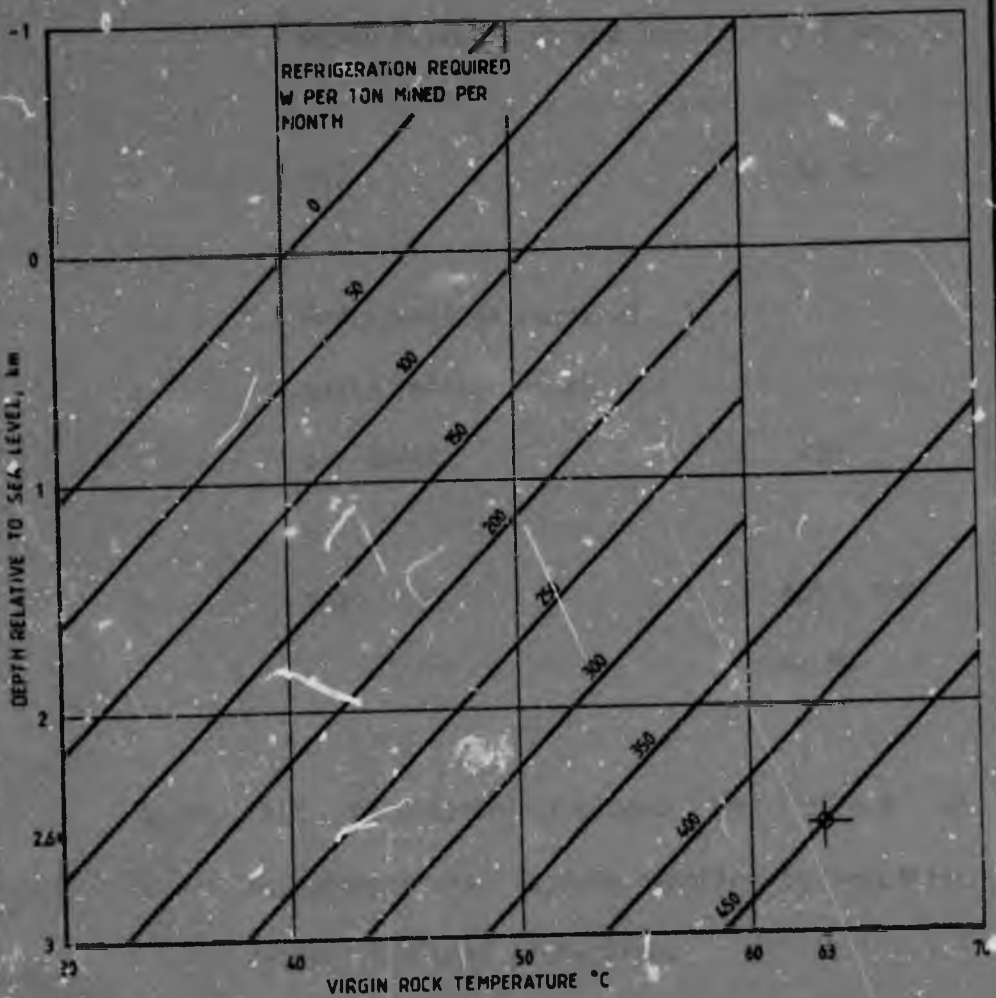


FIGURE 10.4

REFRIGERATION VS. VRT, WHILLIER METHOD

For Tertiary assumptions this method gives a heat load of:

- Depth relative to sea level : 2,6 km
- VRT : 63 °C
- Refrigeration required, watts per ton mined per month : 450
- Heat Load = 450 x 180
  = 81 MW

<u>NOTE</u>: This estimation includes the effect of auto-compression as refrigeration is specified rather than heat production.

10.2.2.4 Heat load estimation - BERENBRUG METHOD (2)

From measurements taken on mines, Barenbrug developed a series of graphs shown in Figure 10.5. Heat production in watts per ton mined per month are shown as a function of virgin rock temperatures and the mining method used. In addition to this, the residual cooling or the amount of heat that could be removed by ventilation air was recorded as a function of depth below surface (Environmenatal Engineering in South African Mines: 606).

Using the method for the Tertiary condition:

- at V.R.T 63°C and scattered mining;

  heat production = 450 W/ton/month x 180 000 tpm

  = 81 MW

- cooling from ventilation air = 30 W/ton/month x 180 000 based on 2,4 m^3/s/kt/month

  = 5,4 MW

therefore cooling required = 75,6 MW

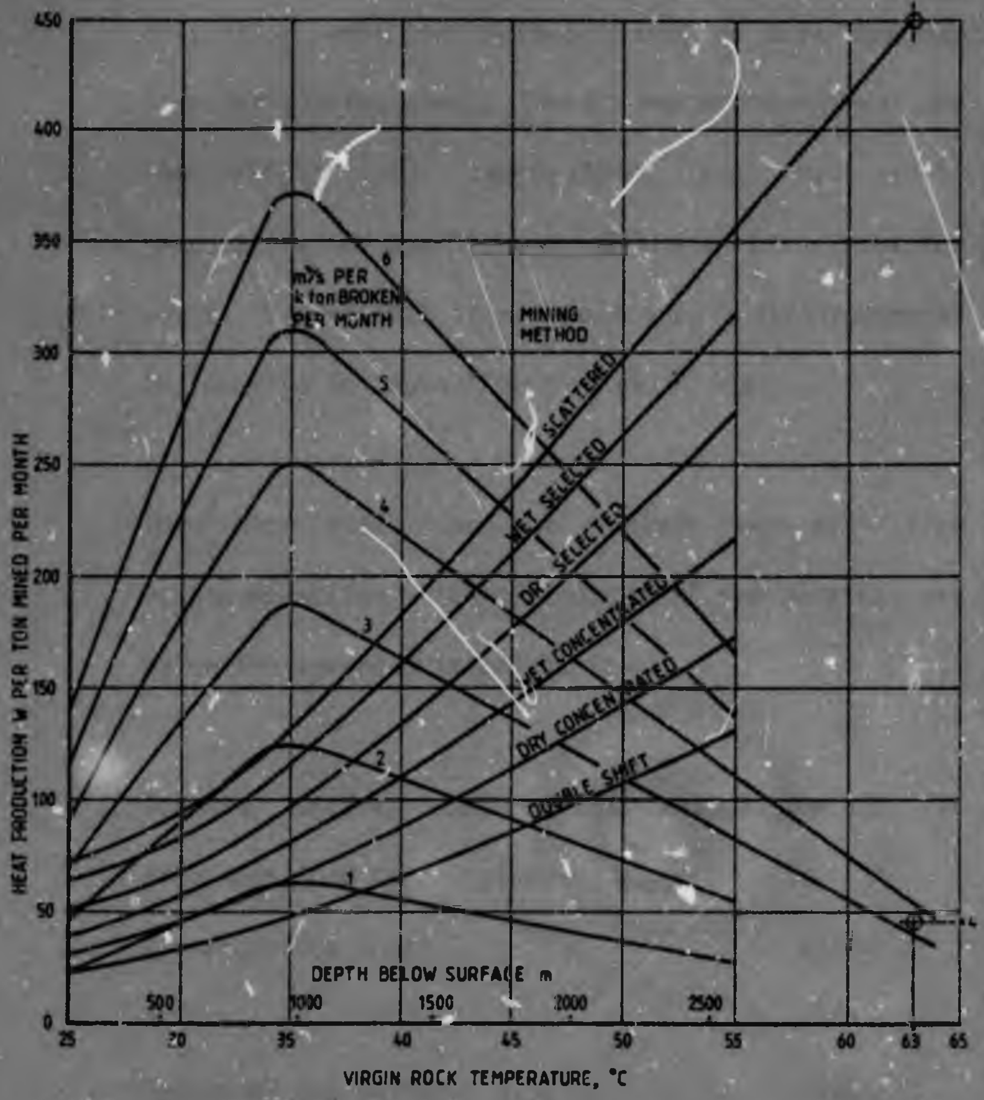


FIGURE 10.5

HEAT PRODUCTION VS. VRT, BARENBRUG METHOD 2

10.2.2.5 Heat load estimation - ENVIRONMENTAL ENGINEERING IN S.A. MINES

This is an empirical method, based on data obtained from surrounding mines. The parameters considered are the virgin rock temperature, the air reject temperature and the distance of the workings from the shaft. Figure 10.6 shows the method. (Environmental Engineering in South African Mines: 959).

This heat load does not include any heat from auto-compression which will occur if the workings are below the second critical horizon.

For the Tertiary scenario this method predicts the mine heat load to be, given:

Distance from shaft	2500m
VRT	63°C
Reject WB correction @ 27°C	+12%

308kW/kt/month x 180kt/month

= 55,4 MW x 1,12

= 62,1 MW

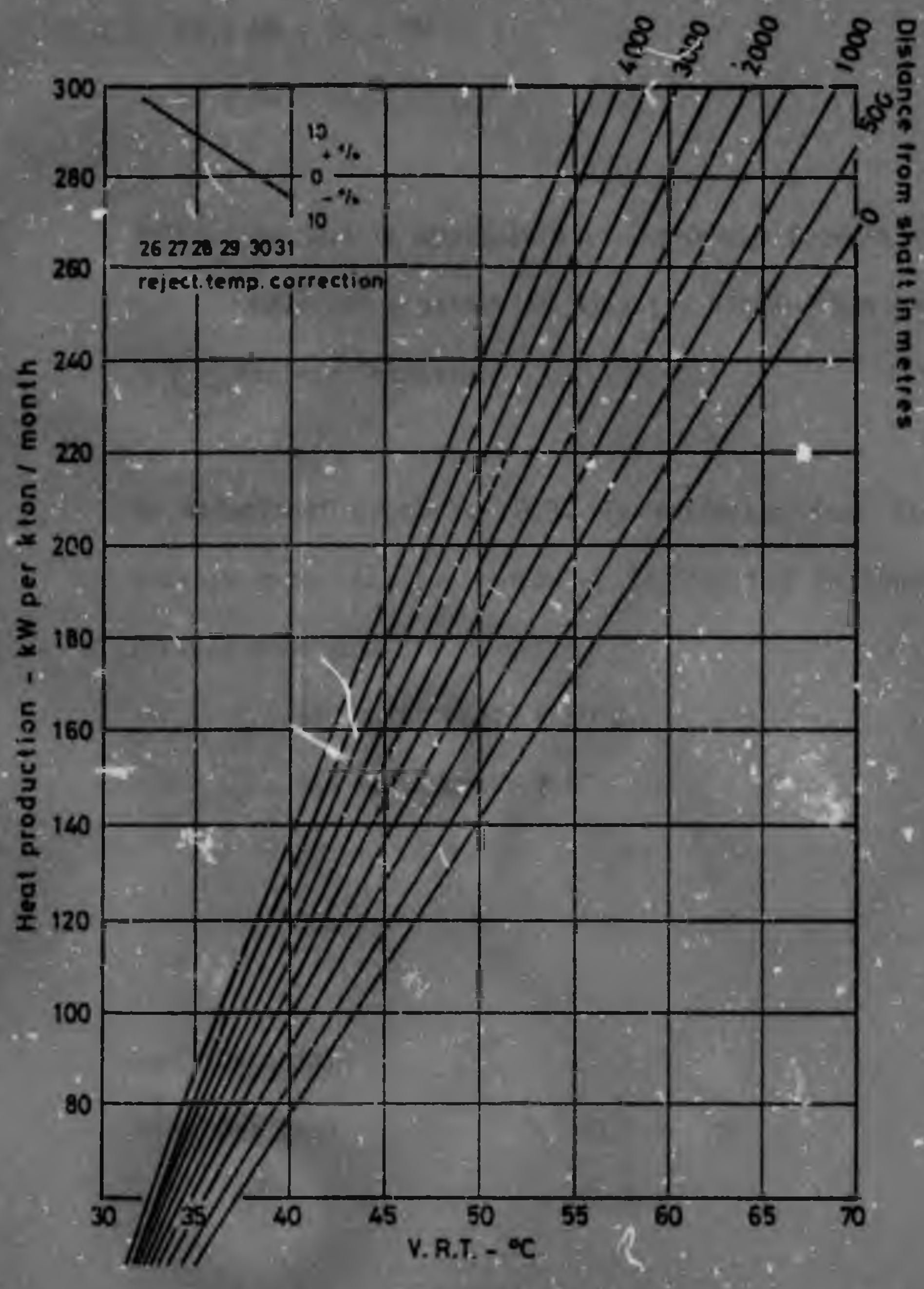


FIGURE 10.6

HEAT PRODUCTION VS. VRT, ENVIRONMENTAL ENGINEERING IN S.A. MINES

Adding auto-compression:

62,1 MW + 22,2 MW

= 84,3 MW

NOTE: The data is applicable to the Orange Free State whose VRT gradient is slightly higher than the Western Transvaal

An adjustment factor of 0,90 is estimated from the data on page 572 (Environmental Engineering in South African Mines).

330 kW/kt/month @ 50°C

365 kW/kt/month @ 67°C

$$\frac{330}{365} = 0{,}90$$

Applying this:

Mine heat load = 62,1 x 0,90

= 55,9

Total heat load = 55,9 + 22,2

= 78,1 MW

10.2.2.6 Software program - P C FRIDGE

This IBM Personal Computer program gives a rough estimate for the design parameters of a mine, given the bare minimum input information. The origin of the computer program is unknown however, a full listing is available.

Supplying relevant Tertiary shaft information the results are as follows:

INPUT DATA

1.	Area	Western Transvaal
2.	Monthly Stope Tons	130 000
3.	Average Depth (m)	3 842

OUTPUT DATA

1.	Refrigeration required (MW)	64,6

NOTE:

Due to the program - fixed percentage, 20% of development to stope tons, the calculated heat loads for the Tertiary scenario are felt to be underestimated (64,6 MW).

- By increasing the stope tonnage from 130 000 tons per month to 150 000 tons per month to compensate for the Tertiary 40% ratio, gives a refrigeration estimate of 74,6 MW.

10.2.2.7 Software program - FRIDGE 5

Written by the Chamber of Mines Research Organization, Environmental Engineering Division, and updated in 1984, this computer program predicts the refrigeration requirements per level, in South African Gold Mines (COMRO i.e 271). The computer program is suited for the Hewlett Packard 67 or 97 calculator and the H.P 41 series.

A model representative of the Tertiary configuration yields the following:

<u>INPUT DATA</u>

1.	Level wet-bulb temperature	27°C
2.	Level Pressure	132 kPa
3.	Air Volume	3,3 m^3/s per kton/month
4.	Centares per month	40 123 m^2/month
5.	Stope face advance	20 m/month

6.	Stoping width	1,2 m
7.	Ratio development to stopping	40 %
8.	Virgin rock temperature	63,3°C
9.	Tons water/ton rock	0,5 tons/ton
10.	Design face wet-bulb	27°C
11.	Temperature of service water before cooling	27°C
12.	Fissure water expectation	0,1 tons/ton
13.	Age of stoped out area	12 month
14.	Temperature of service water after cooling	0,5 °C
15.	Height difference between refrigeration plant and level(+ve from refrigeration plant down)	3843 m

OUTPUT DATA

1.	Ventilation air cooling	0
2.	Chilled service water cooling	4458 kW
3.	Total heat load	51643 kW
4.	Heat from ventilation air	0
5.	Heat from recent development	18275 kW
6.	Heat from established development	1342 kW
7.	Heat from worked area	20776 kW
8.	Other heat loads	5608 kW
9.	Heat from service water	1513 kW
10.	Heat from fissure water	1221 kW
11.	Heat from rock to water	2906 kW

NOTE:

The mine heat load is calculated to be 51,6 MW excluding the effect of auto-compression. Total mine heat load would therefore be:

73,8 MW including auto-compression.

10.2.2.8 Software program - OPTIMIZE

Development of the computer program began in-house during 1974 with work done by [illegible] Howes (Howes). This was an attempt to develop a full optimization study in which five cost functions were defined as follows:

(a) Ventilation. This cost function included the costs of the shafts and the main mine fans necessary to circulate air through the shafts and mine. The cost was a function of shaft diameter and ventilation air quantity.

(b) Refrigeration. The cost of refrigeration required to achieve a desired average face wet bulb temperature with a given quantity of ventilation air.

(c) Productivity. The cost of designing for less than ideal environmental conditions in terms of average face wet bulb temperature and ventilation air quantity (velocity).

(d) Acclimatisation. The cost of having working areas where the maximum wet bulb temperature exceeds 27,5°C.

(e) Dust. The cost of having less air available to dilute the dust formed during normal mining operations.

Based on empirical data of the time, heat gain equations have been modified as more current data became available. Optimize calculates both the heat load of the mine and the optimum diameter of the downcast shaft.

A Tertiary model evaluated using this technique yields:

INPUT DATA

The values of the parameters used in this run were:-

Name of the Mine - Buf Tertiary

Stope tons per month - 130000

Stoping Width - 1,2 metres

Development/Stoping ratio - 40%

Number of Levels - 8

Vertical distance between levels - 75 metres

Monthly face advance - 20 metres

Depth of down cast shaft - 4100 metres

Depth of up-cast shaft - 3000 metres

Virgin Rock temperature - 63°C

Dip of Reef - 27°

Payability - 70%

Density of Rock - 2.7 tons per cubic metre

Thermal conductivity of the rock - 5.6 watts per metre per °C

Surface Barometric pressure - 87.5 kPa

Surface wet bulb temperature - 18°C

Surface dry bulb temperature - 24°C

Fissure water expectation - 0.1 tons per ton

Electrical power consumption - 30 kW HR per ton

Rated capacity of locomotives - 45kW HR

Refrigeration costs - 1100 Rand per kilowatt

Power costs over 15 years - 6500 Rand per kilowatt

Fan capital costs - 400 Rand per kilowatt

Interval between raises - 300 metres

Average daily black wage - [illegible] Rand per day

Ave. development x-sectional area - 11 square metres

OUTPUT DATA

Air quantity - 500 cubic metres per second

Design temperature - 28.5°C

INDIVIDUAL HEAT LOADS, IN KILOWATTS, WILL BE:

Total heat load - 51103

Air cooling power - 0

Amount of refrigeration required - 51103

NOTES:

The programme is structured in such a fashion that the total heat load does not include auto-compression.

In this example the total heat load including auto-compression is:

51,1 MW
+ 22,2 MW
73,3 MW

CAUTION MUST BE EXERCISED IN USING THIS PROGRAM - SOME OF THE ALGORITHMS ARE VERY SPECIFIC - CONSULTATION WITH THE AUTHOR IS ADVISED.

10.2.2.9 Software program - G.R.E.F.

This computer program originates from the Anglo American Corporation and is available through the Computer Sub-Committee of the Mine Ventilation Society of South Africa (Mine Ventilation Society). It computes heat loads and refrigeration requirements in a choice of three main Gold Mining areas in South Africa using an IBM P.C..

A model representative of the Buffelsfontein Tertiary Shaft gave the following results:

Input values as supplied:

1. Mining Area - Vaal Reefs
2. Name of Mine - Buffelsfontein Tertiary
3. Mean Rock Breaking Depth in metres - 4200*
4. Production rate kTons/month - 180
5. Air mass flow rate in kg/s - 600
6. Reject air Temperature in °C - 27
7. Average air travel distance in metres - 3000
8. Surface refrigeration operating - yes

The following values have been set at:

1. Surface Barometric pressure in kPa - 87.5
2. Surface ambient temperature in °C - 18°/25°
3. Air temperature Leaving B.A.C. - 10°/10°
4. Air mass flow rate in kg/s - 600
5. Station Temperature at mean rock breaking depth °C - 29.61/51.11
6. Air reject temperature in °C - 27,0
7. Service water consumption ton/ton - 0.5
8. Surface storage dam capacity in hours - 24
9. Underground dan capacity in hours - 24
10. Energy recovery position metres above M.R.B.D. - 240 m
11. Turbine efficiency - 0.75
12. Underground plant room position metres above M.R.B.D. - 500
13. Water temperature leaving surface pre-cooler - 22 °C
14. Water temperature leaving evaporator - 0.5 °C
15. Water temperature leaving underground workings - 27 °C
16. Underground plant C.O.P. - 4
17. Surface Plant C.O.P. - 6

18. Annual Power cost - R600/kW

19. Capital cost for surface plant - R1100/kW

20. Capital cost for underground plant - R1300/kW

Output mine planning results:

Buffelsfontein Tertiary

Heat Flow Particulars

Virgin rock temperature at M.R.B.D. °C - 63.86

Mine Heat Production kW/kT - 420.13

Total Mine Heat Load kW - 75624

Service water cooling capacity kW - 3436

Circulating air cooling capacity kW - -4615

Underground refrigeration required kW - 76803

NOTES:

1. No listing or printout of the computer program is possible.

2. As a result of the above, a realistic VRT 63,8°C can only be achieved by "Driving" the calculation with a fictitious mean rock breaking depth, 4200m as opposed to 3843m (which gives a VRT of 58°C).

10.2.2.10 Heat load estimation - CHAMBER OF MINES

HEATFLOW (COMRO GUIDE NO.2) makes use of established models based on simplified assumptions that:

- excavations are produced instantaneously.
- air temperature remains constant.
- rock is homogeneous.
- rock has a uniform initial temperature.

The heat sources and sinks are catagorised and previously tested algorithims applied. Linked in a sequential order representative of the mine layout, calculations within the module are segregated into three discrete operations:

(1) determination of the overall heatflow into the module

(2) calculation of the moisture evaporation rate and

(3) calculation of the change in the air condition

A summary of sources and sinks is shown in Table 10.4 (Von Glehn).

The Environmental Engineering Division of COMRO were contracted to assess the heat loads for the Buffelsfontein Gold Mine Tertiary project (C.O.M. December 1988, March 1989). The model was constructed in detail, using the HEATFLOW principles and the following was reported:

Specification of envisaged mining operation and project details.

The analysis has been done for a point in time when the heat loads are likely to be at their worst. The general specifications listed below were defined in close co-operation with the project team.

- Production will be 130 000 tons/month from stoping plus 50 000 tons/month from development.

- Stoping face advance will be 20 m/month with a stoping width of 1,2 m.

MODULE	MODEL USED	EFFECTS CONSIDERED	ASSUMPTIONS	CONSTRAINTS
SHAFTS	JAEGER & CHAMALAUN	• ADIABATIC COMPRESSION • HEAT FLOW FROM SURROUNDING ROCK • HEAT FLOWS FROM ARTIFICIAL SINKS OR SOURCES	• SHAFTS ARE CIRCULAR • RADIAL HEATFLOW ONLY • SURFACE MOISTURE IS EVENLY DISTRIBUTED	
TUNNELS	HEMP	• HEAT FLOW FROM SURROUNDING ROCK • ARTIFICIAL SOURCES AND SINKS • ADIABATIC COMPRESSION	• TUNNELS ARE RECTANGULAR • SURFACE MOISTURE ON FOOTWALL	
STOPES	VON GLEHN AND BLUHM	• ONE DIMENSIONAL CONDUCTION IN PARALLEL SLABS • OTHER HEAT SOURCES – MEN – MACHINERY – EXPLOSIVES – FISSURE & SERVICE WATER – BACKFILL	• OVERALL LAYOUT DIVIDED INTO PRODUCTION AND WORKED OUT	• TOTAL HEAT FLOW INTO THE AIR ONLY, NO SEPARATION BETWEEN SENSIBLE AND LATENT HEAT
DEVELOPMENT ENDS	WHILLIER AND RAMSDEN	• HEAT FLOW FROM ROCK • SERVICE AND FISSURE WATER • OTHER HEAT SINKS OR SOURCES	• TREATED AS MODULES WITH SINGLE INLETS AND OUTLETS • THREE TYPES CAN BE MODELED	• TOTAL HEAT WITHOUT SENSIBLE AND LATENT DISCRIMINATION
AIR COOLERS	STANDARD EQUATIONS	• HEAT TRANSFER DETERMINED EITHER – FOR ESTIMATE OF COOLING DUTY – ESTIMATE OF OUTLET AIR TEMPERATURE	• AIR LEAVING IS SATURATED	
HOISTS	STANDARD EQUATIONS	• SENSIBLE HEAT GAIN ONLY	• HEAT LOAD TO BE A PROPORTION OF THE RATED POWER	
FANS	STANDARD EQUATIONS	• SENSITIVE HEAT GAIN ONLY	• TOTAL INPUT FAN POWER TRANSFERRED TO THE AIR	
SPOT HEAT SOURCES	STANDARD EQUATIONS	• LATENT AND SENSITIVE HEAT GAINS ACCOUNTED FOR	• PROPORTIONED HEAT GAINS DEPENDENT ON SOURCE i.e. ELECTRIC MOTORS, PUMPS ETC.	
SHAFT STATION AREAS	WHILLIER AND RAMSDEN	• SMALLER ITEMS OF HEAT GENERATING EQUIPMENT • THE MAZE OF TUNNELS	• RADIUS FROM SHAFT, 200 m	• LARGER INDIVIDUAL ITEMS THAT HAVE A SIGNIFICANT IMPACT SHOULD BE TREATED SEPARATELY i.e. – Refrigeration machines – Booster fans – Main dewatering pumps
SUB-NETWORKS		• TO MODEL COMPLICATED SUBREGION OF A MINE	DETAILS OF THE SUB-REGION ARE VIEWED INDEPENDENTLY	• MUST NOT CONTAIN FURTHER SUB-NETWORKS • ONE INLET AND ONE OUTLET

TABLE 10.4

C.O.M. HEATFLOW MODULE DESCRIPTION LISTING EFFECTS, ASSUMPTIONS AND CONSTRAINTS

- Production will take place between 42 and 49 Levels with 60 percent coming from 46, 47 and 48 levels.

- Face lengths will vary from 54 to 110 m with a mean of 80 m.

- All mining will be within a 3 000 m boundary (maximum level distances 3 000 m).

- Production will come from five lines spaced at 300 m. Three lines will be producing, one will be under development and one will be almost worked out. (Vamping).

- Dip gulleys will be spaced at 100 m and slusher gulleys on strike will be used in conjunction with continuous scrapers.

- The surface temperature conditions are 18°C wet-bulb and 24°C dry-bulb; the virgin rock temperature at a depth of 4 162 m is 67,1°C.

- Service water consumed in the actual stopes and development will be 0,5 ton of water per ton rock broken.

  The temperature of the service water arriving in the working place will be 18°C. The temperature increase of the service water in the stopes will be 8°C. There is no fissure water expectation.

- All haulages and intake airways will have a cross-section of 11,1 m^2. There will be seven 37 kW diesel locomotives operating on each working level.

- The environmental conditions will be such that a cooling power of at least 300 W/m^2 will be created. (This interprets to a design wet-bulb temperature of not greater than 30°C in areas where the air velocity can be expected to be at least 2 m/s (intake system) and a design wet-bulb of not greater than 27°C in the stoping regions where areas of low air velocity might be encountered).

Method of analysis:

The method of analysis is based on building up the mine layout in the form of a network of heat sources and sinks (in this case a total of 300 building blocks). The heat flow analysis is then done for each building block in turn enabling their interactions to be studied and the minewide heat load determined.

Overall heat load audit:

The total heat load experienced by the system between the point where the air enters the shaft on surface and the various points of return underground, is made up as follows:

- Auto-compression		25,6 MW
- Intake system	- heat from rock	28,4 MW
	- heat from machinery	8,7 MW
- Stoping		16,4 MW
- Development		7,9 MW
		87,0 MW
Residual cooling in ventilation air		-9,0 MW
		78,0 MW

Hence the underground cooling requirement is 78 MW.

Conclusions:

For an average backfill-to-face distance of 3,5 m and 60 percent of the worked out area (excluding gullies) filled and a ventilation flow of 600 kg/s, the underground cooling requirement amounts to 78,0 MW and is made up of:

Primary bulk air cooler	(1 off) total	24,0 MW
Secondary bulk air coolers	(6 off) total	26,0 MW
Tertiary air coolers	(32 off) total	26,5 MW
Average cooling achieved in stopes and developments with chilled service water		1,5 MW

11. SPECIFIC COOLING POWER AS A TERTIARY HEAT STRESS INDEX

11.1 Introduction

Historically no fewer than 90 heat stress indices have been developed to consolidate heat stress factors into a single value that can be used to predict the level of heat stress resulting from exposure to the environmental conditions.

Generally valid for a limited range of environmental conditions very few of the indices proved representative in our deep hot mines. From the work undertaken by J M Steward, he refers and concludes:

"The major problems tackled are the development of a rational heat transfer based method for assessing heat stress, the provision of accurate heat transfer equations, and the formulation of heat stress limits for essentially nude men working in South African gold mines.

It is shown that the cooling power of an environment may be used to quantify heat stress, and that cooling power values also serve as heat stress limits if the values of mean skin temperature and sweat rate, upon which cooling power depends, are linked to an upper safe level of body temperature".(Steward)

The cooling power concept is linked to a maximum safe rectal temperature at which a man will experience a negligible risk of suffering heat stroke. This limit has been set at 40°C associated with a risk of one in a million. It follows then that men will be able to achieve a safe equilibrium rectal temperature in all instances where the calculated cooling power equals or exceeds the rate at which metabolic heat is being generated by the men in question. When metabolic heat generation and cooling power are equal, it follows that different environments having the same cooling power may be assessed as having the same heat stress, in that they will all be associated with the same rectal temperature.(Environmental Engineering: 519)

The cooling power heat stress index more commonly known as SCP or SPECIFIC COOLING POWER has been developed into a sophisticated form which incorporates the following environmental parameters:

(a) Wet Bulb Temperature (ventilated wet bulb)

(b) Dry Bulb Temperature

(c) Radiant Temperature

(d) Air Velocity

(e) Barometric Pressure.

Easily managed and reproducible equations have been programmed to quickly evaluate SPECIFIC COOLING POWER for any combination of input parameters.(Murray-Smith)

11.2 International Standards

At this time the five major institutions who have legislated heat stress exposure indices and limits are embodied in:

(i) Climatic Regulations in the Hardcoal Industry of the Federal Republic of Germany;

(ii) The American Conference of Government Industrial Hygienists (ACGIH);

(iii) Occupational Safety and Health Administration (OSHA);

(iv) Standards Advisory Committe on Heat Stress (SACHS);

(v) American Industrial Hygiene Association (AIHA)

All parties are presently reviewing their criteria and when re-adjusted it is expected that the upper limits of Wet Bulb Globe Temperature (WBGT) and Effective Temperature (ET) will lie within the GENMIN standard of 300 S.C.P. (Kielblock)

11.3 Limits of S.C.P.

A good and safe indication of the rates of metabolic heat production associated with various mining tasks are given in Table 11.1 (Environmental Engineering: 541), which are the result of extensive studies conducted by the Chamber of Mine Industrial Hygiene Division.

LIGHT WORK UP TO 115 W/m²	MODERATE WORK UP TO 180 Wm/²	HARD WORK UP TO 240 W/m²
Winch operation Sweeping Drill assistant Walking Drain cleaning	Building stone walls Operating box holes Drilling Barring Building matt packs Team leaders duties	Tramming Shovelling Timber transport in stopes

TABLE 11.1
CLASSIFICATION OF MINING TASKS ACCORDING TO METABOLIC HEAT PRODUCTION FOR THE USE IN THE ASSESSMENT OF HEAT STRESS

Although higher rates of metabolic heat production than those indicated in this Table for the various tasks may occasionally occur, it is extremely unlikely that they will be sustained for long periods. Table 11.1 may, therefore, be used as a good basis for assessing the safety of the thermal environment in gold mines. Similar tables would be required for other industries.

The International Standards Organization has defined the work categories with a slightly higher limit (that is, more cooling is required) which are shown in Table 11.2 (Environmental Engineering: 258)

ISO CLASSIFICATION	RATE (W/m^2)	CHAMBER OF MINES UPPER LIMIT (W/m^2)
Low	66 - 130	115
Moderate	131 - 200	180
High	201 - 260	240

TABLE 11.2

INTERNATIONAL STANDARDS ORGANIZATION (ISO) AND CHAMBER OF MINES WORK RATE CLASSIFICATION

Other literature has suggested a hard work upper limit of 280 W/m^2 (Whillier and Steward).

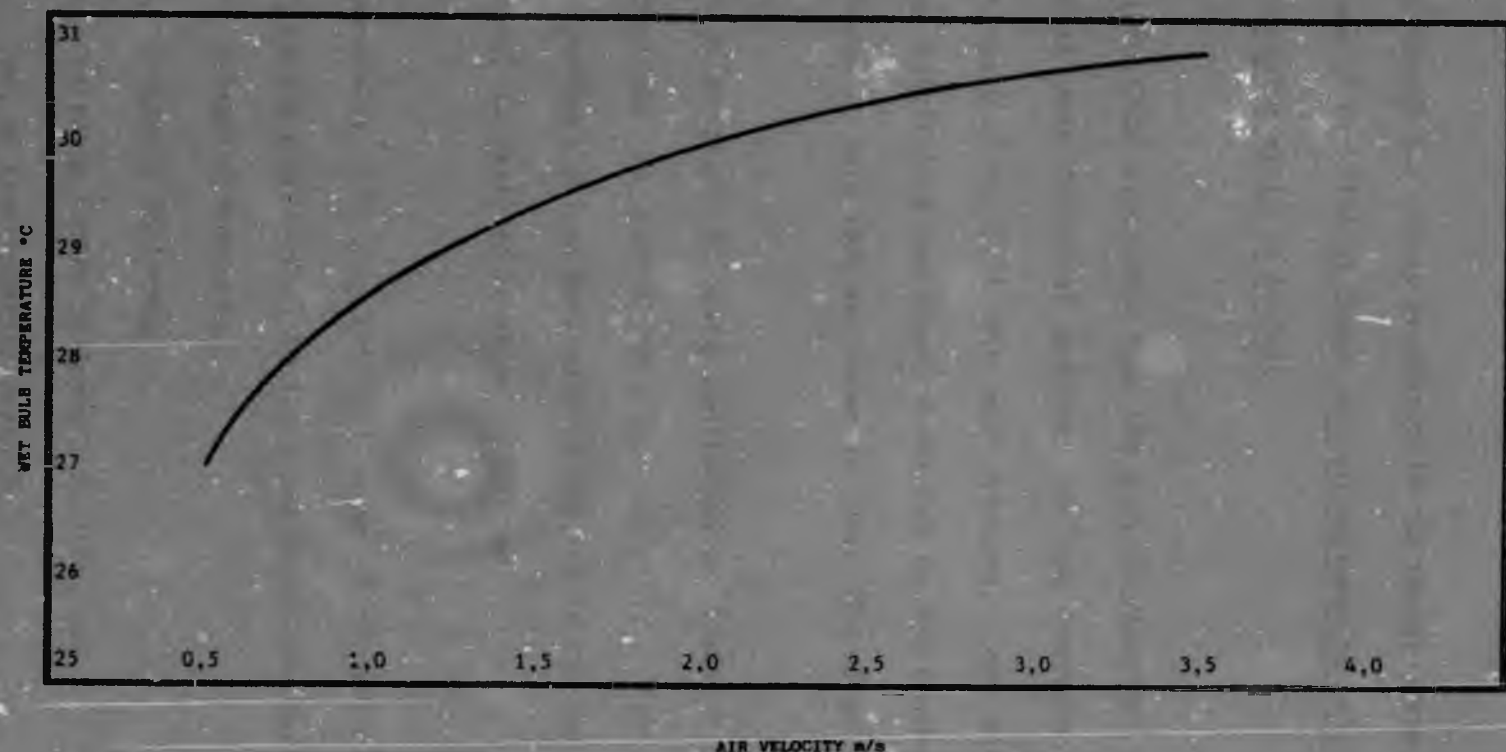
11.4 Equivalent 300 SCP

The following combinations of Wet bulb temperatures and stope face velocities as shown in Figure 11.1 are equivalent to an SCP of 300 W/m^2 and are indicative of conditions in the Tertiary mining area. A graphical representation is also given.

11.5 Implications

The mine heat load and resultant refrigeration requirements were based on the fundamentals of 300 W/m^2 SCP with a design stope reject wet-bulb of 27°C. By allowing stope reject wet bulb to increase to 30°C (but compensating by increased air velocity to maintain 300 W/m^2 SCP) the <u>mine heat load is expected to reduce by approximately 10%.</u>

	AVERAGE STOPE FACE AIR VELOCITY (m/s)																
	0,5	0,6	0,7	0,3	0,9	1,0	1,1	1,2	1,5	2,0	2,5	3,0	3,5	4,0	4,5	5,0	10	20
°C WET BULB	27,0	27,5	27,8	28,1	28,4	28,6	28,8	29,0	29,5	30,0	30,4	30,7	30,9	31,1	31,2	31,5	32,2	32,6



FIGUPE 11.1

STOPE WET BULB TEMPERATURE AND AIR VELOCITIES EQUIVALENT TO 300 W/m^2 SCP

12. BACKFILL

Although not widely practised in the Genmin group, the use of backfill at depth, was an engineering principle accepted early in project.

Industrial experience over the past 10 years has shown that most technical problems in the conveying and stowage of backfill have been overcome. Some details of backfilling operations monitored by the Chamber of Mines to July 1986 are shown in Table 12.1 (Gay).

A pilot plant is being considered at Southern Shaft designed to supply deslimed backfill. Practical experience gained here will be applied in the Tertiary which is designed to stowe 30 000 tpm.

Generally stated, the following occurrences are directly proportional to increased depth from surface:

- Increased Pressure;
- Rockburst Incidence;
- Increased Number of Remnant Pillars, and
- Cost of Environmental Services.

	WESTERN DEEP LEVELS	EAST DRIE-FONTEIN	WEST DRIE-FONTEIN	VAAL REEFS	FREDDIES	RANDFONTEIN ESTATE
Type of fill	Milled Waste Rock	Deslimed Tailings	Deslimed and De-watered Trailings	Deslimed Trailings	Dewatered and Cemented Dewatered Trailings	Cemented Dewatered Trailings
Depth range (km)	2,5 – 3,0	2,1	2,0	2,3	1,5	0,8
Stoping width (m)	1,0	1,0 – 3,0	1,0	1,3	1,0	5 – 15
Mining method	Longwall	Longwall	Longwall	Scattered	Scattered	Room and Pillar
Purpose	Regional support, reduce seismicity, rockburst damage	Local support of middling between reefs, regional support, reduce seismicity, rockburst damage	Regional and local support, reduce seismicity, rockburst damage	Local support, reduce seismicity, rockburst damage	Local support for shale hangingwall	Local support for wide stopes, pillar extraction multi-reef mining
Start of filling	October 1983	1982	August 1981	1980	January 1985	March 1985
Dry tons placed	10 220	300 000	72 000	42 000	11 000	200 000
m^3 placed	6 000	65 000	48 000	22 000	5 000	20 000

TABLE 12.1

BACKFILL STATISTICS FOR SOUTH AFRICAN GOLD MINES

As a result, economic justification for the use of backfill is multi-disciplinary and the application and use of backfill in deep mines has two categories of benefit:

12.1 Primary Benefits

- Reduced Seismicity;
- Reduced Rockburst Damage;
- Reduced Rockfalls;
- Increased Reef Extraction, and
- Improved Environmental Conditions.

12.2 Secondary Benefits

- Increased Transport Capacity;
- Reduced Fire Incidence;
- Improved Gold Recovery, and
- Improved Feeling of Security.

12.3 Environmental Impact

The environmental advantages of backfill include:

- a reduction in heat load (stope heatflow and the mine in general due to less autocompression);
- a reduction in air leakage;
- improved face velocities, and
- creating stope conditions enabling 300 W/m^2 to be easily maintained.

12.4 Tertiary Backfill Specifications

It is envisiaged the backfill geometry in the Tertiary mining block will be determined by rock engineering requirements. The specifications for placement and span are shown in Table 12.2.

PERCENTAGE BACKFILL	60 %
MAXIMUM FILL TO FACE DISTANCE	5,0 m
MINIMUM FILL TO FACE DISTANCE	2,0 m
AVERAGE FILL TO FACE DISTANCE	3,5 m

TABLE 12.2
BACKFILL SPECIFICATIONS FOR THE TERTIARY

The effects of backfill have been estimated in the Tertiary context and both the positive and negative aspects addressed.

12.5 Heat Load Reduction

Two of the greatest influencing parameters on heat reduction in backfilled stopes are:

(1) the percentage of backfill

and (2) the fill to face distance.

From work previously undertaken by Matthews and a study by the Environmental Engineering Laboratory, emperical curves have been developed to show the effect of these parameter changes on the Tertiary mining horizon.

12.5.1 Precentage backfill

Figure 12.1 shows the effect of 60 % backfill on the stope heat (COMRO March 1989). A significant reduction of 35 % from the rock is expected assuming a fill to face distance of 3,5 m. This reduces the potential stope face heat load due to rock from 400 kW to 260 kW. A complete stope and continuous scraper heat prediction is later shown in Table 14.2.

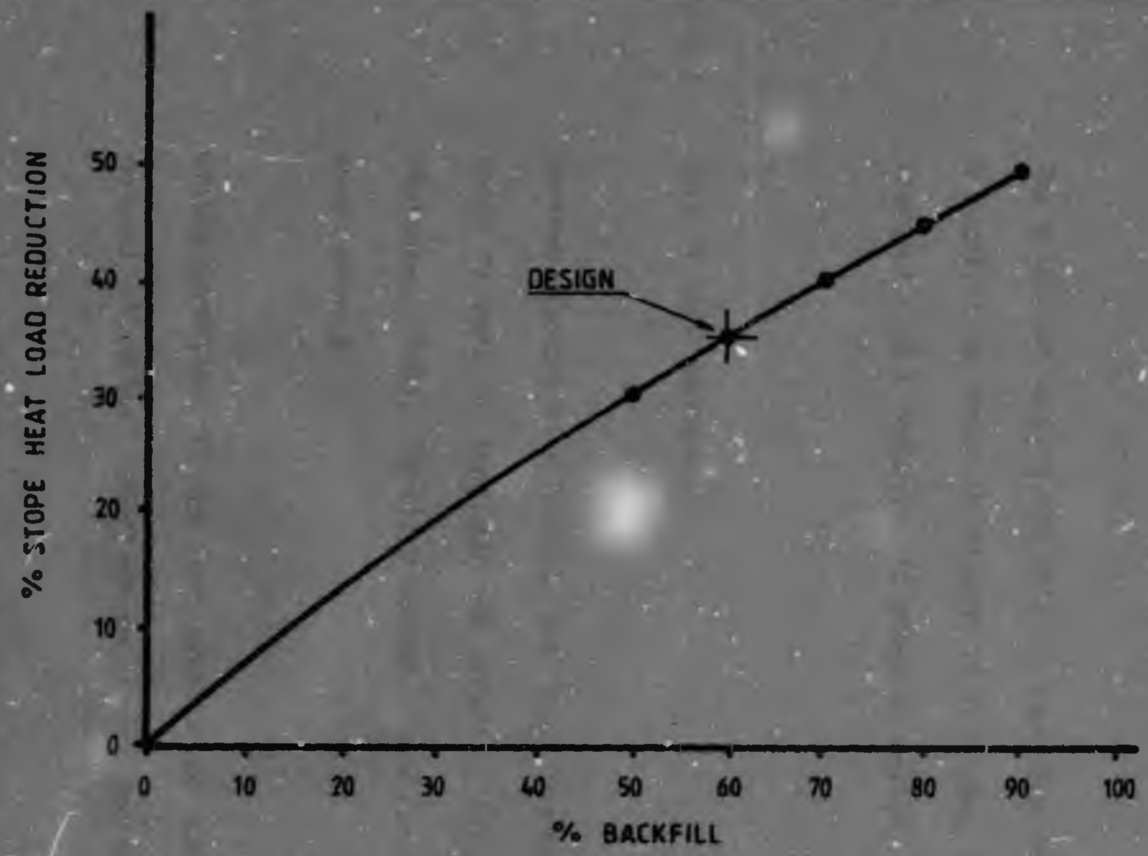


FIGURE 12.1

TERTIARY STOPE HEAT LOAD REDUCTION AS A FUNCTION OF PERCENTAGE BACKFILL

Figure 12.2 is an illustration of the total stope heat components and heat sinks envisiaged in the Tertiary.

12.5.2 Fill to face distance

The effect of varying the fill to face distance at 60 % backfill is shown in Figure 12.3 (Blunm, Vost and Steward) A 35 % reduction is shown at design.

12.5.3 Rate of face advance

Another parameter affecting the heat ingress to a stope is rate of face advance shown in Figure 12.4 (Matthews, 1989). Tertiary design allows for 20 m per month advance.

Mining permitted, if the face advance per month were reduced, stope heat reduction would also decrease.

The merits of percentage backfill are shown by the family of curves showing 90% backfill.

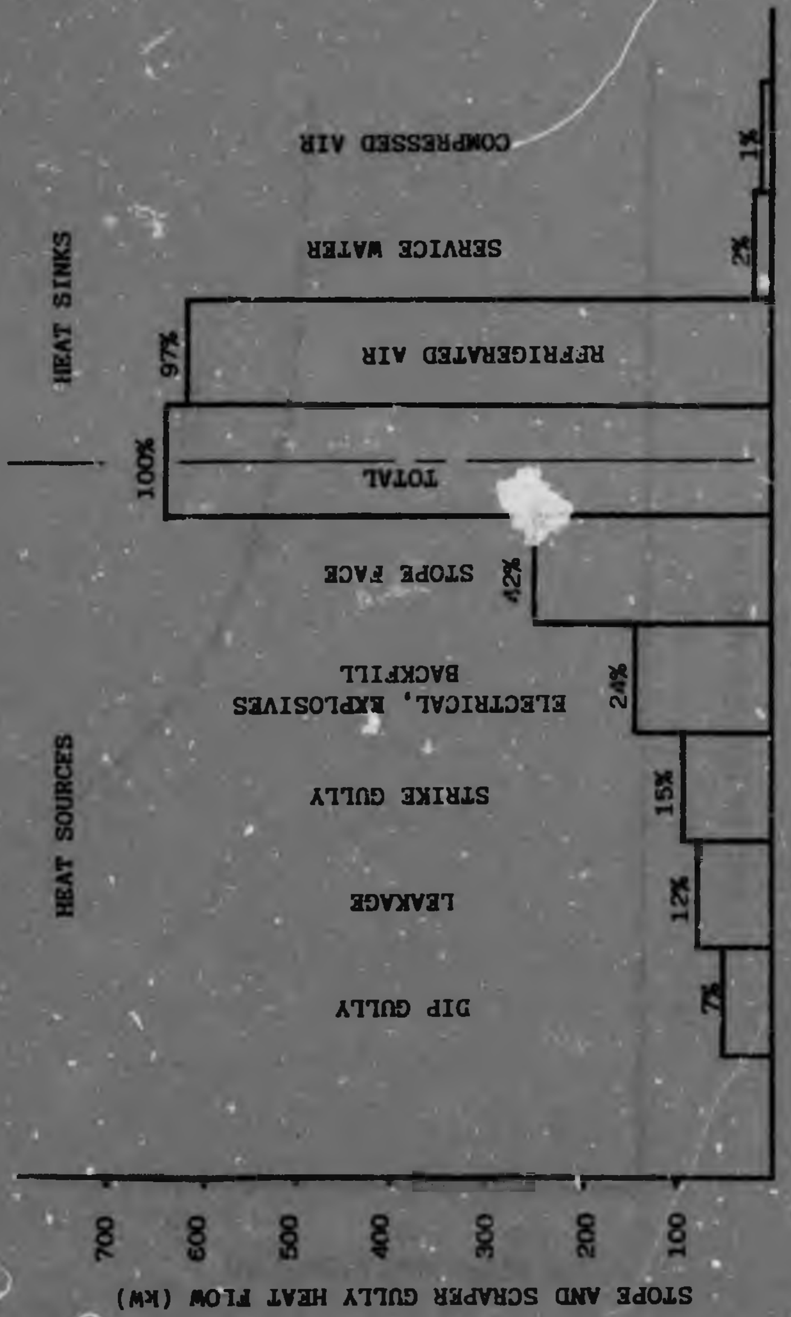


FIGURE 12.2
TERTIARY STOPE HEAT LOADS AND HEAT SINKS WITH BACKFILL

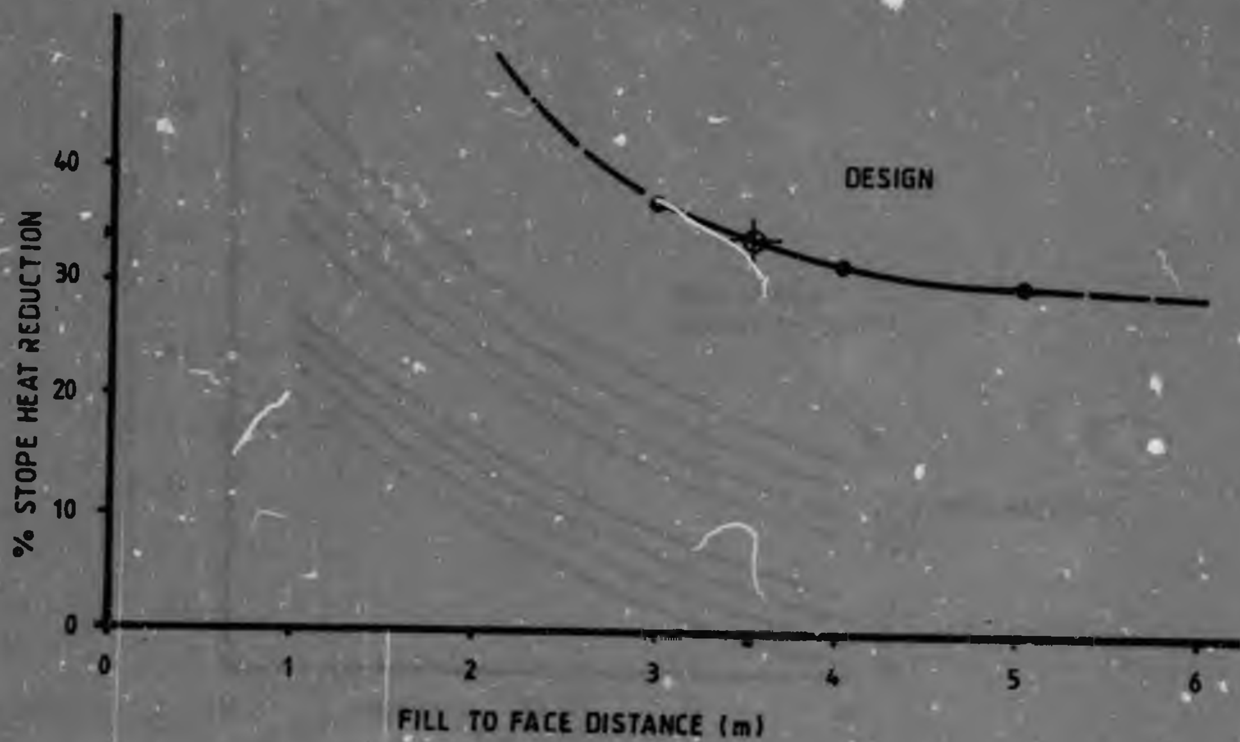


FIGURE 12.3
TERTI[illegible] STOPE HEAT LOAD REDUCTION AS A FUNCTION OF FILL TO FACE DISTANCE

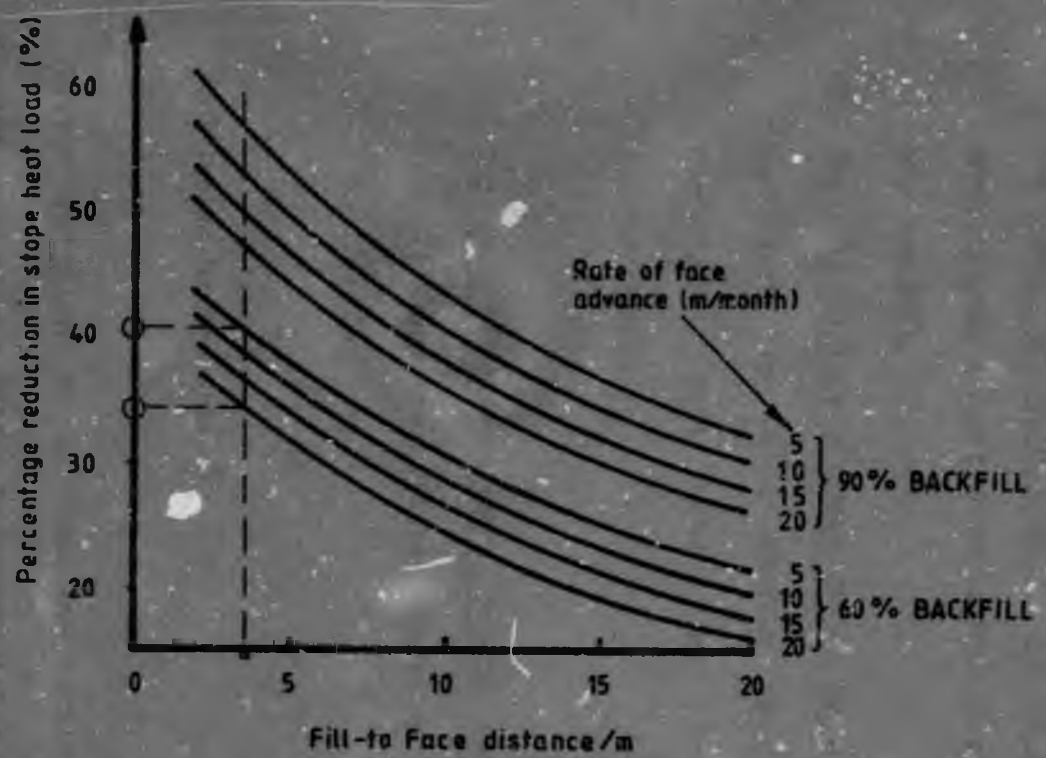


FIGURE 12.4
TERTIARY STOPE HEAT LOAD REDUCTION AS A FUNCTION OF FILL TO FACE DISTANCE, % BACKFILL AND FACE ADVANCE

12.5.4 Summary

Ideally therefore, for stope heat load minimization:

(1) the highest degree of backfill must be practised which becomes a function of logistical placement, time and cost.

(2) the fill to face distance should be minimized, constraints to be a minimum required for operational open area and ventilation considerations.

(3) the rate of face advance reduced (but often impossible due to production requirements).

12.6 Ventilation Requirements

Design air flows in an idealized Tertiary stope configuration are detailed in Section 3.1. A summary of the results is shown in Table 12.3.

	QUANTITY (m^3/s)	MASS FLOW (kg/s) w = 1,[illegible] kg/m^3
AIR INTO CROSSCUT	25	36
AIR DESIGN TO STOPE	19	27
AIR ACTUAL TO FACE	15	21
AIR LEAKAGE	4	6
AIR INTO SLUSHER GULLIES	6	9

TABLE 12.3

TERTIARY AIRFLOW SUMMARY

Due to the limited face cross sectional area of 4,9 m^2 (1,4 m stoping width maximum, and a fill to face distance of 3,5 m) and a maximum design face velocity of 3 m/s (to prevent excessive dust pick up) an effective allowable air quantity of 15 m^3/s is prescribed. The interaction of air velocity and quantity in backfilled Tertiary stopes is shown in Figure 12.5. Similar effects were reported by Matthews (1988).

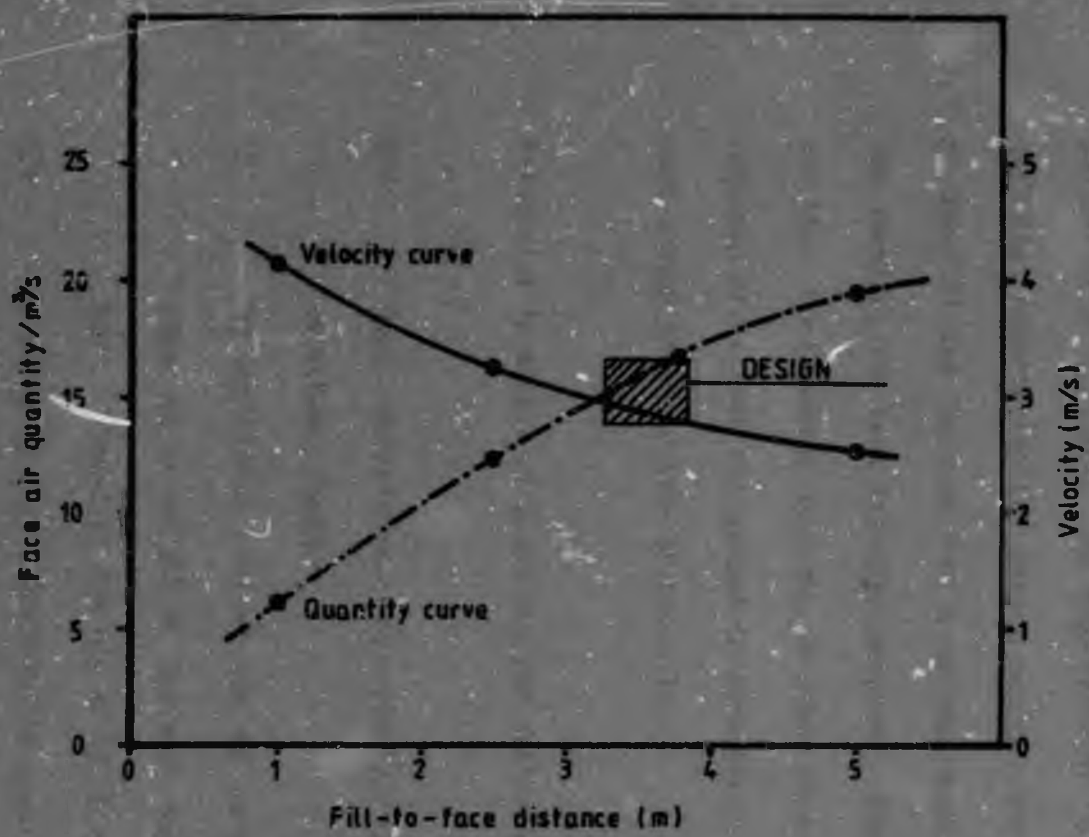


FIGURE 12.5
TERTIARY STOPE FACE AIR QUANTITY AND VELOCITY AS A FUNCTION OF FILL TO FACE DISTANCE

A stope face velocity of 3 m/s has been used as a design, which by industry standards is high. Conventional mining practices reflected in the Chamber of Mines Annual Return report industry average velocities of 0,85 m/s. The high velocity is however, justified considering the mining method, high degree of backfill and the need for a maximum mass flow in the Tertiary for heat transfer purposes (Sections 14 and 15). Furthermore, to date, no other mine has been required to exploit all ground in circumstances using such a high concentration of backfill.

Velocities in excess of 3 m/s are not desirable since:

(1) Dust entrainment is likely to occur (Matthews, 1988).

(2) No additional cooling due to velocity is expected as an SCP of 300 W/m² would have been achieved at 2,0 m/s at 30 °C wet bulb.

(3) Working in high velocities is ergonomically unpleasant.

(4) A power penalty due to high stoping resistance and subsequent pressure drop will be incurred.

12.7 Air Control

Due to the limited design air mass flow to the Tertiary, 600 kg/s or (420 m^3/s), strict air management and control will be imperative in the stoping operation. The backfill strategy is expected to assist in confining air along intake strike gullies with the assistance of well constructed high integrity, properly sealed dip and strike stoppings and well constructed and maintained brattices. An air control efficiency of 85 % is expected shown in Figure 12.6 (Matthews, 1989).

The drop-off in performance at extremely close face-to-fill distance, is due to air being forced to bypass via brattices and seals.

12.8 Tertiary Backfill Resistance

A simplified resistance model of backfilled airways is shown in Figure 12.7

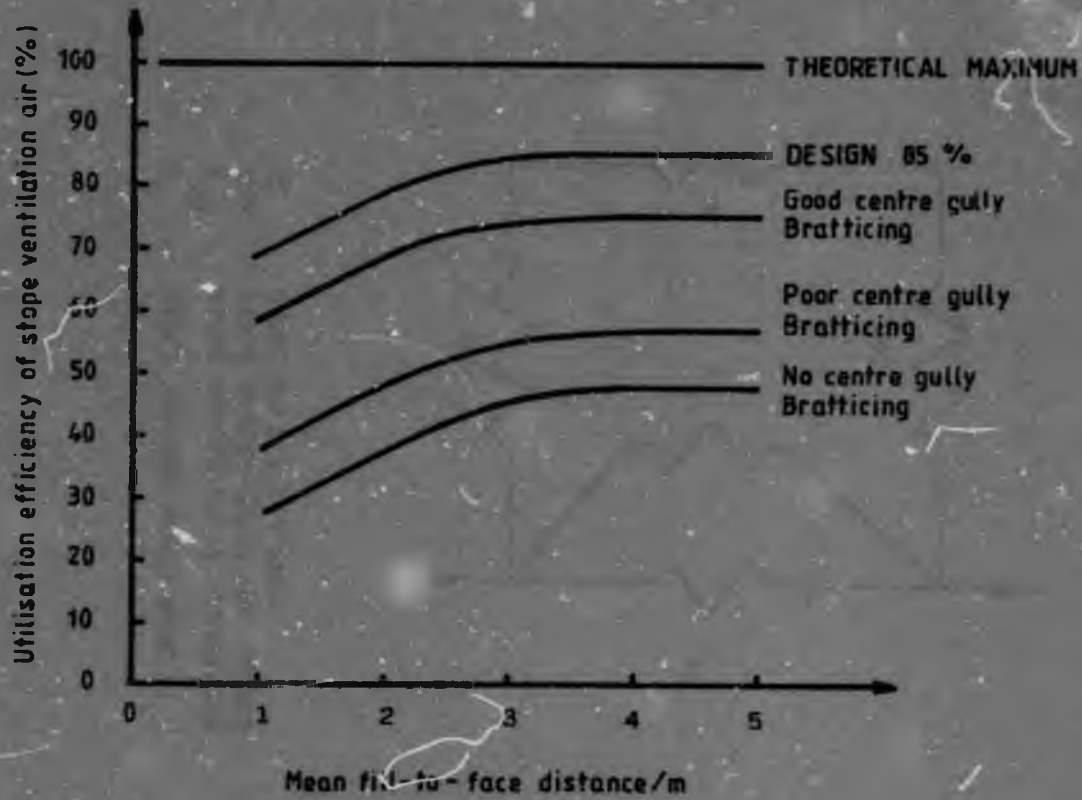
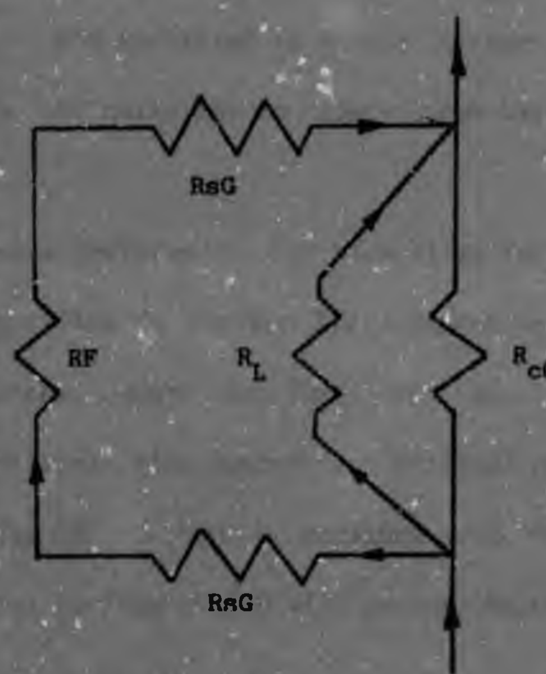


FIGURE 12.6
UTILIZATION OF STOPE AIR AS A FUNCTION OF FILL TO FACE DISTANCE



RF	STOPE FACE RESISTANCE
RsG	STOPE STRIKE GULLY RESISTANCE
R_{cG}	STOPE CENTRE GULLY RESISTANCE
R_L	STOPE LEAKAGE RESISTANCE

FIGURE 12.7
TERTIARY STOPE RESISTANCE

RESISTANCE VALUES:

(1) RF are dependent on fill cycle.

(2) RSG minimized to allow adequate inflow of air.

(3) RCG maximized to reduce leakage.

(4) RL maximized to reduce stope backfill leakage.

System resistances for backfilled Tertiary stopes are quantified in Section 7.2. Depending on a number of factors, such as face profile, stoping width, additional face support and residual ore, the absolute value of 190 Pascals pressure drop for the stope as shown in Figure 12.8 will change (Matthews, 1989).

Despite the relative position of the curve, its rapid steepness increase at fill-to-face distances less than 3,5 m would make stope resistance and pressure drops prohibitively high.

12.9 Conclusion

As a compromise between minimum heat flow and a maximum operating pressure, a mean 3,5 m fill to face distance appears to be optimal.

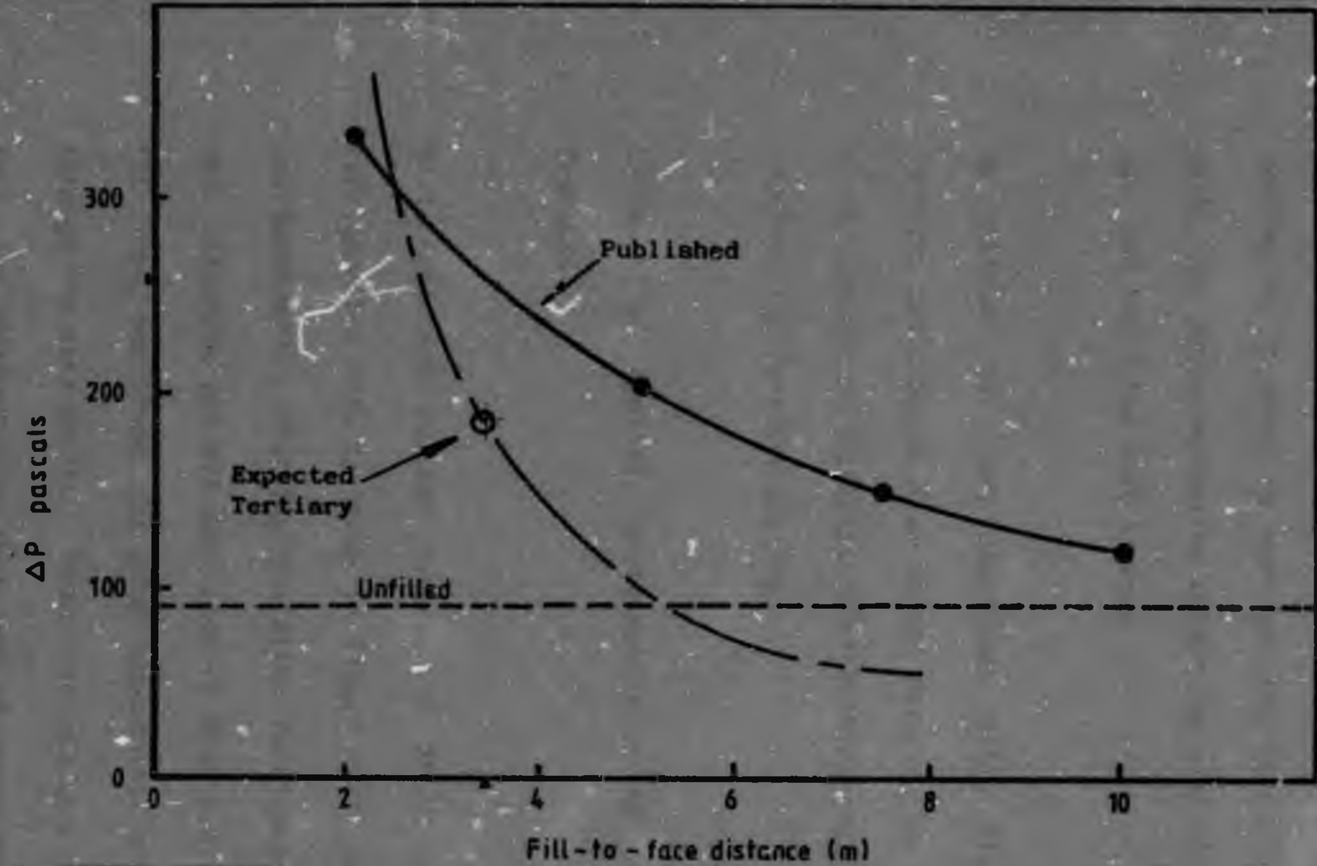


FIGURE 12.8

STOPE PRESSURE LOSS AS A FUNCTION OF FILL TO FACE DISTANCE

13. APPLIED TERTIARY REFRIGERATION

From the mine heat load predictions, 78 Megawatts of cooling is required to overcome the heat flow from rock, the auto-compressive effect on the circulated air, and the other heat sources.

An estimate has been undertaken using historical industry performance, from Annual Returns, as a guide to the amount of installed refrigeration that may be required for Tertiary cooling.

13.1 Estimate of Applied Refrigeration using Industry Averages

Historical and current Chamber of Mines returns' refrigeration data is shown in Figure 13.1 (Appleman and Gardiner). Plotting installed refrigeration in use (kilowatts per kiloton) against mean V.R.T, best fit lines, one linear the other exponential are drawn through the data points. These lines bound a range of installed refrigeration that can be expected to be required to overcome the theoretical mine heat load of 78 MW.

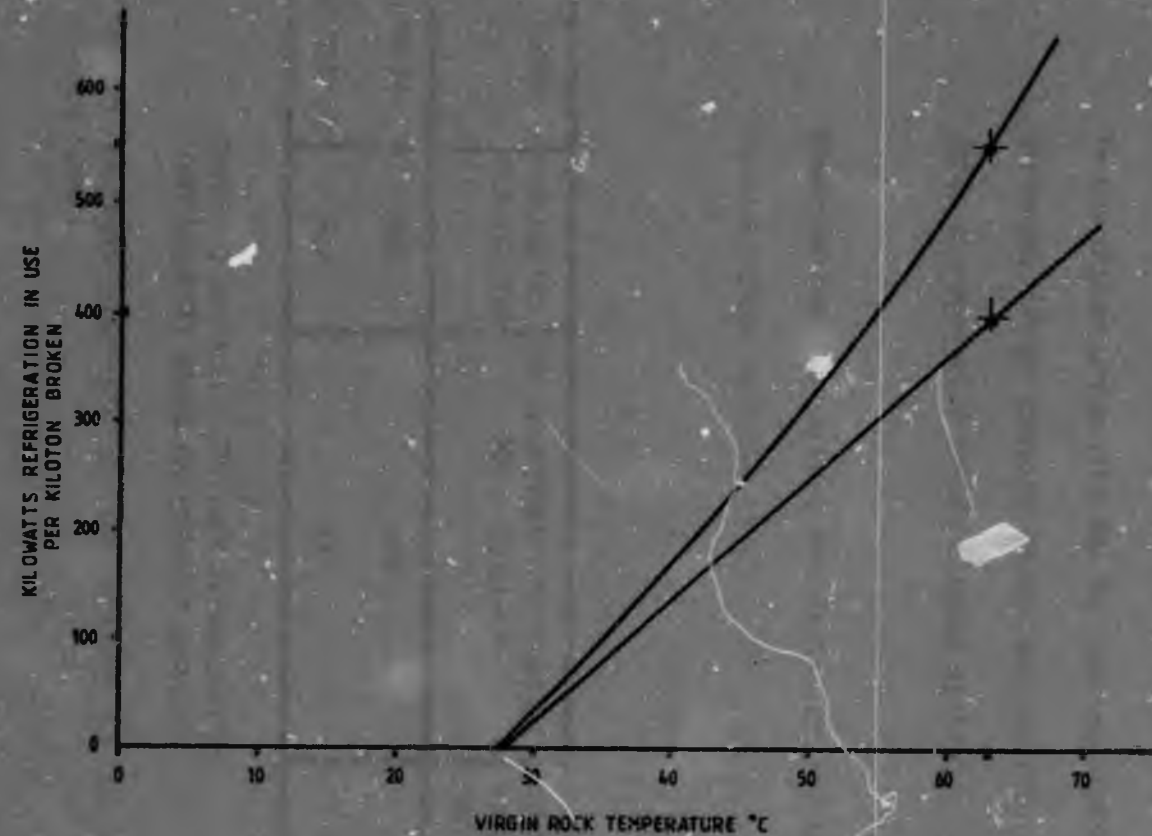


FIGURE 13.1
KILOWATTS OF REFRIGERATION IN USES AS A FUNCTION OF VRT

This approach is only a rough estimate. The losses in any refrigeration system are a function of the system geometry and vary greatly. In the Tertiary case, it is expected that the installed refrigeration will be in the higher proportion due to the long reticulation distances, high pumping heads and multiplicity of water storage wells in the cascade system.

Based on the two curves, the installed refrigeration range is shown in Table 13.1.

CURVE	KILOWATTS PER KILOTON	23% ADJUSTMENT FOR SURFACE PRE - COOLING*	INSTALLED FRIDGE AT 180 kt/m
LINEAR	400	490	88
EXPONENTIAL	550	677	124

TABLE 13.1
EXPECTED INSTALLED TERTIARY REFRIGERATION REQUIREMENTS BASED ON INDUSTRY CAPACITY

* Industry averages show 23% of the cooling to be done by pre-cooling towers.

13.2 Re-use of Existing Plant

Although new technologies such as Hydrolift devices (3 - Chamber Systems) and the Split-Ammonia system present enormous potential, the extension of conventional systems is considered to be the <u>alternative associated with the least technical risk</u>.

Due to the existing refrigeration plant distribution throughout the Southern and Strathmore complex and the in-situ water reticulation networks associated therewith, primary consideration was given to the maximum re-use of this existing infrastructure. Location and duty of plants has been clarified in Section 1.4.2

13.2.1 Alternative 1

The creation of an infrastructure at Eastern, and Strathmore coupled to the Tertiary shaft is possible. By using as much of the existing equipment as practicable at these shafts, in conjunction with available refrigeration plant from Southern shaft, the required maximum 124 MW(R) to achieve the necessary cooling can be assembled.

Notably 45 MW(R) is forseen as a new installation at Eastern shaft supplemented by a full 24 MW(R) at the existing mid-shaft cooling installation. The chilled water from surface will supply the condensers for heat rejection purposes on 46 and 49 levels. Approximately 23 MW(R) and 32 MW(R) are expected to cope with the demands in the Tertiary, and condenser water will be piped to surface at approximately 45°C or higher. From the evaporators on 46 level cold water will be supplied to spray chambers on 46, 45, 44 and 43 levels, while the 49 level installation will serve coolers on 49, 48 and 47 level.

Depending on the depletion programme, availability of refrigeration plant may vary from the schedule, however no serious problems are forseen.

Since the performance of the existing plant cannot be altered significantly, the relatively high evaporating temperature is expected to deliver water at approximately 6°C. With the further unknowns such as spray chamber performance and varying mass flows of air, return water temperatures ranging from 10°C to 20°C may be expected.

The ultimate performance of these machines is somewhat suspect at low returning water temperatures and may have to be supplemented with machines capable of generating water at almost 0°C.

With the high temperature condenser water returning to surface, pre-cooling of this water is expected to have a lot of potential.

13.2.2 Alternative 2

By expanding the surface and underground refrigeration capacity at Southern shaft and by increasing the heat exchange at 27 level mid-shaft cooler, Strathmore, a second option is available. Although feasible the scheme requires almost twice the mass flow of water to effect the heat transfer which results in a considerably higher pumping cost. Three very important advantages are inherent to the scheme;

- No major re-location of refrigeration plant is necessary, simply additions to established plant-rooms.

- No underground plant-rooms are required within the Tertiary block.

- No high temperature return condenser water is piped, which poses a potential hazard in the event of rupture in intake airways.

The major disadvantage lies in the <u>inflexibility of the system.</u> Water arrival temperatures to Secondary and Tertiary coolers are expected to be constant within the region of 7°C. With the uncertainty of the spray chamber performance (of which some 46 are estimated to be required with a minimum duty of 1 MW) and the possible variation of mass flow of air on each level, the design return water temperature to the evaporators, of 19°C may be optimistic. Based on 52 MW(R) required, this implies a minimum water flow rate of 1034 ℓ/s. A 5°C drop in return water temperature has serious implications. Either the feed temperature must compensate, which it can't or the flow rate must increase to:

$$\frac{52\ 000}{(14-7)(4{,}187)} = 1774\ \ell/s \text{ for the same duty}$$

13.2.3 Summary

Both schemes are technically feasible. The mass of water circulated in each proposal however, has a direct effect on the capital and operating cost of both.

In all likelihood a hybrid system exploiting the best features of both must be sought.

GENMIN has a software package called, NETSIM, which combines simulation and C.A.D. techniques to produce mass and thermal energy balances of piping networks. A mathematical model of the process flowsheet can be constructed using this technique which can subsequently evaluate the performance of critical components such as :

- Heat exchangers.
- Cooling coils.
- Spray chambers.
- Pumps.
- Refrigeration Plants.

Refrigeration reticulation scenarios will have to be modelled and evaluated to maximize the installation efficacy and optimize efficiency.

13.3 Other Technical Considerations

13.3.1 Use of ice

A preliminary study by the Environmental Engineering Division of COMRO, (Ramsden and Bluhm) came to the following conclusions:

1. The use of ice would be technically feasible and economically advantageous.

2. Only particulate ice, as made by conventional ice makers should be considered. Slurry ice requires further research before it can be used in a practical system.

3. Clear advantages for the ice system are evident when capital cost and energy consumption are compared. Table 13.2 summarizes the details.

	ICE SYSTEM	WATER SYSTEM
INSTALLED REFRIGERATION	78,0 MW(R)	110 MW(R)
INSTALLED MW(E)	27,7	40,9
CAPITAL COST (R)	104 MILLION	153 MILLION

TABLE 13.2

REFRIG[illegible] IN COSTS COMPAR[illegible]NVENTIONAL AND ICE APPLICATIONS

4. The prop[illegible]ater system is preferred as a practi[illegible] conservative method. The significant cost advantages of the ice system are reduced because of the existence of suitable water chillers that can be inexpensively acquired for the project.

13.3.2 Use of Split-Ammonia Systems

Thermodynamically, the benefits of using a split-ammonia system at great depths are undeniable. An analysis has shown that at Tertiary design depths, a 78 MW(R) ammonia system could cost approximately R130 Million in capital with an installed electrical requirement of 25 MW(E). This by comparison to conventional water and particulate ice is considerably less. A schematic is shown in Figure 13.2 which outlines the process.

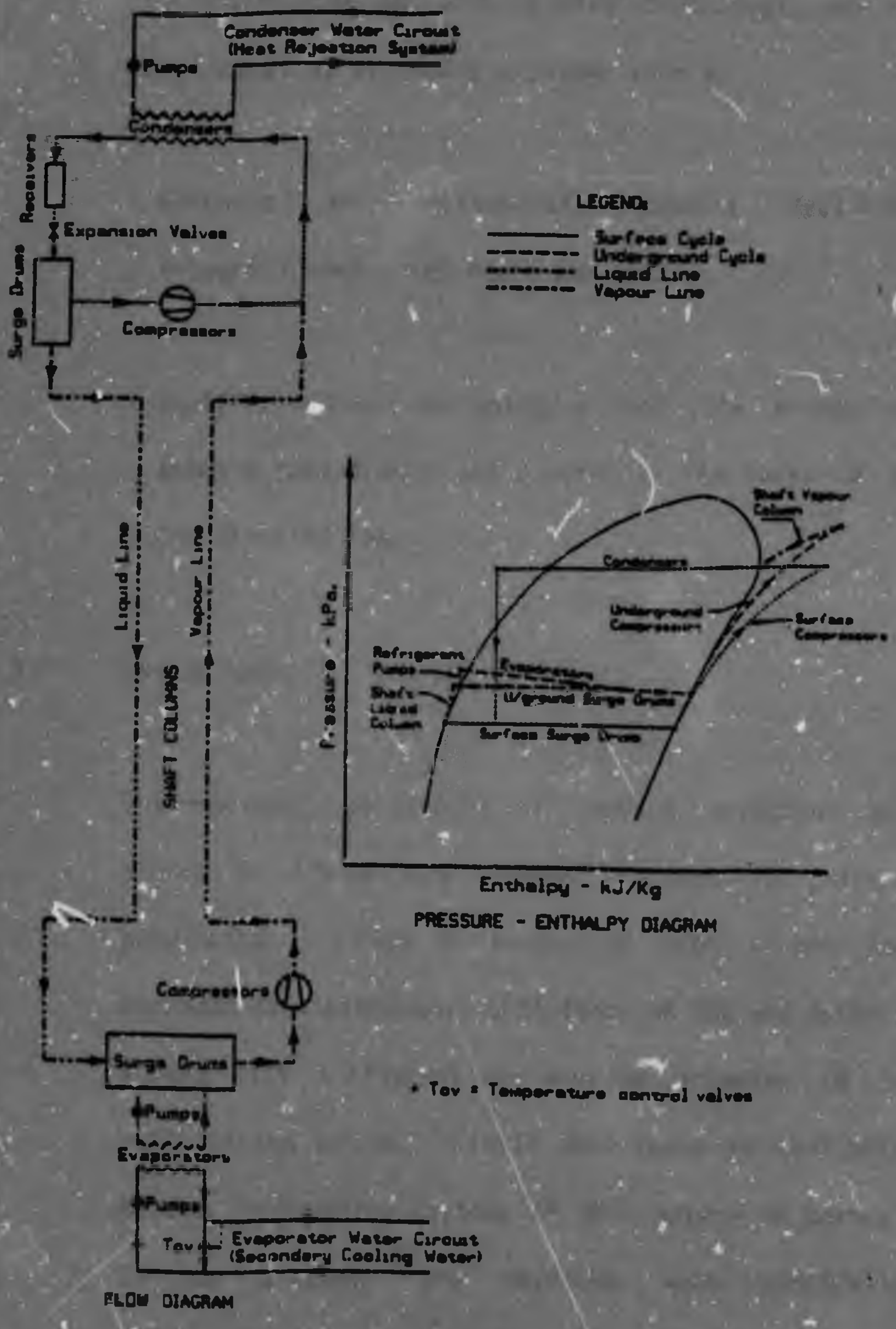


FIGURE 13.2

THE VAN RENSBURG SPLIT-AMMONIA REFRIGERATION PROCESS AT A DEPTH OF 4000M

The application of this untried technology, may be complicated by secondary problems such as :

- Borehole and subsequently ammonia feed-line integrity over great depth and time.
- Implications of designing a fail-safe system of ammonia containment and control in the event of an uncontrolled leak.

10.3.3 Use of hydrolift technology

International acceptance of pumping solutions and slurry in "Three-pipe chamber" systems is gaining popularity. Lifts in excess of 1000 m are not uncommon at a remarkable efficiency of 90% and better. Single lift configurations can be accepted to be reliable and proven. Little experience is available however in coupling systems of this nature in series. Serious problems are realised when vertically connected systems may inadvertently act against one another as a result of valve or control failure. Extreme water pressures could result in pipe rupture.

14. DESIGN MASS FLOW AND COOLING STRATEGY

With the stoping geometry proposed and the tonnage expected, the stope heat load including the tunnel heat contribution from slusher gullies is 470 kW. Figure 12.2 shows the heat load distribution and is summarized as follows:

Stope rock heat	(42%)	=	260 kW
Strike gully heat	(15%)	=	93 kW
Leakage	(12%)	=	74 kW
Dip gully heat	(7%)	=	43 kW
TOTAL			470 kW

An additional heat load from ventilating fans explosives, backfill and continuous scrapers* is estimated to be 150 kW. The total heat load is therefore estimated to be 620 kW for an active stope.

In the high production scenario, approximately 28 raise lines will be active and their functions are classified, shown in Table 14.1.

* Referred to later as fans and scrapers.

NUMBER	STATUS	EXPECTED HEAT LOAD
22	STOPING	22 x 620 kW = 13,6 MW
3	VAMPING	3 x 410 kW = 1,2 MW
3	DEVELOPING	3 x 210 kW = 0,6 MW
28	TOTAL	= 15,4 MW

TABLE 14.1
TERTIARY RAISE LINE HEAT LOADS

Air distribution within an idealized stope is shown in Figure 14.1. Mass flows and quantities are substantiated in Section 3.3.

Assuming an air quality entering the stope of 20 °C wet bulb the expected heat rejection capacities are shown in Table 14.2.

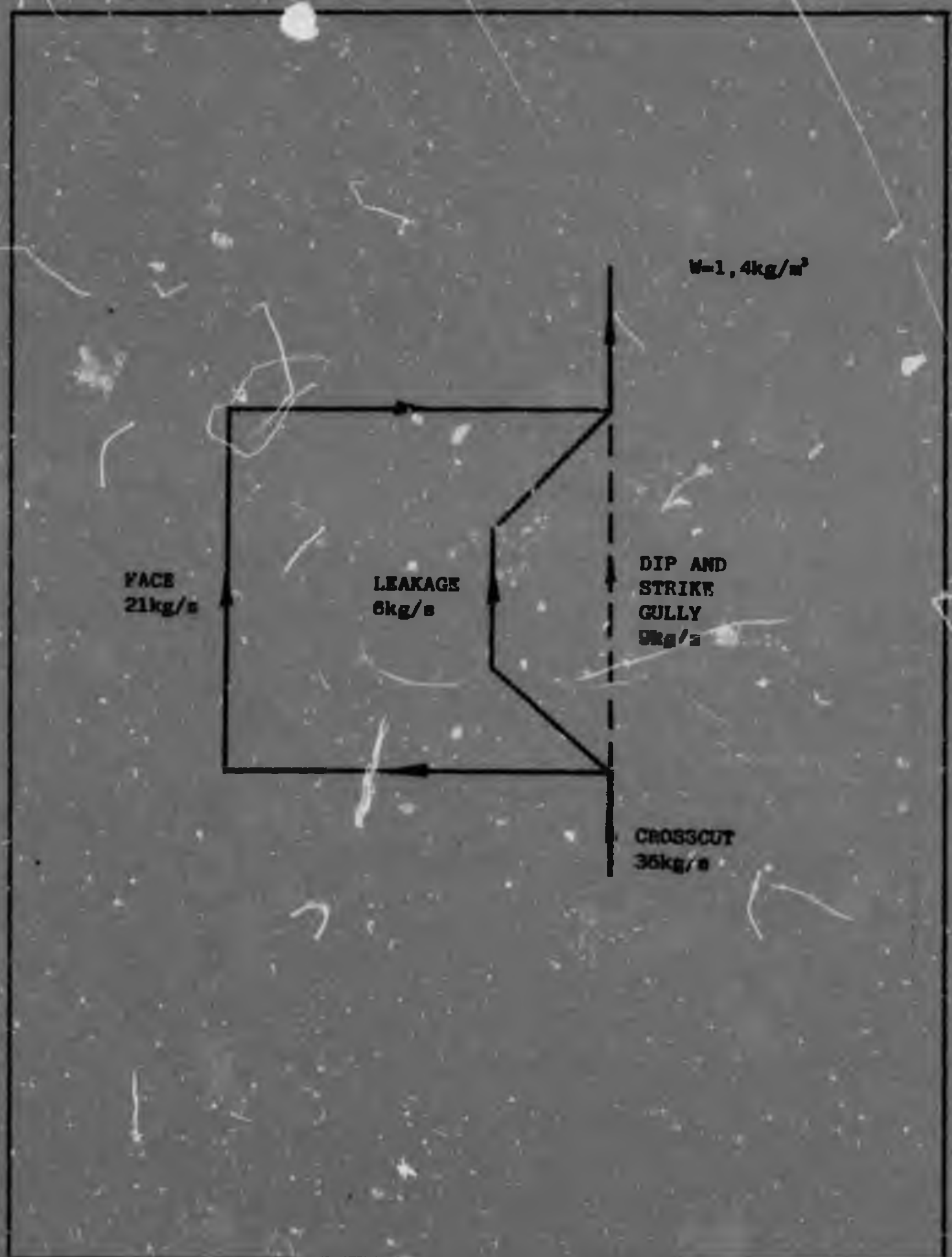


FIGURE 14.1

AIRFLOW DISTRIBUTION IN A TERIARY STOPE

The assumption of 20 °C WB stope entry is qualified in Section 15, (Energy Balance).

	MASS FLOW kg/s	HEAT REJECTION CAPACITY IN kW AT REJECT WET BULB		ESTIMATED HEAT LOAD (kW)	
		27 °C	30 °C	FROM ROCK	FROM FANS AND SCRAPPERS
STOPE FACE	21	460	690	260	NIL
STRIKE GULLYS TOTAL OF 3	7	150	230	90	135
DIP GULLY	2	44	66	45	15
LEAKAGE	6	131	197	75	NIL
TOTAL	36	785	1 183	470	150
				620	

TABLE 14.2
HEAT REJECTION CAPACITY OF STOPE AIR VS ESTIMATED HEAT LOADS

This preliminary heat balance shows a marginal cooling excess at the design reject wet bulb temperature of 27 °C.

No thermal excursions above 30 °C wet bulb are expected and a design reject wet bulb of 27 °C is likely to be maintained.

A mass flow of 36 kg/s (25 m^3/s) in the crosscut will be required to be conditioned prior to stope entry. Based on:

Inlet temperature : 29 °C WB: 77 kJ/kg Sigma Heat

Outlet temperature : 18 °C WB: 42,5 kJ/kg Sigma Heat

Mass Flow : 36 kg/s

Tertiary cooler duty = 1 250 kW

Secondary coolers will be strategically located in main intakes positioned so that Specific Cooling Power of 300 watts per square metre is maintained. Positions of coolers from the shaft will be dependent on mass flows and depth.

15. ENERGY BALANCE FOR THE TERTIARY

An estimate of the Psychrometric conditions of the air within the greater ventilation network at full Tertiary production has been made. The prime objective of the exercise is to identify any areas where excursions of temperature in excess of 30°C WB or violations of Specific Cooling Power less than 300 watts per square metre may occur. Using predicted air mass flows from the VENTFLOW model, a manual computation was performed accounting for major expected heat loads and heat sinks. The energy balance is shown in 3 sections: (1) DOWNCAST (2) STOPES and (3) UPCAST; referring Tables 15.1 to 15.3. The energy balance is compared to the HEATFLOW analysis performed by the COMRO in Tables 15.4 and 15.5.

LOCATION	TEMPERATURE	PRESSURE	SIGMA HEAT	MOISTURE	MASS FLOW	ENERGY
	WB/DB °C	kPa	kj/kg	g/kg	kg/s	M.W.
ATMOSPHERIC DOWNCAST VIA EAST SHAFT	18/24	87,5	55,0	12,5		
EASTERN SHAFT AUTO COMPRESSION	1 502 m		+ 14,7		690	+ 10,1
13L BOTTOM EASTERN	24,4/28	104,0	69,7	19,7		
13L INTAKE	0 m		+ 1,0		640	+ 0,7
13L TOP OF STRATHMORE	24,5/28	104,0	70,7	19,1		
STRATHMORE AUTO-COMPRESSION	922 m		+ 9,1		640	+ 7,6
INTAKE TO 27L MID-SHAFT COOLER	28,6/34	115,0	82,5	19,9		
MID SHAFT COOLING	20 m		-		600	- 23,1
OUTLET FROM 27L COOLER PLUS BYPASS MIX	17,4/18,4	115,0	44,4	10,5		

TABLE 15.1 (a)

DOWNCAST ENERGY BALANCE AND PSYCHROMETRIC CONDITIONS

LOCATION	TEMPERATURE	PRESSURE	SIGMA HEAT	MOISTURE	MASS FLOW	ENERGY
	WB/DB °C	kPa	kj/kg	g/kg	kg/s	M.W.
STRATHMORE AUTO-COMPRESSION	720 m		+ 7,1		600	+ 4,3
INTAKE 35 & 36 L	20,8/23	125,0	51,5	11,5		
WINDERS	0 m		+ 5,0	11,5	600	+ 3,0
TOP OF TERTIARY DOWNCAST	22,5/28,[illegible]	125,0	56,5	11,5		
TERTIARY DOWNCAST AUTO-COMPRESSION	460 m		+ 4,5	12,5	600	+ 2,7
42 LEVEL INTAKE	24,3/28,6	132,0	61,0	12,5		
TERTIARY DOWNCAST AUTO-COMPRESSION	520 m		+ 5,1		300	+ 1,5
50 LEVEL INTAKE	26,7/29,5	139,0	67,1	13,5		
MEAN INTAKE LEVEL IS 46 L	25,6/29	135,0	63,5	12,0		
AUTOCOMPRESSION: 26,2						

TABLE 15.1 (b)

DOWNCAST ENERGY BALANCE AND PSYCHROMETRIC CONDITIONS

LOCATION	TEMPERATURE	PRESSURE	SIGMA HEAT	MOISTURE	MASS FLOW	ENERGY
	WB/DB °C	kPa	kj/kg	g/kg	kg/s	M.W.
46 LEVEL (MAIN INTAKE)	25,6/29	135,0	63,5	12,0		
INTAKE AIRWAY HEAT GAINS	0 m		+ 11,9		600	+ 7,1
INTAKE AIRWAYS TO SECONDARY B.A.C.	29/31,0	135,0	75,1	18,2		
SECONDARY B.A.C. DUTY			- 38,4		600	- 23,0
SECONDARY B.A.C. OUTLET	16/16	135,0	37,0	8,5		
AIRWAY AND DEVELOPMENT HEAT GAIN			+ 42,2		600	+ 25,3
AIR INTO TERTIARY B.A.C.	30/32	135,0	79,2	18,9		
TERTIARY B.A.C. DUTY			- 42,2		600	- 25,3
TERTIARY B.A.C. OUTLET	16/16	135,0	37,0	8,5		
AIRWAY HEAT GAINS			+ 7,3	+ 8,6	600	+ 4,4
AIR ENTERING STOPE	19/23	135,0	44,4	8,6		
STOPE HEAT GAIN			+ 27,3	-	600	+ 16,4
AIR EXITING STOPE	28/30	135,0	71,7	17,1		

TABLE 15.2

STOPE ENERGY BALANCE AND PSYCHROMETRIC CONDITIONS

LOCATION	TEMPERATURE	PRESSURE	SIGMA HEAT	MOISTURE	MASS FLOW	ENERGY
	WB/DB °C	kPa	kj/kg	g/kg	kg/s	M.W.
EXIT STOPE AIR	28/30	135,0	71,7	17,1		
RETURN AIRWAY HEAT GAIN			+ 7,5		600	+ 4,5
46 L RETURN AIR ENTERING TERTIARY UPCAST	30/32	135,0	79,2	19,3		
42 LEVEL TO 37L AUTO-DECOMPRESSION	265 m		- 2,6		600	- 1,5
AIR ENTERING 37L BOOSTERS	28,4/28,4	127,0	76,6	19,3		
HEAT INPUT BY BOOSTER FANS			+ 4,2	-	600	+ 2,5
AIR LEAVING 37L BOOSTER FANS	30/34	130,0	80,8	19,3		
37 LEVEL TO 27L AUTODECOM-PRESSION	915 m		- 9,0		600	- 5,4

TABLE 15.3 (a)

UPCAST ENERGY BALANCE AND PSYCHROMETRIC CONDITIONS

LOCATION	TEMPERATURE	PRESSURE	SIGMA HEAT	MOISTURE	MASS FLOW	ENERGY
	WB/DB °C	kPa	kj/kg	g/kg	kg/s	M.W.
AIR ENTERING 27 L CONDENSER TOWER	26/26	115,0	71,8	18,7		
27L CONDENSER DUTY			+ 40,0	-		
AIR LEAVING 27L CONDENSER	34,8/34,8	115,0	111,8	31,5		
27L TO 23L AUTODECOMPRESSION	240 m		- 2,[illegible]		640	- 1,5
23L TOP OF STRATHMORE UPCAST	33,8/33,8	110,0	109,5	31,5		
HEAT LOSS IN R.A.W's 23L			- 3,8		640	- 2,4
23L BOTTOM OF EASTERN UPCAST	32,8/32,8	108,0	105,7			
AIR FROM SOUTH SHAFT VIA 23L	32,8/32,89	108,0	105,7		90	+ 4,6
23L TO EASTERN MAIN FANS AUTODECOMPRESSION	2 185 m		- 21,4		730	- 15,6
AIR ENTERING EASTERN UPCAST FANS	24,8/24,8	83,0	84,3	24,5		
AIR TO ATMOSPHERE			- 29,3		730	- 21,4
ATMOSPHERE	18/24	87,5	55,0	12,5		

TABLE 15.3 (b)

UPCAST ENERGY BALANCE AND PSYCHROMETRIC CONDITIONS

HEAT LOAD	C.O.M (MW)	ENERGY BALANCE (MW)
AUTO-COMPRESSION	25,6	26,2
INTAKES:		
ROCK + DEVELOPMENT	36,3	37,5
MACHINERY	8,7	5,5
STOPES	16,4	16,4
RESIDUAL COOLING	- 9,0	- 4,9
T O T A L	78,0	80,7

TABLE 15.4
COMPARISON OF COM AND ENERGY BALANCE HEAT LOADS

HEAT SINK	C.O.M (MW)	ENERGY BALANCE (MW)
MID SHAFT COOLER	24,0	23,1
SECONDARY COOLERS	26,0	23,0
TERTIARY COOLERS	26,6	25,3
T O T A L	76,6	71,4

TABLE 15.5
COMPARISON OF COM AND ENERGY BALANCE HEAT SINKS

Variations between the two analyses are primarily due to different mass flows in the calculations.

15.1 Conclusion

From the analysis, maximum wet-bulb temperatures are calculated and in conjunction with design air velocities for the region specified, specific cooling power are listed in Table 15.6. No infringement of the GENMIN philosophy of SCP equal or greateter than $300W/m^2$ is forseen.

COMPONENT	LOCATION	MAXIMUM WET BULB °C	DESIGN VELOCITY m/s	SCP W/m^2
DOWNCAST	27L MID SHAFT INTAKE	28,6	7	392
	50 LEVEL INTAKE	26,7	3	396
INTAKE AIRWAYS	INTAKE TO TERTIARY BAC	30,0	3	311
STOPES	STOPE EXIT	28,0	3	364
RETURN AIRWAYS	27L CONDENSER OUTLET	34,8	6	130*

TABLE 15.6
EXPECTED SCP's IN THE TERTIARY VENTILATION CIRCUIT

* CAUTION

Strategic placement of heat exchangers ensures the compliance of the primary environmental criterion of 300 watts per square metre, where persons are required to work, and no temperatures in excess of 30 °C wet bulb are expected.

A global overview of the Tertiary supports the expectation that the heat absorption capacity of the air at design mass flows has the ability to negate the heat loads.

Table 15.7 summaries the energy balance for the mine. Source(+) and sink(-) energy values refer to sums of each respectively from Tables 15.1 to 15.3. Datum is atmosphere, at 18 °C WB.

COMPONENT	HEAT SOURCES (+) MW	HEAT SINKS (-) MW
DOWNCAST	29,9	23,1
STOPE	53,2	48,3
UPCAST	35,6	47,9
T O T A L	118,7	119,3

TABLE 15.7
TERTIARY GLOBAL ENERGY BALANCE

16. CONCLUSIONS AND RECOMMENDATIONS

16.1 General

Exploitation of the Buffelsfontein Tertiary ore reserves can, from the environmental viewpoint, be undertaken with nominal technical risk. Over and above the conservative, traditional ventilation and cooling philosophies, New Generation Strategies of Primary through to Tertiary controlled air cooling offer the most cost effective and ultimately, lowest risk, heat exchange principles.

16.2 Specific Conclusions

Within the scope of the design conditions and constraints outlined the following conclusions emerged:

(a) The Tertiary mine heat load including autocompression, at mining depths and conditions prescribed (Section 16) is estimated to be 78 Megawatts. Ten independent techniques were employed for heat load determination (Section 10) giving an average of 76,5 MW ± [illegible],[illegible] MW.

Although all estimates are within 6% of the mean a greater degree of confidence is placed on results of techniques which address the most number of relevant input parameters. Accuracy of the project was specified at ± 10% (definitive estimate) and as such, required maximum parameter consultation. The most current algorithms for heat prediction were also felt to be necessary. In this case the Chamber of Mines heat flow programme fulfilled this requirement.

Other techniques are useful for first order estimates to gauge a zone or envelope of most probable heat load. Graphical techniques, 1 to 5 (Section 10.2.2) are a good starting point.

Therefore depending on the:

- degree of accuracy required
- amount of information available
- time constraints (availability)

Various heat prediction methods may be used.

(b) an air mass flow of 600 kg/s circulating in the tertiary mining horizon will:

- Adequately dilute gas and dust concentrations to acceptable levels

- provide a sufficient heat transfer medium to supply cooling and reject heat as required

- Provide air velocities at the face to comply with an SCP of 300 W/m^2.

A greater circulated mass flow may be desirable. However thereby increasing the cost of the project.

An air mass flow of 600 kg/s relates to a volumetric equivalent of 2,2 m^3/s/kiloton/month which is adequate and acceptable in current deep mining philosophy.

(c) The investigation into strategic airway security showed the return airways between Strachmore and Eastern shaft to pose the greatest concern. Additional secure airways, possibly on 23L, should be developed with 4 being used as main returns handling the full Tertiary quantity of 600 kg/s.

Based entirely on ventilation criteria the main shaft diameters optimized at:

1 x 4,5 m diameter equipped downcast

1 x 6,8 m diameter split shaft

However, due to hoisting requirements the final shafts are:

1 x 6,8 m diameter equipped downcast

1 x 7,8 m diameter bratticed split shaft with

downcast area of 27,8 m^2 (equipped)

and upcast area of 20,0 m^2

(d) In order for the Eastern shaft fans to cope with an increased mass flow to 600 kg/s to provide the design mass flow in the Tertiary circuit, booster fans with pressure capabilities of 4,5 kPa will be required. Omission of the boosters will:

- Cause the eastern fans to operate dangerously close to stall.

- Starve the Tertiary block of air by only drawing approximately 400 kg/s through the mining horizon.

- Limit the extent to which mining can proceed. A much narrower operating band would exist in maximum depth, distance to strike and maximum tonnage per month.

(e) As a result of the length of the existing water reticulation system, the number of intermittent storage dams and the high residence time of water in the system, thermal losses are high and the system efficiency low. Using this as a foundation inadvertently leads to an undesirable condition from the outset.

Various conventional cooling water configurations were considered and the best compromise necessitates the application of 125 MW(R) to achieve the required 78 MW(R) effective.

This proportion is not unusually high when compared to other similar installations.

(f) In the case of ultra deep level mining the use of unconventional cooling strategies appears to be financially attractive.

Two such alternatives are:

- Direct expansion ammonia at depth.
- 3 Chamber system.

An indication of total costs for such methods are shown in Figure 16.1.

Although the financial implications are apparent <u>the most important costs of safety and system integrity are not included as they presently are not quantifiable.</u>

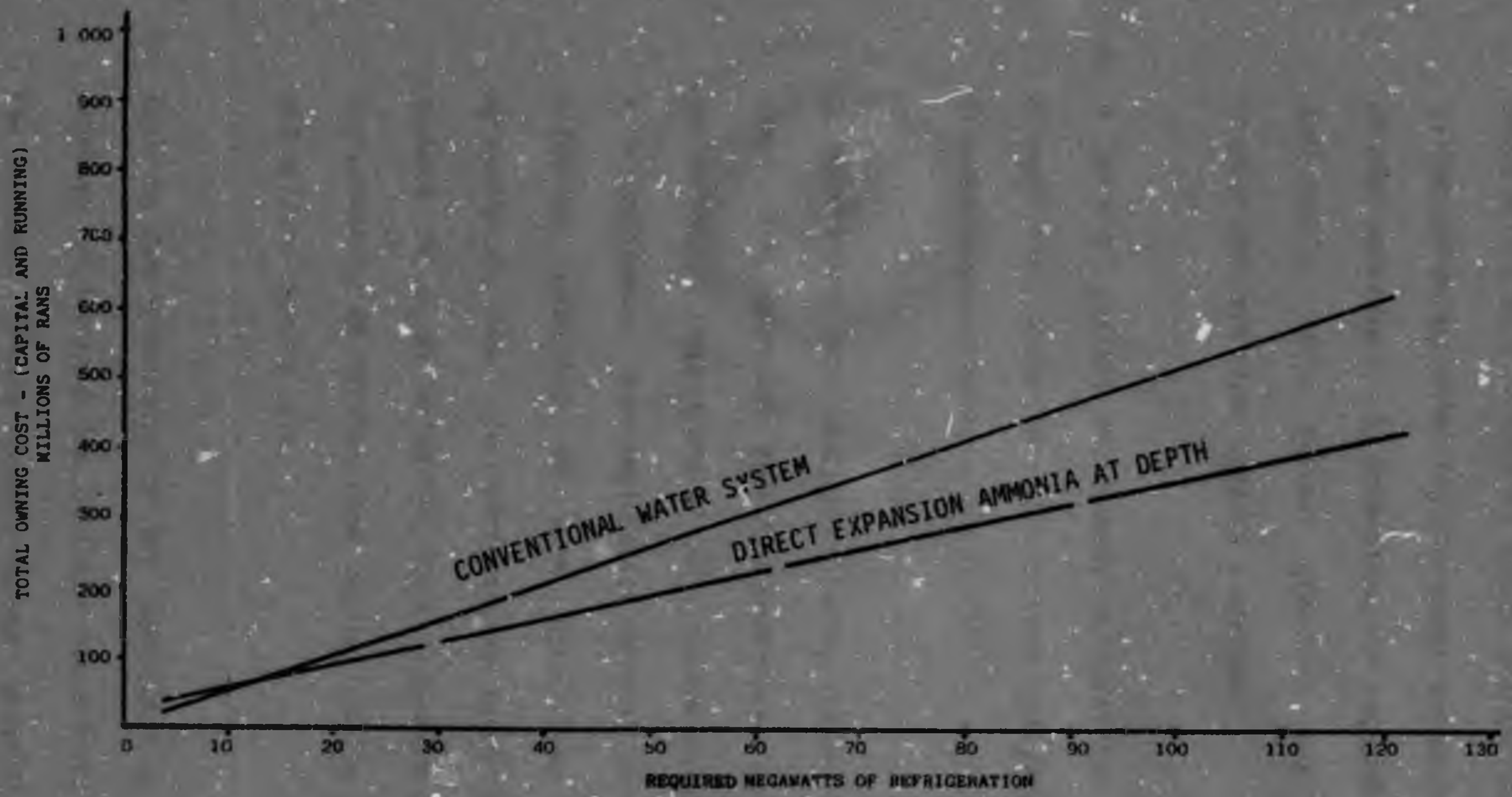


FIGURE 16.1

TOTAL COST OF REFRIGERATION OPTIONS AT A DEPTH OF 4000M

Every effort must be made to reduce thermal losses in the system by optimizing the number and position of heat exchangers. By moderating air temperatures judiciously, large thermal gradients are not created and heat flows kept to a minimum.

(g) A 300 W/m² cooling environment is sufficient to prevent heat stroke for all acclimatized persons proceeding underground.

Using 300 W/m² as a design criteria, rather than simply 27°C wet bulb reject temperature, a new design reject temperature of 30°C can be accepted with confidence. The implication of this is the effective reduction in mine heat load by approximately 10%. In refrigeration terms this amounts to 8 MW which in turn translates to approximately R8 Million rand capital saving and R10 million running cost.

(h) Applying the principles of Air Recirculation is expected to increase the ultimate flexibility of the project. Greater depth, further penetration and larger tonnages are possible if eventually required.

Although the technology has been applied successfully elsewhere a critical analysis is imperative addressing the factors of:

- extreme depth viz. - escape strategy.
- failure to safety.
- radioactive radon daughter entrainment.

(i) The use of backfill has demonstrated its theoretical implications. Undoubtedly advantageous to the reduction of heat flow, leakage control and as a mechanism of air strike and dip control, the placement of fill has a negative impact on the resistance of the mine which however can be adequately overcome with booster fans. The practical implications of transporting and stowing backfill at depths of 4000 m requires some study.

16.3 Recommendations for future work

(a) The thermodynamic benefits of expanding ammonia at depth are apparant. The safety and integrity of the scheme however presents serious concerns. Further work is encouraged into evaluating long - column suspended pipe technology, deep small diameter borehole integrity and fundamental safety systems for underground ammonia plants.

(b) In order to validate the previous work or future scenarios, COMRO's new ENVIRON program is recommended. Where Heatflow and Ventflow were used in isolation in the past, Environ interacts the functions of the two.

(c) NETSIM should be used where possible to augment data for the Environ program.

(d) The cost of a 3 shaft (2 downcast and 1 upcast) should should be further investigated. A preliminary exercise suggests a 3 Shaft configuration as opposed to a downcast plus split shaft to be marginally more costly. The main advantage with the 3 hole option is the elimination of the shaft brattice and the logistics of putting it in place.

(e) The application of controlled recirculation merits further investigation. The Tertiary configuration lends itself ideally to both small and large scale installations. The relief in mass flow in the upper Strathmore and Eastern circuits would result in a significant reduction in system pressure loss. This may be sufficient to obviate the necessity of high pressure booster fans. The reduction in the autocompressive heat load would also be significant, possibly as much as 11 MW if a 50% recirculation were achieved.

(f) A full study of the worst case power failure scenarios should be undertaken. Preliminary calculations show no danger exists provided persons exit the stope as quickly as possible and proceed to the station. The residual coolth in the water reticulation system is sufficient to maintain adequate environmental conditions in intakes and stations for up to six hours.

Gas detection networks, escape strategy, second outlets and safe waiting places must be critically addressed.

(g) Adopting the philosophy of Primary, Secondary and Tertiary cooling a large potential exists for the elimination of insulation from chilled water pipes. Notably, each installation must be evaluated on its own thermodynamic envelope, which not only reduces capital cost but reduced the environmental risk due to fire.

16.4 Closing Remarks

The definitive estimate by the task force concluded that the project capital cost would be approximately 1 100 Million Rand. The Ventilation and Refrigeration Costs amounted to 20%. This proportion being as large as it is, will require discerning systems management with diligent monitoring, and application with minimum wastage, of coolth and circulated air.

APPENDIX A

HAULAGE LAYOUTS BASED ON STRUCTURAL GEOLOGY OF 42 TO 49 LEVEL