HEAT LOAD IN A LEDGING STOPE AT FREE STATE GEDULD GOLD MINE

Ernst Jakobus de Jager

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

Johannesburg 1989

DECLARATION

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

this 15th day of SEPTEMBER 1989

ABSTRACT

The purpose of this study is to gain an understanding of the dynamics pertaining to heat loads in an underground ledging stope. The results of an experiment is compared to that of HEATFLOW and discussed.

The results of the experiment concluded that heat flow from rock in the stope contributed 92,7% of the total heat load. The remainder was made up from heat flow from men, material and machinery. Ventilation air passing through the stope removed 79,3 % of the heat produced. Chilled service water used in the stope removed 15,8% of the heat while the remainder was removed by compressed air.

A 2,0°C rise in the air temperature was observed during the main blast. This rise in temperature was due to the use of explosives.

ACKNOWLEDGEMENTS

This research report arises from work carried out as part of the research programme of the Chamber of Mines of South Africa Research Organisation.

I, would like to thank the Director, Mr S.J. Bluhm and the staff of the Environmental Engineering Laboratory of the COMRO for their assistance during the course of this study,

The assistance of the Management and Environmental staff of Free State Geduld mine is acknowledged.

DEDICATION

This research is dedicated to my wife Linda and daughters Lauren and Carmen.

CONTENTS

LIST OF FIGURES

25x

LIST OF TABLES

25x

LIST OF SYMBOLS

k	-	Thermal conductivity W/mK
p	-	Density of rock kg/m^3
c	-	Specific heat kJ/kg k
S_{fa}	-	Stope face advance m
ß	-	Temperature correction factor (VRT-Twb)/(VRT-Tdb)
dT	-	Temperature driving force (VRT-Tdb) °C
Sw	-	Stoping width m
Q	-	Heat kW
Q_{stopes}	-	Heat pick up from rock per metre of face length kW
Tdb	-	Average dry-bulb temperature in the stope, °C
MFA	-	Average monthly face advance, m
span	-	Average distance from centre gully to the stope face, m

25x

CHAPTER 1

AIMS AND OUTLINE OF STUDY

1.1 Introduction

In 1840 a statement was made at the Manchester Geological Society which read: "One thing is certain that from the increase of temperature downwards, a limit will be fixed to all our mining speculations at a depth of two miles."[1] As can be seen, the problem of heat was already anticipated at that stage.

Higher virgin rock temperatures (VRT) and an increase in heat flow is experienced when the working levels of a mine are developed deeper. This in turn leads to decreased worker productivity since an inverse relationship exist between productivity and the ambient temperatures in underground working places.[2]

In order to create a safe underground thermal environment and to increase productivity it is imperative to quantify heat loads.

The maximum VRT recorded in the South African mining industry during 1986 was 66,1°C having been recorded 3 264 m below surface[3]. There were stopes with wet bulb temperatures exceeding 32,0°C wet bulb while the industry mean face wet bulb temperature was 27,9°C for the same period.

In the quest for acceptable environmental conditions in the underground workings of a mine, refrigeration is installed to provide coolth. In 1986 the rated capacity of installed refrigeration was 1 125 MW. The cost of owning a refrigeration plant amounts to a present value of R2000 per kW cooling.[4]

As previously stated it is of paramount importance to quantify underground heat flows. The design of mine layouts, backfill of stopes, refrigeration and insulation of airways are means to combat the underground heat load. Heat flow from rock, machinery, men etc, has been identified as heat sources underground.[5] The proper understanding of heat flow dynamics will assist in designing systems where heat flow is reduced and consequently a reduction in the demand for refrigeration.

The objective of this study is to obtain an understanding of the various heat flow elements in a ledging stope. The data gathered during this study is added to an existing data base at COMRO and is used to verify a computer program HEATFLOW which is in use in industry. This program predicts the heat flow of a mine.

This study examines all the activities in a stope contributing to heat flow.

1.2 Mining Cycle In A Stope

Various gold bearing reefs can normally be mined. Basal, leader, "A" and "B" reef are some worth mentioning. This particular stope was mined on basal reef which is approximately 1,0 metre wide and varies at an inclined angle of between 0 and 90 degrees to the horizontal. Blasting with explosives has to be done to remove the reef from the surrounding rock. A high concentration of manual labour is performed during the drilling, face preparation and supporting cycles.

When production is commenced underground, the working face would be supported with timber and made safe. After removing loose rock and the face cleaned, drill hole positions would be marked off by means of paint.

Drilling of the blast holes is normally done using pneumatic drills. Hydraulically powered rock drills are used by exception[6]. Pneumatic drills were used and chilled service water was employed to suppress dust, flush the drill hole and cool the drill bit. Ventilation control, support and blasting barricades are installed during this shift.

When the aforementioned cycle is completed, the drilled holes are charged with explosives after which the area is cleared of all workers. The blast takes place after all workers concerned have been removed to the surface of the mine. The blast is normally scheduled to take place between 16h00 and 17h00 daily. Rocks are thrown into the stope void during the blast. The area between the blasting barricade and face (normally 2 metres wide) is filled with broken rock. A re-entry period of 2 to 3 hours is observed after the blast during which time the clearing of blasting fumes and dust are done.

The newly exposed rock of the face, hanging wall, footwall and the broken rock is now at a higher temperature than the previously exposed rock surfaces.

When the workers return to the working place after the blast, it is made safe by removing dangerously placed rocks. Water is sprayed on the effected area to allay dust. The broken rock is removed using electrical powered winches fitted with metal scrapers. Hand lashing of the broken rock then takes place after which the face is washed with water and the cycle is repeated.

The face advances in discrete steps and is stationary for certain periods that depend on the mining cycle. The greatest number of people is present in the working face of the stope during the drilling and supporting phase. Most of the heat is present after the main blast and the broken rock remains in the stope for at least half of the total cycle.

1.3 Scope Of This Study

In this study the method of measuring the various elements of heat flow is described. Several activities which take place during the mining cycle are described. The effect of chilled service water as a heat sink is investigated and the total heat flow and heat sink are calculated and compared with the results of HEATFLOW, a computer program devised by the Chamber of Mines Research Organisation. The results are then discussed.

CHAPTER 2

LITERATURE REVIEW

2.1 Introduction

The objective of this chapter is to review previous theoretical and experimental studies of heat flow in a stope.

2.1.1 Theoretical studies

2.1.1.1 Heat transfer mechanisms - Theory of conduction

Conduction is the transfer of heat from one part of a body at a higher temperature to another part of the same body at a lower temperature. This interaction takes place at molecular level where the transfer of energy is from more energetic molecules to lower energetic levels. In solids that are non conductors of electricity, heat is conducted by lattice waves caused by anatomic motion. Net kinetic energy is transported by the vibrations at very high frequencies and in pulses called phonons. Phonons represents states of excitations characteristic of the thermal capacity of the substance.

Heat flux at macroscopic level is proportional to the temperature gradient. The proportionality constant k is the thermal conductivity of the material.

$$q'' = -k\, dT/dx$$

The minus sign indicates that the heat is transfered in the direction of decreasing temperature. The above equation is the one dimensional form of Fourier's law of heat conduction.

If consideration is given to a one dimensional heat flow along the x direction, then integration shows that

$$q = kA/dx\ (T2 - T1)$$

where the thermal conductivity is considered constant, dx is the wall thickness, and T1 and T2 are the wall-face temperatures. $q/A = q''$ where q is the heat transfer rate through an area A.

Now q = T2 - T1/(dx/kA) = T2 - T1/Rth = thermal potential difference / thermal resistance where dx/kA assumes the role of thermal resistance. The aforementioned relationship is equivalent to that of Ohm's law in electricity.

Semi conductors or non metallic materials have very few electrons path of the electrons, atomic collisions and the scattering limiting their travel in a net direction along the temperature gradient. The internal resistance of a solid is one of the factors that dictates it's conductivity.

2.1.1.2 One dimensional heat flow in a semi infinite solid

The working face of an underground stope advances in discrete steps with the blasting thereof. The footwall and hanging wall can be assumed to consist of vertical slabs, each with a thickness equivalent to the face advance per blast . The slabs are assumed to have been exposed to the air interface for a period which depends on it's distance from the relevant face. The heat flow within each such slab is evaluated using the One Dimensional Conduction in a semi-infinite solid. The total heat flow is then a summation of all the individual slabs. Different convective film coefficients are used to account for different boundary conditions and air velocities.

In the case of the heat flow at the working face an infinite surface heat transfer coefficient is assumed to simulate turbulent air flow and service water usage. The surface temperature of the exposed rock in this zone is estimated by using the wetbulb temperature of the air as an equivalent. The heat flow from the face as well as the broken rock is quantified by assuming that cooling takes place between the ambient air temperature and the virgin rock temperature.

Changes in the boundary conditions during the mining cycle is ignored while it assumed that that the face advance is continuous hence a continuous heat flow into the stope over the mining cycle. The cooling effect of service water is sumilated by assuming that it leaves the stope at a temperature equivalent to that of the inlet wetbulb temperature.

2.1.1.3 Two dimensional model

Time is the independent variable in the heat conduction equation.The prevailing conditions that cause heat to be transfered must start at a point in time.It is for this reason that transients must be considered. When the environment is steady with respect to time, the body temperature is transient due to sudden exposure to the steady environment whose temperature differs from initial temperature state of the solid. After a prolonged exposure the body temperature distribution also becomes steady; this represents steady state heat conduction. The condition of steady state body temperature is always preceded by a starting transient of finite duration. A quasi-steady state heat conduction situation is attained when the environment temperature remains constant but the body temperature changes cyclically.

The two dimensional model has limitations in the sense that it cannot simulate different mining activities or cooling practices. The total heat flow in the working place is determined by using this model. A numerical model of a cross section of a face zone was developed previously.[12] Finite element and finite difference procedures were used to solve the transient two dimensional conduction equation.

The model allows for the different mining activities, which are explained in the previous chapter, by allowing boundary conditions on the rock surface to vary with time.

The body geometry in question in an underground situation is that of a semi-infinite solid. The infinite solid has large dimensions in the vertical, horisontal and $x \geq 0$ directions. Internal plane where $x=0$ represents the contact surface and contains the heat source.

The total face length is divided into sections to enable the heat flow into the face zone to be calculated. Conditions are assumed to be constant in each of these sections. Interpolation is used to calculate the heat and mass transfer in each section.

2.1.1.4 Differential equations of the temperature field

Fourier's law of heat conduction relates heat flow $q = \delta Q/\delta t$ across conducting area A to the negative temperature gradient $\delta T/\delta n$ in a direction normal to A.

$qn = -kA\ \delta T/\delta n$ where k is the thermal conductivity of the conducting material and q is the heat rate. The aforementioned equation is normally expressed as $q''n = -k\ \delta T/\delta n$ where $q'' = q'a$ is the heat flux. If $T = T(x,t)$ the heat flux is transient, while if $T = T(x)$ only the heatflux is constant at a given station x. T as a dependant variable temperature is related to t an independent variable time and spatial locations x,y,z in the conducting body by means of heat balance differential equations derived from Fouriers law of heat conduction.

2.1.1.5. Geometric boundary conditions -Plane Boundaries

The spatial locations in a body with plane surfaces are determined using points p(x,y,z) expressed in rectangular coordinates. The partial differential equation expressing conservation of energy in this coordinate system is as follows.

$\delta/\delta x(k\ \delta T/\delta x) + \delta/\delta y(k\ \delta T/\delta y) + \delta/\delta z(k\ \delta T/\delta z) + q''' = \rho c\ \delta T/\delta t$ where ρ is the density of the body material, c is the specific heat and k the thermal conductivity of the material. q''' represents a uniformly distributed heat source.

If k is constant then the aforementioned equation reduces to $\delta^2T/\delta x^2 + \delta^2T/\delta y^2 + \delta^2T/\delta z^2 + q'''/k = 1/\alpha\ \delta T/\delta t$, where $\alpha = k/\rho c$ is thermal diffusivity.

2.2 Experimental Studies.

2.2.1 Introduction.

Previously, various heat flow equations were developed during experiments conducted by COMRO [7,8,9]. A similar experiment was done at Hartebeestfontein Gold Mine during 1986[5] where it was found that 94 percent of the heat flow in a stope can be ascribed to heat flow from rock.

Heat flow in stopes was studied by Starfield[10] who solved the problem for a stope advancing in discrete steps, assuming one-dimensional conduction from the face, hanging and footwall. He showed that a recurring cycle was present in a stope. A solution in two-dimensions was done by Gould[11].

Von Glen suggested that the effect of chilled service water on air temperatures should be investigated as well as different patterns of chilled service water usage[12].

2.2.2 The computation of stope heatflow - Starfield. (10,13,14)

The fact that the heatflow profile in the stope is cyclical recurring, was concluded as described by Von Glen. (12)

A program was devised by Starfield to calculate the heatflow from rock.(14) The environmental conditions for a section of a stope is determined by using psychrometric relations as well as a wetness factor to denote the quantum of wet rock and a fictitious air temperature in which the latent and sensible heat are accounted for. In simulating the broken rock, use is made of a time varying heat source. Water flow and temperature cannot be quantified, therefor it is ignored as a heatsink or heat source. Duhamel's theorem is used to take into account the temperature history of the ventilating air.

The total length of face advance is divided into areas of daily face advances. This is to simulate the different temperatures of the stope with the stope being advanced. The area exposed to the air is used to calculate the environmental conditions as the heat flows from the rock to the air.

In this program only the most important factor in heatflow namely heatflow from rock is considered. Heatflow from machinery, explosives, timber, men and material are ignored.

2.2.3 Work of Lambrechts

Heat flow in the stope is depended on the difference between VRT and the wet bulb temperature of the air. This was concluded by Lambrechts after analyzing data from a large number of underground observations. The driving force for heat flow was increased with an increased face advance since a higher rock skin temperature was present. The skin temperature of the rock is dependent on the face advance. Normally true VRT would only be present 6 metres away from the rock face. If the face advance is at a higher rate than normal, a higher skin temperature is present due to an increased temperature gradient. This would result in a higher heat flow from rock. Service water was not taken into consideration and the sampling method was not consistent. This is one of the reasons why his work is not widely used.

25x

2.2.4 Other methods for predicting heat flow in stopes

Curves depicting heat flow from rock in stopes were devised by Whillier[15], where the difference between VRT and the dry-bulb temperature is acknowledged as the driving force. Curves are available for different distances of face to centre gully and different face lengths. The underlying principles are that heat from the broken rock is dissipated uniformly into the ventilation air over the mining cycle, heat flow from rock to air is one-dimensional and fixed face temperatures between blasts.

The following equation was derived by Whillier [16]

$$Q_{stopes} = 0,11\ (1 + 0,05.\ SW)(VRT - Tdb)\ \left(\frac{MFA}{10} - \frac{Span}{50} - \frac{kpC}{13x10^6}\right)^{\frac{1}{2}}$$

Gould[11] determined that an approximation for the increase in wet-bulb temperature along the stope face is a function of a value of twice the heat transfer coefficient of an equivalent smooth pipe, with a diameter equal to four times the area of the stope cross-section divided by this perimeter of the stope cross-section. He also concluded that watering down of the working face has benefits if the face was kept wet over the complete mining cycle. If watering down was done directly after the blast increased benefits can be expected.

The water rock thermal balance was devised by Van der Walt and Whillier[17]. The possibility of supplying a sufficient amount of chilled service water to the working face to offset all heat produced in that area is recommended. The cyclical activities of stoping were not considered since the model was based on mean conditions. Chilled service water was taken into account, but only to do the energy balance.

This project compares favourable with experimental data obtained by Bluhm et al[5] and Matthews[9].

2.3 Conclusion

The nature of heat flow is complicated and extremely dynamic. It is for this reason that oversimplified models are referred to in literature. More recent studies [5,9,12] does take cognisance of the heat flow boundaries and if these proposed models are widely used, it would result in decreased heat flow in the underground workings.

This research report will deal with an experiment done underground where the results obtained will be compared with that of HEATFLOW and then discussed.

CHAPTER 3

EXPERIMENTAL PROCEDURE

3.1 Introduction

Acceptable underground thermal conditions has traditionally been accomplished by the installation of refrigeration[4]. In essence this approach accepts high heat loads as inevitable. It is envisaged that mining activities in future would extend to depths of 5 000 metres below surface, into rock temperatures of 70°C.

It is of major importance to decrease the heat loads in a mine since the cost of installing refrigeration is exorbitant as was previously stated. Bottomley[4] investigated insulation of intake airways while Matthews[4] investigated the effect of stope back filling. The information obtained during this experiment will be used to improve on HEATFLOW and in turn could lead to decreased heatflow when used to design new underground ventilation systems.

The stope heat load consists of heat flow from rock, the operation of machinery, oxidation of timber, explosives and metabolic heat from men. The heat sinks are ventilation air, compressed air and chilled service water. The energy balance concept is used to quantify the elements of heat flow.

The following describes an empirical study which took place in a ledging stope at the Free State Geduld Gold Mine. The elements concerned were monitored continually over a 2 month period. Data capturing was achieved by a hard wired instrumentation system and a computer based data logger installed at the stope entrance.

3.2 Description Of The Test Site

The test site was located in Welkom at the Free State Geduld Gold Mine at a depth of 1 570 metres below surface (No 1 Shaft, 51 Level, 44 Stope). The rock production rate was 2070 tons/month with a face advance of 4 metres/month. The mining method was double sided ledging with both south and north faces working simultaneously. The stope had a single intake and return airway where the VRT was 42,5°C.

The ore body in the stope dips at 10 degrees from the horizontal in a west to east direction. The average stoping width was 1 metre. The broken rock cascaded through 3 ore passes to the cross-cut below the stope. No geological faulting was encountered in the stope.

Mining took place on an eleven day fortnight basis. Drilling was done during day shift using five pneumatic drills while night shift cleaning was afforded by two 37 kW electrically powered winches.

The ventilation air was supplied via a travelling way on 51 level to the top of the stope. The air downcasted along the working faces to 53 level. The ventilation air was not cooled prior to use in the stope and no ventilation controls were installed in the stope. A ventilation door was, however, installed on 51 level to assist in directing the airflow into the stope.

The service water in the stope, although originally cooled remotely from the stope, arrived at an average temperature of 28,5°C. The cross-cut service water pipes were not insulated. Table 3.1 lists the average mining parameters and conditions in the stope over the monitored period.

Table 3.1 Average Mining Parameters And Conditions In The Stope

Rock Production : Centares (m²)	782 m²/month
Tons	2 070 tons/month
Face Advance	4 metres
Average Stoping Width	1,0 m
Depth Below Surface	1 570 m
Virgin Rock Temperature	42,5°C
Footwall Rock Density	2 689 kg/m^3
Reef Density	2 696 kg/m3
Footwall Thermal Conductivity	7,39 W/mK
Reef Thermal Conductivity	7,66 W/mK
No of Electrical Winches (37 kW)	2
No of Pneumatic Drills	5
In-stope Ventilation Control	0

3.3 Description Of Instrumentation

The monitoring instrumentation was installed on 51 level at the inlet to the stope and the outlet of the stope below 51 level. A total of eleven parameters were monitored as is shown in table 3.2.

Table 3.2 Schedule Of Measured Parameters And Instrumentation

DESCRIPTION	INSTRUMENT
Inlet Wet Bulb Temperature	Resistance Temperature Device
Inlet Dry Bulb Temperature	Resistance Temperature Device
Inlet Air Flow Rate	Vortex Anemometer
Outlet Wet Bulb Temperature	Resistance Temperature Device
Outlet Dry Bulb Temperature	Resistance Temperature Device
Service Water Flow Rate	Turbine Pulse Generator
Service Water Inlet Temperature	Resistance Temperature Device
Service Water Outlet Temperature	Resistance Temperature Device
Compressed Air Flow	Turbine Pulse Generator
Compressed Air Temperatures	Resistance Temperature Device
Power Consumption	Power Transducer & Integrator

A scan of the instruments every 2,5 minutes and averaged at 15 minute intervals, during the entire period of the project, was done. Electrical power consumption was monitored continuously, integrated with time and then sampled every 15 minutes. The instruments were hard wired to a computer based data logging system which was housed in an air conditioned room on 51 level.

3.4 Computer Based Data Logging System.

A Hewlett-Packard 85 desk computer with a tape drive was linked in series with a Fluke data logger to capture the data. Incorporated in the data logger was a printer and display. Programming could be done in situ using the built in keyboard.

3.5 Experimental Procedure Used

Once all the instrumentation had been installed and the data logging system commissioned, monitoring was carried out for two consecutive months from 14 April 1987 to 18 June 1987. All the instruments were carefully calibrated before the start of the test period, with the calibrations being regularly checked and updated throughout the monitoring period.(Appendix 1) The data was recorded on magnetic tapes which were replaced weekly, and taken to surface for processing.

3.6 Other Measurements

Besides the continuous monitoring exercise other measurements were done using hand held instruments. These are mentioned below and discussed in Chapter 5

3.6.1 Virgin Rock Temperature

In order to determine the virgin rock temperature a measurement was taken in a 50 mm 6m long hole adjacent to the stope.

3.6.2 Stope Surveys

Approximately twenty stope surveys were conducted, enabling information to be gathered on in-stope activities. Rock mined, explosives used and men at work were taken from mine records.

3.6.3 Thermal conductivity and density of rock

Rock samples were taken and the thermal conductivity and density were measured on surface.

3.6.4 Drilled hole temperature measurements

Temperatures were measured at 0,3 metre intervals in a drilled hole to determine the virgin rock temperature decay.

3.6.5 Varying chilled service water flow rate

The use of chilled service water was increased by watering down the face with hoses. The effect was monitored on the computer based data logging system.

3.7 Conclusion

In this chapter the experimental procedure was discussed. In the next chapter the information collected on the data logger is discussed.

CHAPTER 4

INFORMATION GATHERED

4.1 INTRODUCTION

In this chapter the information collected in the field measurement is discussed. Eleven parameters were measured in a ledging stope over a period of two consecutive months. This stope was situated on Free State Geduld Mine in the Welkom area. The stope layout and positions of the instrumentation can be seen in Figure 4.1. The schedule of measured parameters and instrumentation used were shown in Table 3.2.

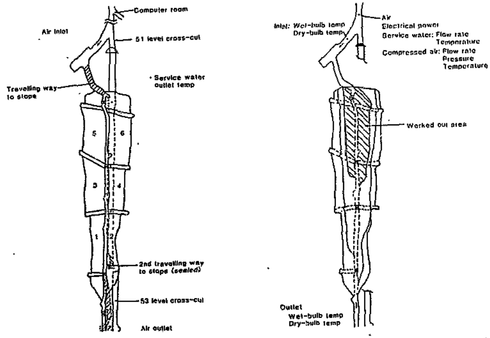

Figure 4.1 Stope Layout

4.2 Air Flow Rate

The overall air flow rate was 10,12 kg/s over the full monitored period. The air flow rate varied from a low of 4,7 kg/s to a high of 17,06 kg/s during the same period. The aforementioned was due to ventilation doors spragged open underground in the same ventilation district. The measurements were done by using a vortex anemometer.

Typical variations in the air flow rate can be seen in Figure 4.2.

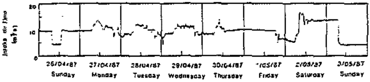

Figure 4.2 Typical variations in the airflow rate.

4.3 Inlet And Outlet Air Temperatures

The average inlet air temperature over the full monitored period was 28,8/32,5°C wb/db and the average outlet temperature was 30,2/32,0°C wb/db.

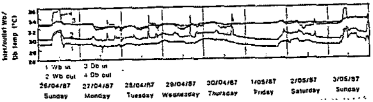

Figure 4.3 Average inlet air temperature.

Figure 4.3 depicts an 8 day period in which the fluctuations in temperature can be studied. The sudden rise in both wet and dry bulb can be seen at blasting times (16h00 - 17h00 daily). This is caused by the heat liberated from explosives. There was a noticeable drop in outlet temperatures with the use of chilled service water. It is, however, not quantified.

4.4 Service Water Flow Rate And Inlet Temperature

The average flow rate of the water was 0,28 l/s overall with 0,86 l/s used during a day shift and 0,02 l/s used during night shift. This flow is extremely low (0,35 tons of water per ton rock mined) if compared to figures in excess of 1,5 ton/ton used in industry for planning purposes. The flow was 'artificially' increased by spraying chilled service water in the working face. This was done for periods of approximately 2 hours daily during the main shift over a five day period. The water usage was increased to a maximum of 6,41 l/s, but the net effect was disappointing.

The water was delivered in the stope of an overall average temperature of 25,9°C with a maximum of 34,0°C and a low of 7,31°C. During day shift the average temperature was 18,6°C with a maximum of 33,12°C and a low of 7,31°C. Since water usage during night shift was minimal the corresponding figures were 28,0°C average, 33,1°C maximum and a low of 13,7°C.

The water temperature decreased with the increased use of the water. The cross-cut chilled service water pipes were not insulated. The temperature of the water in the pipe therefore reached an integral between the wet and dry bulb air temperatures at times of zero or little water usage.

The outlet water temperatures were measured at the chutes of the boxholes. The overall average outlet temperature amounted to 30,6°C while this figure was 29,9°C during day shift and 30,9°C during night shift. It is noteworthy that the outlet water temperature and outlet wet bulb temperature correlated closely. (Figure 4.4)

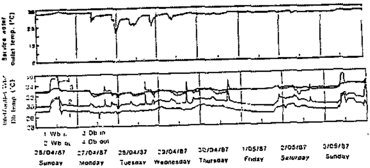

Figure 4.4 Outlet water and return wetbulb air temperature comparison

4.5 Compressed Air Flow Rate, Pressure And Temperature

The average flow rate over the face monitored period was *0,09 kg/s* with a maximum of 0,6 kg/s and a low of 0 kg/s. Only 20% of the total compressed air was used, was consumed during night shift. The drilling shift used an average of 0,2 kg/s.

The pressure of the compressed air *was at an average of* 547 kPa with 573,8 as the average during day shift and 586 kPa during night shift. Figure 4.5 shows typical conditions of an 8 day period.

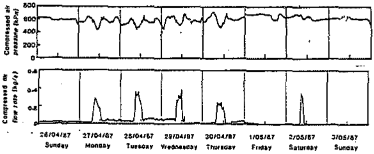

Figure 4.5 Compressed air pressures.

4.6 Electrical Power Consumption

The average total power consumption over the full monitoring period was 3,46 kW. During day shift this figure was 1,14 kW with a maximum of 17,64 kW. The corresponding figures for night shift was 3,46 kW with a maximum of 28,92kW.

The above figures are very low since it was a new stope where the hauling of rock took place over short distances. The rated installed capacity of the electrical motors was 74 kW resulting in the overall utilisation of 39 percent during peak use at night shift. The variations in power consumption is depicted in Figure 4.6.

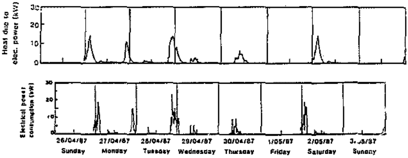

Figure 4.6 Variations in power consumption

4.7 Conclusion

The information gather on the computer based data logger was analysed on surface.

CHAPTER 5

INFORMATION COLLECTED FROM MINE RECORDS AND CONCURRENT EXPERIMENTS

5.1 Introduction

Information was extracted from mine records and used in heat flow calculations. Two experiments were also conducted to establish the virgin rock temperatures and density as well as the thermal conductivity of the rock.

5.2 Mine Records

5.2.1. Rock production

The rock production was obtained from the Survey Department who measured the advance of the working face four times during the monitored period.

The average rock production over the period was 762 centares/month (2 070 tons/month).

5.2.2. Explosives consumption

Dynagel, a nitro-glycerine based explosive was used for rock breaking in the stope. Drill holes had a depth of 0,9 metres, drilled at 70° to the face and spaced 0,3 m apart. Approximately four sticks of Dynagel, 200 mm long with a mass of 1,25 g each were placed in each hole before the blast. A mean total of 1,14 kg of explosives per ton of rock broken was calculated.

5.2.3 Men in stope

Records were kept by the miner in charge on a daily basis. The shift times were known and the heat production by metabolic heat could be calculated using standard figures of heat production. (Metabolic heat production varies with workload, 400 W for hard work to 100 W at rest. In this experiment 300 W was used for calculation purposes.)

5.3 Concurrent Experiments

5.3.1 Virgin rock temperature measurement

A 32 mm diameter diamond drill drilled hole, 6 m long, drilled in an adjacent cross-cut was used for the measurements. A RTD mounted onto a telescopic probe was inserted into the hole and left for 24 hours. Several readings were taken and VRT of 42,5°C, 1 570 m below surface was established.

5.3.2 Thermal conductivity of the rock

Two 26 mm diameter quartz disks were prepared by diamond drilling samples from both footwall and reef. The diamond drilled core was put in a lathe and ends were ground parallel using a diamond grinding wheel and finally polished by hand using carborundum powder. A micrometer was used to measure the flatness of the sample and ensuring a tolerance of less than 15 μm.

The rock samples were saturated with water under vacuum and inserted in a stack which forms part of the measuring instrument.

The thermal conductivity was then determined by using a divided bar apparatus at the Bernard Price Institute of Geophysical Research, University of the Witwatersrand.

5.3.3 Density of the rock

The samples described previously were weighed on a Sartorius scale and the volumes were calculated. From this the following were calculated.

Table 5.1 **Thermal conductivity and density of rock**

	REEF	FOOTWALL
Thermal Conductivity W/mK	7,66	7,39
Density kg/m^3	2 696	2 689

CHAPTER 6

COMPONENTS OF HEAT FLOW - THE ENERGY BALANCE

6.1 INTRODUCTION

Heat flow in stopes is liberated from five major sources: Rock, explosives, electrical equipment, men and the power from pneumatic rock drills.

The heat sinks are ventilation air, compressed air and chilled service water.

Each heat source or sink, except heat flow from rock, can be individually quantified from measurements. Computing a heat balance and equating the total heat removal rate and heat production rate, the heat flow from rock can be deduced. Other heat sources eg. oxidation of timber are not individually identified but are included as heat flow from rock.

6.2 REMOVAL OF HEAT FROM THE STOPE - HEAT SINKS

6.2.1 Ventilation Air

The heat absorbed by the ventilation is calculated as the anthalpy difference between the inlet and outlet of the stope. A time lag of 2 minutes is provided for. This is the time it would take the air to travel through the stope.

The average heat removed by the air over the full monitored period is 80,9 kW with a maximum of 322 kW.

During day shift the average was 92,7 kW and during night shift it was 75,4 kW. The air heat removal rate is shown in Figure 6.1.

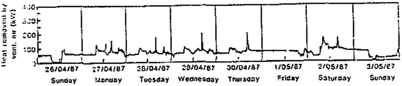

Figure 6.1 Air heat removal rate

6.2.2 Chilled service water

The chilled service water at the start of the shift was normally at a temperature of between that of the ventilation air intake wet and dry bulb temperature. The water temperature decreased with an increased use of water in the stope. The water temperature was normally below that of the exposed rock and while used, below the ambient temperature of the air.

The contribution of chilled service water as a heat sink was calculated as the difference in enthalpy of the water flow between inlet and outlet of the stope. The outlet temperature of the water varied and the relationship to that and the outlet air was shown in Figure 6.2.

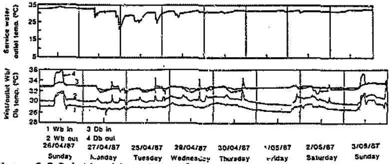

Figure 6.2 Relationship water and air temperature.

It was assumed that the water spent one hour in the stope and that heat and mass transfer took place throughout this period. To account for the aforementioned a one hour exponential delay to outlet temperature was considered in quantifying the heat sink.

The overall average heat removal rate of the water was 15,5 kW, during day shift this amounted to 49,7 kW and at night shift to 0,64 kW. The variation in heat removal rate can be seen in Figure 6.3.

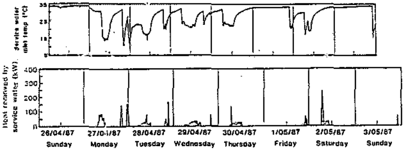

Figure 6.3 Variation in heat removal rate

The use of water in this particular stope was 0,35 ton of water per ton of rock mined. It resulted in a heat removal of 15,5 kW average over the monitored period. The use of the water in the stope was increased by washing down the working area by means of hoses. The consumption doubled to 0,7 ton/ton and as was expected the heat removal increased to 31 kW.

It was concluded during this exercise that the use of chilled service water as a heat sink is very limited.

6.2.3 Compressed air

Compressed air has a dual function in providing instantaneous cooling and providing mechanical power to the drilling machines. All mechanical input to a rock drill is liberated as heat into the ventilation air.

The net effect of compressed air is cooling which is equal to the difference in enthalpy of the compressed air inlet and outlet. It was taken that the average cooling effect was 6 kW as found by Bluhm[5].

The cooling effect of the compressed air is shown in Figure 6.4.

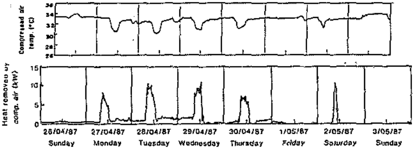

Figure 6.4 Cooling effect of compressed air.

6.3 FLOW OF HEAT INTO STOPES - HEAT SOURCES

6.3.1 Explosives

The energy liberated during the blast is in the form of heat and shock waves. Approximately 15 percent of total energy liberated is shock waves, which has no effect on the thermal environment[18]. The heat liberated by Dynagel was estimated at 4 104 kJ/kg.

Whillier [19] reported that all the energy liberated by explosives is dissipated within a short period after the blast. Hemp[20] found that the heat from explosives is absorbed by the broken and standing rock and is slowly released into the environment. In this exercise sharp peaks occurred in both the outlet air wet and dry bulb temperature of the stope. The change in temperature depends on the quantity of explosives used, the duration of the blast and the air flow rate.

A sample period of 15 days revealed that the average increase in wet bulb during the blast was 1,7°C and 2,1°C dry bulb temperature. An hour after the blast, the wet bulb was an average within 0,6 percent (0,2°C) and the dry bulb within 1,5 percent (0,5°C) of the temperatures prior to the blast. The average heat liberated within a hour after the blast was 610,9 MJ. This figure includes heat from ignitor cord, fuses and heat liberated by broken rock and newly exposed rock surfaces. The percentage heat liberated by the blast is as follows:

TABLE 6.1 Percentage heat liberated by explosives

PERCENTAGE HEAT	TIME AFTER THE BLAST (Minutes)
66	15
23	30
8	45
3	60

The test results obtained indicates that all the energy from the blast is dissipated within a short period after the blast. The change in heat flow can be seen in Figure 6.5

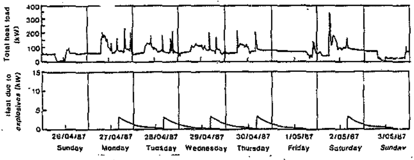

Figure 6.5 Change in heatflow rate

The release of energy from explosives is depicted in Figure 6.6.

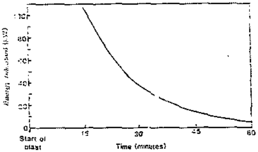

Figure 6.6 Release of energy from explosives

An exponential delay with a time constant of 8 hours has be n assumed as an approximation for calculation of heat flow from explosives. The overall average heat load was 2,6 kW with 0,9 kW during day shift and 2,7 kW during night shift.

6.3.2 Men

A nominal average metabolic heat production rate of 300 W per person is used to cater for all stoping activities[20]. The number of men present in the stope and shift times were recorded. The average heat load from men was calculated to be 2,7 kW over the monitored period. The day shift average was 7,4 kW and the night shift average was 2,7 kW (Figure 6.7).

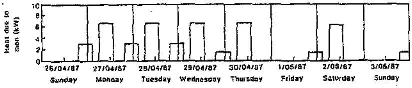

Figure 6.7 Heat production by men.

6.3.3 Electrical equipment

All electrical power used in the stope is converted into heat energy. The heat is released in a two fold action. A certain amount is released into the rock being scraped while the rest is released almost immediately into the air as equipment inefficiency. Since the mechanics of heat transfer in this case are complex, it has been assumed that all heat from electrical power is dissipated in the stope and that the release of this delays exponentially with a time constant of 1 hour[21]. This can be seen in Figure 6.8.

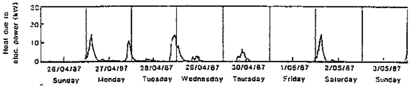

Figure 6.8 Electrical power.

6.3.4 Power derived from compressed air

Compressed air is primarily used to power rock drills and to a lesser extend in the cleaning out of drilled holes. Heat is liberated into the air stream by the energy used in powering the rock drill. The amount of he 'iberated is determined by the change in enthalpy of the compress d air across the rock drill.

6.3.5 Heatflow from rock

The heat flow from rock was calculated by the difference in the total heat flow removed from the stope and the measured heat sources.

The average heat flow from rock over the monitored period was 90,5 kW and the average over day shift was 140 kW while this figure was 65,1 kW. The variations in heat load from rock can be seen in Figure 6.9.

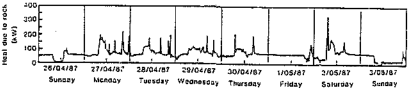

Figure 6.9 Heatflow from rock

CHAPTER 7

DISCUSSION AND CONCLUSION

7.1 Introduction

The two dominant factors in this study were heat flow from rock which contributed 93,4 percent of the total heat flow and the ventilation air which removed 77,3 percent of the heat. The relative magnitudes of different heat sources and sinks can be seen in Figure 7.1 and Table 7.1.

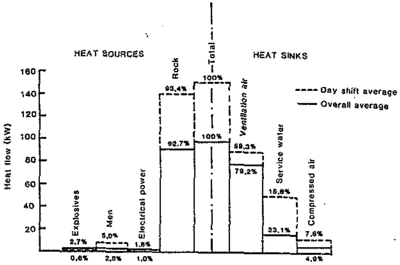

Figure 7.1 Relative magnitudes of heat sources and sinks

Table 7.1 Magnitudes of heat flow components in test stope

	HEAT SINKS (kW)				HEAT SOURCES (kW)			
	VENTILATION AIR	SERVICE WATER	COMPRESSED AIR	TOTAL	EXPLOSIVES	ELECTRICAL POWER	MEN	ROCK
Overall Average	77,3	15,5	4,8	97,6	2,6	1,8	2,7	90,5
Day Shift Average	88,9	49,6	11,4	149,9	0,9	1,5	7,5	140,0
Night Shift Average	72,0	0,7	3,0	75,7	2,7	5,2	2,7	65,1
Working Day	79,7	18,7	5,4	103,8	3,0	2,1	3,3	95,4

7.2 Heat Flow

Heat flow from rock on average makes up for 92,7 percent of the total heat load. The heat load from exposed rock, broken rock and from other unspecified heat sources is included in the heat flow from rock term.

93,4 percent of the total heat load was from rock during day shift and 95,1 percent during night shift.

Artificial heat is the collective term for small heat loads which includes heat from explosives, men and electric power. The artificial heat load in the stope contributed 7,1 kW (7,0 mJ/ton) of rock broken overall. This figure was 9,9 kW (9,3 mJ/ton) during day shift and 10,6 kW (10,5 mJ/ton) during night shift.

7.3 Heat Sinks

Ventilation air was the most dominant heat sink and removed on average 79,2 percent of the total heat load. During the day shift, 59,3 percent of the day shift heat load and on night shift 95,1 percent of the heat load was removed by ventilation air.

The chilled service water removed 15,8 percent of the total heat load overall. The value for day shift was 23,1 percent and for night shift 0,9 percent. The heat removed by compressed air during day shift amounted to 7,6 percent while the overall contribution was only 4,9 percent.

7.4 OVERALL DISCUSSION

If compared to previous experiments by Matthews [9] and Bluhm [5] the relative magnitudes of heat flow components are in agreement. However, the total heat loads differs greatly since this experiment was done in a ledging stope. The results are also in agreement with Heatflow as were discussed previously.

There was a large difference between rock heat load during day and night shift. Since the 'artificial heat load' for day and night shift differed by less than 1 kW, and the increase in heat removed by the ventilation air on day shift (16,9 kW) may be accounted for by the increase in mass flow rate. Ventilation doors left open in this ventilation district had a bearing on the varying mass flow rates. The indication is that the increase in heat flow from rock can be attributed to the use of chilled service water. The removal of rock from the stope during the cleaning cycle is another possible reason.

The variations in rock heat load as in Figure 6.9 exhibits peaks in rock heat flow at times when stoping activities are taking place. This correlates well with the use of chilled service water, as well as blasting of explosives.

7.5 CONCLUSION

Although the results of previous heat flow monitoring exercises in conventional and a back filled stope are comparable, the relative effect of individual heat flow components on the total heat load are more apparent in a ledging stope.

Important tendencies on the release of heat from explosives has been identified.

The use of chilled service water has a positive effect on the average face wet bulb temperature. Further analysis of the effect of chilled service water on the stope environment is required since its current contribution does not seem worthwhile.

CHAPTER 8

HEATFLOW - THE COMPUTER PROGRAM

8.1 INTRODUCTION

In this chapter the computer program, HEATFLOW is discussed and used to predict the outcome of the experimental data. The input parameters to Heatflow is what were measured underground. The output of the program is then compared to the actual results obtained. The discussions are limited to the stoping section of Heatflow only.

8.2 Heatflow - The COMRO Computer Program

8.2.1. Brief description

Heatflow is a program that has been developed by the Environmental Engineering Laboratory of the Research Organisation. The mine air-path network is reduced to a network of heat sources and heat sinks. The relative location of the various boxes and the correct airflow directions are achieved by assigning node numbers to the start and end of the boxes.

The outlet temperature of each box is calculated by determining the heatflow from rock as well as the other traditional sources of heat within that particular box. The calculated outlet temperature of the box then become the intake temperature of the box on the downstream air side.

8.2.2. Algorithms used to determine the heatflow in a stope

The calculations within the stoping zone are done on the basis of one dimensional conduction in parallel slabs. The broken rock left in the stope after the blast is assumed to be removed from the stope at a temperature equal to the intake drybulb temperature of the ventilation air.

Only the total heatflow in the airstream is predicted. No differentiation is made between latent and sensible heat. The algorithms used for the stoping section of Heatflow are as follows:

Footwall and Hangingwall

$$Q = 4{,}9 \frac{kpc}{\pi}^{.5} (Sfa)0.5\ \beta\ dT$$

Face and broken rock

$$Q = S_{fa}\ Sw\ pc\ dT$$

8.3 Discussion

The input parameters of the experimental stope were used to calculate the expected heatflow. The results obtained are shown in table 8.1. below and are highly significant.

Table 8.1. Validation of Heatflow using the experimental data.

DESCRIPTION	EXPERIMENTAL DATA	HEATFLOW
INPUT		
Intake Wet Bulb Temperature °C	28.8	28.8
Intake Dry Bulb Temperature °C	32.5	32.5
Mass Flow kg/s	10.1	10.1
Rock production tons/month	895.0	895.0
Service water ton/ton	0.3	0.3
Service water inlet temperature	28.0	28.8
Depth of stope m	1539.0	1539.0
OUTPUT		
Return Wet Bulb Temperature °C	30.49	30.4
Return Dry Bulb Temperature °C	32.18	32.1
Total heat produced kW	79.1	78.7

25x

The average conditions measured underground during the experiment were used in the comparison. The experimental error can be ascribed to be the cause of some differences. The results of the Heatflow stoping section is, however, very acceptable when used in real life to predict the same doing a mining planning exercise

8.4 Conclusion

The validity of the stoping section of Heatflow is hereby confirmed. The data collected during this experiment will be used by Comro to further the understanding of heatflow in a stope. The quantifying of underground heatloads during the planning stages are of paramount importance. The understanding of heatflow underground in turn will lead to the better design of underground layouts to minimise unwanted heatflow.

Further work is required to quantify the use of chilled service water as a direct cooling medium in the stope. It is thought that more than 50% of the total water usage underground is used in working places other than the production faces. The result being that heat is removed inefficiently since it could be used at an unwanted location.

It is also concluded that the rock produced by the blast, in the working face, should be watered down as soon as possible after the blast to remove the heat. Alternatively the broken rock should be removed as soon as possible from the face after the blast. This would then be the removal of the heat source and not the removal of the heat from the source.

APPENDIX 1.

CALIBRATION OF INSTRUMENTS

1. VORTEX ANEMOMETERS

The average velocity of the air in the cross cut were determined using a vane anemometer in the traversing method. Spot readings were then taken at the proposed vortex anemometer position. The vortex anemometer was installed at the position where the spot reading equaled the average velocity in the cross cut. Several similar exercises were done throughout the experiment at the vortex anemometer positions to confirm their accuracy.

To determine the quantity of ventilation air at any such position the cross sectional area was measured and incorporated in the calculations. The aforementioned measurements were done several times throughout the experiment.

The vortex anemometers itself were calibrated prior to the experiment under laboratory conditions. The conversion and correction factors were used in the calculations.

2. PSYCHROMETERS

The psychrometers consisted of two Resistance Temperature Devices (RTD) each. The RTD's were calibrated under laboratory conditions prior to the experiment and certificates issued. Additional checks were, however, undertaken during the experiment by using an Asmin Precision Psychrometer held next to an installed psychrometer. This was done to confirm the wetbulb temperatura readings of the psychrometer.

3. WATER FLOWMETER.

Readings were taken simultaneously on the flowmeter and data logger. At the same time the water flow was discharged into a 200 liter drum while the time to fill it was measured. The results were satisfactory and no adjustments had to be done.

REFERENCES

1. GEBLER, W.F. Presidental Address. A Review of Mine Cooling Practices Through the Ages. Journal of the Mine Ventilation Society of South Africa. December 1980.

2. HOWES, M.J., The Development of a Functional Relationship between Productivity and the Thermal Environment. Journal of the Mine Ventilation Society of South Africa. Vol. 31. No. 2. Feb 1978. p.21.

3. APPELMAN, G. and GARDINER, L.R., Annual Ventilation Report for the Period 1985 to September 1986. Unpublished Data. Personal Communication. Comro. 1987.

4. BOTTOMLEY, P. The Thermal Insulation of Mine Airways. MSC Dissertation. University of the Witwatersrand. 1987.

5. BLUHM, S.J., ALEXANDER, N.A., MARCH, T.W., BOTTOMLEY, P. and VON GLEN, F., The Measurement of Heat Loads in a Deep Level Stope in the Klerksdorp Goldfields. Journal of the Mine Ventilation Society of South Africa. Vol.39. No. 10. October 1986. pp.129-140.

6. JOUGHIN, N.C., Production Trail of Small Hydraulic Rock Drills - An Overview. Unpublished Data. Personal Communication. Comro. 1984.

7. LAMBRECHTS, J. de V., An Empirical Study of Heat Flow in Stopes in South African Gold Mines. Journal of the South African Institute of Mining and Metallurgy. Vol. 59. January 1959. p.285.

8. LAMBRECHTS, J. de V., Wet Bulb Temperature Gradients in Stope Ventilation. Journal of the Mine Ventilation Society of South Africa. Vol. 19. February 1966. p.13.

9. MATTHEWS, M., McCREADIE, H.M. and MARCH, T.W. The Measurement of Heat Loads in a Backfilled Stope. Journal of the Mine Ventilation Society of South Africa. August 1987.

10. STARFIELD, A.M., A Theoretical Study of Heat Flow in Rock Surrounding Underground Excavations. Phd Thesis, University of the Witwatersrand. 1968.

11. GOULD, M.J., The Determination of Parameters Concerned with Heat Flow into Underground Excavations. MSC Thesis. University of the Witwatersrand. 1968.

12. VON GLEN, F.H., A Study of Heat Exchange in Advancing Stopes. MSC Dissertation. University of Witwatersrand. 1987.

13. STARFIELD, A.M., Heat Flow into the Advancing Stope. Journal of Mine Ventilation Society of South Africa. Vol. 19. Feb 1966. p.13.

14. STARFIELD, A.M., The Computation of Air Temperature Increase in Advancing Stopes. Journal of the Mine Ventilation Society of South Africa. Vol. 19. December 1966.

15. WHILLIER, A., Estimation of Heat Pickup by the Ventilation Air in Stopes. Unpublished Data. Personal Communication. Comro. 1973.

16. WHILLIER, A. and RAMSDEN, R., Sources of Heat in Deep Mines and the Use of Mine Service Water for Cooling. First International Mine Ventilation Congress. Johannesburg. September 1975. p.342.

17. VAN DER WALT, J. and WHILLIER, A., Heat Pickup from Rock in Gold Mines - The Water Rock Thermal Balance and the Thermal Efficiency of Production. Journal of the Mine Ventilation Society of South Africa. Vol. 32. July 1979.

18. LOWNDS, C.M. and DU PLESSIS, M.P., The Behavior of Explosives in Intermediate Diameter Boreholes. Proceedings Annual Conference of the South African Institute for Quarrying. March 1984.

19. WHILLIER, A. and RAMSDEN, R., Sources of Heat in Deep Mines and the Use of Mine Service Water for Cooling. Proceedings International Mine Ventilation Society of South Africa. Johannesburg. 1975.

20. HEMP, R and DEGLON, P. A Heat Balance in a Section of a Mine. Proceedings 2nd International Mine Ventilation Congress. Reno Nevada. 1979.

21. McCREADIE, H.N., DE JAGER, E.J. and MARCH, T.C.W., The Measurement of Heat Flow in a Ledging Stope. Unpublished Data. Comro. 1988.