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Analysis of Heat Sources in a Large Mechanized Development End at Mount Isa Mine

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The T62 decline at Mount Isa is a nominally 5.5 m wide and 4.7 m high heading driven at a grade of one in eight which is to be used as a main electric truck haulage for the 3000 and 3500 orebodies below the current shaft hoisting systems. The measured and expected virgin rock temperatures are 48°C at the main tipple 1000 m below surface and 58°C at the main loading loop 1500 m below surface. The portion of the decline considered in the study comprised a 75 m long face section between the face and the first storage bay and a 300 m long tunnel section from this storage bay to a point of through ventilation. This heading was one of several available to the same development crew, advanced an average of 3.5 m/day and was worked intermittently during the three shift operation.

The face was drilled using a two boom electric hydraulic drill jumbo and was blasted using ANFO explosive. A 231 kW diesel powered loader was used to clear the face to the storage bay recess. This was also used to load a 287 kW diesel powered truck which removed the broken rock from the heading to a waste rock pass 700 m further up the decline. Two 1.22 m diameter flexible ventilation ducts were used to supply ventilation air to within 30 m of the face. Two observers were continuously present during the 120 hour study period and they completed 37 air quantity and air temperature surveys covering both the face and the tunnel sections of the development. The observers also monitored and timed the period of operation of the various items of mining equipment in both sections of the heading.

Measurements of the average heat produced by the diesel powered equipment whilst undergoing typical load cycles were taken prior to the main study by measuring the fuel consumption. A bucket hydraulic line pressure transducer which was calibrated using lead weights was used to determine the average broken rock load in the bucket. An analysis of the measurements taken during the study clearly demonstrated the storage and release of heat by the rock surfaces when the diesel engine is operating and stopped. The average amount of heat removed from the heading was 254 kW of which the rock contributed less than half at 118 kW, diesel powered equipment 105 kW, electric equipment (excluding the ventilation fans) 11 kW, explosives 9 kW and the broken rock removed from the heading 12 kW.

The study showed that where cyclic mining operations take place using diesel powered equipment, using a heat load averaged over 24 hours will provide a reasonable estimate of the conditions that may be expected when the diesel equipment is not in use. During the short periods of diesel equipment operation, multiplying the average heat load by a factor of between two and three will indicate the maximum temperatures that may be expected. Hot environments at Mount Isa are managed by the use of shortened shifts as a warning and stopping work completely when the limiting heat stress condition is encountered. The normal design is therefore to provide optimum ventilation and cooling relative to thermal productivity for the mining operations where diesel powered equipment is not used and, to confirm that this design will not result in six hour shift conditions when the diesel powered equipment is in use.

INTRODUCTION

The high transient heat loads associated with the operation of diesel powered mining equipment in mechanised mining systems presents difficulties when designing the ventilation and cooling arrangements that will meet the required thermal environmental standards. This is particularly so in development headings where the rock removal part of the cycle is relatively short, however, the total rated diesel power of the equipment used can be several hundred kilowatts.

The study described in this paper was conducted at Mount Isa Mine by Dr A.D.S. Gillies of the Department of Mining and Metallurgical Engineering at the University of Queensland with the support of vacational students from the University. Mount Isa Mine is located 1600 km north west of Brisbane in the state of Queensland, Australia. Silver, lead, zinc and copper are mined from orebodies covering an area approximately 3.5 km long, one km wide and one km deep. The study was undertaken in the access decline to the 3000/3500 orebodies. These are lensy copper deposits that occur beneath the existing mine development which is 1100 m below the surface and extend to a known depth of 1600 m below surface. The access decline is located in a shale pillar which lies between the two orebodies. The virgin rock temperature is 48°C at the start of the decline and, with a geothermal gradient of approximately 1°C every 50 m, is expected to be 60°C at the bottom of the access decline. Planning the ventilation and cooling strategy for this extension of mining at depth was started in 1981 and is described in greater detail elsewhere[1]. During this planning period, it was recognised that the operation of large diesel powered equipment resulted in localised "hot" spots which, if designed for by assuming continuous operation, would result in a significant overestimate of the cooling requirements. The purpose of the study was therefore to identify the extent of the transient heat loads relative to the mining cycle and to provide data which could be subsequently used for ventilation and refrigeration planning.

DESIGN OF THE STUDY

Location

The measurements were undertaken in the T62 decline, a one in eight slope development end, being advanced by multi-shift operations off 21C sub which is 1056 m below surface. Figure 1 is a plan of the experimental site and also shows the excavation dates. For the study, the development tunnel was assumed to start 15 m beyond the last cross-cut which carried through ventilation air. The face area was assumed to extend from station A to the new face and varied in length from 68 m at the beginning of the study to 86 m at its completion 120 h later.

Measurements were taken at stations A, B, C and D which were evenly spaced at 100 m as illustrated in Figure 1. Mining of the development end commenced in January 1988 and the rate of face advance (including loading bays) averaged 1 m per working day over the complete period. This average rate included some significant periods when no mining occurred.

Fresh air was supplied into the face area using two 1220 mm diameter flexible ducts with provision to measure the air temperatures and velocity at all the stations. Generally, the decline appeared dry except when the three sumps provided to store water for pumping overflowed. Service water was supplied for both drilling and watering down freshly broken rock. The footwall, for a distance of 10 m from the face, was observed to be wet except when there was no face activity.

Rock removal from the development face took place in stages. Firstly the freshly broken rock was cleared from the face to the first loading bay at A using a diesel powered load–haul–dump (LHD) machine. This allowed access to the face area for subsequent operations to proceed. From the loading bay at A and depending on the availability of a diesel powered truck, the rock was either loaded into the truck and removed from the heading to the waste rock dump or, transferred to the loading bay at C using the LHD where it would be subsequently removed from the heading using the truck.

Mining Activities

The face mining activity during the study period is summarised in Figure 2 and the waste rock haulage activity in Figure 3. Details of the mining equipment used and some typical study data values are given in Table 1. The thermal properties of the rock given in Table 1 were obtained by measuring the virgin rock temperature and the surface heat transfer coefficient[2] and by averaging the figures previously obtained for the pyritic Urquhart shales[3].

Throughout the study period, two observers were present to undertake the necessary air temperature and airflow

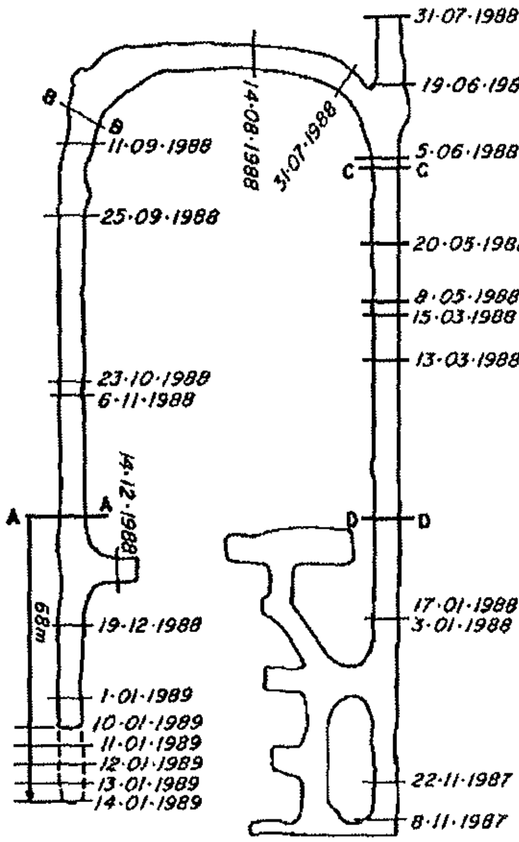

Figure 1. T62 decline

measurements and also to detail the mining cycle and equipment operating periods. In addition, data acquisition systems were used to monitor water usage, compressed air flow and pressure and electric power consumption. The mining cycle activities obtained from the observations taken during the study period are compared in Table 2 to those obtained from a separate investigation during a two week period prior to the study.

Table 1
Typical development data values

Rock properties	
Rock type: Pyritic Urquhart shale	
Surface rock heat transfer coefficient	4.0 W/m²K
Face virgin rock temperature	49°C
Rock density	2850 kg/m3
Rock thermal conductivity	3.67 W/mK
Rock thermal capacity	0.622 kJ/kgK
General information	
Heading average size	6.72 m x 5.00 m
Face area average size	6.23 m x 5.28 m
Fan air quantity	35.3 m³/s
Face air quantity	14.5 m³/s
Supply air temperatures	22.0/40.0°C
Barometric pressure	109 kPa
Footwall water temperature	40°C
Truck details	287 kW, 36 tonne, Cat D400
Loader details	231 kW, 6.1 m³, Elphinstone
Drill details	72 kW, two boom Tamrock
Service water	1.5 l/s, 40°C
Compressed air	0.05 kg/s, 740 kPa, 40°C
Depth below collar	1101 m
Depth below sea level	694 m

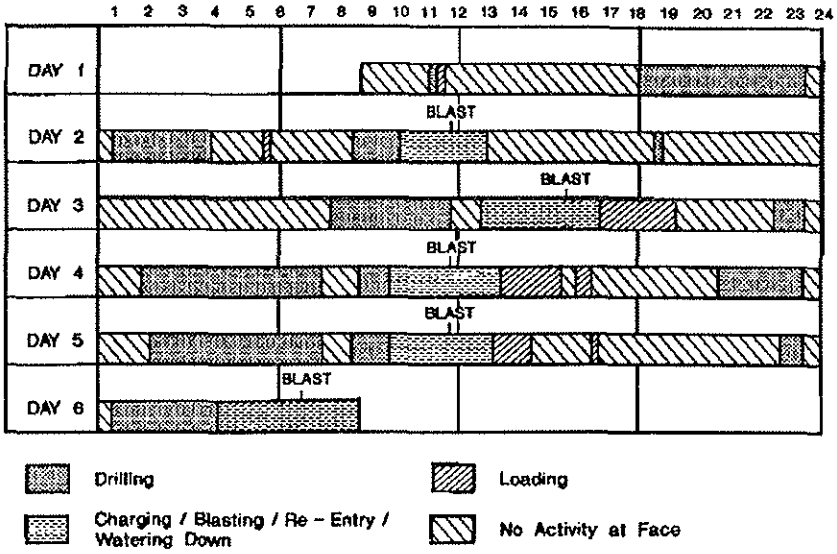

Figure 2. T62 decline tunnel face area activities

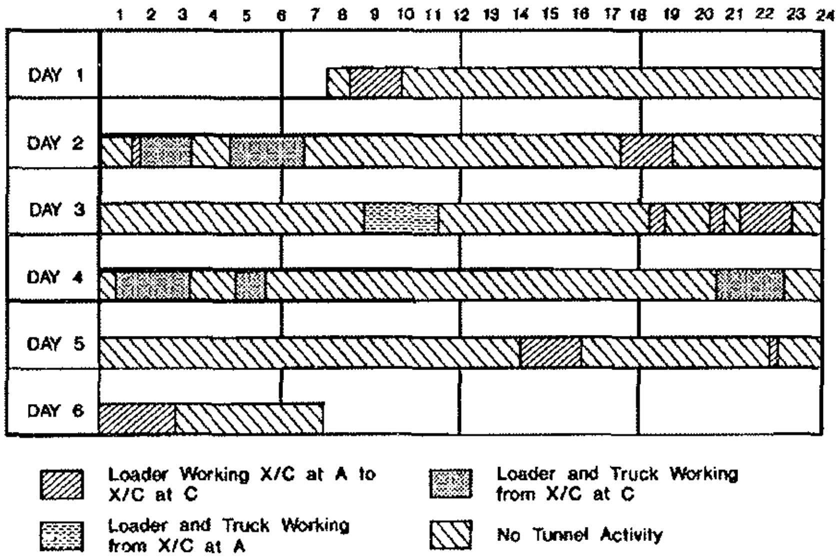

Figure 3. T62 development tunnel waste rock haulage activity

Table 2
Mining cycle activity times

Activity	Proportion of the mining cycle (%)	
	Heat load study	Independent study
Drilling	31.2	30.2
Charge and blast	17.7	19.2
Loading	13.9*	7.5†
Other and no work	35.6	35.4
Six hour shifts	1.6	7.7

* includes machine idling at face
† excludes machine idling at face

Table 3
Diesel fuel consumption and activity

Activity	Cycle time (min)	Fuel consumption (l/min)	Heat produced (kW)
Loader idling	–	0.067	43
Truck idling	–	0.1	64
Loader, face to A	2.33	0.625	401
Loader A to C	4.67	0.74	475
Truck from A	20.0	0.4	256
Truck from C	15.0	0.393	252

Diesel Heat Loads

The heat resulting from the operation of the diesel loader and the truck was calculated from the measured fuel consumption and by assuming complete combustion of the diesel fuel. A calorific value (sensible and latent heat) of 45.6 MJ/kg and a density of 0.845 kg/l was assumed for the diesel fuel. The measured diesel fuel consumption varied with the rock removal activity i.e. whether both the truck and loader, or only the loader was in use. Some typical values are given in Table 3. The amount of rock transported by both the loader and the truck was obtained by fitting a bucket hydraulic line transducer to the loader. Lead weights were used to calibrate this system. Normal "full" bucket loads were observed to range between 10 and 14 tonnes with an average of 12 tonnes. For a rock density of 2850 kg/m^3 and a bucket volume of 6.1 m^3, the swell factor is 1.45.

Although the tray capacity of the truck is 36 tonnes, to avoid damage to the flexible ventilation ducting, the truck was filled with two loader buckets i.e. an average of 24 tonnes. The reduced truck load resulted in a greater clearance between the ventilation ducting and the broken rock in the truck.

Electric Power, Water and Compressed Air

The electric power consumed by the electric hydraulic two boom drill jumbo during the study period is illustrated in Figure 4. When determining the heat sources it was assumed that all energy was dissipated in the face area in the form of kinetic energy or machine inefficiencies.

Figure 5 shows the results of the metered water consumption during the study period.

Compressed air can have a significant cooling effect on the surroundings as a result of the difference in heat content between the compressed air (assumed to enter in a saturated condition) and the ventilating air. The measured air pressures and consumption during the study period are illustrated in Figure 6. Compressed air was only used to flush out drill holes and to drive a face sump pump and, as a consequence, its cooling effect relative to that in a development heading using pneumatic powered drills was very small.

Explosives

Face blasts took place approximately every 24 hours with the heat output from the exothermic reaction estimated to be 3200 kJ/kg of explosive, using the manufacturer's data. Diverse opinions are expressed with respect to the release path of explosive blast energy. It can be considered[4] that essentially the entire available energy of the explosives appears (ultimately) as heat.

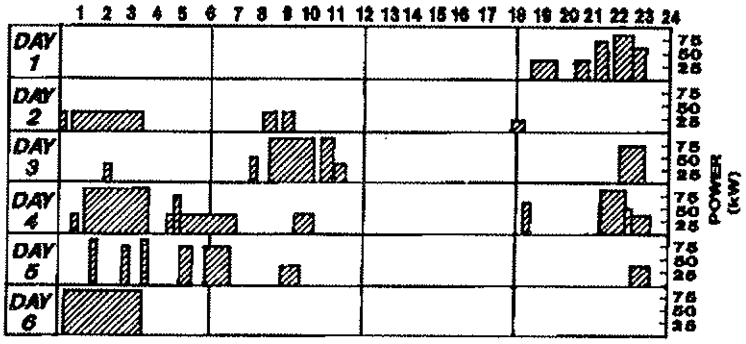

Figure 4. Electric power consumed by electric hydraulic drill jumbo

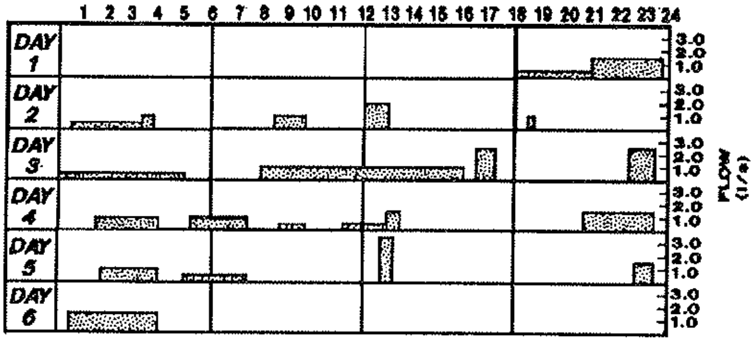

Figure 5. Water consumption

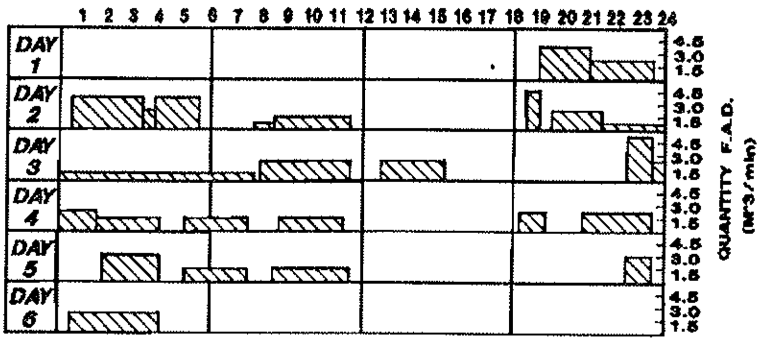

Figure 6. Compressed air consumption

In South African gold mines it was proposed[5] that the heat from explosives was not significant because most of it was removed by the ventilation air within an hour or two of the blast. In a study concerning the heat exchange processes in a South African gold mine stope[6], it was maintained that the explosives did not result in a significant air temperature increase after the blast but that the heat must be transferred to the rock and then to the air over a longer period.

It is probable that some of the heat produced will be carried away with the blasting fumes out of the development end and some will remain in the broken rock which may be released prior to and during rock removal. The proportion of heat removed by each process is a function of rock fragmentation, the ventilation arrangements and the mining cycle. For this study, the release path for blasting energy was estimated to be[7] energy directly to the air 35 per cent, seismic energy 10 per cent, energy into the surrounding rock 5 per cent and the balance of 50 per cent of the energy into the rock broken by the blast. It was also assumed that 90 per cent of the total blast energy would appear as heat in the development end by one or other of these pathways.

ANALYSIS OF HEAT SOURCES

The main sources of heat in any development end are diesel and electric powered equipment, explosives, the surrounding rock, fissure water and service water. For future predictive work it is important to relate the estimates obtained using information such as design load cycles with the actual measured values. For a 3.8 m face advance per blast the amount of rock to be removed after each blast was taken to be 360 tonnes.

Air Quantities

When determining the balance between the heat sources and sinks and the variation in heat and moisture removal from the development heading, it was essential to establish the amount of air at each measuring station as accurately as possible. Over the extended period of the study, damage to one or both of the ventilation ducts would increase the leakage and, although the total quantity of air supplied to the development heading would only change slightly, the amount of air delivered to the face section could be significantly reduced.

During the study period, centre point readings were taken using a vane anemometer in both the duct and the heading at each measuring location. These were adjusted with correction factors of 0.84 and 0.68 for the ducts and heading respectively using the results of pitot tube traverses undertaken prior to the study. There were no significant changes in ventilation during the study and measured values (obtained from 436 measurements) are summarised in Table 4.

Table 4
Airflow rates and duct leakages, m^3/s

Predicted	Face	A	B	C	D	Fan
Duct 1	7.8	8.0	9.1	11.5	13.8	18.3
Duct 2	6.7	6.9	7.9	10.3	12.5	16.9
Total	14.5	14.9	17.0	21.8	26.3	35.2
Measured Total		14.5	16.0	22.2	26.5	

Also given in this table are the predicted airflows in the ducts based on the fan performance and a uniform duct leakage. This fitted the measured airflow rates where the surface area of the holes in the duct causing the leakage varied between 230 and 250 mm^2/m^2. For this type of flexible ducting in 20 m lengths using bell and ppigot couplings, the expected leakage is 200 mm^2/m^2 when the duct is new. For ducting in average to poor condition the leakage will increase by factors of two and five respectively.

Bends in the ducts were included by adding a total equivalent length of 75 m to the actual duct length of 475 m between the fans and the discharge at the face. The mass flow rate of air entering the development face section at station A during the study period was 18 kg/s and that entering the heading at station D was 32 kg/s.

Diesel Powered Equipment

The estimated cycle time and weighted mean engine load for a 231 kW loader with a bucket capacity of 12 tonnes and clearing broken rock from the face to the storage bay at station A which is an average of 77 m from the face, is 146 s and 85 per cent (99 s load and haul up at 100 per cent, 12 s manoeuvre at 70 per cent and 35 s dump and return at 50 per cent). Allowing for a total of 34 (30 plus 4 face cleaning) cycles, a total of 82.7 minutes is the minimum time required to clean the face of broken rock after a blast.

The actual cleaning time required for four blasts during the study period varied between 126 and 145 minutes with an average of 134 minutes. Assuming that when the loader is not removing rock it is idling at 7 per cent load, the average heat output during the face cleaning period is estimated to be 382 kW. This is based on a 33 per cent diesel engine efficiency and is within 5 per cent of the 401 kW given in Table 3.

For the same loader, moving broken rock between the storage bays at stations A and C which are 200 m apart, the mean load is estimated to be 75 per cent over a 276 s cycle. Allowing 30 cycles to move the 360 tonnes of rock produced in a blast results in a minimum estimated time of 138 minutes for the operation to be completed. The storage bay capacities at stations A and C are greater than the 360 tonnes produced by a blast and consequently the number of cycles each time this operation takes place can be significantly different from the long term average of 30.

The actual time involved in moving broken rock between the two storage bays using the loader on four separate occasions resulted in operational times of between 110 and 218 minutes with an average of 162 minutes. Again, assuming that during the balance of the operating time the loader is idling, the average heat output is estimated to be 450 kW which is close to being within 5 per cent of the 475 kW given in Table 3.

The removal of the broken rock from the development end to the main waste rock dump was carried out using a 287 kW truck which was loaded with broken rock from the storage bays at either stations A or C. The overall cycle times for the truck dumping in a waste pass some 700 m further up the decline was estimated to be 17.5 and 14.5 minutes from the loading bays at stations A and C respectively.

With the truck idling whilst being loaded and the loader idling whilst the truck is hauling, the total heat output in the heading (excluding the 700 m to the waste dump) was estimated to be 294 kW and 211 kW respectively. These are within approximately 15 per cent of the measured values of 256 kW and 252 kW given in Table 3. During the first 90 hours of the study, based on these estimates, the total heat output resulting from the operation of the diesel powered equipment in the heading was calculated to be 9445 kWh or an average heat release of 105 kW throughout the study period.

Electric Power

The drill jumbo was the only user of electric power in this development end. The ventilation supply air fans were excluded by taking measurements in the ducts and heading beyond the fans and could easily be accounted for in future predictive work if necessary.

The energy required during the drilling cycle for the four face advances varied between 200 and 320 kWh with an average of 270 kWh. In addition to the face holes, drilling included that for ground support with rock bolts and is the main reason for the significant variation in energy used during the drilling cycle. Over the first 90 hours of the study, the total energy used was 975 kWh or an average power of 11 kW.

Explosives

For the nominal 3.8 m face advance, a total of seventy 4.0 m long holes were drilled in a centre burn cut drill pattern. The explosives used for each blast were 280 kg of ANFO, 25 kg of ISANOL (50 per cent polystyrene and 50 per cent ANFO) and seventy powergel primers and detonators resulting in a total of 1080 MJ of energy released during each blast. For the three blasts occurring during the first 90 hours of the study, and assuming that 90 per cent of the total blast energy appears as heat, the average heat load was 9 kW.

Broken Rock

The rock broken during blasting will be partially cooled by the ventilating air prior to its removal from the heading. A total of 8 trucking cycles were involved in moving the broken rock from storage bay A and 42 cycles from storage bay C. Assuming a truck load of 24 tonnes, a total of 1200 tonnes was removed from the heading during the first 90 hours of the study. The heat load to the ventilation air resulting from cooling this mass of rock which has a thermal capacity of 0.622 kJ/kg is approximately 2.4 kW per degree centigrade change in rock temperature.

The air dry bulb temperature during the study period varied between 40°C and 45°C and, for a virgin rock temperature of 49°C, the probable heat load was 5 x 2.4 or 12 kW. Measurements of the temperature at the surface of the rock pile were between 39°C and 40°C and one measurement within the rock pile shortly after blasting indicated a temperature of 51°C.

Water and Compressed Air

Service water was used during drilling, blasting and for watering down during rock removal. The flow rate varied between 0.2 and 3.5 l/s and averaged 0.45 l/s during the first 90 hours of the study. For each 1°C change in the water temperature between entering and leaving the development end, the sensible heating effect is approximately 2 kW.

The service water supplied to the development heading was not chilled and entered and left the heading at approximately 40°C (close to the air dry bulb temperature) and therefore contributed a negligible sensible heating or cooling effect. Fissure water other than surface seepage was not encountered in measurable quantities.

Compressed air consumption varied between 0.02 and 0.10 kg/s at gauge pressures between 700 and 750 kPa. The average consumption over the first 90 hours of the study was 0.023 kg/s at a gauge pressure of 730 kPa. The temperature of the compressed air entering was 40°C (close to the air dry bulb temperature). Approximately half of the compressed air supplied to the heading was used to drive the face pump and, assuming an overall pump efficiency of 25 per cent, compressed air cooling in the development heading was less than 0.5 kW.

Rock Surfaces

The heat flow from within a rock mass to the rock surface and then to the ventilating air is a complex process dependent on the rock and air thermal properties, the time since the excavation was created, the ventilation history, the amount of moisture present and equipment heat loads. The absence of obvious moisture on a rock surface does not mean that the airway is "dry"—it could mean that the evaporation rate is exceeding any water seepage rate.

The expected heat and moisture transfer in the heading was estimated using a heat and moisture simulation programme developed for mines such as Mount Isa[8]. In a development heading ventilated with an auxiliary fan and ducting system, there will be heat exchange from the rock to the air and also between the air in the duct and that in the heading. The simulation allowed for this heat transfer and also that from the progressively more recently exposed rock surfaces towards the face of the development heading.

Previous work[8] carried out where large equipment and in particular diesel powered equipment operates, has demonstrated a thermal flywheel effect for the surrounding rock surfaces. This in turn influences the total amount of heat released from the rock surfaces to the ventilating air. The values given in Table 5 for heat flow and moisture transfer illustrate the extent of the influence of rock surface wetness and equipment operation.

The high and low values for heat and moisture given in Table 5 cover the range of wet and dry bulb temperatures encountered during the study period. Full equipment operation is defined here as the 231 kW loader operating continuously at full load and the moisture resulting from the complete combustion of the hydrogen in the diesel fuel is excluded.

Table 5
Expected heat and moisture exchange

Airway condition	Rock heat flow(kW) Low	High	Water evaporated (g/s) Low	High
No equipment				
0.1 wetness	104	114	52	62
0.2 wetness	124	138	74	88
Full equipment				
0.1 wetness	38	58	94	89
0.2 wetness	64	86	127	122

HEAT AND MOISTURE REMOVED BY THE VENTILATION AIR

The heat and moisture removed from the development heading by the ventilation air was obtained from the measured air temperatures entering in the two ducts and those measured leaving in the tunnel at station D, a barometric pressure of 109 kPa and a mass flow rate of air of 32 kg/s.

Cyclic Variations

The variation in intake and return wet bulb temperatures for the first 90 hours of the study is illustrated in Figure 7. The final 30 hours of the overall study is not included because the interval between intake and return wet and dry bulb temperatures was too great when compared to the duration of the different mining operations. A certain licence has been taken in interpolating between spot measurements to give a continuous trace. Any errors incurred should balance out over the full period of the study.

Also illustrated in this figure is the calculated amount of heat removed by the ventilation air and the amount of heat added as a result of the usage of both diesel and electrical powered mining equipment. The interrelationships between the measured air temperatures and the cyclic mining operations inherent in a development heading are evident from the figure.

Heat Balance

The equipment heat sources and the ventilation air heat sink in kWh for the first 90 hours of the study are summarised in Table 6. To obtain the expected heat flow from the rock, the average heat balance of 139 kW must be reduced by 9 kW and 12 kW respectively to allow for that from the two other main contributors i.e. the explosives and the broken rock. When the resultant value of 118 kW is compared to the expected heat flow from the surrounding rock summarised in Table 5, it lies, as expected, between the predicted heat flow with no equipment operating and that with the full equipment power in use.

Moisture Increases

The measured wet and dry bulb temperatures were used to obtain the increase in moisture content of the air in the heading. The values in Table 7 summarise the values

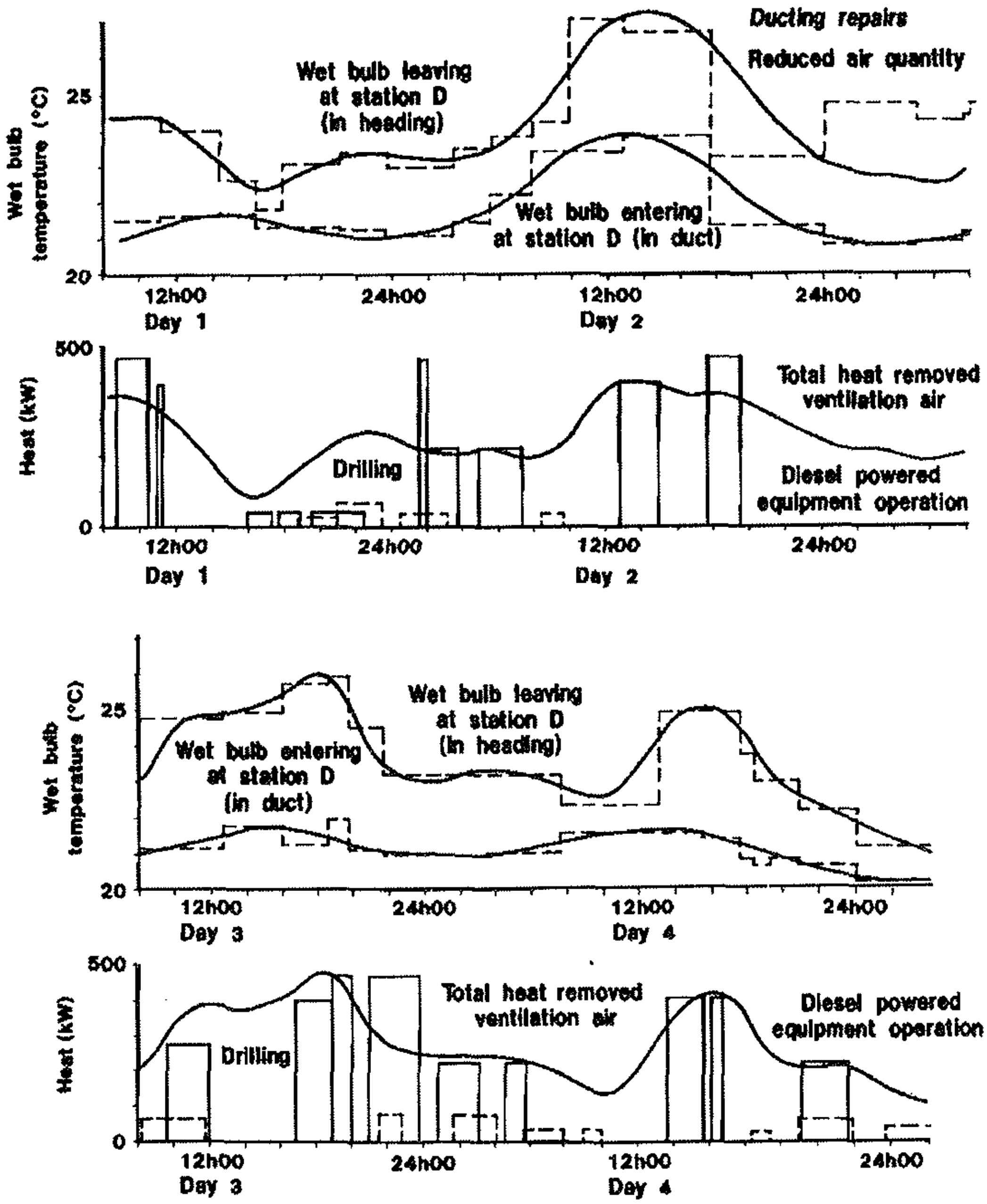

Figure 7. Variation in entering and leaving wet-bulb temperatures, heat removed by the ventilation air and equipment operation during study period

Table 6
Heat balance during the study

Time period (h)	Heat removed (kWh)	Diesel energy (kWh)	Electric energy (kWh)	Heat balance (kWh)	Heat balance (kW)
0–15	3265	1170	143	1952	130
16–39	6200	2840	149	3211	134
40–63	7175	3165	262	3748	156
64-90	6235	2270	403	3562	132
0–90	22875	9445	957	12473	139

Table 7
Measured increases in moisture content

Activity	Range(g/s)	Average (g/s)
Loader, face to A	165 to 195	140
Loader, A to C	155 to 170	120
Truck from A		125
Truck from C	95 to 105	80
Other (drilling)	50 to 95	75

obtained for the different mining operations. The first four items involved diesel powered equipment operation and the average values given exclude the water vapour resulting from complete combustion of the fuel at a rate of 0.09 g/kW.

When comparing these values with the predicted results given in Table 5, it is evident that a wetness fraction of greater than 0.1 (10 per cent of the airway perimeter is wet) and probably closer to 0.2 is appropriate. In a previous survey of a 460 m long drive with through ventilation and the appearance of being absolutely dry, there was an increase in moisture content of the air equivalent to a wetness fraction of 0.05.

It is also evident that when the diesel powered equipment is operating, the actual moisture increases are higher than expected. This is probably caused by increased localised evaporation in the vicinity of the diesel exhaust discharge. This can be accounted for in the predictive work by arbitrarily increasing the amount of water vapour resulting from combustion of the fuel by a suitable factor.

APPLICATION OF THE RESULTS

Predictive Methods

A requirement of the study was to use the results to improve the predictive methods currently employed at Mount Isa and used for ventilation and cooling planning purposes. From the measured temperatures recorded throughout the study period, the following average wet bulb temperature increases were observed:

Face conditions	– Diesel LHD operation	+ 4.8°C
	– Drilling	+ 2.0°C
	– Other work	+ 1.8°C
Air leaving at D	– Face LHD operation	+ 3.7°C
	– A to C LHD operation	+ 3.0°C
	– Trucking from A	+ 3.0°C
	– Trucking from C	+ 2.0°C
	– Drill and other work	+ 1.8°C

During the study period the average amount of heat released into the development heading as a result of the operation of diesel equipment was 105 kW. The average amount released in the face area was 31 kW (a spot heat load) with the balance distributed along the heading (a linear heat load) depending on the method of removing the rock from the heading and the storage bays.

The face heat load (spot) was increased by an average value of 20 kW to allow for the heat released when drilling and from the explosives during blasting. The heat estimated to be released from the broken rock prior to its removal from the heading (12 kW) was assumed to be uniformly distributed along the heading (linear load). The values presented in Table 8 are the predicted increases in wet bulb temperature, heat and moisture transfer in the heading using the identified heat loads and multiples of this.

Table 8
Predicted heat and moisture increases

Heat load multiplier	1	2	3	4
Face wet bulb (°C)	1.3	2.2	3.0	3.6
Wet bulb at A (°C)	1.6	2.6	3.4	4.2
Wet bulb out at D (°C)	2.2	3.1	4.0	5.0
Total heat increase (kW)	252	364	482	606
Heat flow from rock (kW)	115	90	71	58
Total moisture gain (g/s)	107	141	190	256
Water evaporation (g/s)	96	120	158	213

Comparing the study results with the predicted values given in Table 8 confirms the general criteria already used for design at Mount Isa. Using the 24 hour average heat loads in the predictive routines will result in a reasonable estimate of the conditions that may be expected in the tunnel when the diesel powered equipment is not in use. During the relatively short periods of operation of the diesel powered equipment, using heat loads which are three times the 24 hour average will indicate the maximum temperatures that may be expected in the tunnel.

It is also evident from Table 8 that there is a significant underestimate of the actual face wet bulb temperatures. This arises from the presumption in the predictive routines that the duct delivers all of the air available at its discharge to the face. The actual distance between the duct termination and the face is between 20 and 30 m and between a half and two thirds of the air available at the end of the duct actually reaches the face. This would then result in local wet bulb temperature increases at the face some 50 per cent greater than expected i.e. 3.0 x 1.5 = 4.5°C.

Heat Stress Limits

Managing hot environments at Mount Isa is achieved by the application of reduced shift length or stopping work completely if the designated conditions are exceeded. The shortened shift length is a warning that the heat stress limit where the work will be stopped is being approached. The original work[9] was based on the predicted fourth hour sweat rate (P4SR) which was developed as a result of the work undertaken between 1944 and 1946 at the National Hospital for Nervous Diseases in Queen Square, London and validated by a series of experiments between 1948 and 1953 in Singapore.

Two factors are important when considering this index, the first is that it arose from a recognition that other indices available at that time such as effective temperature did not provide a suitable measure for the range of conditions encountered and secondly, that the index was primarily based on a wartime activity where the requirement was to determine the maximum activity that could be tolerated. Mining does not fit into this category in that life does not depend on a continued high activity at limiting heat stress levels.

Despite this limitation, the index gives a good indication of what constitutes a limiting heat stress condition. It incorporates metabolic (work) rate and the influence of clothing and, for mining, a limiting heat stress or stop job condition can be defined by a P4SR of between 4.5 and 5.0. The conditions relating to a shortened or six hour shift were based on a P4SR of 3.8.

The charts produced to determine six hour or stop job conditions were wet and dry bulb temperature dependent for air velocities greater than 0.5 m/s, between 0.2 and 0.5 m/s and below 0.2 m/s. The limits on the charts equate approximately to air cooling powers which are corrected for clothing[10] of 140 and 115 W/m^2 for the six hour and stop job limits respectively. Whereas a stop job condition has no time limitation, the six hour shift is normally only awarded when working in that condition has exceeded two hours. In a development end, the time taken to clear the face of broken rock is of the order of 2 hours and the six hour shift condition would therefore provide an upper ventilation and cooling design limit.

An air cooling power of 140 W/m^2 would have reduced thermal productivity implications if applicable for the full mining cycle. Fortunately, as the study has demonstrated, the air cooling power would significantly improve when the loading cycle is completed and the diesel powered equipment is no longer used in the development heading. The normal design is therefore to provide optimum ventilation and cooling relative to thermal productivity for the mining operations where diesel powered equipment is not used and, to confirm that this design will not result in six hour shift conditions when the diesel powered equipment is in use.

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