

RESOURCE REQUIREMENT FOR DRIFTING IN UNDERGROUND MINES

by

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ABSTRACT

Design of tunnelling projects involves many machine and rock parameters, such as rock types, machine capacity, penetration rate, maintenance schedule, etc. Capital investment, resource requirement and project scheduling will all play an important role in defining the economics of the overall project.

Knowledge of the cost sensitivity of different project parameters will improve overall success of any particular drifting project provided this information is incorporated into an effective management plan.

In this thesis, a critical review of the current approaches to design drilling & blasting, mucking, transportation and ventilation aspects of drifting is given. By examining essential parameters such as tunnel area, type of rock, required number of holes, a computer model has been developed to facilitate the selection of a drill jumbo, a Load-Haul-Dump Unit, type and size of a locomotive with mine cars, estimation of airflow required at the face, calculation of number of fans needed and the selection of fan and duct types. In addition, the model incorporates a cost minimization procedure for a drilling and blasting round. Accordingly, nonlinear programming is applied to solve a nonlinear objective function subject to nonlinear equality constraints derived from the correlations among the parameters above.

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CHAPTER ONE

1. INTRODUCTION

The excavation of drifts is an important aspect of underground mining. It is not unusual for the percentage of rock broken during development, for example in a mine where sub-level caving is practiced to be as much as 25% of the total (Holmberg, 1982). If one also considers the amount broken for ventilation, exploration and transportation drifts, it can easily be understood that planning and excavation of drifts play a major part in the total economics of the mine.

Increased mechanization and distances between the shafts and the working areas, coupled with ventilation requirements with increasing mining depth, demand larger tunnel areas and longer distances for ventilation and transportation of men and machinery.

Owing to a number of factors influencing the design, loading, location and building of tunnels, various tunnelling methods have been developed. A wide range of tunnelling options is now available. Depending on the ground conditions a contractor may opt to excavate using conventional techniques with hydraulic or pneumatic drills, tunnel-boring machines, shield-machines, slurry type machines or road-headers. The reasons for choosing a particular excavation method are varied and often difficult to assess. However, when reviewing a variety of jobs

appropriate to either drill and blast or tunnel-boring excavation, several common factors emerge. When it comes to making a choice, one must consider time, cost, geology, tunnel length and available equipment.

Brockway (1982) indicated that for shorter drives in hard rock, the conventional method has a cost advantage. In all cases, a tunnel-boring machine has the cost advantage when the tunnel length reaches such a distance that the tunnel-borer's productivity comes into play and reduces the effective tunnel cost. Despite the many advantages of using tunnel-boring machines, drilling and blasting is often preferred in most hard mining applications (Nord et al., 1979).

Some computer programs have been developed for examining various sub-systems of tunnel excavations such as: fragmentation (conventional or tunnel boring), material handling, ground support and ventilation (Sharp et al., 1983). A review of the existing models show that:

- most of the programs are designed to stand alone, that is each program looks only at limited aspects of tunnel operations,

- a great majority of them do not consider optimization of resource requirements.

The objective of this research is to analyse drilling, blasting, mucking, transportation and ventilation aspects of conventional tunnelling method and to develop a single computer model that would take care of entire tunnelling operations and give optimum resource

requirements.

A computer model TUNOPR has been developed that estimates required number of shotholes, bit diameter, hole depth and diameter of cut hole in a burn cut in order to minimize the cost of drilling and blasting round; the best combination of equipment to minimize the total cost of the operation is also established. Some parts of this research have been published elsewhere (Ozan and Allen, 1987).

In Chapter 2, factors influencing drilling and blasting operation of the conventional tunnelling method are discussed. Types of cuts, their advantages and limitations are presented.

Chapter 3 contains the examination of two mucking systems; Overshot Loaders (OL) and Load-Haul-Dump Units (LHD) and their applications in tunnelling operations.

In Chapter 4, the applications of rail transportation system in tunnel excavations and its productivity are examined. Calculations of tractive effort, ideal gradient and required power are given.

In Chapter 5, methods of temporary tunnel ventilation are reviewed. The factors that determine the quantity of air required at the face are listed. Design calculations are given by assuming a uniformly distributed leakage and analysing the leaky duct mathematically.

In Chapter 6, the model TUNOPR is described. The model formulation of the objective function and constraints to minimize the cost of a round are illustrated. This

chapter also contains the description of the mathematical technique which is used for the model solution.

In Chapter 7, practical implications of the model are demonstrated by presenting parametric analysis of the Tyee Lake Hydroelectric Project of Alaska.

Finally, in Chapter 8, conclusions and suggestions for further research are presented.

CHAPTER TWO

DRILLING AND BLASTING

2.1. INTRODUCTION

In order to describe a drilling operation adequately, the type of rock and drilling equipment should be characterized. The engineer is primarily interested in those properties which affect drillability and the siting of drill holes and the direction of attack. The last may be the direction in which to drive a tunnel. Drillability factors in the strict sense, are largely properties which can be determined from the hand specimen of the rock; in a wider sense drillability can refer to the ease with which rock can be drilled and blasted over the duration of a project. Certain structures and relationships occurring in the field must also be taken into account, these include faults and folds (McGregor, 1967).

2.2. ROCK PROPERTIES

The fundamental rock properties affecting penetration rate are hardness, toughness, abrasiveness, grain size and uniformity (Tamrock, 1983; McGregor, 1967).

Hardness : This is a somewhat ambiguous term which is commonly used but not always understood (Shepherd, 1953). It can be defined as the resistance to the power of a drill to affect penetration. In a mineral species, a relative hardness may be determined from a single scratch test and assigned a position on Moh's scale of hardness which is

based on the scratch properties of ten minerals on a numerical scale 1 to 10. The hardness of a rock depends not only on the hardnesses of the individual grains of which it is comprised but also on the coherence between the particles.

Toughness : This is a property which is related to hardness but it is more difficult to determine. According to McGregor (1967) the strength of the bond of sedimentary rocks can be regarded as a measure of toughness: the hardness would be controlled by the true hardness of the minerals in the rocks. In igneous rocks, it is the texture of the rock (i.e. the shapes and relationships of the crystals) and in metamorphic rocks often the structure of the rock (i.e. the relationships between two or more different textures) which determine the toughness.

Shepherd (1953) claims that toughness is a property which primarily depends on the impact strength of the cementing material, shape and size of the minerals.

Abrasiveness : This can be regarded as the ability of the rock to wear away drill-bit materials. It is a relative quality and it is intimately connected with hardness, but it is also affected by particle shape, the tenacity of the mineral fragments and the cleavage.

The size and shape of the particles of rock influence the type of abrasion, as also does the torque and thrust applied to the bit. A relative large, sharp-edged rock fragment held firmly is likely to cause greater wear

than would fine dust, provided that the shear forces applied to the bit are insufficient to break or dislodge the fragment (McGregor, 1967).

With percussive drilling, the only rocks which need to be considered as highly abrasive are those containing :

- a. quartz, e.g. quartzite, sandstone and acid-igneous rocks,
- b. other forms of silica, e.g. flint and wood-opal,
- c. olivine, e.g. dunite and some forms of basalt, and
- d. garnet, e.g. garnetiferous gneiss.

With rotary drag-bit drilling, abrasion is a more critical factor than in percussive drilling and many other rocks must therefore be considered abrasive (Tamrock, 1983; McGregor, 1967).

Grain size and Uniformity : In general coarse even-grained rocks are drilled faster than fine-grained rocks or rocks having a very variable particle size. Geologically, grain size is a useful factor in determining the method of formation. In igneous rocks a coarse grain indicates either that the material hardened under a thick cover of overlaying strata and therefore cooled slowly allowing ample time for crystal growth, or that the liquid from which the rock was formed was exceptionally rich in certain minerals. Conversely, a fine grain indicates rapid cooling, a shallow depth or a small mass (McGregor, 1967).

2.3. MACHINE CHARACTERISTICS

There are several types of boring and drilling machines distinguished by the primary breakage motion. Percussive,

Rotary and both Rotary and Percussive (R-P). Percussive drilling can be defined as " a drilling form in which the rock is penetrated by the repeat impact of a reciprocating bit ", while rotary drilling is " a form of drilling in which rock is penetrated by the continuous rotation of the bit under a heavy feeding pressure ".

There is also Down-the-Hole drill (DTH) type, a percussive drill with the drive machine located directly behind the bit. Percussive drills have some rotation to provide indexing and rotary drills with roller bits have a percussive equivalent (Adler and Krishnan, 1983).

The percussive drills are used in mines with rock having a compressive strength exceeding 138 MPa (20000 psi). The rotary-percussive drills are used in rocks with a compressive strength between 82.7 - 165.5 MPa (12000 - 24000 psi) including some limestone and lead/zinc mines. The rotary drills are equipped with drag bits and are used in rocks with a compressive strength between 55.2 and 110.3 MPa (8000 - 16000 psi) (Roos, 1982).

Apart from compressive strength of the rock, the presence of materials such as graphite, lead and clays as well as fracturing characteristics of the mineral crystals affect the choice of drill type.

A recent development is the use of hydraulic percussion drills on jumbos. The principal advantages of the hydraulic percussion drills over air drills are their higher efficiency and their ability to concentrate more

impact energy into a given space (Roos, 1982). A hydraulic drifter is claimed to have a 50% higher penetration rate than a corresponding pneumatic drifter (Tamrock, 1983). The real content of this claim is that due to the more favourable shape of the hydraulic piston it does not stress the drill steel as much as a pneumatic piston does. A hydraulic drifter therefore can feed through a certain size of drill steel 50% more energy without reducing the life of the steel.

Tamrock (1983) points out that due to the unequal distribution of mass along the piston in a pneumatic drifter, the energy transmitted to the steel during one stroke of the piston is not equal along the drill steel. Therefore, peaks of stress, which cause fatigue in the steel without adding much to the energy transmission are formed.

Hydraulic drifters have some other advantages over pneumatic drifters such as: the difference in drill steel costs, energy costs, spares and maintenance costs, component life span, etc. (Roos, 1982; Tamrock, 1983).

2.4. ROCK CLASSIFICATION

In order to obtain the operating and the performance parameters of the drill as a function of rock type, it is necessary to classify each rock class appropriately. Since rock toughness is one of the properties of rock affecting penetration, rocks can be classified according to strength values. Deere-Miller Rock Classification divides rocks into five different classes depending on their compressive strengths (See Table 2.1). These classes are:

1. very high (where compressive strength is bigger than 220 MPa),
2. high (compressive strength ranges from 110 to 220 MPa),
3. medium (compressive strength is 55 - 110 MPa),
4. low (where compressive strength is between 28 - 55 MPa) and
5. very low (compressive strength is less than 28 MPa).

The classification of rocks for drilling helps in estimation of various operating and performance parameters for each rock class and drill type, i.e. expected penetration rate, bit life, etc. (See Table 2.2).

2.5. ROCK DRILL CLASSIFICATION

To meet the variety of conditions encountered in rock drilling, several distinct types of drills have been developed. In general, rock drills may be classified as either hand-held or mounted, with hand-held machines including jackhammers or sinkers, jackdrill or jackleg and stoper (Kurt, 1982). The mounted drills are commonly known as "drifters".

The jackhammer or sinker is used primarily for general mine utility work such as drilling anchor holes, pin holes (usually horizontal holes to fasten sheaves to side walls), blasting large boulders, and similar applications. Jackhammers are classified according to weight, and they range from 7 to 30 kg (Kurt, 1982). Because of their light weight and versatility, jackdrills are very effective in

small drifts, small tunnel headings and stoping.

2.6. DRILL STEEL EQUIPMENT

There will always be several alternatives to choose from when selecting drill steel equipment for any particular drilling situation. Factors to be considered are the hole size and length, the type of rock, the size and power of the rock drill and the past experience.

The drill steel must last under impact stresses, torque loads and bending. It is also attacked by corrosion and rust. Wear resistance must also be good. These things must be considered in the choice of the material and heat treatment method. There are several types of drill steel equipment available: the high carbon steels which also contain some amount of manganese and molybdenum, the medium and low carbon steels with chromium, nickel and manganese, fully carburized rods and integral rods which are made of carburized steels (Tamrock, 1983).

There are two basic types of drill steel equipment:

- extension steels,
- shanked rods.

Extension drill steel equipment has the following parts:

- shank adapters,
- couplings,
- extension rod,
- drill bit.

Drill bits are offered in blade and button types. Blade bits can be subdivided into cross or X-bits. The latter form is used with hole diameters of 64 mm and above. Whether button or blade bits are more suitable mainly depends on the rock properties in any given situation. However, the use of button bits seems to be increasing in a wide variety of rock formations and especially, though not always, with larger holes due to continued improvements in cemented carbides (hard metal inserts) (Atlas Copco, 1980; Tamrock, 1983).

Table 2.3a and 2.3b illustrate the expected service life of some drill steel equipment. It can be seen from the tables that expected equipment life is less in abrasive rock compared with non-abrasive or slightly abrasive rock. These figures will be used in the model development.

Table 2.1 Deere-Miller Rock Classification (after Adler and Krishnan, 1983)

Rock Strength Class	Compressive Strength (psi)	Modulus Ratio	Description
Very High	> 32,000	200-500	Quartzite Diabase Dense Basalts
High	16,000-32,000	200-500 (> 500)	Majority of igneous rocks, stronger metamorphic rocks, well cemented sandstones, hard shales, majority of limestones and dolomites
Medium	8,000-16,000	200-500 (< 200)	Shales, porous sandstones and limestones, schistose metamorphic rocks.
Low	4,000-8,000	200-500 (< 200)	Porous or low density rocks (friable sandstone, porous tuff), clayshales, rock salt
Very Low	< 4,000	< 200	Weathered or altered rocks of any lithology

Table 2.2 Performance Parameters (after Adler and Krishnan, 1983)

Rock Strength Class	Drill Performance Parameters					
	Penetration Rate (fph)				Bit Life (ft.)	
	Percussive	Rotary	Rotary-Percussive	D-H-D	Percussive or D-H-D	Rotary or R-P
Very High	20 - 40	10 - 30	15 - 30	15 - 30	500 - 1500	500 - 1500
High	30 - 50	20 - 30	15 - 50	15 - 30 (60)	500 - 1500	500 - 2000
Medium	30 - 60	30 - 70	40 - 80	-	1500 - 4000 (No D-H-D)	2000 - 5000
Low	85 - 120	70 - 150 (300 - 600)	70 - 160	-	4000 - 18,000 (No D-H-D)	5000 - 20,000
Very Low	← Auger, dig, rip, cave, etc. →					

Note: Do not use rotary when rock SiO₂ content exceeds 50%.

() extreme values

Table 2.3a. Service Life of Cemented Carbide Products (after Atlas Copco, 1980)

Product	In abrasion rock (drilled metre)	In non-abrasion or slightly abr. (drilled metre)
Integral steel		
Regrinding interval	20 - 25	150
Service life	200 - 300	750 - 850
Threaded Insert Bits		
Regrinding interval	20 - 25	150
Service life	250 - 300	900 - 1200
Threaded Button Bits		
Regrinding interval	100 - 150	
Service life	300 - 600	1200 - 1500

Table 2.3b. Service Life of Steel Products in Tunnelling (after Atlas Copco, 1980)

	<u>Drilled Metre</u>
Drifter extension rods	800 - 1200
Shank rods	600 - 800
Coupling sleeves	100% or rod life
Shank adapters	1200 - 1600

2.7. ROCK DRILLABILITY

The drillability of rocks is defined as the rate of penetration of a drill bit into the rock in question (m/min). When this is measured in the field for different rocks, the value must be comparable, i.e. the same drilling equipment should be used (Tamrock, 1983).

The different properties of rocks have a varying effect on different drilling equipment.

- Tough rocks tend to slow the rotation of a drill bit and so result in bit breakage and low penetration rate
- Abrasive rocks tend to wear the bits
- Hard rocks require powerful percussion as well as rotation
- Rock fissures and cracks are likely to lead the drill steels and cause jamming
- Soft grounds cause flushing problems.

Igneous rocks are hard but massive and relatively easy to drill with percussive drills provided that they are not too badly fissured and decomposed. They are usually too hard for rotary drilling at least in underground applications (McGregor, 1967).

Sandstones are abrasive and therefore they are not usually drilled with rotary drills. They are normally easy to drill with percussive drills.

Limestones vary greatly in hardness, from chalk to carboniferous limestones. Bit wear is not a problem. Hard limestones can be drilled with percussive drills or with

heavy rotary drills, whilst soft limestones are better drilled with rotary drills.

There are several methods developed to test drillability of rocks, e.g., the drop test, the Siever's miniature drill test, etc. (Movinkel and Johannessen, 1986).

2.8. APPLICATIONS OF DRILL JUMBOS

The drilling jumbo is a drilling unit equipped with one or more rock drills and mounted on a mechanical conveyance. Jumbos range from single drill ring units mounted on simple skids to sophisticated, computerized multiple drill units mounted on diesel engine powered carriers and equipped with automatic controls (Roos, 1982).

Individual types of jumbos are usually designed for specific purposes such as fan drilling in sublevel caving operations. Some units, such as development jumbos, can be utilized for several functions in addition to their normal applications.

2.8.1. GENERAL SELECTION CRITERIA

Since working conditions vary in underground mines, a jumbo must be selected to cope with the individual characteristics of the mine. The necessary considerations include manoeuvrability within the working place, gradients, ambient temperatures, characteristics of the rock and the task jumbo will be used for.

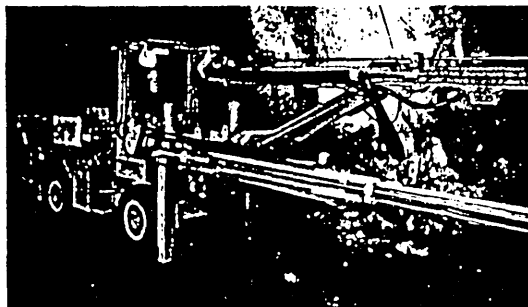
2.8.2. APPLICATIONS

Application of jumbos in mining industry can be classified as; shaft sinking, production, roof bolting and development.

The development drifts in mines have cross sections that vary from 2.1 m high by 2.1 m wide to more than 5.5 m high by 6.7 m wide (Roos, 1982). The drifts may be relatively straight and level, they may slope steeply, they may have sharp turns or they may follow a spiral course. Therefore a variety of specialized development jumbos have been developed to meet these conditions. Tamrock drifting and tunnelling jumbos with their specifications are illustrated in Table 2.4. The table illustrates the range of areas where each jumbo can work, number of booms, sizes of the jumbos, tramming speeds and gradeability. This information is used for further calculations.

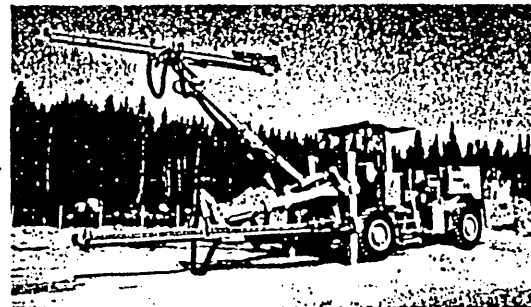
The MINIMATIC all-hydraulic drilling jumbo is designed especially for small headings where space is limited. The job-proven standard components which make up the Tamrock low-pressure hydraulics ensure trouble-free operation.

MINIMATIC HS 205 M



The PARAMATIC is the optimum solution for most drilling purposes thanks to its wide range of applications: 3-boom version for efficient tunnelling and 2-boom version for fast drifting and short-hole production drilling in mines.

PARAMATIC HS 205 L



The MAXIMATIC is ideal for large-scale tunnelling work. Efficient rock drills on three heavy-duty booms guarantee unbeatable drilling capacity.

MAXIMATIC HS 305 B

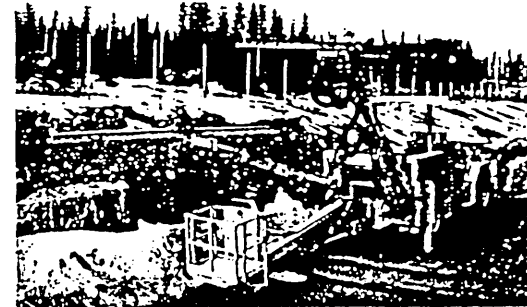


Table 2.4 Tamrock Drifting and Tunnelling Jumbo series

JUMBOS		3.3 m	5.5 m	9.4 m	11.8 m	12.3 m
		Micromatic	Monomatic	Minimatic	Paramatic	Maximatic
Coverage	m ²	4...10	5...43	8...55	14...85	16...101
Booms		1×FB 201 H	1×MR 600 1×ZR 700	2×MR 600 2×ZR 700	2 or 3×MR 600 or ZR 700 2×ZR 1000	2 or 3×ZR 1000
Chain feeds		1×KS 30 H	1×CF 145 H 1×EXT 140 H	2×CF 145 H 2×EXT 140 H	2 or 3×CF 145 H 2 or 3×EXT 140 H	2 or 3×CF 145 H 2 or 3×EXT 140 H
Drilling system		300-series	500-series	500-series	500-series	500-series
Carrier		Centre-articulated	Centre-articulated	Centre-articulated	Centre-articulated	Centre-articulated
Jumbo length	mm	7550	9050	10800	13180	13715
Jumbo width	mm	1300	1700	1940	2500	2880
Tramming height	mm	1600	1600	2400	2690	3200
Tramming speed horizontal	km/h	15.0	9.0	10.0	9.0	10.0
Tramming speed on 8° = 1:7 slope	km/h	2.0	7.0	3.7	2.0	3.0
Gradeability		20° = 1:2.8	18° = 1:3.0	20° = 1:2.8	18° = 1:3.0	15° = 1:3.7
Turning radius outside/inside	m	4.2/2.9	5.6/3.4	5.0/2.6	6.7/3.7	7.0/3.3

2.9. TYPES OF CUT

The most important part of any tunnel round is the cut. If it does not operate efficiently it will not pull to full depth and either the whole round will fail or unsatisfactory results will be obtained. In order to maximize the performance of the various types of cut, it is necessary to understand fully their mode of operation and thus their requirements.

In order to achieve the desired results when blasting in tunnels and rock cavities, it is important that the first shot known as the cut hole should initiate the break. Some of the most common types of cuts are described here.

2.9.1. ORDINARY CUT

The holes are placed symmetrically in relation to the vertical centre line of the section. The drill holes are horizontal and the angle towards the working face is large (See Figure 2.1). The symmetrical holes meet at the bottom.

Advantages of this cut can be listed as; the cut is symmetrical therefore drilling can be divided equally between two drill runners. The cut requires a small number of holes and may be applied for "easily broken" rock.

Disadvantage of this cut is that only a small advance rate is achieved unless abnormally large quantities of explosives are used.

2.9.2. PYRAMID CUT

This cut has received its name from the shape of the initial opening. Three or four wedge shaped holes are so directed that they meet at a point furthest in. The pyramid cut is

mainly employed in raises and for shaft sinking but it is not recommended for horizontal operations where a machine set up for a definite direction of the four holes cannot be easily obtained (See Figure 2.2).

Advantages:

The cut is symmetrical and easily drilled. In other respects it is similar to the ordinary cut.

Disadvantages

The cut is not suitable for horizontal headings.

2.9.3. FAN CUT

Fan shaped distribution of the initial holes exercise the effect of breaking up the rock in layers and thus overcoming the internal resistance. The shot holes are arranged in horizontal rows. These cuts are basically variations of the wedge cut and are generally employed, although not always, in narrow drivages where there is insufficient room for a wedge cut. They are particularly useful in overcoming excessive throw when an angled hole is preferred. The fan cut is also effective in locating the rockpile to one side of the roadway which may be of use when certain types of muck clearance system are being used, e.g. a side tipping bucket feeding a conveyor which is located at the side of the roadway (ICI). Figure 2.3 illustrates Fan cut.

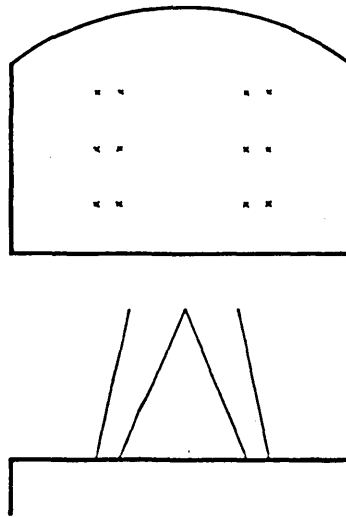


Figure 2.1 Ordinary Cut

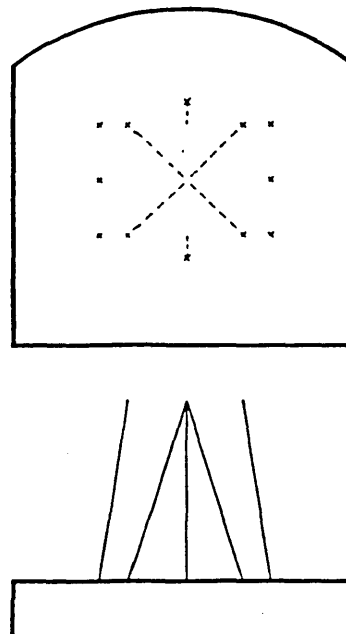


Figure 2.2 Pyramid Cut

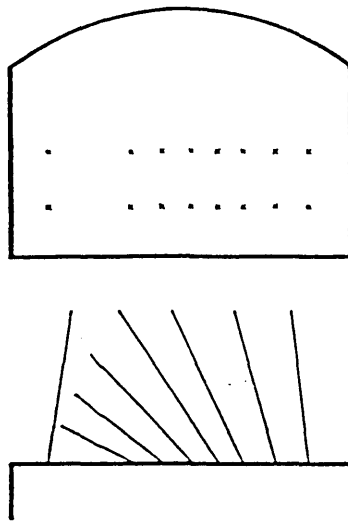


Figure 2.3 Fan Cut

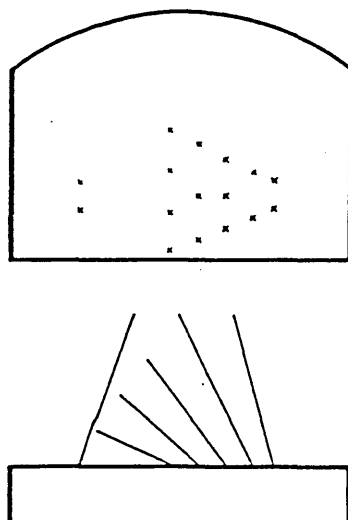


Figure 2.4 Triangle Cut

2.9.4. TRIANGLE CUT

The characteristic feature of triangle cut lies in the fact that the drill holes are arranged in zig-zag manner. In this way a larger opening is obtained as the drill holes can break out between preceding row of holes. Each vertical row of holes breaks out a layer. If the front holes do not break out to the full depth, the burn-out holes indicate the direction of break for the following row of holes since the holes are arranged in zig-zag (See Figure 2.4). The name triangle is due to distribution of the holes at the working face and the form of the initial opening (Fraenkel, 1953).

Advantages

Total drilling per m^3 of rock is less than for the fan cut owing to the reduction of the number of holes inclined to the working face. Explosive consumption is less than for the fan cut. The cut is employed to the best advantage in very hard rock or in ores.

Disadvantages

As in the case of the fan cut, drilling work is unequally divided and the cut cannot be employed in narrow headings.

2.9.5. V-CUT

For tunnel areas bigger than approximately 30 m^2 the V-cut can be used (Atlas Copco, 1980 ; Fraenkel, 1953). In this type of cut, all the holes are of the same diameter but are angled to form a wedge. Consequently the rock is blasted out at an angle to the face. The wedge angle is gradually reduced so that the holes nearest the tunnel contour are

virtually straight ahead in the direction of the tunnel. With V-cuts and Fan cuts where angled holes are drilled, the advance rate is strictly dependent upon the width of the tunnel (See Figure 2.5).

2.9.6. BURN CUT

The principle behind the burn cut is that small diameter holes are drilled with great precision around a larger hole (Holmberg, 1982). The larger empty hole serves as a free face for the smaller holes and the opening is enlarged gradually until the stoping holes take over (See Figure 2.6). The diameter of empty hole (cut hole) usually ranges from 65 to 175 mm.

The obvious advantages of using burn cut are that no attention has to be paid to the tunnel width, the cut is much easier to drill with machine since there is no need to change the angle of the boom (Holmberg, 1982; Fraenkel, 1953; ICI).

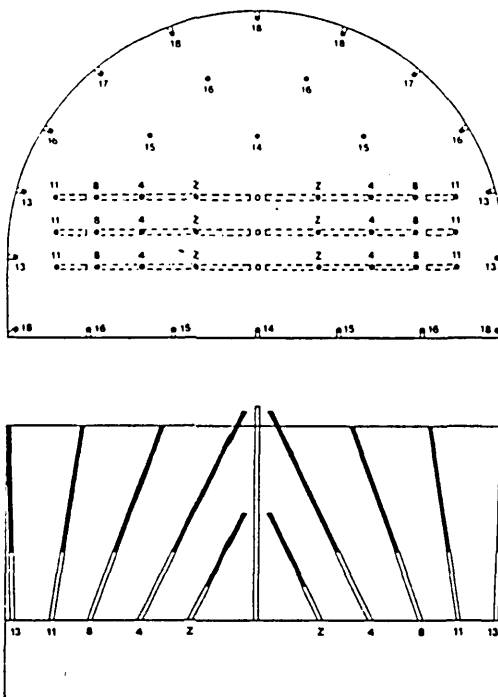


Figure 2.5 V-Cut

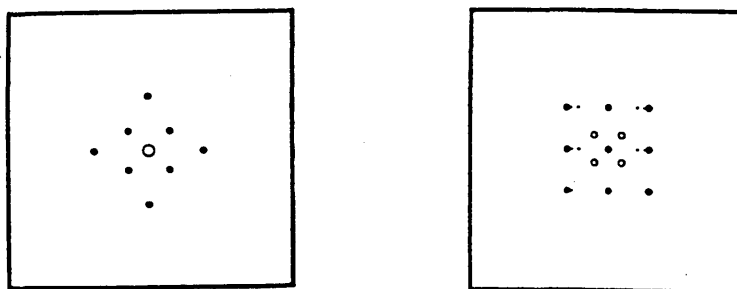


Figure 2.6 Burn Cut

2.10. NUMBER OF SHOTHOLES REQUIRED

Number of shotholes needed to distribute the charge mainly depends on:

- type of rock
- type of explosive
- hole size
- smooth blasting requirements
- type of cut
- blasting vibration restrictions (Tamrock, 1983)
- desired material size after blasting

The total number of shotholes in the face area is determined from various empirical methods relying heavily upon the past experience (Kurt, 1982). Although Kurt mentions in his paper that number of holes is a function of the rock type, bit size, the tunnel, etc., he makes a simple assumption to calculate number of holes. According to his assumption, one hole accounts for 0.6 to 0.8 m² of face area whilst Some authors find relationship among number of holes, bit diameter and tunnel area (Atlas Copco, 1980; Langefors and Kihlstrom, 1963). Figure 2.7 illustrates the relationship between tunnel area and number of holes required whereas Figure 2.8 is a graph of tunnel area and number of holes examining bit diameter and rock compressive strength.

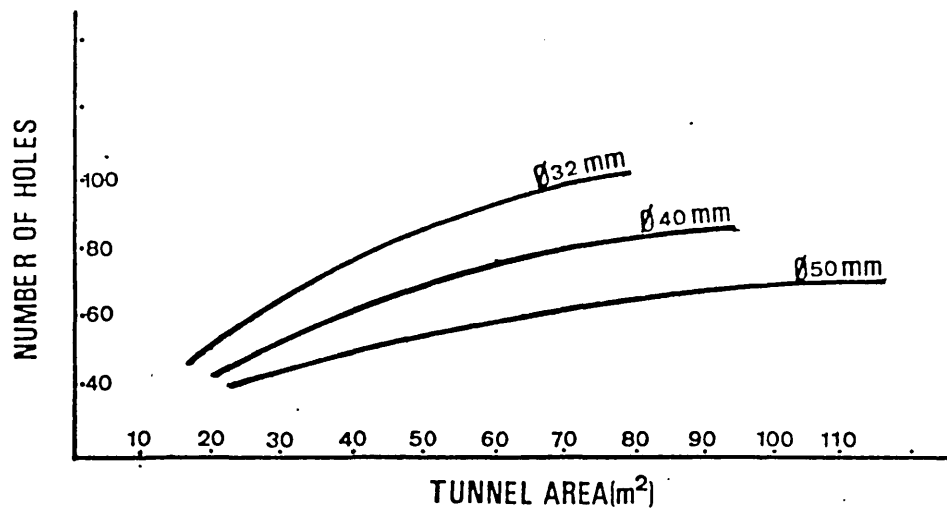


Figure 2.7 Relationship between tunnel area and number of holes (after Atlas Copco, 1980)

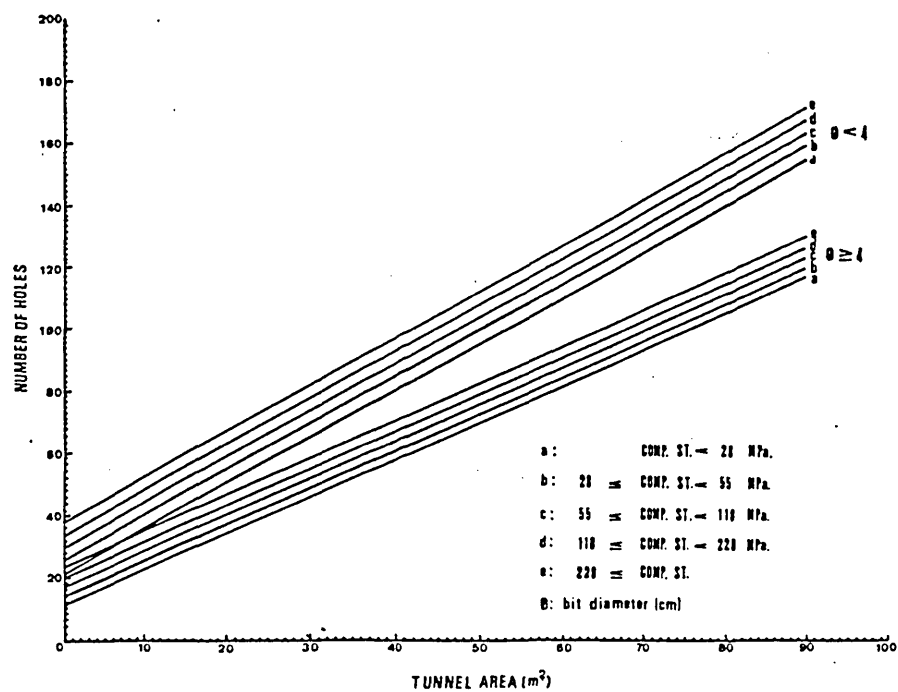


Figure 2.8 Graph of number of holes vs tunnel area

Shotholes are generally drilled with diameters of 26 to 40 mm (Szechy, 1967 ; Tamrock, 1983). Hole sizes from 27 mm to 38 mm are often considered to belong to the small-hole, while 40 to 64 mm hole sizes belong to the large-hole.

2.11. CALCULATION OF HOLE DEPTH AND ADVANCE RATE

Shotholes are generally drilled to a depth of 1.2 to 3.5 m and it is not worthwhile to drill holes deeper than 2.5 m and as the drilling time of a 3.6 m deep hole is equal to that of two holes of a depth of 2.4 m (Szechy, 1967). Szechy states that holes should never exceed the width of the tunnel, and the round pulled by the holes is generally shorter by about 30 - 40 cm than the hole depth (supported by Mayo, 1981). He also gives an empirical equation which is:

$$H = (A - 1) \cdot \alpha + 0.5 \quad (2.1)$$

where

H = the hole depth (m)

A = cross sectional area of the tunnel (m^2)

α = a coefficient the value of which depends on the diameter (d) of the shothole varying as follows:

$d = 60 - 80$ mm $\alpha = 0.20$

$d = 45 - 60$ mm $\alpha = 0.18$

$d = 20 - 30$ mm $\alpha = 0.10$

If it is assumed to drive a tunnel with a cross sectional area of 30 m^2 and the bit diameter used to be 45 mm, estimated value for hole depth can be found as 5.72 m from Eq.2.1. This value is bigger than the range suggested by

Szechy (1967). In many cases, shorter rounds give a better working cycle and rapid advance per day (Langefors and Kihlstrom, 1963).

According to Holmberg (1982), the advance rate is restricted by the diameter of the cut hole (empty hole) and the deviation for the smaller diameter holes in a burn cut. Drifting becomes very expensive if the advance rate is less than 95% of the hole depth. Holmberg (1982) finds a relationship between the hole depth and the diameter of the cut hole that is :

$$H = 0.15 + 34.1 \Phi - 39.4 \Phi^2 \quad (2.2)$$

where

H = hole depth (m)

Φ = diameter of the cut hole (m)

And the expected advance rate I (m/round) is :

$$I = 0.95 H \quad (2.3)$$

Equations 2.2 and 2.3 are valid only when hole deviation does not exceeding 2% (Holmberg, 1982).

In general diameter of empty hole ranges from 64 to 120 mm (Atlas Copco, 1980). By using the Equation 2.2 for the lower and upper limit values of the cut hole, hole depth ranges from 2.17 to 3.67 m which is in the range suggested by Szechy (1967).

Driving with maximum advances often involves over estimation of the results. It is essential to know not only the possibilities but also practical limitations and their causes. A blasting round with a drilling pattern planned to give an advance of 5.4 m that actually only gives

approximately 4.2 m, is quite unsuccessful and probably not as good as the one designed for 3.3 m (Langefors and Kihlstrom, 1963).

2.12. DRILLING AND BLASTING PATTERN

Shotholes are generally drilled according to a predetermined drilling pattern and fired in a predetermined sequence of rounds. The drilling patterns to be used depends on the depth of the holes, the stratification of the the rock and its behaviour (Szechy, 1967). The sequence of firing rounds varies according to the nature of the rock, operational technique and the position of the headings. It is practical to start firing with the cut holes placed in the centre of the tunnel area and to finish with the outer holes as in this case all the subsequent shots will have at least two free faces (Hemphill, 1981; Langefors and Kihlstrom, 1963).

The overall effect of blasting and especially rock fragmentation is also beneficially affected by the application of delay firing, controlled by the length of the delay periods. The general range is from 50 to 500 ms, with the larger delays used for more difficult breakage geometry.

CHAPTER THREE

MUCKING SYSTEMS

3.1. INTRODUCTION

The cross sectional area of small tunnels in mines, power stations, water supply systems, sewage and oil pipelines may often be determined simply by the working space required during construction. The most economical tunnel cross section is then chosen primarily with regard to two factors, namely the size of the job, which means generally the length of the tunnel and the time available for the work.

Where time has minor importance, short and small size tunnels (1.8-2.0 m²) can be constructed most cheaply by using simple equipment. Loading should be performed by hand, using wheel barrows or side-dumping cars (Wanhainen, 1953). The need for mechanized loading and transportation becomes more apparent as the length and the size of the tunnel increases.

There are several types of mucking systems used in tunnelling, such as overshot loaders either track or trackless, continuous gathering arm loaders, crawler mounted or rubber tyred front-end loaders and Load-Haul-Dump Units (LHD).

In this chapter two types of mucking systems are examined by considering the tunnel face area, type of muck and job conditions. These are overshot loaders and LHD vehicles.

3.2. OVERSHOT LOADERS

Overshot loaders are loading machines which fill a front mounted bucket by crowding and bucket lifting, passing the loaded bucket over the machine and throwing the load into a haulage unit into the rear. The load is retained in the bucket as the load mass is accelerated in its path over the machine. At the point of discharge, the bucket is stopped suddenly and the load leaves the bucket (Sundeen and Wenberg, 1982). Figure 3.1 shows a typical overshot loader of this class.

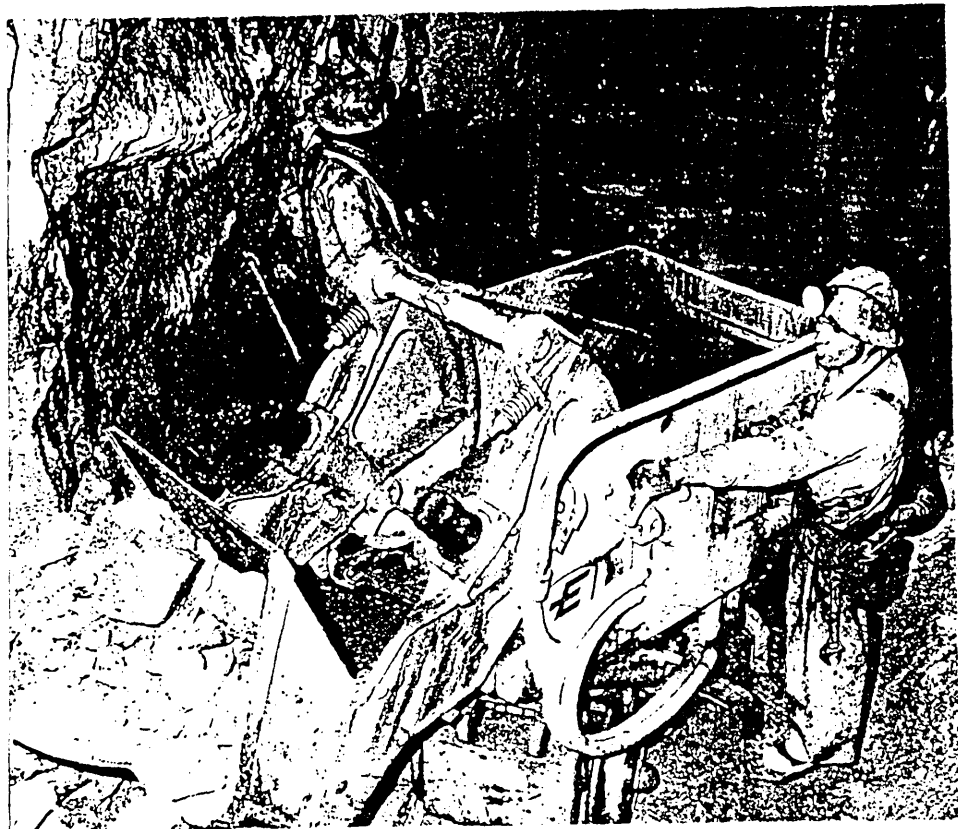


Figure 3.1. A typical Eimco overshot loader.

Overshot loaders were developed for loading in narrow drift headings and drawpoints, where it is impossible to place or pass a haulage unit alongside the loader. Haulage

access is to the rear of all overshot loaders. The early machines were rail mounted, air powered and small loading capacities (Wanhainen, 1953). Today, the same basic design configuration exists, and all successful overshot loaders still use compressed air. Bucket sizes range from 0.14 to 0.59 m³ (Sundeen and Wenberg, 1982). The most commonly used machines are rail-mounted. Crawler versions, however, are popular for special applications, especially in shaft sinking.

3.2.1. CONDITIONS AFFECTING APPLICATION OF OVERSHOT LOADERS

Rail-mounted loaders are used where soft ground requires rail support or where the long haul is accomplished with rail. This situation usually occurs in civil tunnels or in mine drifts where transportation will continue to utilize rail haulage.

Crawler mounted loaders are used where the roadway surface is rough and lateral travel beyond the rail-bound machine is required. Figure 3.2 illustrates a typical crawler mounted overshot loader.

The largest rail-bound loaders can clean roadways as wide as 4.0 m from a single track. Wider headings require multiple trackage. Headroom requirements depend on the path described by the bucket as it passes over the machine.

The largest heading that can be driven by overshot loaders from a single set of rails is approximately 4 m wide and a minimum of 3.5 m high above the rail whereas the smallest rail-bound loaders can work efficiently in headings

as small as 1.8 m wide by 1.8 m high above the rail (Sundeen and Wenberg, 1982). It means that tunnel area, in which a rail-bound overshot loader can be applied is in the range 3.24 m² to 14.0 m².

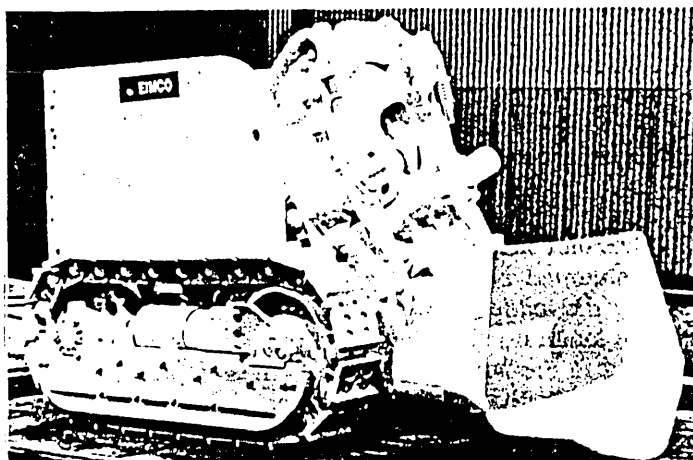


Figure 3.2. A typical crawler mounted overshot loader, Eimco 630.

Rail-bound machines are restricted to relatively flat gradients, usually not in excess of 2%. Crawler mounted loader can work on gradients as steep as 20%. Most overshot loaders are powered by compressed air and these machines do not require ventilation especially for normal operation. The only ventilation needed is for the operator.

3.2.2. PRODUCTIVITY OF OVERSHOT LOADERS

Productivity varies greatly depending on the physical limitations in the working areas. Under optimum conditions, small loaders in a typical rail heading load out from 25-30 m³/hr with using a 0.14 m³ bucket. Large loaders with upto 0.59 m³ buckets load as much as 110 to 120 m³/hr (Sundeen

and Wenberg, 1982). Theoretical production rates under optimum conditions can be calculated using a cycle time of 20 sec. per bucket load (180 bucket loads per 60 min) and multiplying this by the theoretical cubic capacity of the loader. Their productivity depends on the type of material being mucked (fine, coarse or chunks), working time, degree of automation and availability of power.

Wanhainen (1953) listed advantages and disadvantages of using rail-bound loaders compared with other types of loaders.

Advantages:

- they can be used for both finely and coarsely broken material,
- they clean up the tunnel satisfactorily and might be used to some extent for digging out insufficiently loosened rock,
- these loaders are of compact design, therefore occupying little space,
- easy to move,
- they can load on curves,
- low initial cost.

Disadvantages:

- relatively limited momentary capacity,
- require continuous track laying,
- the filling of the bucket takes a relatively longer time,
- since the mucking machine advances during the digging movement, there is also a risk of derailment,

- rather complicated design. The machines are not entirely reliable in operation, although different makes vary considerably in this respect.

3.3. LOAD-HAUL-DUMP UNITS (LHD)

In the early 1950s, underground miners and tunnelling contractors were experimenting with ways to supplement or eliminate track-bound loading and hauling equipment. Diesel-powered crawler and rubber-tyred front-end loaders and trucks designed for use on the surface were being modified and tried with limited encouraging success.

By the mid 1960s, LHD vehicles were firmly established as a fundamental part of what has become known as the "trackless" mining concept, used in many mines and in many countries around the world. The flexibility, mobility and versatility of these units have given the industry a useful tool and have added new dimensions to mine development and production (Stevens and Acuna, 1982).

LHD vehicles combine certain characteristics of conventional front-end loaders and dump trucks specially designed for material handling in underground mining and tunnelling operations. The design purpose is to provide one vehicle with one man, with the vehicle loading itself, hauling the load over level or inclined haulage ways, and dumping the load. Figure 3.3 illustrates an LHD vehicle.

In general, the LHD vehicles available from most of the manufacturers share the same basic features. Table 3.1 lists some popular LHD sizes of one US manufacturer,

together with the principle dimensions that influence the choice of a loader (Section 3.3.2). This will form the basis of applications of different LHDs in the model development.

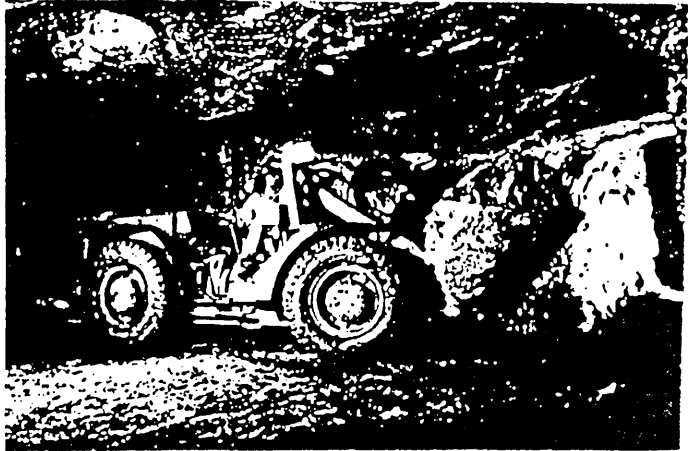


Figure 3.3 Load-Haul-Dump Unit

Table 3.1. LHD vehicle capacities and sizes (after Stevens and Acuna, 1982)

Rated Payload Capacities				Operator Height		Overall Width		Approx. Turn Radius	
m ³	cu yd	t	st	mm	in.	mm	in.	m	ft
0.8	1.0	1.4	1.5	1830	72	1220	48	2.4	8.0
1.5	2.0	2.7	3.0	1930	76	1550	61	3.7	12.1
2.7	3.5	4.8	5.3	1730	68	1830	72	4.1	13.3
3.8	5.0 A	6.8	7.5	2110	83	2440	96	4.7	15.5
3.8	5.0 B	6.8	7.5	2140	84	2140	84	4.8	15.9
3.8	5.0 D	6.8	7.5	1980	78	2510	99	5.0	16.4
6.1	8.0	10.9	12.0	2260	89	2490	98	6.1	19.9
9.9	13.0	17.7	19.5	2540	100	3050	120	5.8	19.1

There are other intermediate capacities and dimensional variations by other manufacturers (Caterpillar, 1983).

3.3.1. ECONOMICS OF LHD

In mine production, the basic measure of the economics of mucking is the cost per ton, or alternatively, the cost per unit volume to move from one place to another. In development, time required to move a specific amount of muck is the usual measurement of productivity. In both cases, total distance of mucking is an important factor.

Another important economic factor in trackless mining concerns the adequate support facilities and experienced personnel. Proper preventive and proper repair maintenance must be provided at a moderately sophisticated level (Spencer and Gordon, 1986). The flexibility, mobility and versatility of LHD vehicles may have a strong impact on the distance to which the trackless system remains viable (Stevens and Acuna, 1982).

These are important design features inherent in LHD units, expanding the scope of their usefulness in the overall mining method. Table 3.2 lists critical areas in improving LHD efficiencies and operation costs, their effects and solutions.

As it can be seen from Table 3.2 that the most significant area affecting LHD costs and efficiencies is in the loading cycle and the two most critical areas are the bucket capacities, conditions and loading.

Table 3.2 Problem areas that affect LHD efficiencies
(after Spencer and Gordon, 1986)

<u>Problem area</u>	<u>Effect</u>	<u>Solution</u>
a.Underfilling of bucket	Reduced tonnage	Operator training
b.Under capacity of bucket	Reduced tonnage	Check bucket capacity
c.Poor tyre life	High operating costs	Operator training check roadways
d.Damaged tyres	Excessive tyre costs	Prevent bad loading practices in water logged areas.
e.Worn buckets and multiple pass loading	Reduced tyre life Reduced tonnage Increased fuel consumption Heat generation and ventilation pollution Increased maintenance costs	Replace bucket Operator training

Often LHDs are supplied with capacities that may not be suitable for the applications of the mines (Spencer and Gordon, 1986). For example, an LHD which has a tramming capacity of 4500 kg is employed in a mine where in-situ rock density is 2.7 and density of broken bulk material is 1.7 t/m³. Therefore, this LHD could have a heaped bucket size requirement of 2.7 m³.

$$(4.5 / 1.7 = 2.7 \text{ m}^3)$$

However, if the mine was not sure of the density of broken material another LHD could have been supplied with a bucket capacity of 2 m³. This would have led to an undercapacity tonnage of 3.4 t in comparison with 4.5 t (a

loss of 24% per load).

Knowing the operating conditions of LHD, one can easily calculate the loss due to an undersized bucket. Spencer and Gordon (1986) examined the factors that affect LHD efficiencies during loading and suggested some solutions to improve LHD efficiencies and operational costs.

3.3.2. SELECTION OF LHD VEHICLES

In selecting an LHD vehicle, the first consideration is whether or not the unit will fit into the working space. Reasonable and regulated clearances for the operator and vehicle must be employed, including auxiliary installations such as air, water and ventilation lines. These conditions have a direct influence on the tramming speeds that are attainable. Stevens and Acuna (1982) stated that in general, 0.9 to 1.2 m is considered to be the minimum clearance between the vehicle and haulageway walls and 0.5 m the minimum clearance between the operator's helmet and the haulageway back. At least one known regulation requires a minimum of 1.5 m between the vehicle and the walls. Consequently, the mine opening should be wide and high enough to accommodate the vehicle with the operator.

It is a common practice to select the largest capacity vehicle that could be accommodated by the mine opening dimensional constraints. Largely the acquisition and operating costs do not increase necessarily in direct proportion to increased vehicle capacity. Although the acquisition cost of a bigger size LHD is higher than that of

a small size one, the productivity of bigger size of LHD is much greater than a vehicle with small capacity. Therefore in tunnelling operations, one of the criterions that affect the choice of LHD vehicles, is the area of the tunnel. If several sizes of LHDs could be used in a heading, usually the one which has the biggest capacity among them will be the most economical.

The following demonstrates how to find out the applications of LHDs that have, for example, payload capacities of 0.8, 1.5, 3.8, 6.1 and 9.9 m³ (See Table 3.1). If it is assumed that the minimum clearance between the vehicle and haulageway back is 0.5 m and the minimum clearance between the walls and the vehicle is 1.5 m, the sizes of the smallest tunnel areas where the listed LHDs can be used are determined to be : (See Table 3.3)

Table 3.3 Tunnel areas where different LHDs can be used

Rated payload capacities		Operator height	Overall width	Tunnel area
m ³	ton	m	m	m ²
0.8	1.4	1.83	1.22	9.83
1.5	2.7	1.93	1.55	11.00
3.8	6.8	2.11	2.14	14.09
6.1	10.9	2.26	2.40	15.15
9.9	17.7	2.54	3.05	18.40

3.3.3. PRODUCTION ESTIMATION FOR LHD VEHICLES

Accurate estimation of LHD productivity depends on the understanding and application of a number of variable factors that are known as the "job conditions".

The job conditions are a group of factors that affect both the productivity of the vehicles and the ultimate cost of moving materials. Stevens and Acuna (1982) reviewed the more significant job conditions and classified them as excellent, severe: average conditions assume that various factors offset each other to approach an average between the excellent and severe conditions.

Excellent conditions:

Under these conditions, the LHD vehicle carries ample lighting to illuminate the floor, roof and the walls. The loading area of the floor is nearly level, kept free of spillage and well drained to provide good traction. The muck is well blasted and free of large sizes that require secondary blasting. There is an adequate flow of ventilating air to the face, providing a comfortable working environment for a well trained operator.

The main haulageways are sufficiently wide and high. They are maintained to reasonable smoothness and are kept free of spillage. Tight and sharp turns are minimized.

Under these conditions, a spacious dumping point consistently handles the full production of LHD vehicles that are in operation.

Having defined the excellent conditions , according to availability of listed working conditions, job conditions can be classified as excellent, average or severe.

Severe conditions:

Under these conditions, a minimum of vehicle lights leave the operator driving the vehicle in a restricted tunnel of light, thereby inviting collision with the walls. The floor may have adverse grade, be slippery or be littered with spillage. The muck is poorly fragmented, containing large boulders that must be worked out and taken away for secondary blasting. The main haulageways are of minimum width and height, slippery or soft with areas of deep standing water. There may be sharp and tight turns to be manoeuvred.

Under these conditions, there is only a minimum of manoeuvring room at the dumping point. Restrictions to dumping may prevent a clean dump cycle.

As the job conditions approach to excellent, the greater will be the productivity and the lower will be the cost.

The variables in the job conditions could be applied to the estimation of productivity with the formula given below :

$$P_r = \frac{(t_o \times b)}{(t_f \times t_v)} \quad (3.1)$$

where

P_r = rate of production (t/hr)

t_o = operating time (min/hr)

b = bucket payload (ton)

t_f = fixed time in minutes per cycle (min)

t_v = total variable time in minutes required to travel between the loading and the dumping points (min)

Operating minutes per hour :

In Equation 3.1, t_o reduces an available 60-min hour to a lower number of operating minutes to account for various unavoidable delays to production. These delays might cover such things as moving from one working place to another during the shift, periodically inspecting the vehicle, making minor adjustments, stopping for fuelling or stopping for operator comfort.

Bucket payload carried per cycle :

Since material weight per cubic measure varies widely in mining and tunnelling, the true basis for rated capacity is its tramming or payload capacity (Stevens and Acuna, 1982). The bucket volume should be reviewed in favour of matching the rated tramming capacity to the known loose weight of the material to be moved.

Fixed cycle time :

In the production equation, t_f is the portion of the cycle time spent in loading and dumping the bucket, including the direction changes and other fixed cycle time for different job conditions.

In the production formula, t_v is the total variable time between the loading and the dumping points. Where there is only one haulage segment in the complete cycle,

loaded and empty speeds are approximately the same.

$$t_v = \frac{2 S}{v} \quad (3.2)$$

where

t_v = total variable time (min)

S = one way hauling distance (m)

v = average speed (m/min)

Table 3.4 Operational and fixed time for LHD (after Stevens and Acuna, 1982)

<u>Job conditions</u>	<u>Op.time (min/hr)</u>	<u>Fixed time (min/cycle)</u>
Excellent	55	0.80
Average	50	1.10
Severe	45	1.40

CHAPTER FOUR

RAIL TRANSPORTATION

4.1. INTRODUCTION

Underground transport of mineral, men and material is a critical activity in the mining process. Over the past two decades substantial investment of money and skills have created high potential capacities in both face and development workings. The outputs of these high production units are particularly vulnerable to any failures clearing the product or providing input resources, stemming from the transport systems (Watt, 1982).

Watt (1982) points out that the vulnerability is exacerbated by the progressively increasing distance from the shaft or drift to the typical face. This distance already averages over 3.9 km and by the middle of the present decade, a third of National Coal Board's faces will be in excess of 5 km from the shaft.

Underground transport is seen to be demanding of equipment and manpower and to comprise activities of considerable magnitude particularly in the fields of mineral transportation, manriding and the conveyance of face equipment both in and out of the mine.

In this Chapter, rail haulage is examined and its applications and specifications are listed.

4.2. RAIL HAULAGE

The principal application of rail haulage in underground mining is the movement of ore or coal from the production areas or material from the development ends or tunnel faces to a processing plant or to a dumping area. In some cases, such production haulage may end at a shaft tip or an incline depending on the type of mine.

The basic reasons for considering rail haulage are high tonnage production per shift and long distances. The choice between rail haulage and other systems is not easy and requires a complete study of many factors: tonnage, distance, grade, product size, number of production areas, ore reserve, capital cost, operating and maintenance costs (Buckeridge et al., 1982; Athaus, 1985; Watt, 1982).

4.2.1. MAIN-LINE HAULAGE

Projecting a main-line rail haulage system for comparison with other systems requires a choice of many variables which are interrelated. The project must start with consideration of the mining system, production rate, distances, loading and dumping conditions.

Athaus (1985) compared rail haulage with belt conveyor system in terms of economical use and concluded as:

- rail haulage preferably used in extended roadway networks with more than one main level and with variable lengths, directions, capacity and functional demands of the rail road system over time.

- rail haulage is exclusively operated in mines, eg. coal mines where different coal types are produced or in

anthracite mines in order to minimize degradation.

- track haulage is restricted to horizontal drivage of roads by floor stability and requirement of curves with sufficient radii.

- belt conveyor systems operated in sequence, are predominant in mines with one main level characterized by a high degree of regional concentration in seam roadways and inclines. Compared to rail haulage the capital costs of belt conveyors are high but on the other hand labour costs are low.

The advantages of rail main-line haulage are high tonnage capacity, flexibility, reliability, safety, low operational and maintenance costs, and dual usage of the systems (eg., supply haulage). The disadvantages are high capital investment, high installation costs and requirements for more maintenance facilities.

4.2.2. SUPPLY HAULAGE

Supply or materials haulage is a basic requirement of any mine, but the type and volume of supplies vary with the type of mineral being mined, the mining methods and the size of the mine. A mine development for rail haulage gains supply haulage through the addition of a secondary track system to extend the main-line closer to the production areas.

When belt conveyors are chosen for production haulage, a rail supply haulage system is often installed rather than using rubber-tyred vehicles. While haulage grades of more than 6% and the movement beyond the end of

rail system favour rubber-tyred vehicles, longer haulage distances and difficulty in maintaining a smooth roadway favour rail (Buckeridge et al., 1982).

4.2.3. PERSONNEL MOVEMENT

Owing to increasing distances between the shafts and working areas (average 6.2 km in W. Germany (Athaus, 1985) and 3.8 km in the UK (Watt, 1982)) and the negative impact on air conditions, when moving to deeper areas, necessitate economical manriding systems. The efficient and comfortable movement of the miner to and from his work area has a significant effect on his daily output. Minimizing travelling time also becomes more and more important as hourly wages increase.

Personnel are generally moved over a rail system by one of the following; a specialized personnel carrier to which a certain number of men who work in a certain area are assigned; a combination vehicle, used as a light supply/utility locomotive during a working shift; and man-trip cars towed by a supply locomotive. The first and second systems are used in coal mines where long distances make speed and comfort important. The third system is more common in metal mines (Buckeridge et al., 1982).

4.2.4. EQUIPMENT MOVEMENT

With the advancement of mining technology many mining machines and their sub components have grown in size. This trend needs careful consideration for adding specialized carriers to handle the movement of larger equipment (raise borers, ventilation fans, drill jumbos, etc.). The

selection of the weight, speed and operating characteristics of supply locomotives used to move equipment is important.

4.3. APPLICATION OF LOCOMOTIVES

4.3.1. TYPE OF LOCOMOTIVE

The first basic decision in selecting a locomotive concerns its intended use, main-line haulage or supply haulage. Production haulage requires heavier locomotives, higher tractive effort and the probable continuous duty cycle requirements.

There are four types of locomotives used underground;

- diesel powered
- trolley locos with overhaul power supply
- battery locos with lead-acid batteries (or nickel-iron batteries which have been used occasionally)
- combined trolley-battery engines (Athaus, 1985; Green, 1953).

Today the choice between diesel and electric locomotives is governed to a large degree by the requirements of state and federal mine laws, ventilation requirements (See Chapter 5), and the acceptance of diesel locomotives by the miners. If a diesel locomotive is selected, initial capital cost will be lower than that of the electric type, but the operation and maintenance costs will be higher. Often the service life of electric locomotives can be more than 40 years. It is doubtful that a diesel locomotive will be able to match that service life (Buckeridge et al., 1982).

If an electric locomotive is selected, a choice must be made from three types; trolley, battery and a combination trolley-battery. A battery locomotive offers the greatest flexibility. It is the only one that can match the diesel by eliminating the cost of trolley-wire installation. Figure 4.1, 4.2 and 4.3 illustrate specifications of battery, trolley and diesel tunnelling locomotives.

The first step in choosing an appropriate type of locomotive is to examine the factors that govern the operation of the haulage system. These include size, disposition, length, gradient and stability of roads, arrangement of pit bottom and duty in terms of time, load and mine car capacity (Green , 1953). Following working principles will determine the choice of locomotive:

- a. a road subject to movement of either floor or roof is unsuitable for trolley locomotives.

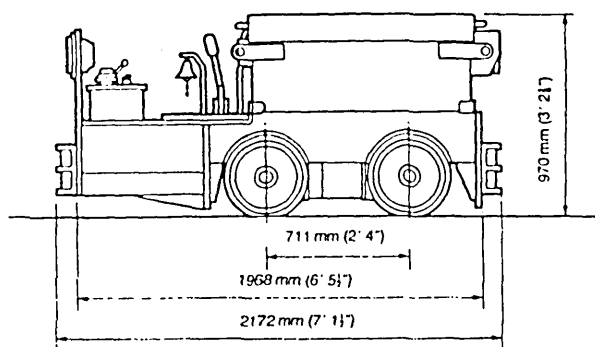
- b. a road of small size may require enlarging to provide necessary clearance for trolley locomotives, and the cost of this may be prohibitive.

- c. a road with severe gradients (steeper than 1 in 15) is unsuitable for battery locomotives (Lunnon, 1976). It may be suitable for trolley locomotives.

- d. deep hot mines may be unsuitable for diesel locomotives due to the heat produced by the diesel engine.

- e. in a mine where diesel fumes cannot be tolerated, diesel locomotives may be undesirable.

- f. trolley locomotives are not appropriate in very wet mines.



SPECIFICATION

Motor

Clayton 7hp 1 hour rated on 48 volts

Controller

Cam contactor type giving four notches of control with dead man's handle arranged to cut traction

Resistance

Unbreakable alloy strip type

Battery

Traction batteries between 9 and 12.6 kwh capacity at the 6 hour rated can be provided

Battery Container

Ventilated and weather-proof fitted with heavy duty Colton plug and socket

Transmission

Totally enclosed, axle mounted wormdrive gearboxes

Brakes

Quick acting lever brake

Wheels and Axles

Heat treated, 355 mm (14") dia. wheels. Alloy steel axles

Axleboxes

Roller bearing type

Couplers

link and pin standard

Frame

All steel welded construction

Warning device

Brass bell

Optional Extras

Electric horn
Easily removable wheels
Thyristor controller
Convertible gauge
Battery box rollers

Rail gauge

up to 457 mm (18") fixed
609 mm (24") fixed
457/609 mm (18/24")
variable

Overall Width

686 mm (27")
762 mm (30")
762 mm (30")

Approval of locomotive for a specific duty should be obtained from Clayton Equipment

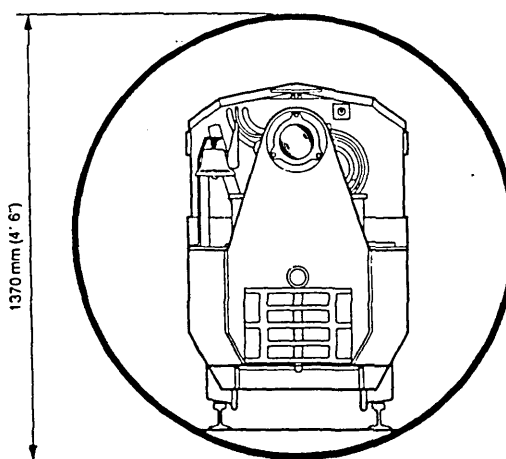
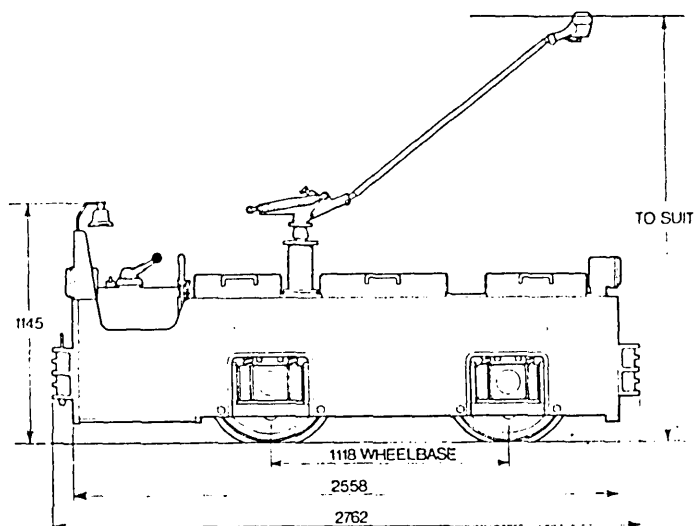


Figure 4.1. Specifications of a 1.75 ton Battery Powered Tunnelling Locomotive (Clayton Equipment)



SPECIFICATION

Motor
Clayton 20 hp 1 hour rated on 250 volts D.C. split field type

Controller
Cam contactor type giving 6 notches of control with deadman's handle arranged to cut traction

Resistance
Edge wound alloy strip type

Transmission
Totally enclosed axle-mounted heavy duty, traction type worm gearboxes. Cardan shaft drive to leading axle

Brakes
Quick acting lever brake operating Ferodo shoes on transmission

Wheels and Axles
Wear-resistant 560 mm dia. wheels pressed on to alloy steel axles

Axleboxes
Roller bearing type fitted with grease seals

Coupler
Link and pin standard

Suspension
Coil springs in pockets in axleboxes

Torque Arms
Steel link fitted with rubber bushes anchored to frame

Frame
Robust all-welded steel frame

Warning Device
Brass bell

Headlamps
Sealed beam type

Current Collector
Slipper type

Exterior Finish
Safety yellow

Optional Extras:
Easily removable wheels
Convertible gauge
Dynamic braking
Pantograph current collector
Electric horn
Automatic couplers
Sanding equipment

Rail Gauge **Overall Width**
up to 762 mm (30") 1120 mm (3'8")
 914 mm (36") 1120 mm (3'8")

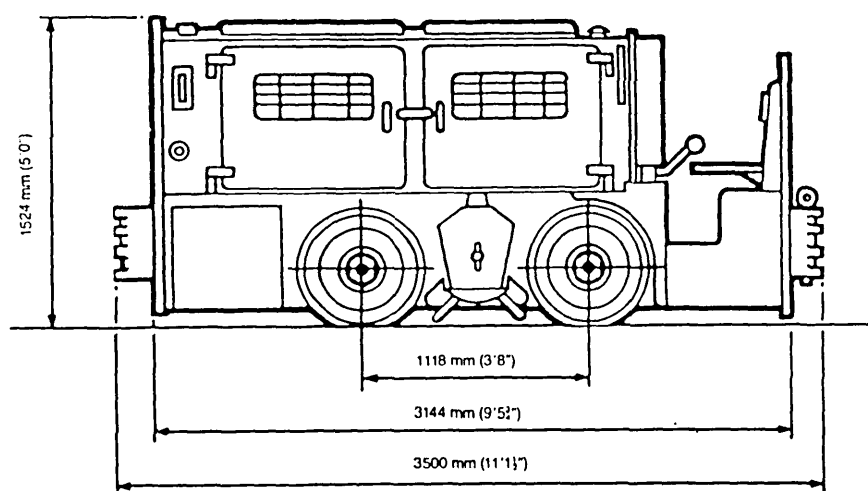
* This locomotive has outside wheels.

HAULAGE CAPACITY CHART

Speed km/h	Tractive Effort kg	Loads hauled in tonnes on straight track					
		Level	1 in 200	1 in 150	1 in 100	1 in 50	1 in 30
0-6.5	750	80	50	45	36	23	15
8	440	46	28	25	20	12	7
9	340	35	21	19	15	9	5
10	270	27	16	14	11	6	3
11	210	20	12	10	8	4	2
12	160	15	8	7	5	3	1

All the above figures based on a starting and rolling resistance of 9 kg/tonne.
Approval of locomotive for a specific duty should be obtained from Clayton Equipment.

Figure 4.2. Specifications of a 3 ton Trolley Mining & Tunnelling Locomotive (Clayton Equipment)



SPECIFICATION

Weight
7.25 tonne

Engine
Deutz air-cooled, type F4L912W

B.H.P.
(traction rating) 55 at 2,300 r.p.m.

Transmission
Hydrostatic

Pump
Axial piston – variable displacement

Motor
Axial piston – fixed displacement

Gears
Totally enclosed, traction wormgear

Brakes
Dynamic through transmission

Parking and Emergency
Quick acting lever type

Wheels
559 mm (22") dia.

Suspension
Rubber chevron type

Controls
Engine speed lever. For forward and reverse
speed control lever

Painting
Safety Yellow

Standard Equipment:
Warning horn
12 volt Battery
Alternator
Ammeter
Engine temperature gauge
Engine oil pressure gauge
91 litre (20 gallon) Fuel Tank
3 pocket type Buffer/Coupler
Dry Type Air Cleaner

Optional Extras:
Automatic couplers
Radio control

Rail Gauge **Overall Width**
600 mm (24") to 1118 mm (3'8")
914 mm (36") variable

HAULAGE CAPACITY CHART

Speed km/h	Tractive Effort		Loads hauled in tonnes on straight track			
	Kg.	Level	1 in 200	1 in 150	1 in 100	1 in 50
0-7	1812	194	121	107	87	54
8	1565	166	104	92	74	46
10	1250	131	81	71	57	35
12	1020	105	65	57	45	27
14	860	87	53	47	37	21

All the above figures are based on a starting and rolling resistance of 9kg/Tonne.
Approval of locomotive for specific duty should be obtained from Clayton Equipment.

Figure 4.3. Specifications of a 7.25 ton Diesel Tunnelling Locomotive (Clayton Equipment)

4.3.2. SIZE AND SPECIFICATIONS OF LOCOMOTIVES

Finding the size and specifications of a locomotive may require the consideration of many factors, particularly if safety and auxiliary equipment are also involved. The first and most important step is to decide on the required locomotive weight necessary to develop the tractive effort to move the train load.

A locomotive with steel tyres on clean dry rails can produce a tractive effort of as much as 26% of its weight. It should also be able to produce approximately 15% more tractive effort than required by the trailing load in order to provide a reasonable acceleration rate (Buckeridge et al., 1982; Goedhals and Donnelly, 1960). Before starting the design calculations, definition of the ideal gradient should be made.

4.4. DESIGN CALCULATIONS

4.4.1. GRADIENT AND TRACTIVE EFFORT CALCULATIONS

In the ideal haulage, the slope or inclination of the tracks is such that the load on the locomotive is the same for transporting the full hoppers to the tip as for transporting the empty hoppers back to the loading place, and the locomotive must, in emergency, be capable of taking the full train back to the loading place without damaging the locomotive motors.

$$(\text{Train resistance}) \quad R_{t_{\text{up}}} = R_r + (g \cdot 9.08) \quad (4.1)$$

$$(\text{Train resistance}) \quad R_{t_{\text{down}}} = R_r + (g \cdot 9.08) \quad (4.2)$$

where up = upgrade and down = downgrade

R_r is rolling resistance of 9.08 kg (20 lb) per ton

maximum for antifriction bearings, g is grade in percent (Buckeridge et al., 1982).

$$TE = (W_l \cdot R_t) + (W_{tr} \cdot R_t) \quad (4.3)$$

where

$$TE = \text{tractive effort (kg)}$$

$$W_l = \text{weight of locomotive (ton)}$$

$$W_{tr} = \text{weight of train (cars + material) (ton)}$$

$$R_t = \text{train resistance (kg/ton)}$$

By using the definition of ideal gradient, where tractive effort of full and empty trains are the same, the ideal gradient can be found as:

$$W_{tr} = W_{car} + W_{load} \quad (4.4)$$

$$TE_{full} = TE_{empty} \quad (4.5)$$

$$(W_l + W_{car} + W_{load}) \cdot R_{t_{down}} = (W_l + W_{car}) \cdot R_{t_{up}} \quad (4.6)$$

where

$$TE_{full} = \text{tractive effort for full train (kg)}$$

$$TE_{empty} = \text{tractive effort for empty train (kg)}$$

$$W_{car} = \text{total car weight (ton)}$$

Substituting Equations 4.1 and 4.2 into Equation 4.6;

$$(W_l + W_{car} + W_{load}) \cdot (R_r - (9.08 \cdot g)) = (W_l + W_{car}) \cdot (R_r + (9.08 \cdot g)) \quad (4.7)$$

$$g = \frac{((W_l + W_{car} + W_{load}) - (W_l + W_{car})) \cdot R_r}{9.08 \cdot ((W_l + W_{car} + W_{load}) + (W_l + W_{car}))} \quad (4.8)$$

If the full train load is F_l and empty train weight is E_l ,

$$F_l = (W_l + W_{car} + W_{load}) \quad (\text{ton}) \quad (4.9)$$

$$E_l = (W_l + W_{car}) \quad (\text{ton}) \quad (4.10)$$

Equation 4.8 (ideal gradient) becomes,

$$g = \frac{(F_1 - E_1) \cdot R_r}{9.08 \cdot (F_1 + E_1)} \quad (\%) \quad (4.11)$$

Having established the ideal gradient, tractive efforts (loaded and empty train) can be estimated from Equation 4.3. Theoretical tractive effort (TTE) of a given locomotive;

$$TTE = W_1 \cdot 0.26 \cdot 1000 \quad (\text{kg}) \quad (4.12)$$

TTE may also referred to as available tractive effort. The difference between available tractive effort (TTE) and required tractive effort (TE) is the amount which will be used for acceleration. As mentioned earlier this difference should be greater than 15%.

4.4.2. DETERMINATION OF POWER REQUIRED

To calculate required power for the locomotive, it is necessary to know the tractive effort (TE) (Section 4.4.1), then the following formula can be applied:

$$P = \frac{TE \cdot v}{3.6 \cdot E} \quad (\text{kgfm/s}) \quad (4.13)$$

where

P = power (kgfm/s)

v = speed of locomotive (km/hr)

E = motor efficiency (%)

TE = tractive effort (kg)

If the result is wanted in kW, Equation 4.13 becomes :
(since 1kW = 102 kgfm/s);

$$P = \frac{TE \cdot v}{367.2 \cdot E} \quad (\text{kW}) \quad (4.14)$$

P is the actual required power. To determine the power

consumed, the distribution efficiency must be included in the calculations (Buckeridge et al., 1982).

CHAPTER FIVE

AUXILIARY TUNNEL VENTILATION

5.1. INTRODUCTION

In order to provide acceptable environmental conditions in a mine, it is essential to estimate the ventilation and refrigeration requirements. Unless these estimates are made at an early stage in the life of a mine, it may be difficult to provide and maintain satisfactory conditions (Burrows et al., 1982).

5.2. METHODS OF TEMPORARY TUNNEL VENTILATION

There are three types of temporary tunnel ventilation used during construction: (1) blowing or force (2) exhaust and (3) a combination of the two.

5.2.1. BLOWING (FORCE)

In this method, the fresh air is blown directly to the face by means of a duct. Thus the crew at the face is always surrounded by clean air, and pollutants are quickly swept away to the section of the tunnel further back. However, since the cross section of the tunnel is considerably larger than that of the duct, the outward velocity is reduced. Thus the foul air from the face moves back very slowly through the tunnel and inundates it. Personnel working behind the face, such as those erecting scaffolding or installing permanent support, are exposed to this contaminated atmosphere (See Figure 5.1).

Occasionally the foul air will move at a similar velocity as diesel vehicles operating in the tunnel, and when vehicles are leaving the tunnel, they will travel in a relatively stagnant air mass, with consequent pollutant build-up and re-breathing by the engines (Singh, 1982).

5.2.2. EXHAUST

In the exhaust method of ventilation, the contaminated face air is drawn out through a duct and the fresh air moves to the face through the main tunnel. The air at the face is swept slowly by this procedure, and the fresh air distribution may not be very uniform. Since temporary duct extensions are used to keep the pipe close to the face, some fresh air leakage into the return duct is inevitable. Furthermore, fresh air from portal or ventilation shafts may pick-up heat and moisture from the surrounding rock. In order to overcome some of these drawbacks, an auxiliary fan may be used with a short duct to increase the air velocity near the face and improve the cleaning efficiency; this, however, adds to the cost (Szechy, 1967). Power consumption is higher, as the exhausted air is less dense, and the fans generally are not as efficient in this mode (See Figure 5.2).

5.2.3. BLOWER-EXHAUST COMBINATION

Often a combination of blowing and exhaust ventilation systems may be used, with fresh air blown in the face areas and foul air exhausted out of the working areas behind the face. Laying two ducts along the length of the tunnel may, however, be quite expensive and occupy valuable space.

Sometimes it is possible to use the same fan in the blower-exhaust mode at different times in the mining cycle. If the fan is a non-reversing type, this can be accomplished by bi-furcating the duct connection to the fan and opening and closing appropriate valves, as shown in Figure 5.3.

Generally the fresh air is blown to within about 30 m (100 ft) of the tunnel face by means of a duct (Singh, 1982). After a blast, however, the fan is reversed and the fumes are exhausted for a period of 10 minutes. Then the fan is reversed again into the blower mode. This tends to move whatever fumes are left near the face towards the tunnel mouth. Meanwhile, the crew is mounted on a train, near where they have retreated for safety during the blast (well behind the ventilation duct opening), and then quickly taken to the face.

5.3. AIRFLOW REQUIREMENTS

The estimation of how much airflow is required within the working areas of a mine ventilation network is still the most empirical aspect of modern ventilation planning. The majority of such assessments continue to be based on local experience of gas emissions and heat load. They are still often quoted in terms of m^3/s per ton of mineral output. Corrections can be applied for variations in the age of the mine, the extent of old workings, distances from the shaft bottom, depth and rates of production (Burrows et al., 1982).

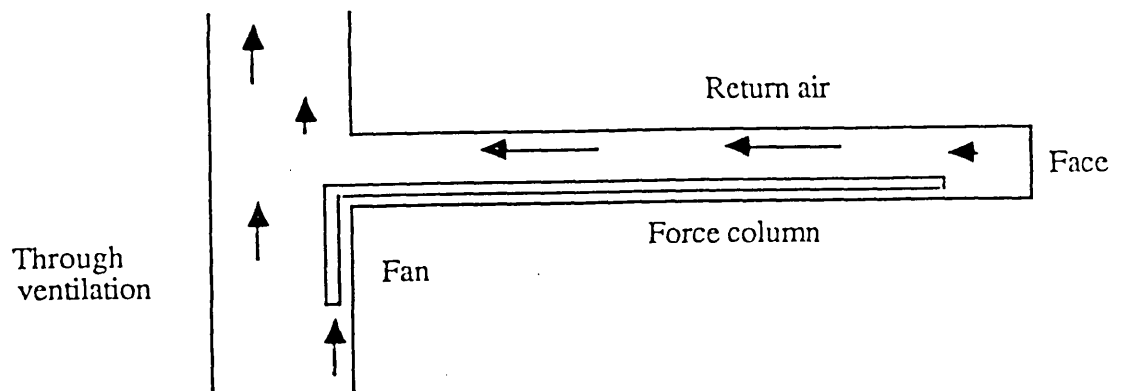


Figure 5.1. Blowing face ventilation system (after Thorp, 1982)

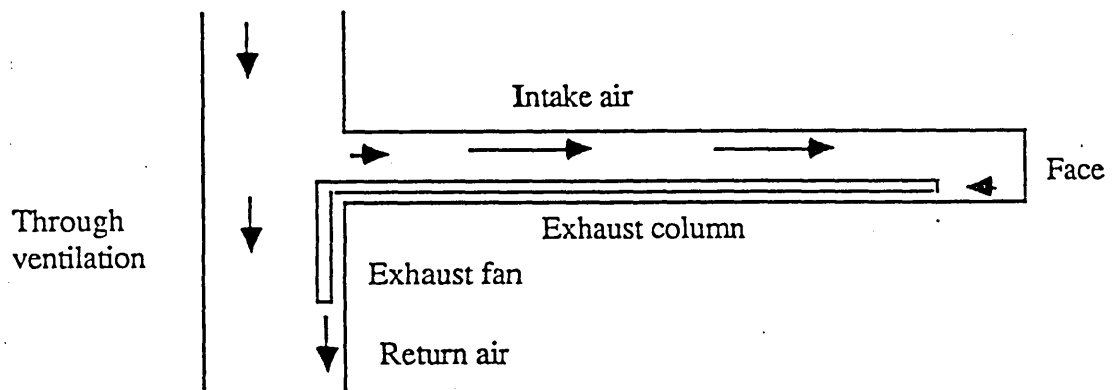


Figure 5.2. Exhaust face ventilation system (after Thorp, 1982)

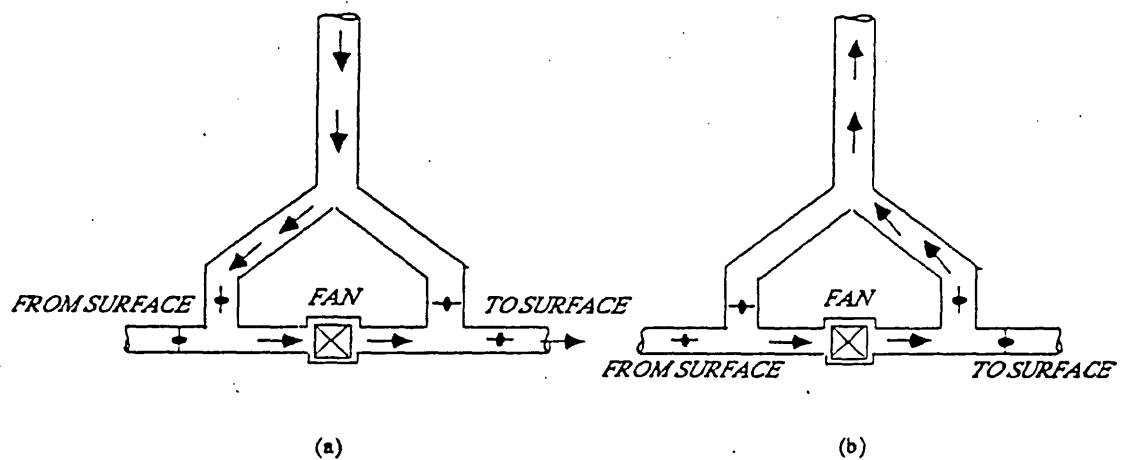


Figure 5.3. Schematic diagram of airflow reversal (a) Exhaust action (b) Blowing action (after Szechy, 1967)

The factors that need to be considered in estimating the quantity of fresh air for a tunnel include;

- 1- Number of people working per shift,
- 2- Gas Emissions,
- 3- Dust,
- 4- Type of haulage used,
- 5- Type and amount of explosive employed,
- 6- Methods of tunnelling,
- 7- Diesel Equipment,

8- Length of tunnel (Szechy, 1967; Singh, 1982; McPherson, 1984).

5.3.1. NUMBER OF PEOPLE WORKING PER SHIFT

The minimum quantity of air required per person to maintain a healthy environment is $0.009 \text{ m}^3/\text{s}$ (20 cfm), based on CO_2 and oxygen threshold limit value (TLV) requirements (Singh, 1982).

In order to dilute fumes and dust, however, the Occupational Safety and Health Administration (OSHA) specifies that at least $0.095 \text{ m}^3/\text{s}$ (200 cfm) for each employee underground must be provided (Singh, 1982).

5.3.2. GAS EMISSION

Most of the models that estimate gas flow into mine workings are essentially finite element programs and consider emissions from rectangular blocks above and below the working horizon. There is, however, considerable variation in the methods used to assess the rate of methane emissions from source horizons in roof and floor strata. Some of the

more sophisticated programs take into account the presence of methane drainage holes, and the decay of emission from any source with respect to time (McPherson, 1984).

The total volume of gases produced from a district in a mine may be assessed by monitoring concentrations and volume flows in the return airways and also in any methane drainage lines that may be present. It may also be possible to use the recorded rates of production on the working face.

Whichever technique is employed to predict the rate of gas emission, the final calculation of airflow requirement is

$$Q = G_e \cdot P / C_n \cdot h \cdot 36 \quad (5.1)$$

where

Q = required air volume (m^3/s)

G_e = gas emission rate (m^3/t)

P = mineral produced in h hours (ton)

C_n = concentration (percentage by volume) to which the gas is to be diluted.

In the case of methane , C_n is often taken to be one half of the concentration stipulated by the law (McPherson, 1984).

5.3.3. DUST

Dust is formed by most mining and rock handling processes. Due to the effects of pneumoconiosis and more particularly silicosis, it is probably one of the most severe and costly operational health problem in the mining industry (Sandys and Quilliam, 1982).

There are many techniques of reducing dust concentrations in mines ranging from water infusion of the solid mineral through to dust suppression by water sprays and air filtration systems (Mundell et al., 1979; Hamilton et al., 1982). Nevertheless, with current mining methods, it is inevitable that dust particles will be dispersed into the air at all places where rock fragmentation occurs at the rock winning work place and throughout the mineral transportation route. Dilution of airborne particles by ventilation remains the primary method of controlling dust concentrations in underground mines.

In this case, it is primarily the velocity rather than the quantity of airflow that is important. In any airway where dust is produced at a given rate, the distance over which the particles are carried by the air stream depends upon the air velocity and the settling rate of the individual dust particles. The latter depends on the density, size and the shape of the particles as well as the psychrometric conditions of the air.

$$Q = E.P.1000 / C.3600 \quad (5.2)$$

where

Q = airflow required (m^3/s)

E = emission rate of respirable dust (g/t)

P = rate of mineral production (t/h)

C = increase in concentration of respirable dust (mg/m^3)

Equation 5.2 calculates the quantity of air required to dilute the general body concentration of respirable dust. Dust particles in the respirable range ($\leq 5 \mu m$) will settle

out at a negligible rate (McPherson, 1984). For dilution purposes respirable dust may be treated as a gas.

5.3.4. TYPE OF HAULAGE

Electric haulage systems need air for cooling the motors, but this requirement may be small. Battery locomotives or trams require fresh air especially at charging stations. Diesel locomotives or LHDs need air to dilute exhaust gases released. Pneumatic-haulage systems may not require extra air, since this is introduced by expanded compressed air, which has a cooling effect (Singh, 1982).

5.3.5. EXPLOSIVES

Hard-rock tunnels are excavated mainly by drilling & blasting procedures. Most explosives, when detonated, produce large quantities of toxic fumes, comprising primarily carbon monoxide (CO), oxides of nitrogen and sulphur dioxide (SO₂).

The exact amount of fumes generated during blasts depends not only on the type of explosive but also on the total weight of explosive in each blast, completeness of combustion, age of the explosive, degree of confinement, manner of detonation, type of rock, and the moisture content of the air.

Observation by Szechy(1967) indicated that the area contaminated by the fumes of explosives is less than 21 m deep within the specified minimum safe interval of 10 minutes between successive explosions. It should be more than enough to exchange this volume of air five times within

the 10 minutes of waiting period specified in mining regulations,

$$Q = 21.A.N / \tau \quad (\text{m}^3/\text{s}) \quad (5.3)$$

where, τ is time in minutes, N is number of times the air should be exchanged per amount of time τ , and A is cross sectional area of tunnel in m^2 . If N is taken as 5, as recommended by Szechy and τ is 10 minutes of blowing, Eq.5.3 becomes

$$Q = 0.175.A \quad (\text{m}^3/\text{s}) \quad (5.4)$$

Generally, smoke in a tunnel diffuses at a rate of about 0.13 m/s (Singh, 1982). In US practice, it is common to have an air velocity that is twice this diffusion rate to obtain rapid dispersal. Unless a lower velocity has been determined, to control methane and respirable dust by the Mine Health and Safety Manager, the minimum mean entry air velocity must be 0.25 to 0.30 m/s (Mundell, 1977; Meakin, 1982; Singh, 1982).

5.3.6. METHODS OF TUNNELLING

The problems of ventilation encountered in drilling & blasting excavations are quite different from those that may be met whilst boring. Lubricants in the various types of machines, jumbos, etc. generate fumes at high temperature. Timber used for lagging or for temporary support tends to produce considerable amount of CO and H_2S as it decomposes.

5.3.7. DIESEL EQUIPMENT

Operation of diesel equipment in underground mines provides the potential for serious health and safety hazards from the gases, vapours, and particulates in the exhaust. Therefore,

various countries have rules and regulations for the safe deployment of diesel equipment (Kenzy and Ramani, 1979; McPherson, 1984). The basic stipulation is that there should be sufficient ventilation to dilute exhaust gases and particulates below each of their respective threshold limit values.

The criterion that has been used most widely for initial estimates of required ventilation is that of m^3/s of air per 100 kW rated power. Values quoted in the literature have varied from 3 to 10 m^3/s per 100 kW (McPherson, 1984). This range reflects difference in permitted Threshold Limit Values (TLVs) and engine design, and also the position at which airflows are measured. McPherson (1984) stated that as a first estimate, many ventilation planners now employ at least 6 m^3/s of airflow per 100 kW rated power, measured at the machine, all diesel equipment being cumulative in any one split.

5.3.8. TUNNEL LENGTH

The longer the tunnel, the more difficult it is to ventilate. It takes a longer time for the blast fumes to be cleared, and the friction losses vary in airflow directly with length for the case with exhaust systems (See section 5.5).

5.4. MINE HEAT LOAD

The mine heat load is the sum of all heat transfers that take place into the ventilating airstream. Heat load may be calculated for a complete mine or for individual sections or

airways. The heat load of a mine varies with the condition of the air at inlet (McPherson, 1984).

Since cooling and de-humidification are the processes most frequently employed in mines when air conditioning is involved, the sources of heat in mines become particularly important (Hartman, 1982a).

5.4.1. "AUTOCOMPRESSION"

"Autocompression" results in an increase in temperature of either air or water as a result of the conversion of potential energy into enthalpy when air is entering a mine through a shaft and not as a result of heat flow from an external source. However, autocompression is of such importance in mine ventilation and cooling that it must be included in any discussions on source of heat in mines (Hemp, 1982).

"Autocompression" has been studied for many years. Hartman (1982b) assumed that if no interchange in heat or moisture content of air takes place in the shaft, the compression occurs adiabatically and used the adiabatic law to calculate heat load, whereas, McPherson(1984) used steady flow equations by taking moisture content of the air into account. In the majority of cases, the shaft is not completely dry, thus pick up of both moisture and heat from the wall rock can occur. The airflow is, therefore, non-adiabatic.

5.4.2. GEOTHERMAL GRADIENT

The temperature of subsurface rock rises steadily with depth. In most climates, virgin rock temperature ceases to

be affected by surface temperature changes and is taken as constant for reference purposes to about 15 m beneath the surface. It then increases with depth at an approximately uniform rate in a given locality, the rise being termed the geothermal gradient. However, even within a single mining district, it varies and seldom can be considered as a constant. The age of the rocks, their thermal properties, and their proximity to recent igneous activity or hot springs largely determine the gradient.

Although, it varies in different mining districts and can be determined accurately only by field measurements, the earth's heat flux ranges from 0.04 to 0.06 W/m² (0.013 to 0.019 Btu/h.ft²) and averages 0.05 W/m² (0.016 Btu/h.ft²) (Hartman, 1982b).

Knowing the heat flux and thermal conductivity, **k**, (in W/m.°C) of the various rock formations encountered, the geothermal gradient in a locality can be calculated from conductive-heat-transfer theory (by Whillier, 1974)

$$q = k A \frac{\Delta t}{\Delta Z} \quad \Rightarrow \quad q/A = k \frac{\Delta t}{\Delta Z} \quad (5.5)$$

where q/A is heat flux (heat flow per unit area) and $\Delta t/\Delta Z$ is geothermal gradient (temperature change per unit depth) in °C/m for a given formulation.

$$\Delta t = \text{depth} * \text{geothermal gradient}$$

$$VRT = \Delta t + t_s \quad (5.6)$$

where

$$\Delta t = \text{temperature gain}$$

VRT = virgin rock temperature

t_s = surface rock temperature

When the temperature reaches the effective temperature, autocompression heating effects alone dictate that air conditioning must be installed (Hartman, 1982a).

5.4.3. ELECTRICAL MACHINES

All the electrical power delivered to underground working appears as a heat load on the mine environmental system apart from any work that is done against gravity (McPherson, 1984). The effect of the machine efficiency is relevant, in two ways within this context. Firstly, the heat produced can be reduced only if the machine is replaced by another of greater efficiency to give the same mechanical work output at a reduced power consumption. Secondly, the efficiency of the machine determines the distribution of the heat produced. The greater the efficiency, the lower the heat produced at the motor and transmission.

5.4.4. DIESEL MACHINES

A diesel engine is a heat engine, that converts a portion of heat into mechanical power. The remaining heat input is dissipated directly as heat. The amount of heat produced is a function of the machine efficiency. Any useful work done will reduce the heat dissipated.

The easiest way to estimate heat loads arising from diesel equipment is to consider the quantity of fuel consumed. This will present little problem if an existing situation is being investigated, but is not as easy to use for future planning (Hemp, 1982). Under these circumstances

heat loads must be estimated from the total rated power of the diesel equipment and for information on how it is used.

The internal combustion engines of diesel equipments have an overall efficiency approximately 30% (McPherson, 1984; Hemp, 1982) of that achieved by electrical units. Hence, diesels will produce approximately three times as much heat as electrical equipment of the same mechanical work output. This can be demonstrated by taking a typical rate of fuel consumption to be 0.3 litres per rated kW per hour. At a calorific value of 34000 kJ/l for the diesel fuel, the heat produced is

$$\frac{0.3 * 34000}{3600} = 2.83 \text{ kW of heat per kW work output}$$

5.4.5. FISSURE WATER

Water has a thermal conductivity of 0.62 W/m.°C which is considerably lower than that of most rocks. Fissure water is often emitted at a temperature close to the virgin rock temperature (VRT). In circumstances of local vulcanism or radioactive decay it may even be higher. The heat load on the mine environment can be calculated from the flow rate and drop in water temperature between the points of emission and effective exit from the mine ventilation system (Johnson et al., 1982).

5.4.6. OXIDATION

Coal and sulphide ore mines are particularly liable to suffer from a heat load arising from the oxidation of fractured rock. This can be predicted from the following

assumptions. Each 1 kg of oxygen consumed oxidizes 12/32 kg of carbon. Taking the calorific value of carbon as 33800 kJ/kg (McPherson, 1984; Hemp, 1982) gives a corresponding heat production.

5.4.7. EXPLOSIVES

The heat produced during blasting varies with the type of explosive used and charge density. The amount of heat released for most explosives employed in mining operations falls within the range 3700 kJ/kg for ANFO to 5800 kJ/kg for Nitro-glycerin (Whillier, 1974; Hemp, 1982).

5.4.8. HUMAN METABOLISM

The processes of human metabolism produce heat, since the body temperature is normally above that of the atmosphere, body waste heat flows into the air. Metabolic heat produced per labouring man ranges from 234 to 703 W (Hartman, 1982b). Heat, both sensible and latent (from evaporating sweat), is contributed to the mine air by workers under normal conditions, the more significant in hot, humid atmospheres being latent. If the amount of ventilating air is relatively large in proportion to the number of people it serves, this source can be ignored (Johnson et al., 1982).

5.4.9. HEAT PICK-UP IN DEVELOPMENT ENDS

When tunnels are being developed the working face moves further and further away from the main air stream and therefore the condition of the air reaching the face deteriorates as development progresses. Mining Engineers, when faced with this difficulty, may use different methods

to keep the temperature of the face at a level where reasonable working efficiency is possible (Jacobs, 1967).

With increasing depth there is a rise in the rate at which heat is picked-up by the air supplied to a development end. Such heat input is reflected by an increase in temperature of the air between entry and exit. It can be kept down by increasing the quantity of ventilation air, but when ventilation columns are long, the extra fan power required may aggravate the situation.

Patterson (1982) argues that there exists an optimum air quantity which corresponds to a minimum rise in wet-bulb temperature in a given development end and estimates the optimum air quantity.

When designing auxiliary ventilation systems (Section 5.5), one of the objectives to achieve is to supply enough air for the workers and to dilute gases from diesel engines to well below their threshold limit values (TLV).

Computer programs have been developed to estimate the heat pick-up and the effect on air temperatures for two distinct zones in tunnelling. One zone is at the working face itself including a portion of the tunnel and stretches back from the face for a distance of about 50 m (Goch, 1973), and other is for the completed portion of the tunnel that serves as the intake airway to the working zone at the far end (Lawrance and Whillier, 1975).

The heat pick-up in the vicinity of the working face of a developing tunnel depends on many parameters that are not easily specified in advance. The equation given by Whillier

and Ramsden (1975) may be used to estimate this heat pick-up approximately ;

$$H = DFA \cdot (t_{vr} - t_{wb}) \cdot (P/12)^{1.3} \cdot (k\rho C/13 \cdot 10^6)^{0.5} \quad (5.7)$$

where

H = rate of heat pick-up by the ventilation air in the vicinity of the working face of the development end.

DFA = daily face advance (m)

P = perimeter of the tunnel area (m)

k = thermal conductivity of rock (W/m°C)

ρ = density of rock (kg/m³)

C = specific heat of rock (J/kg °C)

The constant 13×10^6 is a typical value of $k\rho C$ for quartzite (Whillier and Ramsden, 1975).

This calculated value must be added to the heat produced by the fans in the tunnel as well as any electric or diesel power used for other purposes, in order to estimate total heat load.

5.5. AIR REACHING THE FACE OF DEVELOPMENT ENDS

When tunnels are developed in hot rock the thermal environment at the working face becomes worse as the face moves away from the point of the ventilation supply; and may be of particularly concern for high speed advances in hot rock.

Wet-bulb temperature is the primary factor that determines the cooling power of the air in the development ends and therefore it is desirable to keep the wet-bulb temperature of the air arriving at the face as low as

possible (Whillier and Ramsden, 1975). This can be accomplished several ways ;

a. increasing the quantity of air reaching the face; which not always be possible. As mentioned earlier, increased volume flux of air demands greater fan power.

b. using refrigeration (Johnson et al., 1982).

c. drying out thoroughly the footwall of the completed sections of the tunnel through which the incoming air travels (Jacobs, 1967).

d. insulating the rock surface of airways (Hamilton et al., 1982).

Insulating the rock surface of the airways reduces the heat flowing into the underground environment directly by providing a thermal barrier between the rock and the cooler ventilation air. In a fully insulated airway, for example, the heat flow from the rock into the the air could be reduced by as much as 50 percent (CM Research Report, 1987).

Insulation has also some other benefits :

a. Reduction in the fan power required due to lower friction factors along airways.

b. Reduction in air leakage by using insulation as ventilation sealant.

c. Prevention of fissure water entering the airways.

d. Reinforcement for ground support.

5.6. DESIGN OF AUXILIARY TUNNEL VENTILATION SYSTEMS

For the design of an auxiliary ventilation system, the volume rate of the airflow that will be required at the face and the maximum length of the drivage dictate the necessary

size and type of the fan and the duct. The quantity of air required at the face can be calculated by using the methods given in Section 5.3.

Auxiliary ventilation relies upon fabricated ducting for the passage of either intake or return air. Because of the nature of ducting and numerous joints, complete elimination of leakage from or into the ducting is impossible in practice (Browning, 1983; Le Roux, 1979).

5.6.1. AIRFLOW ANALYSIS IN LEAKY DUCTS

Three methods are available for analyses of airflow problems in a leaky duct (Vutukuri and Lama, 1986).

a. assuming a uniformly distributed leakage and analysing the leaky duct mathematically ;

b. assuming a number of discrete leakage paths and treating the leaky duct, i.e. duct and leakage paths, as a ventilation network;

c. assuming a number of discrete leakage paths and treating the airflow through the leaky duct as a series-parallel combination of airflows along the duct and through leakage paths.

a. Mathematical analysis of airflow in ducts with a uniformly distributed leakage:

The rate of airflow in a leakless duct can be calculated by using Atkinson's formula:

$$Q = (P \cdot A^3 / k \cdot L \cdot C)^{0.5} \quad (5.8)$$

where

Q = rate of airflow (m^3/s)

P = pressure loss (Pa)

k = friction factor for the duct (Ns^2/m^4)

A = cross sectional area of duct (m^2)

L = length of duct (m)

C = perimeter of the duct (m)

For a given duct, kLC/A^3 is constant and usually substituted by the term R , the resistance coefficient with units Ns^2/m^8 . Hence,

$$Q = \sqrt{(P / R)} \quad (\text{m}^2) \quad (5.9)$$

In a leaky duct, in addition to the variation of the pressure, the flow rate varies along the duct and the calculations are not straight forward.

The flow of air along a uniformly leaky duct can be defined by two differential equations which may be solved simultaneously to calculate leakage and resistance coefficients from the pressure and the flow distribution (Browning, 1983). A scientific method of expressing the magnitude of the leakage from a column is by means of the "leakage coefficient" as defined by the National Coal Board (Le Roux, 1979).

Unlike a mine airway, where it is assumed that pressure only varies along the length, in a leaky duct both quantity and pressure will vary. Two constants are required to define the duct characteristics, namely, the resistance coefficient of the duct and resistance coefficient of leakage paths.

Browning (1983) describes the resistance constant as being similar to an airway resistance. He expresses it as

the pressure loss for a given length at a given flow and the leakage coefficient as the difference in flow (leakage) for a given length at a given pressure. The unit adopted for leakage is $\text{m}^3/\text{s} / 100 \text{ m}$ length at a pressure of 1 kPa.

Vutukuri (1983) finds a relationship between leakage coefficient and resistance coefficient of leakage paths as:

$$R_1 = 1000 / (L_C)^2 \quad (5.10)$$

where

R_1 = resistance coefficient of leakage paths (Ns^2/m^8)

L_C = leakage coefficient ($\text{m}^3/\text{s} / 100 \text{ m}$ at 1 kPa)

Calculation of R_1 of a duct is complicated because in practice ducts are never under uniform pressure. According to Le Roux (1979), the pressure is always greatest nearer the fan and consequently 100 m of duct near the fan will leak much more than 100 m of duct some distance away from the fan. All problems of this nature can be calculated by means of the nomogram illustrated in Figure 5.4 which was constructed by National Coal Board.

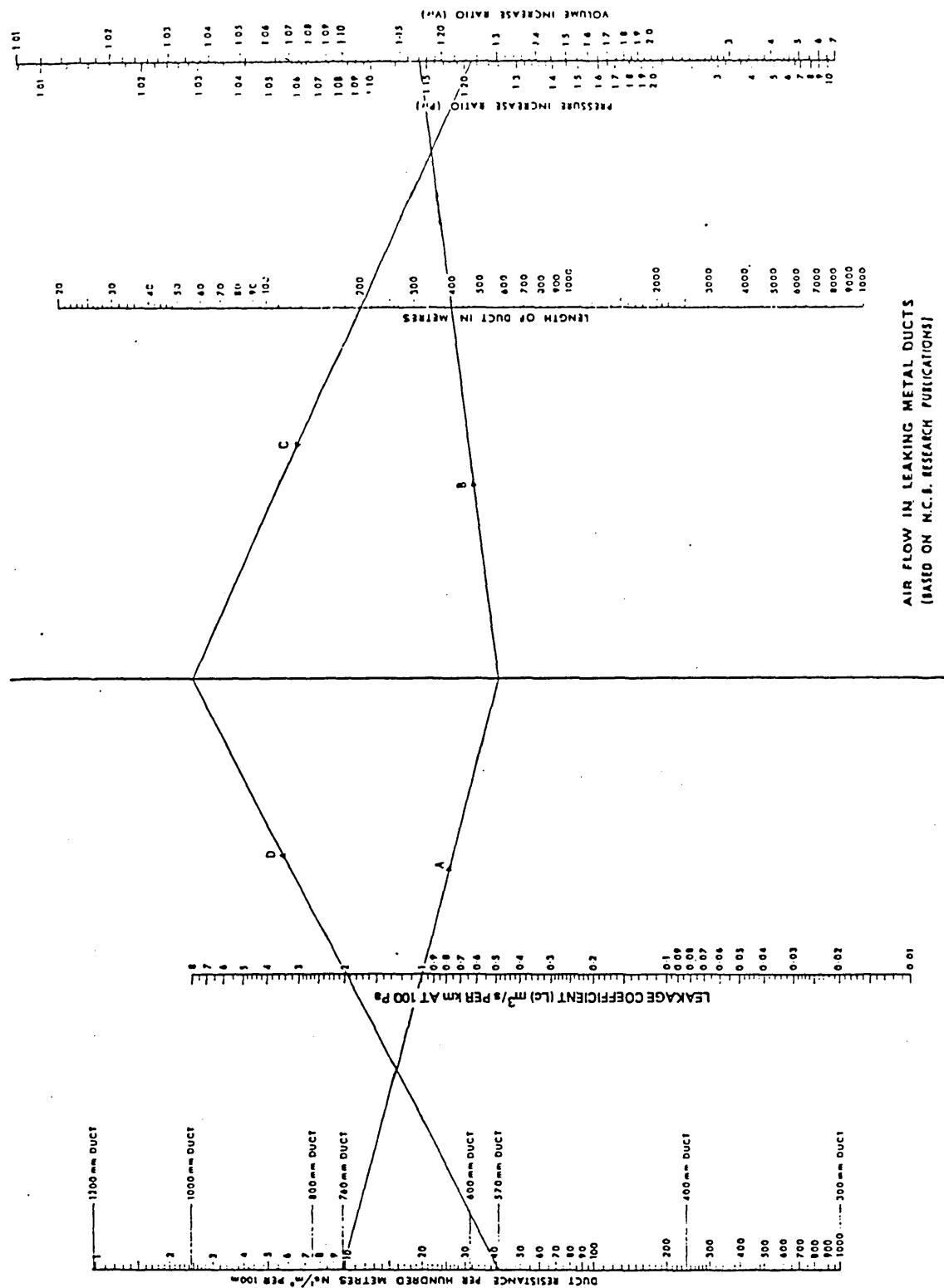


Figure 5.4. Airflow in leaking metal ducts (after Le Roux, 1979)

Example: A 570 mm duct, 200 m long, has an intake volume of 3 m³/s and delivery volume of 2.4 m³/s. What would be the leakage coefficient ?

Solution : The volume increase ratio (VIR) = 3.0/2.4 = 1.25, Draw a line (C on the nomogram, Figure 5.4) from VIR = 1.25 through the 200 m duct length to intersect the blank line in the centre. From this intersection draw another line (D) to the specific resistance volume of the 570 mm duct. Resistance coefficient of leakage paths is found to be 41 Ns²/m⁸.

The airflow in a duct with a uniformly distributed leakage may be presented by the following two equations (Vutukuri and Lama, 1986).

$$\frac{dP}{dx} = R_d \cdot Q^2 \quad (5.11)$$

and

$$\frac{dQ}{dx} = \sqrt{(P / R_l)} \quad (5.12)$$

where

P = pressure at a distance x along the duct

Q = volume flow rate at a distance x

R_d = resistance coefficient of the duct

R_l = resistance coefficient of leakage paths

x = distance along the duct measured in the direction of increasing volume flow rate and pressure

These two equations are valid for the cases of both forcing and exhausting systems, provided that P and Q are

regarded as positive and x is measured in the direction of increasing P . The validity of these two equations depends on the following assumptions:

- air density is constant
- pressure losses at ends are neglected
- diameter of the duct is uniform

By eliminating Q from Equation 5.11 and 5.12, the following equation is obtained (Vutukuri and Lama, 1986).

$$\left(\frac{dP}{dx}\right)^{1/2} \cdot \frac{d^2P}{dx^2} = \left(\frac{R_d}{R_l}\right)^{0.5} \cdot 2 \cdot \left(\frac{dP}{dx}\right) \cdot P^{1/2} \quad (5.13)$$

By taking the boundary conditions, eg., $P=0$ and $Q_0=0$ (Q_0 = volume flow rate at the face), $x=0$, Vutukuri (1983) introduced dimensionless parameters u and t .

$$u = \left(\frac{2}{R_l^{0.5} \cdot R_d}\right)^{2/3} \frac{P}{Q_0^2} \quad (5.14)$$

$$t = \left(2 \left(\frac{R_d}{R_l}\right)^{1/2}\right)^{2/3} x \quad (5.15)$$

Equations 5.14 and 5.15 are used to calculate pressure increase and volume increase ratios. Pressure increase ratio (PIR) is defined as the ratio of pressure required to deliver Q_0 under leakless conditions. The volume increase ratio (VIR) is defined as the ratio of volume flow rate handled by the fan to volume flow rate reaching the face (Vutukuri, 1983).

$$PIR = 1.0203344 - 0.26817t + 1.068933t^2 - 0.726484t^3 + 0.258632t^4 - 0.0309553t^5 + 0.0006928t^6 \quad (5.16)$$

and

$$\begin{aligned} \text{VIR} = & 1.0088985 - 0.062632t + 0.731745t^2 - 0.466485t^3 + \\ & 0.180633t^4 - 0.0288809t^5 + 0.00155255t^6 \end{aligned} \quad (5.17)$$

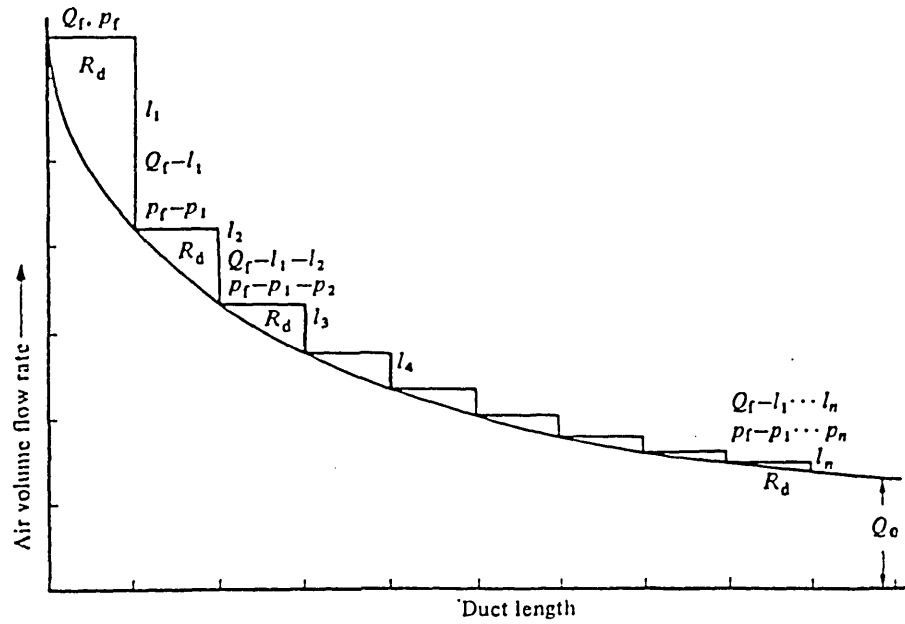


Figure 5.5 Schematic diagram for mathematical analysis of a duct with a uniformly distributed leakage (after Vutukuri and Lama, 1986)

$$\text{PIR} = \frac{P_1}{P_o} \quad (5.18)$$

where

P_1 = pressure developed by the fan in a leaky duct (Pa)

P_o = pressure developed by the fan in a leakless duct (Pa)

$$\text{VIR} = \frac{Q_1}{Q_o} \quad (5.19)$$

where

Q_1 = volume rate handled by the fan (m^3/s)

Q_0 = volume rate reaching the face (m^3/s)

In order to calculate the resistance coefficients of the duct and resistance coefficient of leakage paths, formulas given by Browning (1983) can be used.

$$R_d = \frac{(P_1 - P_2)}{L} \cdot \left(\frac{5}{(2Q_1 + 3Q_2)} \right)^2 \quad (5.20)$$

and resistance coefficient of leakage paths;

$$R_l = \frac{3(Q_1 - Q_2)(P_1 - P_2)}{2L(P_1^{3/2} - P_2^{3/2})} \quad (5.21)$$

where

R_d = resistance coefficient of the duct (Ns^2/m^8)

R_l = resistance coefficient of leakage paths (Ns^2/m^8)

P_1 = upstream pressure (kPa)

P_2 = downstream pressure (kPa)

Q_1 = airflow at P_1 (m^3/s)

Q_2 = airflow at P_2 (m^3/s)

L = Length of duct (m)

The calculations usually involved are:

a. To find the minimum ventilation pressure and volume flow rate to give the required volume flow rate at the face of the heading.

b. To determine the fan operating point and the delivered volume flow rate at the face for a given duct and a fan. The following are required; R_d , R_l , L and fan characteristic curve ($P-Q$).

5.6.2. USE OF MULTIPLE FANS

When a ventilation duct in a development end is extended as the face advances, the air quantity reaching the face decreases due to increasing duct resistance and increasing leakage (See Figure 5.5). When the face air quantity decreases to the minimum acceptable value, another fan is installed in the duct line. The question that then arises is whether the second fan should be installed next to the existing fan at the one end of the duct or some distance away.

If the second fan is installed next to the first one, the air volume delivered at the face increases but the leakage also increases. Therefore the second fan is usually installed some distance away from the first one in the duct line (Le Roux, 1979). The important criteria in this case is that no circulation takes place between the air flowing in the duct and the surrounding air. For a constant duct length, the problem can be approached in two ways:

a. by deciding the location for the second fan at the middle of its length and choosing the fans with required characteristics.

b. by determining the optimum location for the second fan of given characteristics (Vutukuri, 1983).

CHAPTER SIX

THE COMPUTER MODEL

6.1. INTRODUCTION

More effective utilization of available resources often creates a narrow margin of competitive advantage:

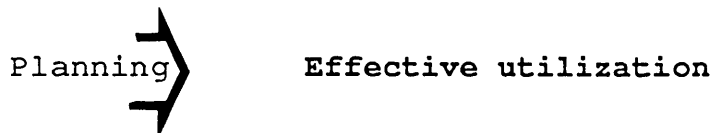
Limiting resources

Men

Money

Materials

Machines



Operations research (OR) seeks determination of the best (optimum) course of action under the restricted resources. The term of operations research is quite often associated almost exclusively with the use of mathematical techniques to model and analyse decision problems. When mathematical models are used properly they have many advantages.

One important advantage of model building is the frame of reference that it provides for consideration of the problem. A model will indicate gaps that are not immediately apparent, and after examining, the nature of the failure might indicate the model's deficiencies (Thierauf et al., 1985).

From a cost stand point, a firm benefits in obvious ways by expressing a complex problem in a mathematical model

that will allow it to change parameters without undertaking actual construction of the project. For example, in a tunnelling operation where a mathematical model is used, selection of drilling and mucking or transportation machines that are not well matched can be avoided. Furthermore models save time, since results (whether favourable or unfavourable) can be obtained within a relatively short time. However, the time needed to evaluate actual results requires the completion of a given project, that means the entire duration of its day-to-day operations.

Once a problem has been expressed in mathematical equations, the user can insert different values into the model and study the behaviour of the system. If properly undertaken, statements about the sensitivity of the system to a change in any of the variables can be made.

6.2. THE COMPUTER MODEL, UGDEV

A series of computer programs have been developed at the Earth Mechanics Institute (EMI) for evaluating the economics of various mining subsystems (Sharp et al., 1983). They are;

1. fragmentation - drilling and blasting
2. tunnel boring
3. material handling - cyclical
4. ground support
5. ventilation

All of these programs were designed to stand alone.

The model UGDEV developed during the course of this research has the main features listed below :

1. calculates the number of blastholes for a given tunnel area, hole diameter and compressive strength of the rock.

2. estimates penetration rate, drilling and blasting time for known rock conditions and type of drilling (rotary or percussive).

3. selects mucking system (either overshot or Load-Haul-Dump Unit (LHD) by taking area, job conditions and type of muck into account and calculates the number of mucking cycles and mucking time.

4. estimates weight of locomotive capable of carrying the blasted material by considering gradient of the road, quantity of the material and daily advance rate.

5. It then estimates required resources such as labour, consumable items, etc. Table 6.1 gives an example output of the model.

This model is straight forward and computes according to input values supplied by the user. In the calculations of the drilling part of the model, type of jumbo, number of booms, bit diameter, and diameter of cut hole in the face are all input values. However, the type of equipment has a big effect on the total economics of the operation. A question arises as to whether some of the parameters such as, number of holes, hole depth, type of jumbo, number of booms, type of mucker, etc. could be examined as variables and any of the existing mathematical

programming techniques could be applied to minimize the cost of operation and to find the best combination of equipment. To take care of this disadvantage, TUNOPR has been developed by the author (Section 6.8).

Table 6.1 Example output of the model UGDEV

+ UNDERGROUND MINE DEVELOPMENT +			
+ +			
+ BY T.T.OZAN & H.E.K.ALLEN +			

6.3. OPTIMIZATION

The fundamental problem of optimization is to arrive at the best possible decision for any given set of circumstances. An optimization problem involves minimizing (maximizing) a function called objective function of several variables, possible restrictions on the values of the variables being defined by a set of constraint functions.

The most important type of operations research models is the symbolic or mathematical model (Taha, 1982). It assumes that all the relevant variables, parameters and constraints as well as the objective are quantifiable. Thus, if x_j , $j=1,2,\dots,n$ are the n decision variables of the problem under study, and if the system is subject to m constraints the general mathematical model can be written in the form ;

$$\text{optimize } z = f(x_1, \dots, x_n) \quad (\text{objective})$$

subject to

$$g_i(x_1, \dots, x_n) \leq b_i \quad i = 1, 2, \dots, m$$

$$x_1, x_2, \dots, x_n \geq 0$$

The constraints $x_1, x_2, \dots, x_n \geq 0$ are called non-negativity restrictions. They restrict the variables to zero or positive values which in most real life situations appear to be a natural requirement.

Optimization is generally taken to signify the maximization or the minimization of the objective function. But this is the extent to which the word of "optimization" goes in unifying operations research (Taha, 1982). This means that two analysts working on the same program

independently may yield two different models with different objective criteria. For example, one may prefer to maximize profit while another might prefer to minimize the cost. The two criteria are not equivalent in the sense that with the same constraints the two models may not produce the same optimum solution. This can be made clear by realizing that, although cost may be under the immediate control of an organization in which the study is made, profit could be affected by uncontrollable factors such as the market situation dictated by competitors.

The main conclusion at this point is that the solution of a model is at best only optimum relative to that model.

6.3.1. OPTIMIZATION TECHNIQUES

Optimization techniques propose the rules for moving from point to point in the track across the feasible region. They dictate how to choose a new direction and how far to move. The appropriate method depends upon the form of the problem, the number of problem variables, whether they are continuous or discrete, and the form of the objective and constraint functions. To introduce the names of the optimization techniques and give an idea of the sorts of problems to which they are appropriate, the titles of the methods are listed below with short statements about their capabilities (See Nicholson, 1971; Taha, 1982; Thierauf et al., 1985).

Calculus and Lagrange multipliers

Calculus can be used on small problems of a few continuous variables with simple objective functions and no constraints. It can also be used when equality constraints are present through the introduction of Lagrangian multipliers. The calculus methods are particularly elegant when applicable (Nicholson, 1971).

Linear programming

Thierauf et al. (1985) define linear programming as "a mathematical technique for determining the allocation of a company's limited resources." A mathematician might offer a more technical definition, calling it "a method for solving problems in which a linear objective function must be minimized or maximized when considering certain linear constraints." Large problems in continuous variables can be solved by linear programming methods provided that the objective function is linear and the constraints are in the form of equalities or inequalities.

Nonlinear optimization

Nonlinear optimization methods are concerned with handling the geometry of a nonlinear objective function subject to nonlinear equality or inequality constraints. The structure of these methods is more complex. Usually, it is only to obtain a local optimum (Nicholson, 1971).

Dynamic programming

Dynamic programming is a mathematical procedure designed primarily to improve the computational efficiency of solving select mathematical programming problems by

decomposing them into smaller, and hence computationally simpler subproblems. Dynamic programming typically solves the problems in stages, with each stage involving exactly one optimizing variable. The computations at different stages are linked through recursive computations in a manner that yields a feasible optimal solution to the entire problem when the last stage is reached.

Thierauf et al. (1985) state that this technique is an extension of the basic linear programming developed by R. Bellman and G.B.Dantzig. In its early years, dynamic programming was commonly called stochastic linear programming, in reference to linear programming problems that dealt with uncertainty. Today, dynamic programming has been developed as a management science model and is used to solve a wide range of business problems.

Branch and bound method

The branch and bound method (also called back-track programming) is a general approach to discrete optimization problems which aims to conduct an intelligent search through all possible solutions, cutting out large groups of possibilities early in a search. It depends on being able to judge in advance which directions of search can be eliminated. Despite the generality of the idea, the method is limited to suitably small problems (Nicholson, 1971).

Heuristic techniques

Although mathematical models seek the determination of optimum solution, sometimes the mathematical formulation

may be too complex to allow an exact solution. Even if the optimum solution can be attained eventually, the required computation may be impractically long. In this case, heuristics can be used to develop good (approximate) solutions. The heuristic method of solution relies on empirical rules that have the potential to determine an improved solution relative to the current one. Actually, heuristics are search procedures that intelligently move from one solution point to another with the objective of improving the value. When no further improvements can be achieved, the " best " attained solution is the approximate solution to the model (Taha, 1982).

Permutation procedures

This method deals with problems which are represented as permutations. It can find local optima of these problems, but the choice of how to define a local optimum must be defined for the particular application. This may be useful freedom, but it also raises uncertainties about the quality of the answers obtained.

The characteristics of the problems which the various methods will solve are summarized in Table 6.2.

Table 6.2 Problem characteristics for various optimization methods (after Nicholson, 1971)

Method	Problem type				
	Number of variables	Form of variables	Form of objective function	Form of constraints	Local or global optima
Calculus and Lagrange multipliers	few	continuous	differentiable	simple and differentiable equalities	local
Linear programming	many	continuous	linear	linear equalities and inequalities	global
Non-linear optimization	several	continuous	continuous and preferably convex	continuous functions and inequalities	local
Dynamic programming	several	preferably discrete	separable as multi-stage process	fairly general	global
Branch and bound methods	few	discrete	general	general	global
Permutation procedures	many	elements	general	general	specialized local
Heuristic techniques	many	discrete	well-structured	well-structured	unspecified

6.3.2. MODEL FORMULATION

The formulation of a generalized model entails four stages: (1) definition of the problem, (2) construction of the model, (3) solution of the model and (4) validation of the model (Thierauf et al., 1985; Taha, 1982).

This sequence is by no means standard but is regarded as generally acceptable. Except for the model solution stage, which is based on generally well-developed techniques, the remaining phases do not seem to follow fixed rules. This stems from the fact that the procedures for these phases depend on the type of the problem under

investigation and the operating environment in which it exists.

The first phase of the model formulation deals with problem definition. This indicates three major aspects;

- a. a description of the goal or the objective of the study,
- b. an identification of the decision alternatives of the system,
- c. a recognition of the limitations, restrictions and requirements of the system.

The second phase of the study deals with model construction depending on the definition of the problem. A model should specify quantitative expressions for the objective and the constraints of problems in terms of the decision variables. If the resulting model fits into one of the mathematical models, a convenient solution can be obtained.

The third phase of a study addresses the model solution. In mathematical models, this is achieved by using optimization techniques. In addition to the solution of the model, one must also secure additional information concerning the behaviour of the solution due to the changes in the system's parameters. This is usually referred to as sensitivity analysis. In particular, such an analysis is needed when the parameters of the system cannot be estimated accurately (Taha, 1982). In this case it is important to study the behaviour of the optimal solution in the

neighbourhood of these estimates.

The last phase of the model formulation is the model verification. A model is valid if, despite its inexactness in representing the system, it can give a reasonable prediction of the system's performance. A common method for testing the validity of a model is to compare its performance with some past data available for the actual system.

6.4. DEFINITION OF OBJECTIVES FOR TUNNELLING OPERATIONS

Tunnels are constructed for several purposes. These comprise in part the permanent transportation network excavated to final dimensions and provided with a permanent lining and in part of the interconnected exploitation drifts and galleries in which only temporary supports are employed.

For the purposes of the mining industry, tunnels are driven for ventilation, exploration, transportation and access to work places.

The objectives of tunnelling operations, where conventional drilling and blasting method are practiced, can be listed as follows:

a. To find out the best combination of number of holes, hole diameter, hole depth and cut hole diameter in order to minimize the cost of a drilling and blasting round for a known area and given rock conditions.

b. To examine the cost and performance of drilling jumbos whose specifications, eg., purchase price, boom number, installed power, etc., are known.

c. To find the best mucking system to match the

selected jumbo type.

d. To choose the best combination of locomotive and rail cars to minimize the cost of transportation.

e. Having selected a drill jumbo, mucking system, locomotive and estimated air quantity required at the face of the tunnel, select the best combination of ventilation fan and duct that would provide the estimated quantity of air at minimum cost and calculate number of fans needed.

6.5. MODEL FORMULATION TO MINIMIZE COST OF A DRILLING AND BLASTING ROUND

The total cost of a drilling and blasting round can be expressed as:

$$TDBC = TDC + TBC \quad (6.1)$$

where

TDBC = total cost of drilling and blasting

TDC = total cost of drilling per round

TBC = total cost of blasting per round

$$TDC = JC \cdot \frac{n_h \cdot H}{n_b \cdot P_{rt}} \quad (6.2)$$

where

JC = cost of operating jumbo per hour (Section 6.7.2)

n_h = number of drilled holes per round

n_b = number of booms mounted on the jumbo

P_{rt} = penetration rate

H = hole depth

$$TBC = n_h \cdot c_{det} + \frac{d^2 \cdot \pi}{4} L_{ch} \cdot d_{ex} \cdot c_{ex} + \frac{0.64 \cdot c_{op} \cdot n_h \cdot H}{P_{rt} \cdot n_b} \quad (6.3)$$

where

c_{det} = cost of detonator

d = diameter of hole

L_{ch} = charge length

d_{ex} = density of explosive

c_{ex} = cost of explosive

c_{op} = labour cost for charging the shotholes

In Equations 6.2 and 6.3, there are relations among some of the parameters, as explained in Chapter 2. These include the number of holes, tunnel area, rock compressive strength and the rock penetration rate. Therefore, these parameters can be examined as decision variables and the objective function can be written as :

$$f(x) = JC \cdot \frac{x_1 \cdot x_5}{P_{rt} \cdot n_b \cdot 60} + x_1 \cdot (x_4 \cdot c_{ex} + c_{det}) + \frac{0.64 \cdot c_{op} \cdot x_1 \cdot x_5}{P_{rt} \cdot n_b \cdot 60} \quad (6.4)$$

where

$f(x)$ = cost of drilling and blasting

x_1 = (n_h) number of holes

x_2 = (d) hole diameter (mm)

x_4 = charge per hole (g)

x_5 = (H) hole depth (m)

c_{op} = labour cost (\$/hr)

From Figure 2.8, Equation 6.5 was developed.

$$x_1 = (1.74366 - 8.9166 \cdot E-3 \cdot x_2) \cdot \text{Area} - \beta \quad (6.5)$$

where

Area = tunnel area (m^2)

β = a constant depending on the rock compressive strength and drilling type (percussive / rotary)

There is a relation between the diameter of the cut hole and the hole depth (Chapter 2, Equation 2.2). If the diameter of the cut hole is expressed with x_3 :

$$x_5 = 0.15 + 34.1 \cdot x_3 - 39.4 \cdot x_3^2 \quad (6.6)$$

Shotholes are usually charged with a column length of two-thirds of the hole depth and in stone drifting, each hole should be charged at less than 1400 g/hole (ICI).

$$x_4 = (x_2^2/4) \cdot \pi \cdot x_5 \cdot 2/3 \cdot d_{ex} \quad (6.7)$$

where

x_2 = bit diameter (mm)

x_3 = diameter of cut hole (m)

x_4 = charge per hole (g)

d_{ex} = density of explosive (g/cm³)

x_5 = hole depth (m)

Equations 6.5, 6.6 and 6.7 can be taken as equality constraints and the decision variables can be defined as :

$$10 \leq x_1 \leq 160 \quad (\text{number of holes}) \quad (6.8)$$

$$28 \leq x_2 \leq 64 \quad (\text{bit diameter}) \quad (6.9)$$

$$0.06 \leq x_3 \leq 0.12 \quad (\text{cut hole diameter}) \quad (6.10)$$

$$x_4 \leq 1400 \quad (\text{charge per hole}) \quad (6.11)$$

$$1.5 \leq x_5 \leq 4.5 \quad (\text{hole depth}) \quad (6.12)$$

Having formulated the model, the next step is to solve the problem. The goal is to minimize the objective function (Equation 6.4) which is a nonlinear equation subject to nonlinear equality constraints (Equations 6.5 to 6.12).

6.6. MODEL SOLUTION

6.6.1. NONLINEARLY CONSTRAINED MINIMIZATION

Methods for minimizing a function subject to nonlinear constraints can be divided broadly into two classes, those which set up an equivalent unconstrained minimization problem by adding a penalty term to either the objective function or the Lagrangian function, and those which seek to generate a sequence of feasible descent steps (Sargent, 1974; Taha, 1982).

For nonlinearly-constrained problems, much of the terminology is almost the same as for the linearly-constrained case. At the solution, x^* (local minimum) of a nonlinearly-constrained problem, some of the constraints will hold as equalities, these constraints compose the set of active or binding constraints at the solution (NAG, 1978).

Assume that there are t active constraints at x^* and let $\hat{A}$ denote the matrix whose columns are the columns of A corresponding to the active constraints, with $\hat{b}$ the vector similarly obtained from b ; then

$$\hat{A}^T x^* = \hat{b}$$

The matrix is defined as an $n \times (n-t)$ matrix satisfying

$$\hat{A}^T Z = 0;$$

$$Z^T Z = I.$$

The columns of Z form an orthogonal basis for the set of vectors orthogonal to the columns of $\hat{A}$.

Gradient vector is described as the vector of first

partial derivatives of $F(x)$ and is denoted by $g(x)$, i.e.

$$g(x) = \left(\frac{\partial F(x)}{\partial x_1}, \frac{\partial F(x)}{\partial x_2}, \dots, \frac{\partial F(x)}{\partial x_n} \right)^T$$

Hessian matrix is the matrix of second partial derivatives of a function. The Hessian matrix of $F(x)$ is denoted by $G(x)$ and its $(i,j)^{th}$ element is given by;

$$\frac{\partial^2 F(x)}{\partial x_i \partial x_j}$$

If $F(x)$ has continuous second derivatives, then $G(x)$ must be positive semi-definite at any unconstrained minimum of F .

The projected gradient vector $g_z(x)$ is the vector $Z^T g(x)$. At the solution x^* of a nonlinearly-constrained problem, the projected gradient must be zero, which implies that the gradient vector $g(x^*)$ can be written as a linear combination of the columns of $\hat{A}$, i.e.

$$g(x^*) = \sum_{i=1}^t \lambda_i \hat{a}_i = \hat{A} \lambda$$

The scalar λ_i is defined as the Lagrange-multiplier.

A simple interpretation of the i -th Lagrange multiplier is that it gives the gradient of $F(x)$ along the i -th active constraint normal; a convenient definition of the Lagrangian-multiplier vector is :

$$\lambda = (\hat{A}^T \hat{A})^{-1} \hat{A}^T g(x^*).$$

The Lagrange function is given by ;

$$L(x, \lambda) = F(x) - \lambda^T \hat{C}(x)$$

g_L is defined as the gradient of the Lagrange function; $G_L(x)$ as its Hessian matrix and $\hat{G}_L(x)$ as its projected Hessian matrix, i.e.;

$$\hat{G}_L = Z^T G_L Z$$

Sufficient conditions for x^* to be a solution of a nonlinearly-constrained problem are :

- (i) x^* is feasible, and $b(x^*) = 0$;
- (ii) $\| g_z(x^*) \| = 0$ or equivalently, $g(x^*) = \hat{A}(x^*)\lambda$;
- (iii) $\hat{G}_L(x^*)$ is positive definite ;
- (iv) $\lambda_i > 0$ if λ_i corresponds to a constraint of the form $\hat{C}_i \geq 0$; the sign of λ_i is immaterial for an equality constraint. Note that condition (ii) implies that the projected gradient of the Lagrangian function must also be zero at x^* , since the application of Z^T eliminates the matrix $\hat{A}(x^*)$ (See NAG, 1978).

6.6.2. MODEL SOLUTION

To minimize the objective function (Equation 6.4) subject to nonlinear constraints, the Conmin algorithm developed by Fletcher and Powell was applied (See Appendix-1). Kuester and Mize (1973) give a listing of a program for the Conmin algorithm. The program is written in ASA FORTRAN IV. Having done some modifications, the program was run on CDC 855 at the Imperial College. Although minimization in the value of the objective function was successful, the lower constraints were violated.

Therefore a program was written to call a routine from the routines library at the Imperial College (Budd, 1986). The routine called E04UAF attempts to find a minimum of a function of several variables subject to fixed bounds on the variables and to general equality and/or inequality

constraints. A sequential Lagrangian method is used , no derivatives are required (NAG, 1978). The results obtained from the routine satisfied the constraints and the function value was minimized. The program was then developed for further calculations (Section 6.8).

6.7. SELECTION OF EQUIPMENT

Every item of production machinery and plant incurs costs from the moment it is acquired until its disposal to another user or sale for scrap. These costs may be classified as ;

a. Acquisition. Not only the purchase price of the machinery but also all additional costs associated with installation, testing and commissioning that may be necessary to work the machine up to its full operating efficiency, before it is surrendered to production.

b. Operating. These are the costs associated with using or operating the machine to provide necessary service.

c. Maintenance. These are the costs incurred in caring for the machine by appropriate service and repair actions.

d. Disposal. When a machine is to be sold, there will be costs associated with organizing and carrying out the removal and dispatch of the machine to the purchaser. The market price less these costs will constitute the disposal cost or, more usually, the disposal value - except in unusual circumstances, where the machine achieves only a scrap metal price which is insufficient to cover its removal costs (King, 1985).

The period of time spanning acquisition to disposal is called the machines life cycle. This is not necessarily the total life of the machine. The acquisition cost is incurred at the start of the life cycle whilst the disposal value arises at its end. Operating and maintenance costs are incurred throughout the life cycle. If these costs are discounted to present value form, using the company's cost of capital as the discount rate then the result is referred to as the life-cycle cost for the machine. This can be reduced to an equivalent annual cost over the life cycle of the machine and is then termed the equivalent annual cost (King, 1985).

The equivalent annual cost is important because it allows cost comparisons to be made between the machines over unequal life cycle times. In such a case, the equipment with the lowest equivalent annual cost will be preferred.

In most tunnelling operations, the drilling and mucking cycle is carried out by a combination of one drilling machine with one mucker. Therefore, the objective is to find the best combination of these machines that will have minimum cost and meet a prescribed advance rate. So the question is how to calculate equivalent annual costs of the machines and be able to calculate the cost of operation.

In practice it is usual to separate the costs of financing the operation (ownership costs) from the operating costs. The reasons for the separation are only technical, as the outflow of cash is independent of the use or purpose to which it is put (Morgan, 1969).

6.7.1. METHODS OF ECONOMIC ANALYSIS

The method of economic analysis will depend on the type of operation under consideration. If the operation is such that all equipment will be kept in service for its full service life, a system expressing the total costs in terms of an annual average can be used.

There are three commonly used methods:

- a. the annual cost method
- b. the present worth method
- c. the rate of return method (Morgan, 1969; Riggs, 1968).

The first two methods compare costs or revenue on the basis of a stipulated minimum acceptable return on invested capital. The alternative with the lowest equivalent annual cost is the choice. The third method compares alternatives on the basis of the percentage return on increments of investment. The amount that an alternative exceeds a minimum standard return with an available investment source, is the criterion for choice.

In TUNOPR, the annual cost method is used. With this method all the receipts and disbursements occurring over a period of time are converted to an equivalent uniform annual charge. It is a popular method because of the wide spread inclination to view a year's gains and losses as a criterion of progress (Riggs, 1968). This method is suitable for constant investments with uniform savings and readily understandable; but it is unsuitable for projects with

large variations in annual cost.

Ownership costs

Capital recovery factor (CRF) is used to determine the amount of each future annuity payment required to accumulate a given present value when the interest rate and number of payments are known.

$$CRF = \frac{i(1+i)^n}{(1+i)^n - 1} \quad (6.13)$$

$$ACC = (PP - Sv).CRF + Sv.i \quad (6.14)$$

where

CRF = capital recovery factor

ACC = annual cost of capital recovery

i = interest rate on investment or required return
on capital

PP = purchase price

Sv = salvage value

n = plant life or study period

The capital recovery (Equation 6.14) gives the total ownership costs of the facility taking into account the following factors:

1. service life of facility
2. salvage value of proposed facility
3. depreciation
4. installed cost
5. investment expenses
6. interest rate on capital

6.7.2. COST OF JUMBO UTILIZATION

Costs of operating a drill jumbo can be classified into three principal categories of ownership costs, operating costs and maintenance costs.

1. The ownership costs include the purchase price, the shipping costs to the mine site, the transportation to the working site, the interest on invested capital, taxes, insurance, equipment depreciation and other overhead costs (Roos, 1982).

2. The operating costs include cost of power (electricity, fuel or compressed air), operating and supervisory labour and consumable items, drill bits, drill steel, fuel, oil, etc.

3. The maintenance costs include labour, maintenance supplies, the replacement parts, the minor and major overhaul parts and labour.

All three cost categories must be examined to determine the cost - benefit performance of a jumbo.

a. Estimation of Jumbos ownership cost

For example a 2-boom drill jumbo is bought to excavate a drift in an underground mine.

Assumptions

Total delivered price	\$ 240000.00
Tyre costs	\$ 4500.00
Expected salvage value	\$ 16000.00
Service life (n)	4 years
Number of hours, jumbo expected to work	4000 hr/yr
Interest rate (%)	17

From Equation 6.15, annual cost can be found as:

$$CR = ((240000 - 4500) - 16000) 0.364 + 1600 0.17$$

$$CR = 82735 \text{ \$/yr.}$$

The ownership cost per hour is:

$$OCJ = \frac{82735}{4000} = 20.6 \text{ \$/hr}$$

Operating and maintenance costs

The operating and maintenance costs of a drill jumbo are calculated on the basis of the length of the drilled holes.

The operation of a typical development jumbo per round can be divided into the following increments:

- tramming the jumbo to the drilling face
- aligning and levelling the jumbo
- connecting supply hoses or cables
- testing the air, water, electric and hydraulic circuits
- inspecting the face for bootlegs, loose rock and obstructions
- positioning the booms, feeds and drills
- extending the feed against the face
- collaring the holes
- drilling the holes
- retracting drill steels
- stowing the booms for tramming
- raising the levelling jacks
- disconnecting the supply hoses
- starting the tramming engine

- tramming the jumbo to the temporary storage area

Roos (1982) gives some values in terms of time (min/round) for each increment listed above.

Labour

cost of labour = wage of an operator x drilling time

where

$$\text{drilling time} = \frac{n_h \cdot H}{60.0 \cdot P_{rt} \cdot n_b}$$

n_h = number of holes

H = hole depth (m)

P_{rt} = penetration rate (m/min)

n_b = number of booms

Power supply = Power x drilling time x cost of electricity

Consumable parts

drill steel = (cost / life (drilled metre)) x n_h x H

couplings = (cost/life) x n_h x H

shanks = (cost/life) x n_h x H

bits = (cost/life) x n_h x H

Replacement parts

Drill steels, feeds and boom parts are replaced during ongoing maintenance, average wage x total drilling time (Roos, 1982).

Maintenance labour

Preventive and emergency maintenance at:

wages of operator x 0.5 hr per round (Roos, 1982).

Minor overhaul (2000 hr)

Approximately 8% of purchase price per 2000 working hour

Major overhaul (12000)

Approximately 30% of the purchase price per 12000 working hour.

The total operating and maintenance cost of a drill jumbo (OPJUM) is the sum of these elements. Then the jumbos total cost per hour can be calculated as:

$$JC = (OPJUM/TDT) \cdot 60.0 + OCJ \quad (6.15)$$

where

JC = total cost of a drill jumbo (\$/hr)

OPJUM = operating and maintenance cost (\$/hr)

TDT = total drilling time (min)

OCJ = ownership cost (\$/hr)

The calculated value of JC (cost of jumbo including the operator) will form one of the coefficients of the objective function (See Equation 6.4).

The same calculations are used to estimate the total cost of using Load-Haul-Dump Units (LHDs) as muckers. After computing the number of holes, hole depth, cut hole diameter, total drilling time and the quantity of blasted material each round, total mucking time and cost from total drilling, blasting and mucking operations are calculated. In this process several types of jumbos and LHDs that can work in the given area, are examined and the best combination is found. The next step is to examine some parameters for rail transportation, such as: estimation of the ideal gradient, and the best combination of locomotive and mine cars among the ones whose specifications are input.

In the design of auxiliary ventilation, the quantity

of air required at the face is calculated depending on the given tunnel area, rock conditions and selected equipment. Having found the air quantity, input fan types and ducts are examined to find the best combination to satisfy the duty of minimum cost.

6.8. DESCRIPTION OF THE MODEL, TUNOPR

A generalized flowchart of the model TUNOPR is illustrated in Figure 6.1. TUNOPR is a FORTRAN 77 program constructed to minimize the cost of a drilling and blasting round subject to some commonly used constraints: bit diameter, hole length, cut hole diameter, etc., to estimate costs of jumbos and LHDs for the optimum combinations and to calculate air requirement at the face.

The program consists of one main program and six subroutines (ENTER, VALUES, COSTS, OPNAG, RAIL, VENTIL, HEAT).

6.8.1. SUBROUTINE ENTER

This subroutine reads a data file created by the user to input the specifications of the operation and different equipment. The model examines three types of jumbos (1, 2 and 3-boom drill jumbos) and five types of different capacities of LHDs. So delivered prices, salvage values, interest rate, life of equipment, etc. are input for the financial considerations.

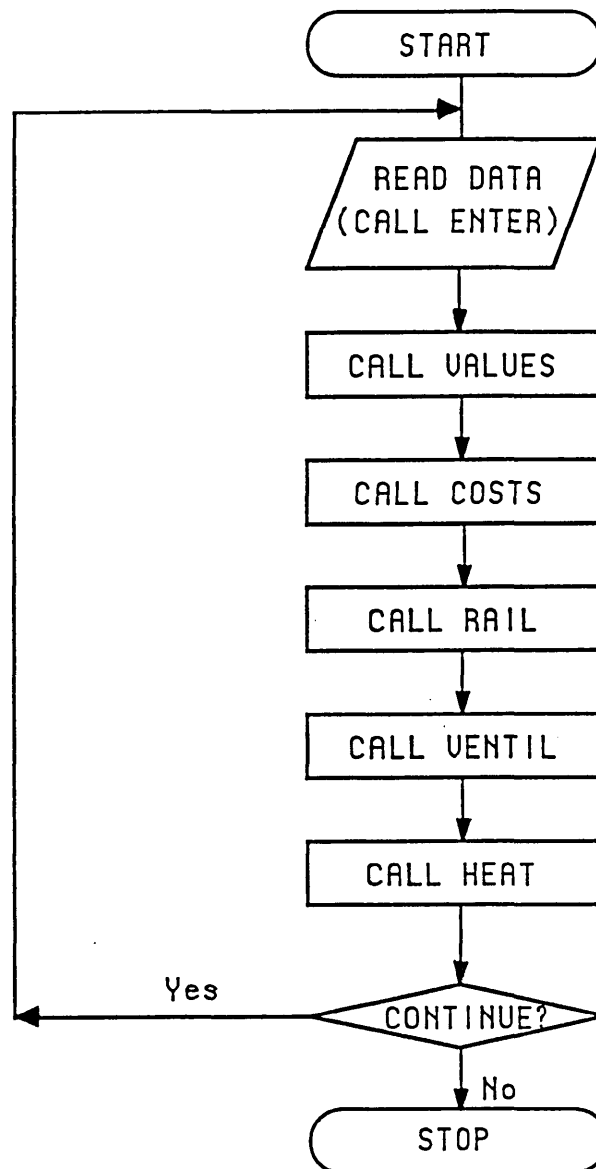


Figure 6.1 A generalized flowchart of the model TUNOPR

For examination of locomotives and mine cars, the numbers of locomotive types and mine cars are subject to user selection and require the provision of specifications with prices to be entered for each system. The same thing applies to the ventilation phase of the operation. The number of fans, ducts, different diameters, together with their prices, lives, and specifications are input.

6.8.2. SUBROUTINE VALUES

Some data is carried from this routine to the main program. The specifications of three types of jumbos (See Table 2.3) and five types of LHDs (See Table 3.1) are stored here. According to the input parameters, values are selected and carried for further calculations. For example, for the type of drilling either percussive or rotary, expected penetration rate, equipment life, such as drill bits, shanks, couplings, etc., are determined and taken to the main program.

6.8.3. SUBROUTINE COSTS

After reading a data file which contains all the input values, the subroutine COSTS is called to compute equipment costs. An example input file is given Appendix-3. It is necessary to assume initial values for the number of holes, bit diameter and cut hole diameter in order to calculate operating costs of different jumbos just before the optimization routine OPNAG is called (See Equation 6.4). Here, three types of jumbos and five types of LHDs are examined to find the best combination.

In the routine, DATSE1 and DATSE2 are two subroutines

containing input data for the optimization routine. Since total cost of a drill jumbo per hour in a drilling round is a coefficient of the objective function, optimum values should be used to calculate the actual operating costs of the jumbos. This recursive procedure is repeated until the input and calculated values of the optimization routine converge.

Having estimated optimum results for a drilling and blasting round and costs of LHDs, the jumbo and LHD types that could be used in the given area are chosen for examination of different combinations. Obviously each combination would have different round length according to their productivity. Therefore, number of rounds in a day and daily advance rate would be different for each combination. The question then arises can stipulated advance rate per day be achieved by any of the combinations at a lower cost? Consequently, advance rate needs to be examined and it forms an equality constraint for the solution of the objective function (Equation 6.4). As a result, the number of decision variables increases from five to six. In the program this is called second step optimization. Estimated values of cost of operation per round and round length are examined and the combination that has minimum cost and shortest round is chosen. In the design calculations of rail transportation and ventilation, calculations are carried out for the chosen combination of jumbo and LHD (See subroutine RAIL and VENTIL). Figure 6.2 illustrates a generalized flowchart of the routine COSTS.

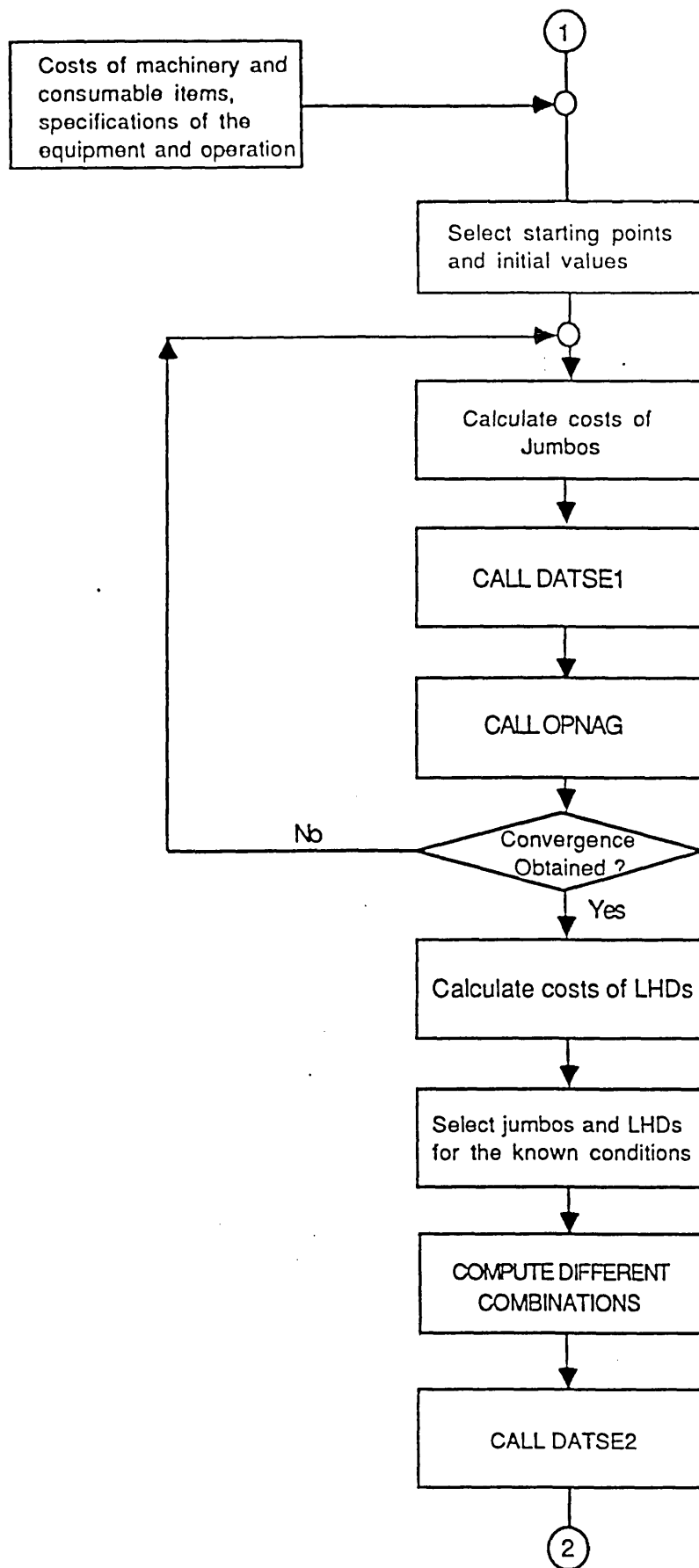


Figure 6.2 Flowchart of the subroutine COSTS

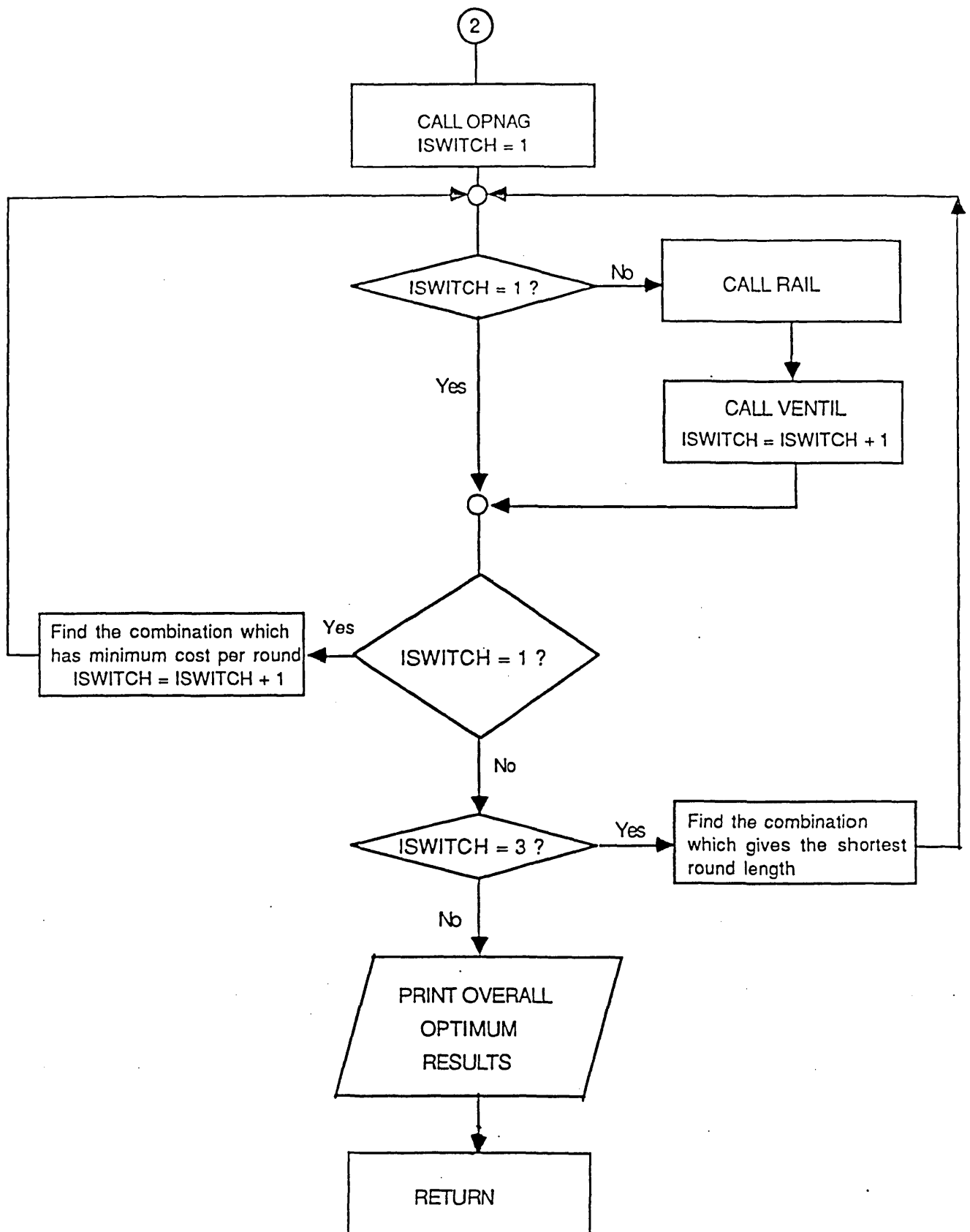


Figure 6.3 Flowchart of the subroutine COSTS (continued)

6.8.4. SUBROUTINE OPNAG

This subroutine is constructed to call the routine named E04UAF from the routine library at the Imperial College (Budd, 1986).

E04UAF attempts to find a minimum of a function of several variables subject to fixed bounds on the variables and/or to general equality constraints. The routine is intended for functions and constraints which have continuous first and second derivatives (although it will usually work even if the derivatives have occasional discontinuities) (NAG, 1978).

Apart from E04UAF, OPNAG contains three more subroutines supplied by the user. FUNCT1 is a routine to calculate the value of the objective function (Equation 6.4). CON1 is supplied to calculate the values of the constraint functions. The third routine AMONIT, monitors the minimization process. All three subroutines are called within the subroutine E04UAF.

6.8.5. SUBROUTINE RAIL

This routine estimates ideal gradient of the tunnel and calculates the required tractive effort for the locomotive and car combinations whose specifications are supplied by the user. The quantity of the material carried each round is calculated for the combination suggested in the subroutine COSTS. After calculating the costs of locomotives, mine cars and the number of cars, cost of transportation and travelling time are computed for every

100 m of advance. Then the best combination of locomotive and mine car is chosen and carried to the subroutine COSTS.

6.8.6. SUBROUTINE VENTIL

Vutukuri and Lama (1986) developed a computer program in Basic for auxiliary ventilation calculations. The program is based on the theory of airflow in leaky ducts with a uniformly distributed leakage. It calculates duct length for known air quantity reaching the face and fan data (fan characteristics). It can also estimate the volume rate of the fan and pressure developed by the fan.

When driving long tunnels, because of increasing resistance of the duct and increasing leakage (Chapter 5), the total fan power required, depending on the fan type, may exceed the capacity of a single installation. Then the question that needs to be answered is where the second fan should be installed? Therefore Vutukuri's program was modified to calculate the required number of fans for the given tunnel length. The subroutine also computes installed power for ventilation, power costs, pipe costs and fan costs in order to select the best combination of fans and ducts. Three types of ventilation ducts; metal, flexible and flexible wire-reinforced are examined here. In the calculations, the following coefficients of friction factor were employed: for metal pipe is $0.001862 \text{ N s}^2/\text{m}^4$, for flexible pipe is $0.002795 \text{ N s}^2/\text{m}^4$ and for flexible wire-reinforced pipe is $0.00373253 \text{ N s}^2/\text{m}^4$ (Bossard, 1983a). Browning (1983) lists leakage coefficients for 610 mm in diameter three types of ducts as follow:

leakage coefficient of metal pipe = $0.14 \text{ m}^3/\text{s}/100 \text{ m}$ at 1 kPa

leakage coefficient of flexible pipe = 0.16

leakage coefficient of flexible wire-reinforced pipe = 0.35

Subroutine VENTIL uses a routine called SUBDUC that estimates coefficients of the second degree polynomial fan curve ($P=aQ^2+bQ+c$) in order to calculate the point where system resistance ($P = R Q^2$) curve intersects the fan characteristic curve.

6.8.7. SUBROUTINE HEAT

The subroutine HEAT is adopted from the program developed by Whillier and Thorpe (1971). The program is described in Research Report No 6/75 (Lawrance and Whillier, 1975). Given the perimeter and overall length of development, the virgin rock temperature, barometric pressure and volume flow rate of the inlet air, as well as the face advance rate, the program calculates the air temperature, humidity and air quantity reaching the face at specified stages of the development.

Whillier and Thorpe (1971) considered three degrees of footwall wetness.

a. 'Bone-dry' ($f=0.0$). The footwall is completely dried out, an ideal situation.

b. 'Practically-dry' ($f=0.4$). A situation typically obtained when efforts have been made to dry out the footwall as development progresses.

c. 'Wet' ($f=1.0$). No attempt has been made to dry out the footwall.

The leakage or short circuiting of air is expressed as the percentage volume of air lost into the exhaust duct per metre of development; the broad classifications being:

- 'no leakage' : zero air loss
- 'small leakage' : 0.03% loss per metre
- 'average leakage' : 0.09% loss per metre
- 'large leakage' : 0.15% loss per metre.

Output parameters are listed at one-tenth intervals of the final development length. In the case of tunnels longer than 450 m, the first interval is further broken down into thirtieths. Output is suspended in cases where the dry-bulb approaches to within 0.1°C of the virgin rock temperature (since heat transfer is negligible) or in case where the wet-bulb temperature rises above 36°C. For a more complete discussion, refer to psychrometry and heat transfer processes texts (eg., Hartman, 1982a).

As mentioned in Chapter 5, in a development end air reaching the face is affected by some parameters such as type of duct, diameter of duct, length of duct, etc. In this program the authors do not examine the leakage as a function of the above parameters. For example, when calculating duct length in the subroutine VENTIL, while leakage is approximately 0.2% per metre in a 75 cm diameter duct, the leakage is approximately 0.34% per metre when the diameter of the duct is 1.0 m. Consequently, even if one assumes any of the leakage type (small, average or large), actual leakage will differ with the type of duct and diameter of duct. Therefore, the output given by the

subroutine HEAT may not give accurate estimations in the calculations of auxiliary ventilation.

6.9. THE OUTPUT OF THE MODEL

The output of the model is written into three files called INTER, VENT and RESULT. The output of the optimization routine OPNAG is written into INTER. Output file VENT contains input data for ventilation calculations and calculated values for each fan-duct combination. The overall results of the model are given in RESULT.

CHAPTER SEVEN

MODEL VALIDATION AND APPLICATIONS

7.1. INTRODUCTION

Definition of the problem, construction of the model and model solution are followed by model validation in Operation Research studies. In this stage, the aim is to test how accurately the model represents reality. Validation is usually accomplished by systematic computer runs that should highlight defects in the logical construction of the model. These tests ensure that all of the model's parts are sound and there are no programming errors.

A model is valid if, despite its inexactness in representing the system, it can give a reasonable prediction of the systems performance. A common validation test of a model is to compare its performance with some past data available for the actual system. The model will be valid if under similar input conditions it can produce the past performance of the system. The problem is that there is no assurance of future performance duplicating past history. Also, since the model is based on careful examination of the past data, the comparison should always reveal favourable results (Taha, 1982).

In this Chapter, for validation of the model, a tunnel which was excavated by conventional drilling and blasting method in Alaska is examined as a case study.

7.2. THE TYEE LAKE HYDROELECTRIC PROJECT

The Alaska Power Authority, Tyee Lake Project is a 30 MW size hydroelectric development utilizing a static head of approximately 425 m provided by Tyee Lake. An above-ground power house is located at the penstock tunnel portals (slightly above sea level) at a distance of 2.3 km from the lake. The project is located in Southeast Alaska near Wrangell (Provost and Ness, 1983).

7.2.1. DESCRIPTION OF THE UNDERGROUND CONSTRUCTION

The underground work consists of 4 main components as shown in Figure 7.1.

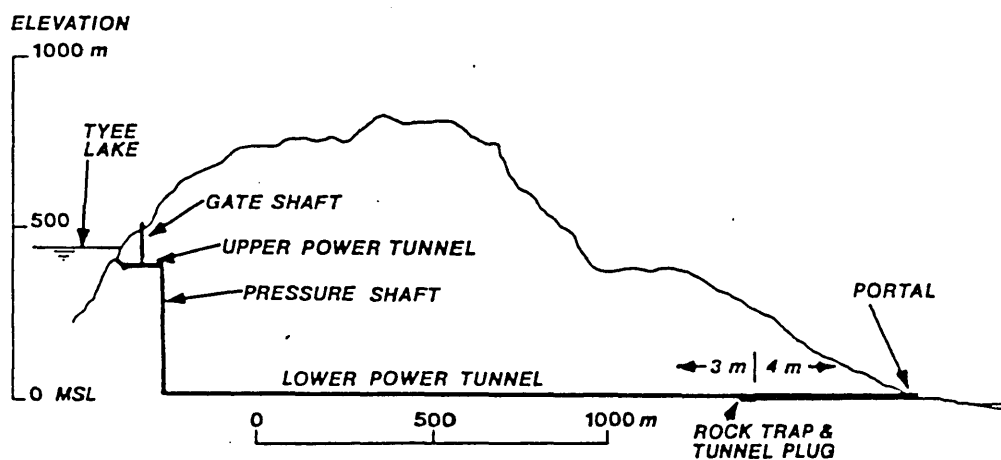


Figure 7.1 Tunnel elevation showing the 4 major components of the underground work at the Tyee Lake

a. The lower power tunnel with approximately 370 m of 4 x 4 m horseshoe tunnel and 1700 m of 3 x 3 m horseshoe tunnel to be excavated in quartz-diorite.

b. The vertical pressure shaft is an approximately 340 m of 3 m diameter raise.

c. The upper power tunnel with approximately 128 m in length by 3.7 m diameter with enlargements at the shaft stations.

7.2.2. ACTUAL OPERATION

The following "major" pieces of tunnelling equipment were used in the construction of the lower power tunnel (Provost and Ness, 1983).

- 1 1981 Tamrock Minematic R2EH, 2-boom jumbo.
- 1 1981 Haggloader 8HR E1 with 4 m wide blades.
- 3 1982 Hagglund HRST-115 CLE/CTE shuttletrains.
- 1 Plymouth DMD24 - 12 ton locomotive
- 1 Plymouth DMD30 - 15 ton locomotive

The tunnel pay line and shuttletrain profile restricted the size of the ventilation pipe, which in turn restricted the size of diesel locomotive and therefore the number of shuttletrain cars that could be transported.

7.2.3. PERFORMANCE AND RESULTS

High rates of advance were achieved under favourable conditions. The overall advance for the power tunnel was approximately 10.7 m/day. The advance rate for 4 m section averaged 10 m/day and 3 m section averaged 10.8 m/day.

In each round, 54 holes were drilled with a depth of 3.2 m; the duration of total round was approximately 362 min., 144 min of which was spent on drilling the holes. It took 22 min to fill three cars.

Provost and Ness (1983) state that the size of hauling and mucking equipment available did not ideally match the tunnel dimensions. The mucking cycle, including in and out, took approximately 100 min at the midpoint of the learning curve and was not reduced considerably because of the increased haul distances.

7.3. MODEL SOLUTION

The program was run for the values of the first part of the lower power tunnel that is 4 x 4 m horseshoe tunnel and 370 m long. Figure 7.2 illustrates the cross section of the area.

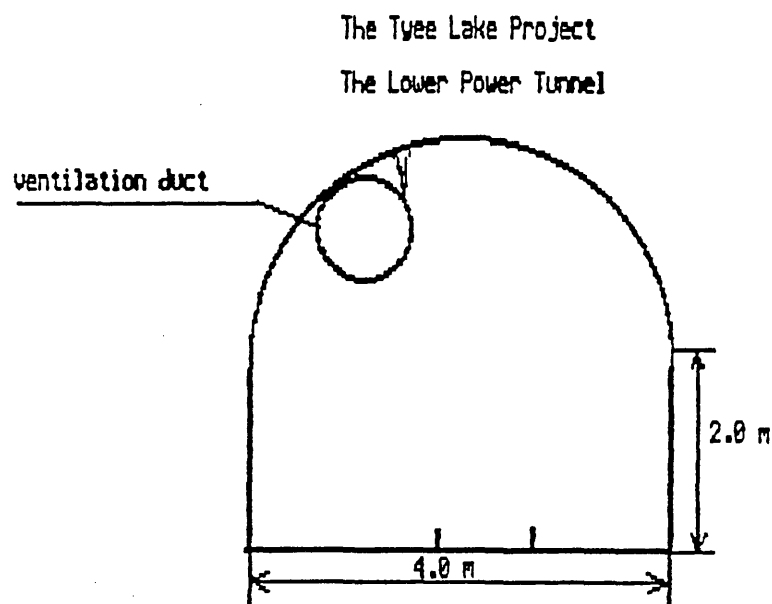


Figure 7.2 Cross section of the lower power tunnel

The compressive strength of the rock was assumed to be approximately 220 MPa from the Deere-Miller Rock Classification (See Table 2.1). Since the rock compressive strength is quite high, percussive drilling was examined

due to its higher penetration ability in rock of this type (See Table 2.2).

Three types of drill jumbos manufactured by the Tampella Ltd. Tamrock were examined: Monomatic (1-boom), Minimatic (2-boom) and Paramatic (3-boom) electro/hydraulic jumbos (Tamrock, 1983).

The type of LHD was chosen from among those whose specifications are given in Table 3.1 and 3.3. In selecting a jumbo or an LHD, as mentioned in the earlier chapters, the size of the area is particularly significant. Figure 7.3 illustrates the range of the area in which different jumbos and LHDs operate.

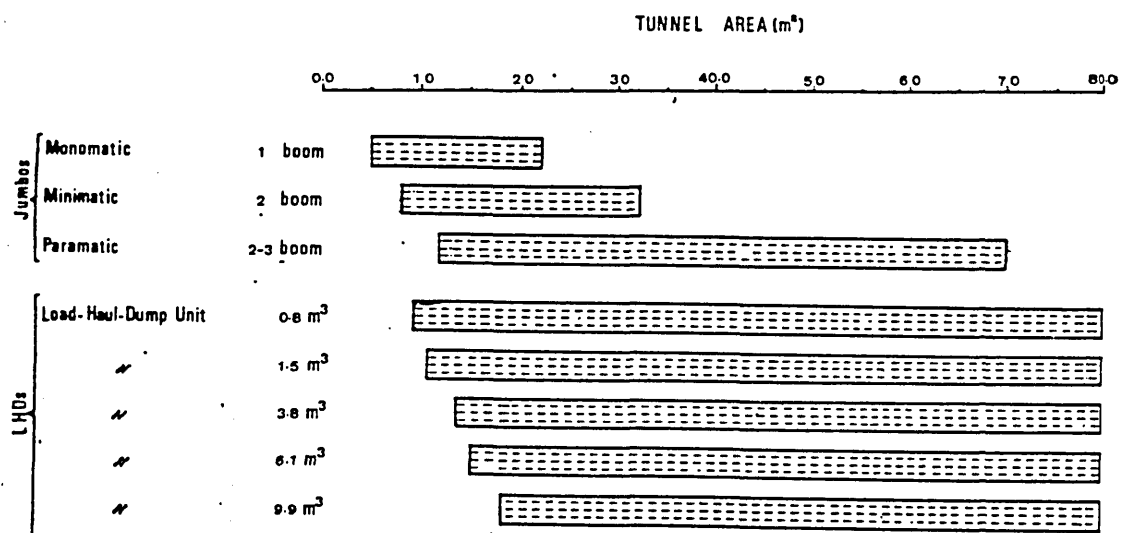


Figure 7.3 Range of areas for different jumbos and LHDs.

The types of equipment that work in the given area can be found from Figure 7.3. For example, Monomatic jumbo (1-boom) could cover areas from 5 to 22 m². Jumbos are compared according to their number of booms while LHDs are

examined according to their bucket sizes.

Three sizes of diesel locomotives (3,7 and 10 ton) were compared with three types of mine cars (2,3 and 4 m³).

Three types of auxiliary fans, the characteristics of which are illustrated in Figure 7.4, were examined with three types of ventilation ducts, metal, flexible and flexible wire-reinforced form for three different sizes (500, 750 and 1000 mm).

The rest of the input values such as purchase prices, lives, salvage values of the machines to be compared, cost of consumable items, etc., are given in Appendix-2. All the cost figures given in the Appendix are assumed values.

The output of the model for the first part (4 x 4 m) of the lower power tunnel is given in Table 7.1 to 7.8.

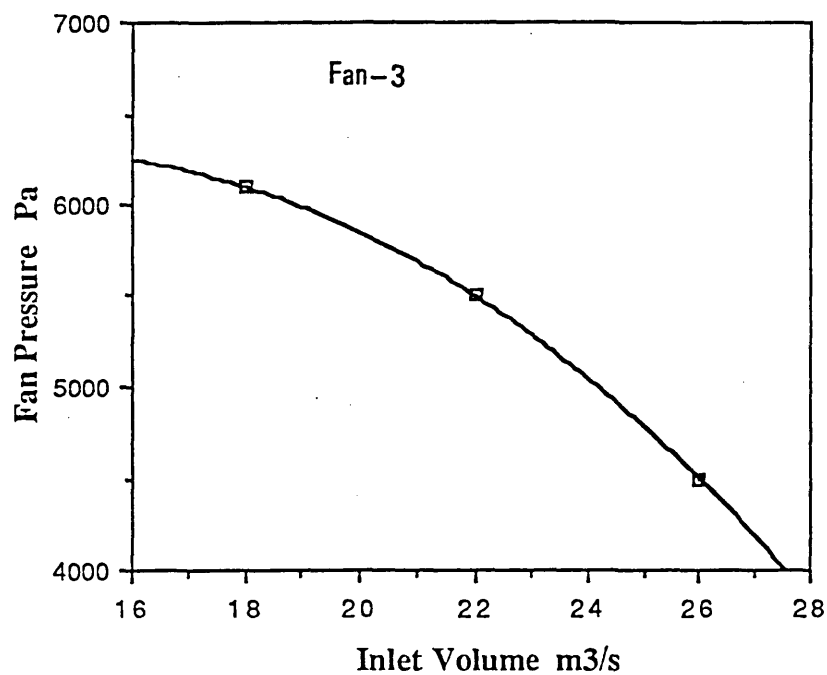
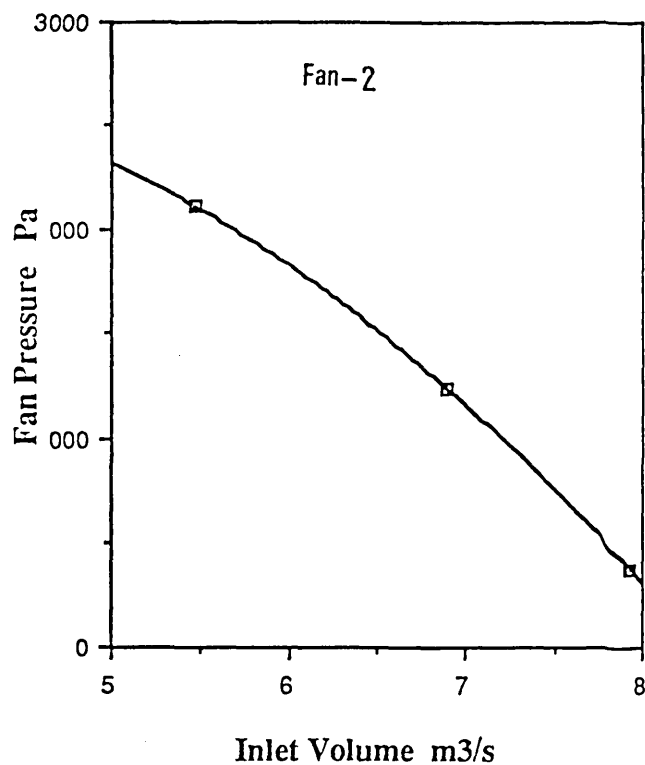
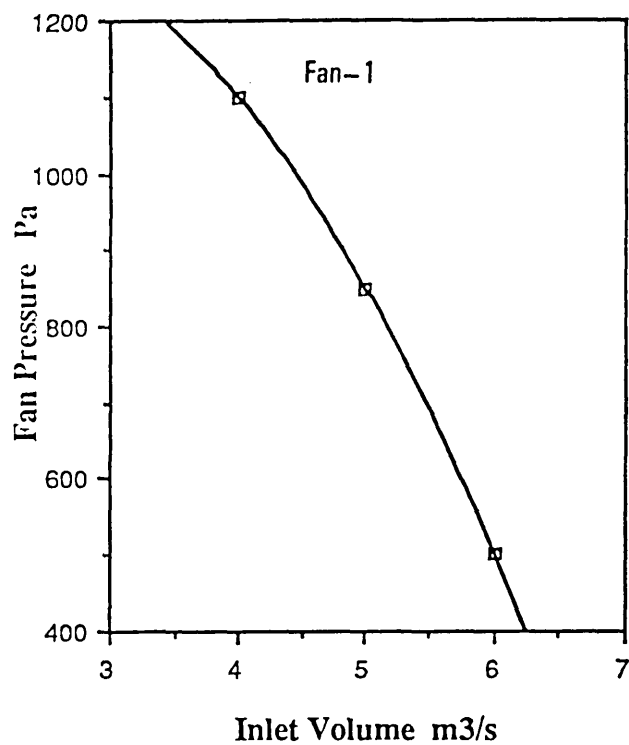


Figure 7.4 Characteristic curves of three auxiliary ventilation fans

Table 7.1 Output of the model for the Tyee Lake Project

DATE : 87/10/15. TIME : 13.20.02.

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===== DRIFTING / TUNNELLING IN UNDERGROUND MINES =====
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===== T . T. OZAN =====
===== RESEARCH ENGINEER =====
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NAME OF THE PROJECT = THE TYEE LAKE PROJECT
TUNNEL AREA = 14.28 SQMT
TUNNEL LENGTH = 370.00 M.
COMPRESSIVE STRENGTH = 220.00 MPA.
RATE OF ADVANCE AIMED TO ACHIEVE = 10.00 M / DAY .
DRILL TYPE = PERCUSSIVE

	NO OF BOOMS	PURCHASE PRICE(\$)	TYRE COSTS(\$)	SERVICE LIFE(YR)	OPERATING HOURS(YR)	RESALE VALUE(\$)
JUMBO(1)	1	170000.00	3000.00	4	4500.00	10000.00
JUMBO(2)	2	240000.00	4500.00	4	4000.00	16000.00
JUMBO(3)	3	450000.00	6000.00	5	4000.00	17000.00

INTEREST RATE % = 17.00

BUC.CAP. M3

LHD(1)	.80	56207.00	3000.00	5	4000.00	5000.00
LHD(2)	1.50	64718.00	3500.00	5	4200.00	7500.00
LHD(3)	3.80	119497.00	4500.00	6	4500.00	10000.00
LHD(4)	6.10	288943.00	6500.00	6	4700.00	13000.00
LHD(5)	9.90	569460.00	7000.00	7	5000.00	25000.00

Table 7.2

OPTIMUM RESULTS FOR DRILLING AND BLASTING ROUND

	NUMBER OF HOLES	BIT DIA (MM)	CUT HOLE DIA (CM)	DEPTH OF HOLE (M)	CHARGE PER HOLE (G)
JUMBO (1)	49	29.000	8.000	2.200	1400.000
JUMBO (2)	49	29.000	8.000	2.200	1400.000
JUMBO (3)	49	29.000	8.000	2.200	1400.000

LENGTH OF DRILLING ROUND (MIN)

JUMBO (1)	638.803
JUMBO (2)	327.735
JUMBO (3)	223.150

OPERATING COSTS (\$/HR)

JUMBOS	LHDS
JUMBO (1)	23.566 LHD (1)
JUMBO (2)	31.473 LHD (2)
JUMBO (3)	49.155 LHD (3)
	LHD (4)
	LHD (5)

ANNUAL COST \$ / YEAR

JUMBO (1)	58931.699	LHD (1)	15917.766
JUMBO (2)	82735.018	LHD (2)	18065.306
JUMBO (3)	136354.770	LHD (3)	30953.718
		LHD (4)	77280.808
		LHD (5)	141273.945

ESTIMATION OF TOTAL COST (\$/HR)

JUMBO (1)	36.662	LHD (1)	22.121
JUMBO (2)	52.157	LHD (2)	24.874
JUMBO (3)	83.244	LHD (3)	33.333
		LHD (4)	54.480
		LHD (5)	79.256

COMBINATION OF JUMBOS AND LHDS

	COST PER ROUND (\$)	LENGTH OF ROUND (MIN)	NUMBER OF CYCLES	MUCKING TIME (MIN)
JUMBO (1) - LHD (1)	659.621	737.703	43	98.90
JUMBO (1) - LHD (2)	645.088	691.703	23	52.90
JUMBO (1) - LHD (3)	634.658	659.503	9	20.70
JUMBO (2) - LHD (1)	524.318	426.635	43	98.90
JUMBO (2) - LHD (2)	509.786	380.635	23	52.90
JUMBO (2) - LHD (3)	499.355	348.435	9	20.70

Table 7.3

SECOND STEP OPTIMIZATION WHERE MINIMUM DAILY ADVANCE RATE IS TAKEN AS A DECISION VARIABLE.

COMBINATION	NO OF HOLES	BIT DIAMETER	CUT HOLE	HOLE DEPTH	CHARGE PH.	ROUND/DAY	ADVANCE (M/D)
JUMBO (1) -LHD (1)	49	29.00	8.00	2.11	1400.00	2	4.01
JUMBO (1) -LHD (2)	49	29.00	8.00	2.11	1400.00	2	4.01
JUMBO (1) -LHD (3)	49	29.00	8.00	2.11	1400.00	2	4.01
JUMBO (2) -LHD (1)	49	29.00	8.00	2.11	1400.00	3	6.02
JUMBO (2) -LHD (2)	49	29.00	8.00	2.11	1400.00	3	6.02
JUMBO (2) -LHD (3)	49	29.00	8.00	2.11	1400.00	3	6.02

RAIL TRANSPORTATION

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INPUT VALUES

	LOCO WEIGHT	PURCHASE PRICE	LIFE (YR)	SALVAGE VALUE	TIME WORKED (HR/YR)	GAUGE (MM)
LOCOMOTIVE (1)	3.00	55000.00	3	2000.00	4000.00	609.60
LOCOMOTIVE (2)	7.00	70000.00	3	3000.00	4500.00	609.60
LOCOMOTIVE (3)	10.00	95000.00	4	3000.00	4000.00	609.60

	WEIGHT (TON)	CAPACITY (M3)	PURCHASE PRICE (\$)	LIFE (YR)	SALVAGE VALUE (\$)	TIME WORKED (HR/YR)	GAUGE (MM)
CAR (1)	2.50	2.00	2390.00	4	.00	3000.00	609.60
CAR (2)	3.00	3.00	3390.00	4	.00	3000.00	609.60
CAR (3)	4.20	4.00	4100.00	4	.00	3500.00	609.60

COST OF OPERATOR (\$/HR) = 6.00

ROLLING RESISTANCE, KG/T = 9.08

QUANTITY OF MATERIAL TO BE CARRIED (TON) = 59.69

DENSITY OF BROKEN ROCK (T/M3) = 2.00

CALCULATIONS

COMBINATIONS	IDEAL GRADE (%)	NUMBER OF CAR	TRACTIVE EFFORT KG FULL	TRACTIVE EFFORT KG EMPTY	RETARDING FORCE KG FULL	THEORETICAL TRAC.EFF KG (*)	PERCENT TRAC.EFFORT USED	PERCENT TRAC.EFFORT AVAILABLE
LOCO (1) - CAR (1)	.22	15	714.16	447.68	1398.20	780.00	91.56	8.44
LOCO (1) - CAR (2)	.24	10	638.48	372.72	1248.31	780.00	81.86	18.14
LOCO (1) - CAR (3)	.24	8	695.64	411.57	1361.80	780.00	89.18	10.82
LOCO (2) - CAR (1)	.21	15	748.40	489.42	1437.26	1820.00	41.12	58.88
LOCO (2) - CAR (2)	.24	10	672.71	415.32	1288.13	1820.00	36.96	63.04
LOCO (2) - CAR (3)	.23	8	729.85	454.01	1401.46	1820.00	40.10	59.90
LOCO (3) - CAR (1)	.21	15	774.15	520.53	1466.46	2600.00	29.77	70.23

Table 7.4

LOCO(3) - CAR(2)	.23	10	698.46	447.02	1317.85	2600.00	26.86	73.14
LOCO(3) - CAR(3)	.23	8	755.59	485.61	1431.08	2600.00	29.06	70.94

(*) THEORETICAL TRACTIVE EFFORT IS ASSUMED TO TO BE 26% OF THE LOCO WEIGHT
AND LOCOMOTIVE SHOULD BE ABLE TO PRODUCE AT LEAST 15% MORE TRACTIVE EFFORT FOR ACCELERATION.

COMBINATIONS	ACCELERATION FULL M/S2	ACCELERATION EMPTY M/S2	DISTANCE TO ACCEL. M	DISTANCE TO STOP M
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LOCO(1) - CAR(1)	.136	.21	28.27	18.29
LOCO(1) - CAR(2)	.132	.21	29.30	18.03
LOCO(1) - CAR(3)	.133	.21	29.05	18.03
LOCO(2) - CAR(1)	.135	.20	48.32	32.46
LOCO(2) - CAR(2)	.130	.20	50.05	32.32
LOCO(2) - CAR(3)	.131	.20	49.61	32.16
LOCO(3) - CAR(1)	.134	.19	64.87	44.65
LOCO(3) - CAR(2)	.129	.19	67.15	44.72
LOCO(3) - CAR(3)	.130	.20	66.53	44.36

LOCOMOTIVE(2) - CAR(3)

DISTANCE (M)	TRAVEL TIME (MIN/ROUND)		COST OF TRANSPORT (\$/ROUND)	POWER OF LOCOMOTIVE (KW)
	FULL	EMPTY		
100.00	.92	.76	.84	32.30
200.00	1.38	1.22	1.30	32.30
300.00	1.84	1.68	1.76	32.30

COST OF RAIL AND TRACK MATERIAL (\$/M) = 70.00

AUXILIARY VENTILATION CALCULATIONS

INPUT VALUES

NUMBER OF DIFFERENT TYPES OF FANS TO BE EXAMINED = 3
NUMBER OF POINTS ON FAN CHARACTERISTIC CURVE = 3

TYPE OF FAN = 1

Q (M3/S)	P (PA)
4.00	1100.00
5.00	800.00
6.00	500.00

Table 7.5

TYPE OF FAN = 2

Q(M3/S)	P (PA)
5.47	2115.20
6.89	1244.00
7.93	373.26

TYPE OF FAN = 3

Q(M3/S)	P (PA)
18.00	6100.00
22.00	5500.00
26.00	4500.00

TYPES OF DUCT TO BE EXAMINED = METAL , FLEXIBLE , WIRE REINFORCED FLEXIBLE
NUMBER OF DIFFERENT DIAMETER OF DUCTS = 3

DUCT TYPE 1

COST OF DUCT (\$ / M) = 10.00
EXPECTED LIFE (YEAR) = 2
FRICTION FACTOR (NS2/M4)= .001863

DUCT TYPE 2

COST OF DUCT (\$ / M) = 15.00
EXPECTED LIFE (YEAR) = 2
FRICTION FACTOR (NS2/M4)= .002795

DUCT TYPE 3

COST OF DUCT (\$ / M) = 16.00
EXPECTED LIFE (YEAR) = 2
FRICTION FACTOR (NS2/M4)= .003725

DIAMETER (M) = .50
DIAMETER (M) = .75
DIAMETER (M) = 1.00

FAN TYPE 1	PRICE (\$)	=	2000.00	LIFE (YEAR)	=	2
FAN TYPE 2	PRICE (\$)	=	3000.00	LIFE (YEAR)	=	2
FAN TYPE 3	PRICE (\$)	=	4500.00	LIFE (YEAR)	=	3

FAN TYPE = 3
DUCT TYPE = 1
DUCT DIAMETER = .50 M.

ADVANCE (M)	DUCT LENGTH (M)	NO.OF FANS	TOTAL POWER (KW)	TOTAL FAN COST (\$/HR)	COST OF DUCT (\$/HR)
208.8	183.8	1	81.54	4.55	.16
370.0	345.0	2	165.53	9.21	.31

Table 7.6

FAN TYPE = 3
 DUCT TYPE = 1
 DUCT DIAMETER = .75 M.

ADVANCE (M)	DUCT LENGTH (M)	NO.OF FANS	TOTAL POWER (KW)	TOTAL FAN COST (\$/HR)	COST OF DUCT (\$/HR)
370.0	345.0	1	109.78	5.96	.31

FAN TYPE = 3
 DUCT TYPE = 1
 DUCT DIAMETER = 1.00 M.

ADVANCE (M)	DUCT LENGTH (M)	NO.OF FANS	TOTAL POWER (KW)	TOTAL FAN COST (\$/HR)	COST OF DUCT (\$/HR)
370.0	345.0	1	160.23	8.48	.31

FAN TYPE = 3
 DUCT TYPE = 2
 DUCT DIAMETER = .50 M.

ADVANCE (M)	DUCT LENGTH (M)	NO.OF FANS	TOTAL POWER (KW)	TOTAL FAN COST (\$/HR)	COST OF DUCT (\$/HR)
148.8	123.8	1	80.48	4.49	.17
268.2	243.2	2	162.76	9.07	.33
370.0	345.0	3	246.95	13.75	.46

FAN TYPE = 3
 DUCT TYPE = 2
 DUCT DIAMETER = .75 M.

ADVANCE (M)	DUCT LENGTH (M)	NO.OF FANS	TOTAL POWER (KW)	TOTAL FAN COST (\$/HR)	COST OF DUCT (\$/HR)
370.0	345.0	1	103.77	5.66	.46

FAN TYPE = 3
 DUCT TYPE = 2
 DUCT DIAMETER = 1.00 M.

ADVANCE (M)	DUCT LENGTH (M)	NO.OF FANS	TOTAL POWER (KW)	TOTAL FAN COST (\$/HR)	COST OF DUCT (\$/HR)
370.0	345.0	1	152.49	8.09	.46

Table 7.7

FAN TYPE = 3
 DUCT TYPE = 3
 DUCT DIAMETER = .50 M.

ADVANCE (M)	DUCT LENGTH (M)	NO.OF FANS	TOTAL POWER (KW)	TOTAL FAN COST (\$/HR)	COST OF DUCT (\$/HR)
116.6	91.6	1	82.33	4.58	.13
202.2	177.2	2	167.94	9.33	.25
283.5	258.5	3	256.25	14.22	.37
360.4	335.4	4	347.48	19.25	.48
370.0	345.0	5	441.89	24.44	.49

FAN TYPE = 3
 DUCT TYPE = 3
 DUCT DIAMETER = .75 M.

ADVANCE (M)	DUCT LENGTH (M)	NO.OF FANS	TOTAL POWER (KW)	TOTAL FAN COST (\$/HR)	COST OF DUCT (\$/HR)
370.0	345.0	1	115.34	6.24	.49

FAN TYPE = 3
 DUCT TYPE = 3
 DUCT DIAMETER = 1.00 M.

ADVANCE (M)	DUCT LENGTH (M)	NO.OF FANS	TOTAL POWER (KW)	TOTAL FAN COST (\$/HR)	COST OF DUCT (\$/HR)
370.0	345.0	1	165.63	8.75	.49

Table 7.8

NAME : THE TYEE LAKE PROJECT

DATE : 87/10/15. TIME : 13.20.39.

OVERALL OPTIMUM RESULTS

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I	I	I	I	I	I	I	I	I	I
I	I	I	I	I	I	I	I	I	I
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I	I	I	I	I	I	I	I	I	I
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I	I	I	I	I	I	I	I	I	I
I	I	I	I	I	I	I	I	I	I
I	I	I	I	I	I	I	I	I	I
I	I	I	I	I	I	I	I	I	I
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I	I	I	I	I	I	I	I	I	I
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I	I	I	I	I	I	I	I	I	I
I	I	I	I	I	I	I	I	I	I
I	I	I	I	I	I	I	I	I	I
I	I	I	I	I	I	I	I	I	I

(*) OPTIMUM RESULTS THAT MINIMIZE THE COST OF A ROUND

Table 7.2 illustrates the optimum results for a drilling and blasting round with combinations of different jumbos and LHDs. Although three types of jumbos and five types of LHDs are input for the model, jumbo 3, LHD 4 and LHD 5 are excluded because of the size of the area.

Length and cost of a round for each combination are shown in Figure 7.5.

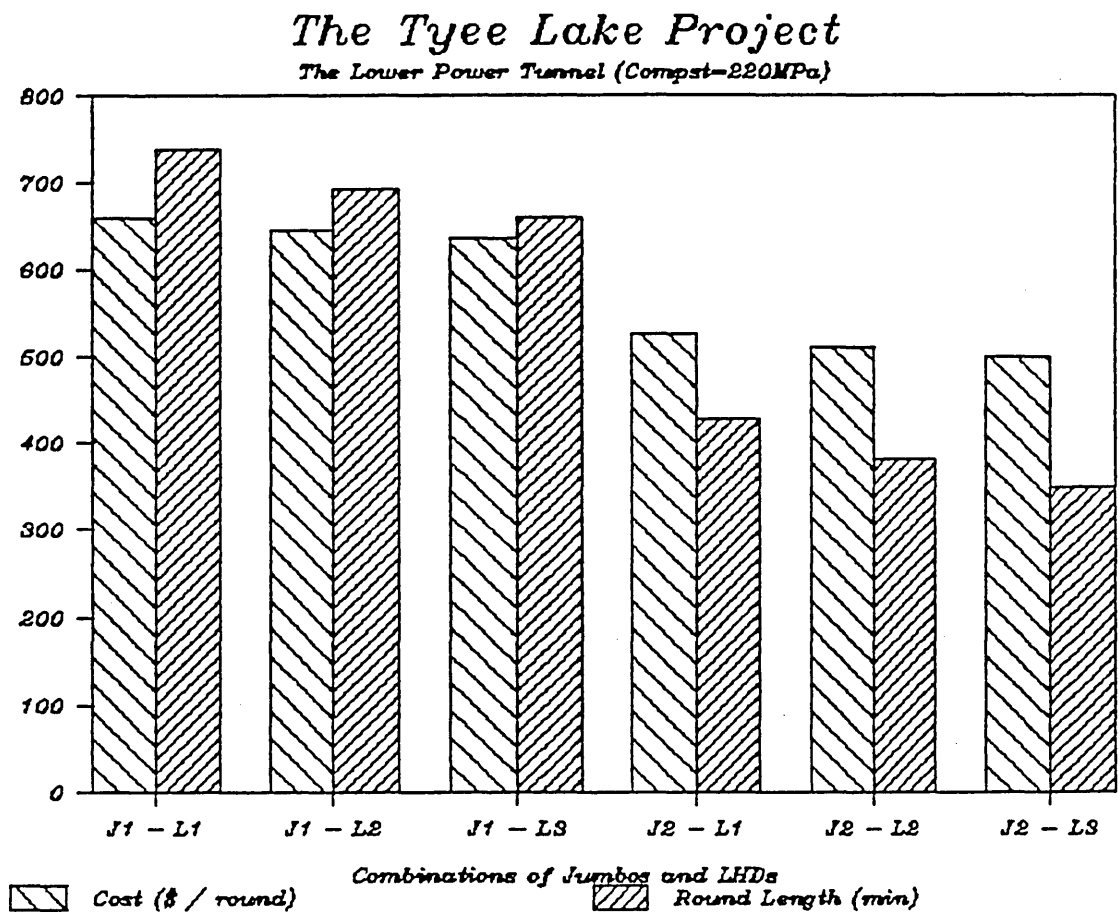


Figure 7.5 Length and cost of a round for different jumbo and LHD combinations (Compressive strength = 220 MPa)

As can be seen from Figure 7.5, the best combination which has the shortest round length and the lowest cost per round, is of jumbo 2 and LHD 3 (J2-L3). That means for the known conditions of the lower power tunnel, a 2-boom jumbo (such as Minimatic) and an LHD with 3.8 m^3 bucket capacity, are the best among these combinations. It can be said that although the optimum choice determined by model TUNOPR matches the jumbo type used in the actual operation, the type of mucker is different.

In the case of the lower power tunnel, where the rock has a compressive strength of 220 MPa (assumed) and the area of the tunnel is 14.28 m^2 , 49 shotholes would be drilled with a bit diameter of 29 mm to the depth of 2.2 m per round. An average advance rate of 2.09 m could be achieved. With this combination, daily advance rate would be 6.02 m. Though, in the operation, the number of holes was 54 and the hole depth was 3.2 m per round, the total drilling time was 144 min which is less than the result obtained by TUNOPR for 49 holes and 2.2 m hole depth. One possible explanation for this result is that the value of compressive strength, a factor that affects penetration rate, was in reality less than the assumed value of 220 MPa.

It can be seen from Table 7.2 that mucking of the blasted material would take less than 21 min with the chosen mucker (LHD 3).

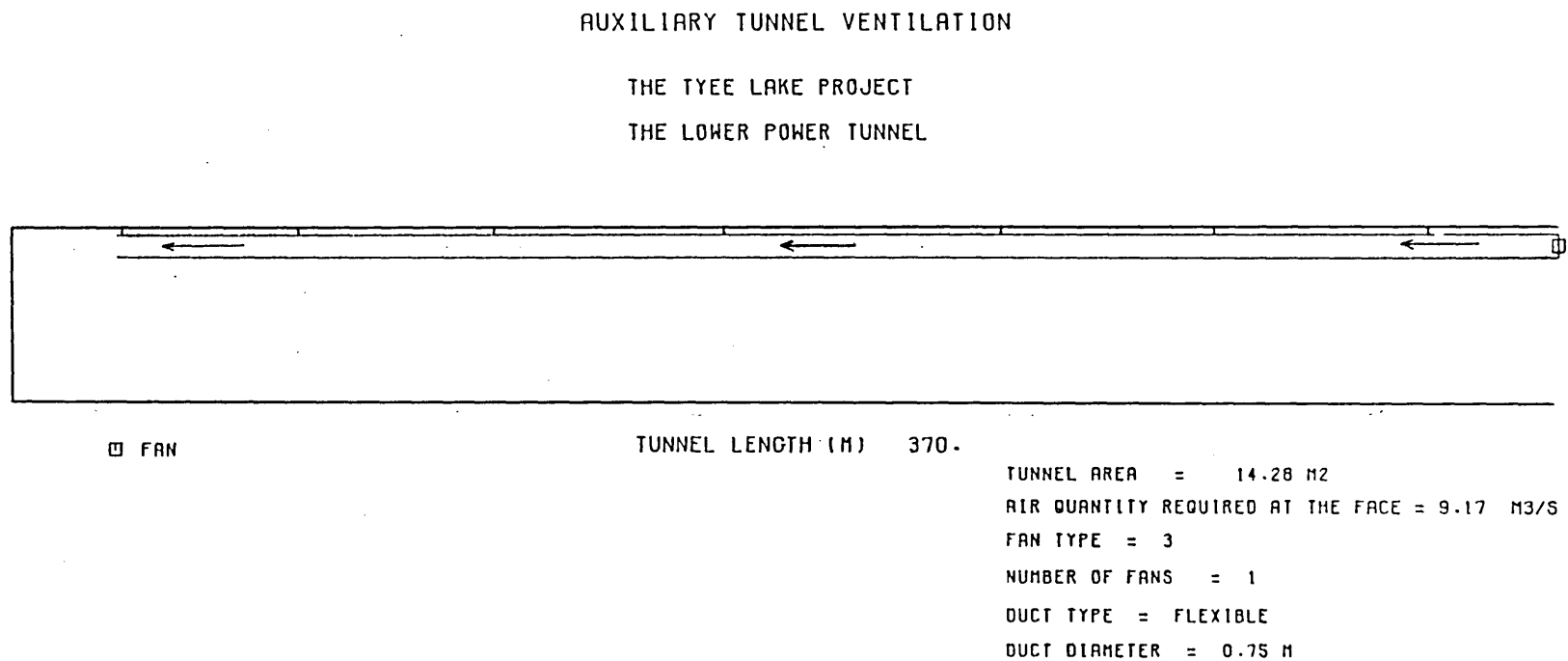
Having calculated optimum parameters for the drilling and blasting operation and chosen the type of jumbo and

loader, combinations of locomotives and mine cars are examined (Table 7.3 and 7.4). Ideal gradient of the tunnel, number of cars required per round and tractive efforts needed (full and empty trains) were estimated. The best combination that would have the lowest transportation cost and produce at least 15% more tractive effort than required was chosen to be locomotive 2 with mine car 3 (4 m^3). That means locomotive 3 (7 ton) could produce the required tractive effort to move the train. Table 7.4 illustrates only the transportation cost of the combination of locomotive 2 and mine car 3.

The air quantity required at the face for the lower power tunnel was found to be $9.17 \text{ m}^3/\text{s}$ by considering size of the area and the selected equipment. It is seen from the output that only fan 3 could supply such a throughput of air. Observations show that one fan (fan 3) would be sufficient and its combination with flexible duct (75 cm diameter) could ventilate the face at the lowest cost (See Table 7.4 to 7.7). The fan would provide $11 \text{ m}^3/\text{sec}$ of air at a pressure of 6231 Pa where volume increase ratio (VIR) is 1.27 and pressure increase ratio was approximately 1.23. The calculated equivalent resistance of the system was approximately $42.85 \text{ N s}^2/\text{m}^8$ and duct efficiency was 64%.

Figure 7.6 Ventilation of the lower power tunnel

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The overall results obtained for the lower power tunnel of the Tyee lake project are given in Table 7.8.

In order to see the effect of the compressive strength of the rock, three more runs were made for the same conditions. Figure 7.7 illustrates the results obtained when the compressive strength is 175 MPa and Table 7.9 shows the overall results.

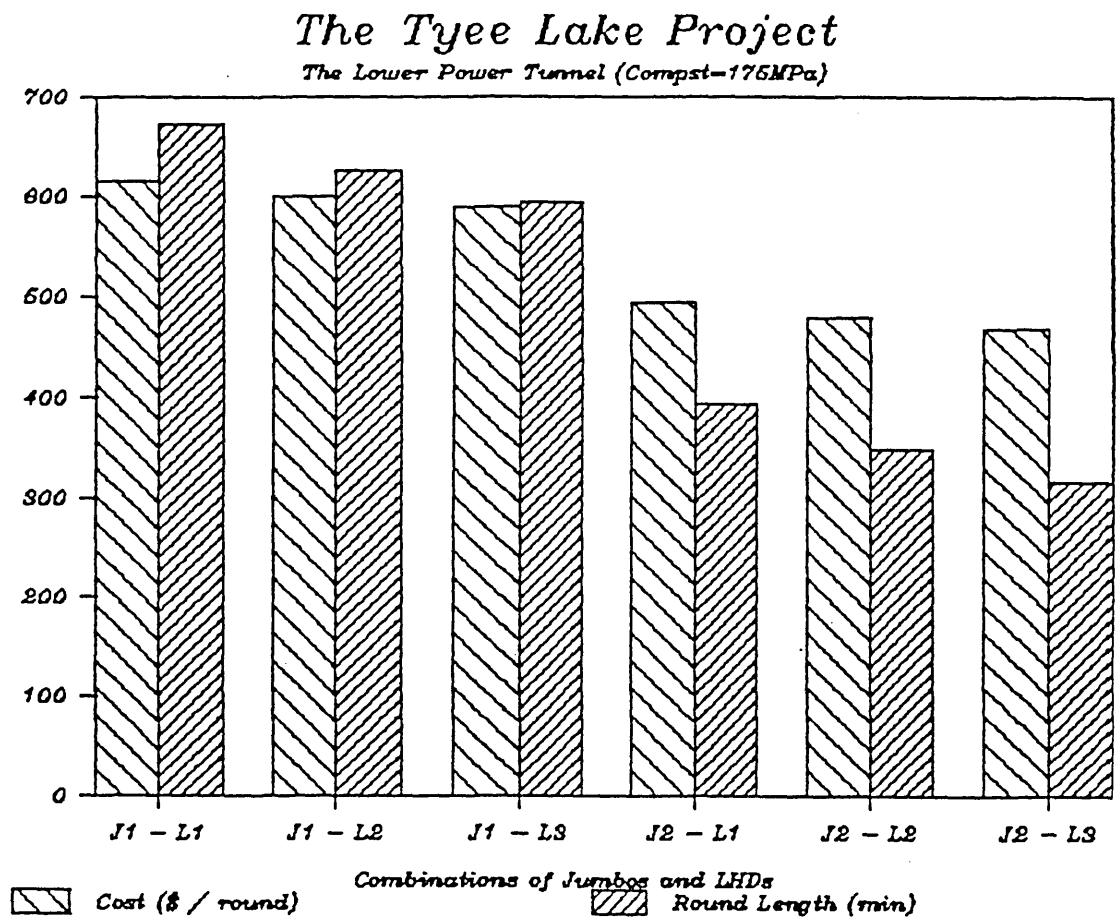


Figure 7.7 Length and cost of a round for different jumbo and LHD combinations (Compressive strength=175 MPa)

Table 7.9 Overall results for the compressive strength value of 175 MPa.

NAME : THE TYEE LAKE PROJECT

DATE : 87/10/16. TIME : 17.29.18.

O V E R A L L O P T I M U M R E S U L T S

I	I	I	I	I	I	I	I	I
I	I TYPE OF	I NUMBER	I HOLE DEPTH	I HOLE CHARGE	I BIT	I CUT HOLE	I TOTAL DRILLING	I
I	I JUMBO	I OF HOLES	I (M)	I (G)	I DIAMETER	I DIAMETER	I TIME (MIN/RND)	I
I	I	I	I	I	I (MM)	I (CM)	I	I
I DRILLING	I	I	I	I	I	I	I	I
I AND	I	I	I	I	I	I	I	I
I BLASTING	I 2	I 49	I 2.20	I 1400.00	I 29.00	I 8.00	I 294.56	I
I	I	I	I	I	I	I	I	I
I	I	I	I	I	I	I	I	I
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I	I	I	I	I	I	I	I	I
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(*) OPTIMUM RESULTS THAT MINIMIZE THE COST OF A ROUND

It was found that when the compressive strength decreases from 220 to 175 MPa, the total drilling, blasting and mucking time diminished from 384 to 315 minutes and the cost of a round decreased from \$499 to \$348. No change was observed in the other results. For this compressive strength value, an advance rate of 10 m/day could be achieved with the combination of jumbo 2 - LHD 3.

Figure 7.8 illustrates the graph constructed for the compressive strength value of 100 MPa and Table 7.10 is the overall optimum results.

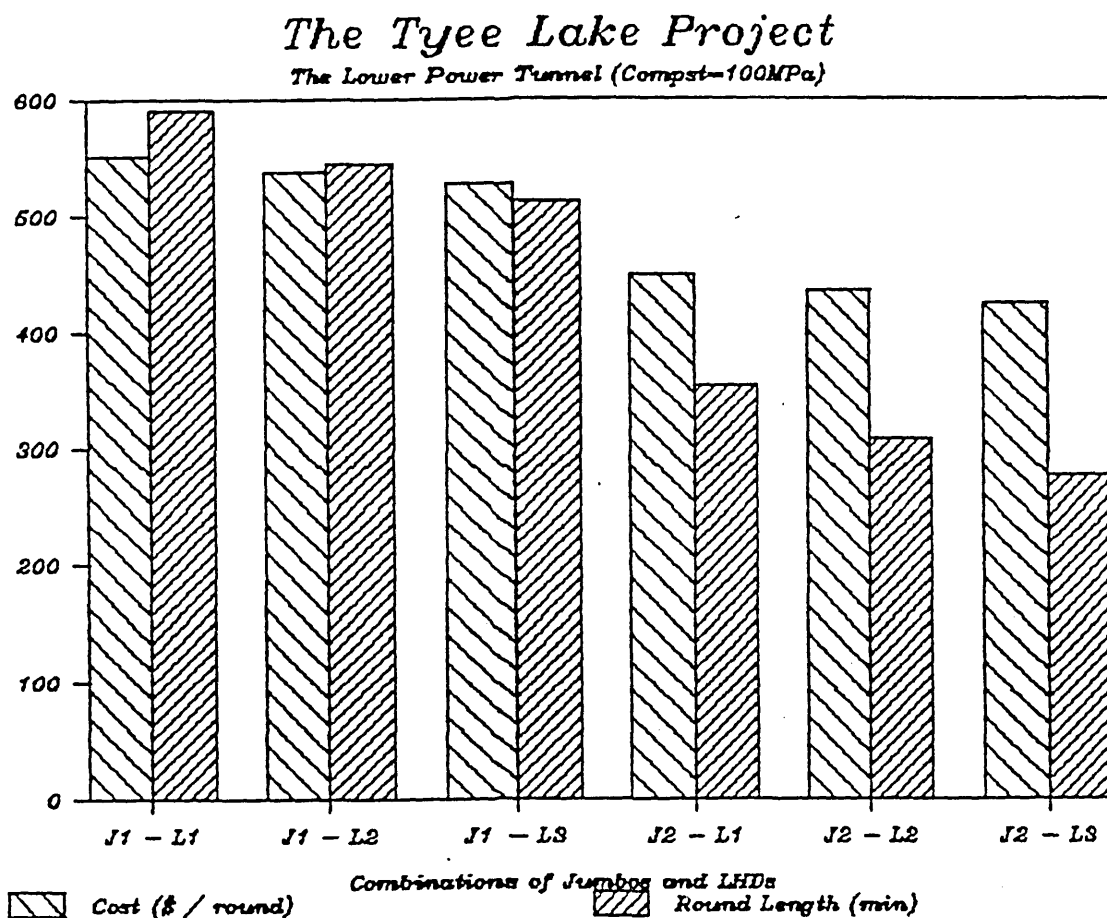


Figure 7.8 Length and cost of a round for different jumbo and LHD combinations (Compressive Strength = 100 MPa)

Table 7.10 Overall results for the compressive strength value of 100 MPa

NAME : THE TYEE LAKE PROJECT

DATE : 87/10/16. TIME : 17.30.28.

OVERALL OPTIMUM RESULTS

I	I	I	I	I	I	I	I	I	I
I	I TYPE OF	I NUMBER	I HOLE DEPTH	I HOLE CHARGE	I BIT	I CUT HOLE	I TOTAL DRILLING	I	I
I	I JUMBO	I OF HOLES	I (M)	I (G)	I DIAMETER	I DIAMETER	I TIME (MIN/RND)	I	I
I	I	I	I	I	I (MM)	I (CM)	I	I	I
I DRILLING	I	I	I	I	I	I	I	I	I
I AND	I	I	I	I	I	I	I	I	I
I BLASTING	I 2	I 47	I 2.20	I 1400.00	I 29.00	I 8.00	I 254.09	I	I
I	I	I	I	I	I	I	I	I	I
I	I TYPE OF	I MATERIAL	I MUCKING	I TOTAL	I COST OF DRILLING BLASTING	I	I	I	I
I	I LHD	I TO BE MUCKED	I TIME (MIN)	I DRILL&MUCK	I AND MUCKING	I	I	I	I
I	I	I (TON)	I	I TIME (MIN)	I (\$/ROUND)	I	I	I	I
I	I	I	I	I	I	I	I	I	I
I MUCKING	I 3	I 59.69	I 20.70	I 274.79	I 424.28	I	I	I	I
I	I	I	I	I	I	I	I	I	I
I	I TYPE OF	I WEIGHT OF	I CAPACITY OF	I NUMBER OF	I IDEAL	I COST OF	I	I	I
I	I LOCOM.	I LOCO (TON)	I CARS (M3)	I CARS	I GRADIENT	I (\$/ROUND)	I	I	I
I	I	I	I	I	I	I	I	I	I
I RAIL	I 3	I 10.00	I 4.00	I 8	I .23	I 1.76	I	I	I
I TRANSPORT	I	I	I	I	I	I	I	I	I
I	I TYPE OF	I TYPE OF	I DUCT	I NUMBER	I TOTAL POWER	I TOTAL VENTILATION COST	I	I	I
I	I FAN	I DUCT	I DIAMETER	I OF FANS	I INSTALLED	I	I	I	I
I	I	I	I (M)	I	I (KW)	I (\$ / HR)	I	I	I
I	I	I	I	I	I	I	I	I	I
I VENTILATION	I 3	I 2	I .8	I 1	I 103.77	I 6.12	I	I	I
I	I	I	I	I	I	I	I	I	I

(*) OPTIMUM RESULTS THAT MINIMIZE THE COST OF A ROUND

Finally the last run was made for a hypothetical compressive strength of 75 MPa. It is obvious that this value is too low for the rock where the lower power tunnel was driven. These are presented to show the model results for an extreme condition (See Figure 7.9 and Table 7.11). The cost of a drilling, blasting and mucking round decreases by 16%, the length of the round decreases by 25% and number of round increases from 3 to 5 per day.

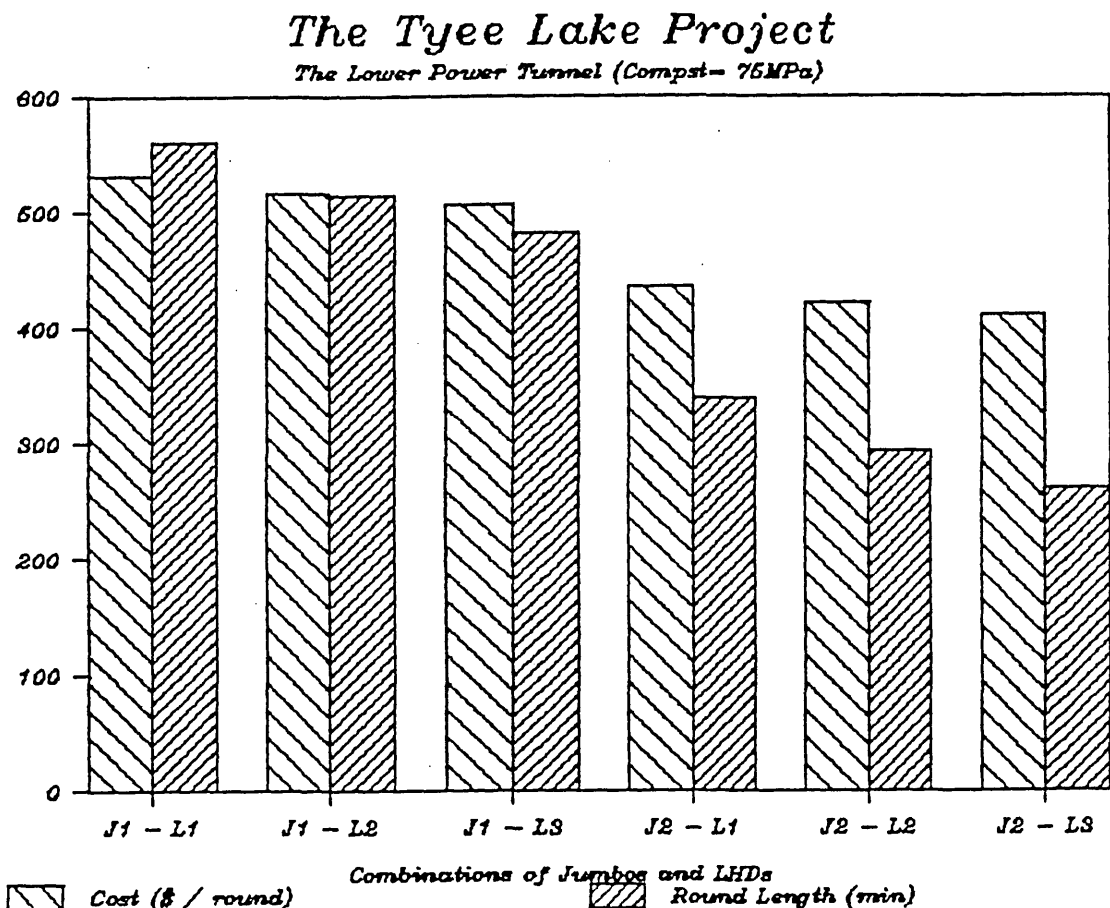


Figure 7.9 Length and cost of a round for different jumbo and LHD combinations (Compressive strength = 75 MPa)

Table 7.11 Overall results for the compressive strength value of 75 MPa.

NAME : THE TYEE LAKE PROJECT

DATE : 87/10/16. TIME : 17.31.56.

OVERALL OPTIMUM RESULTS

I	I	I	I	I	I	I	I	I
I	I TYPE OF	I NUMBER	I HOLE DEPTH	I HOLE CHARGE	I BIT	I CUT HOLE	I TOTAL DRILLING	I
I	I JUMBO	I OF HOLES	I (M)	I (G)	I DIAMETER	I DIAMETER	I TIME (MIN/RND)	I
I	I	I	I	I	I (MM)	I (CM)	I	I
I DRILLING	I	I	I	I	I	I	I	I
I AND	I	I	I	I	I	I	I	I
I BLASTING	I 2	I 47	I 2.20	I 1400.00	I 29.00	I 8.00	I 239.22	I
I	I	I	I	I	I	I	I	I
I	I TYPE OF	I MATERIAL	I MUCKING	I TOTAL	I COST OF DRILLING BLASTING	I	I	I
I	I LHD	I TO BE MUCKED	I TIME (MIN)	I DRILL&MUCK	I AND MUCKING	I	I	I
I	I	I (TON)	I	I TIME (MIN)	I (\$/ROUND)	I	I	I
I	I	I	I	I	I	I	I	I
I MUCKING	I 3	I 59.69	I 20.70	I 259.92	I 410.60	I	I	I
I	I	I	I	I	I	I	I	I
I	I TYPE OF	I WEIGHT OF	I CAPACITY OF	I NUMBER OF	I IDEAL	I COST OF	I	I
I	I LOCOM.	I LOCO (TON)	I CARS (M3)	I CARS	I GRADIENT	I (\$/ROUND)	I	I
I	I	I	I	I	I	I	I	I
I RAIL	I 3	I 10.00	I 4.00	I 8	I .23	I 1.76	I	I
I TRANSPORT	I	I	I	I	I	I	I	I
I	I TYPE OF	I TYPE OF	I DUCT	I NUMBER	I TOTAL POWER	I TOTAL VENTILATION COST	I	I
I	I FAN	I DUCT	I DIAMETER	I OF FANS	I INSTALLED	I	I	I
I	I	I	I (M)	I	I (KW)	I (\$ / HR)	I	I
I	I	I	I	I	I	I	I	I
I VENTILATION	I 3	I 2	I .8	I 1	I 103.77	I 6.12	I	I
I	I	I	I	I	I	I	I	I

(*) OPTIMUM RESULTS THAT MINIMIZE THE COST OF A ROUND

The output from the subroutine HEAT is given in Table 7.12 for some assumed figures.

Assumptions :

Virgin rock temperature	30 °C
Barometric pressure	110 kPa
Wet-bulb temperature	15 °C
Dry-bulb temperature	20 °C
Volume flow rate	9.17 m ³ /s
Daily face advance rate	6.02 m
Rock conductivity	6 W/m °C
Rock heat capacity	2400 kJ/m ³ °C

The output parameters are listed at one-tenth intervals of the final development length. It is important to realize that the table may not be used to estimate the air temperatures. The temperatures given are only face temperatures. For example, the face temperatures at the end of a 200 m tunnel are not the temperatures at the 200 m point of a 370 m tunnel.

Table 7.12

CONDITIONS OF AIR REACHING THE FACE OF DEVELOPMENT END FOR VARIOUS STAGES OF THE TUNNELLING OPERATIONTHE TYEE LAKE PROJECT

OVERALL LENGTH OF DEVELOPMENT = 370.0 m
 PERIMETER = 14.3 m
 DAILY FACE ADVANCE = 6.0 m
 ROCK TEMPERATURE = 30.0 °C
 BAROMETRIC PRESSURE = 110.0 KPa
 AIR QUANTITY = 9.17 m³/s
 INITIAL SPECIFIC VOLUME = 0.77435 m³/kg

DISTANCE COMPLETED (METRES)	ELAPSED TIME (DAYS)	BONE-DRY			PRACTICALLY DRY			WET			AIR AT FACE m ³ /s
		WET BULB °C	DRY BULB °C	MOISTURE IN AIR g/kg	WET BULB °C	DRY BULB °C	MOISTURE IN AIR g/kg	WET BULB °C	DRY BULB °C	MOISTURE IN AIR g/kg	
NO LEAKAGE											
0	0	15.00	20.00	7.72	15.00	20.00	7.72	15.00	20.00	7.72	9.17
37	6	15.71	21.83	7.72	15.71	21.81	7.73	15.71	21.41	7.89	9.17
74	12	16.17	23.04	7.72	16.16	22.98	7.74	16.19	22.23	8.08	9.17
111	18	16.51	23.95	7.72	16.50	23.87	7.75	16.57	22.79	8.27	9.17
148	24	16.78	24.68	7.72	16.77	24.57	7.76	16.90	23.21	8.46	9.17
185	30	17.00	25.28	7.72	17.00	25.15	7.77	17.20	23.54	8.66	9.17
222	36	17.19	25.79	7.72	17.19	25.65	7.78	17.47	23.80	8.86	9.17
259	43	17.35	26.22	7.72	17.35	26.06	7.79	17.71	24.02	9.05	9.17
296	49	17.49	26.60	7.72	17.49	26.43	7.80	17.95	24.21	9.24	9.17
333	55	17.61	26.93	7.72	17.61	26.74	7.81	18.17	24.37	9.44	9.17
370	61	17.71	27.22	7.72	17.72	27.02	7.82	18.38	24.51	9.63	9.17
SMALL LEAKAGE											
0	0	15.00	20.00	7.72	15.00	20.00	7.72	15.00	20.00	7.72	9.17
37	6	15.71	21.83	7.72	15.71	21.81	7.73	15.71	21.41	7.89	9.07
74	12	16.17	23.06	7.72	16.17	23.01	7.74	16.20	22.25	8.08	8.97
111	18	16.53	24.00	7.72	16.52	23.92	7.75	16.60	22.83	8.28	8.87
148	24	16.81	24.77	7.72	16.81	24.66	7.76	16.94	23.27	8.48	8.77
185	30	17.05	25.40	7.72	17.04	25.28	7.77	17.26	23.62	8.69	8.67
222	36	17.25	25.95	7.72	17.24	25.80	7.78	17.55	23.90	8.91	8.58
259	43	17.42	26.42	7.72	17.42	26.25	7.79	17.82	24.14	9.12	8.48
296	49	17.57	26.82	7.72	17.57	26.64	7.80	18.08	24.34	9.34	8.39
333	55	17.70	27.18	7.72	17.70	26.99	7.82	18.32	24.52	9.56	8.30
370	61	17.81	27.49	7.72	17.82	27.29	7.83	18.56	24.68	9.78	8.21
AVERAGE LEAKAGE											
0	0	15.00	20.00	7.72	15.00	20.00	7.72	15.00	20.00	7.72	9.17
37	6	15.71	21.83	7.72	15.71	21.81	7.73	15.71	21.41	7.89	8.87
74	12	16.19	23.11	7.72	16.19	23.06	7.74	16.22	22.28	8.09	8.58
111	18	16.57	24.12	7.72	16.57	24.03	7.75	16.65	22.91	8.30	8.30
148	24	16.88	24.95	7.72	16.88	24.84	7.76	17.03	23.39	8.53	8.03
185	30	17.14	25.66	7.72	17.14	25.53	7.77	17.38	23.78	8.77	7.76
222	36	17.37	26.27	7.72	17.36	26.12	7.78	17.71	24.10	9.02	7.51
259	43	17.56	26.80	7.72	17.56	26.63	7.80	18.03	24.37	9.28	7.26
296	49	17.73	27.26	7.72	17.73	27.07	7.82	18.35	24.61	9.55	7.03
333	55	17.87	27.66	7.72	17.89	27.45	7.83	18.65	24.81	9.83	6.80
370	61	18.00	28.01	7.72	18.02	27.79	7.85	18.95	25.00	10.11	6.57

Table 7.13 (continued)

DISTANCE COMPLETED (METRES)	ELAPSED TIME (DAYS)	BONE-DRY			PRACTICALLY DRY			WET			AIR AT FACE m ³ /s
		WET BULB °C	DRY BULB °C	MOISTURE IN AIR g/kg	WET BULB °C	DRY BULB °C	MOISTURE IN AIR g/kg	WET BULB °C	DRY BULB °C	MOISTURE IN AIR g/kg	
LARGE LEAKAGE											
0	0	15.00	20.00	7.72	15.00	20.00	7.72	15.00	20.00	7.72	9.17
37	6	15.71	21.83	7.72	15.71	21.81	7.73	15.71	21.41	7.89	8.67
74	12	16.21	23.16	7.72	16.21	23.11	7.74	16.24	22.32	8.09	8.21
111	18	16.62	24.23	7.72	16.61	24.15	7.75	16.70	22.99	8.32	7.76
148	24	16.95	25.14	7.72	16.95	25.03	7.76	17.11	23.51	8.57	7.34
185	30	17.24	25.92	7.72	17.23	25.78	7.77	17.51	23.94	8.85	6.95
222	36	17.49	26.59	7.72	17.48	26.43	7.79	17.89	24.29	9.14	6.57
259	43	17.70	27.18	7.72	17.70	27.00	7.81	18.26	24.59	9.45	6.22
296	49	17.88	27.68	7.72	17.89	27.48	7.83	18.63	24.85	9.78	5.88
333	55	18.04	28.11	7.72	18.06	27.89	7.85	18.99	25.08	10.13	5.56
370	61	18.17	28.48	7.72	18.20	28.25	7.88	19.36	25.29	10.49	5.26

7.4. THE MODEL APPLICATIONS

The ability to change and rerun a program enables the user to investigate how the magnitude of changes in specific parameters will affect the overall results. It should be remembered that all the cost figures used to assess the sensitivity of the model were assumed. They do not represent any particular project and were generated only for the purposes of illustration.

Rock properties and Number of holes:

Figure 7.10 is a plot of the number of holes versus tunnel area with variations in the rock compressive strength. If, for example, in Figure 7.10 the rock compressive strength drops from 75 MPa to 50 MPa for the tunnel area of 30 m², then the number of holes will decrease from 70 to 66. The graph is constructed from the optimum values suggested by the model.

Rock properties and Round length:

Figure 7.11 is a graph showing the change in round length as a function of the tunnel area and rock compressive strength. The graph is plotted for the best combination of drill jumbo and LHD Unit. Round length increases with increasing rock compressive strength because of the relationship between required number of holes in each round and the rock compressive strength value (See Figure 7.10).

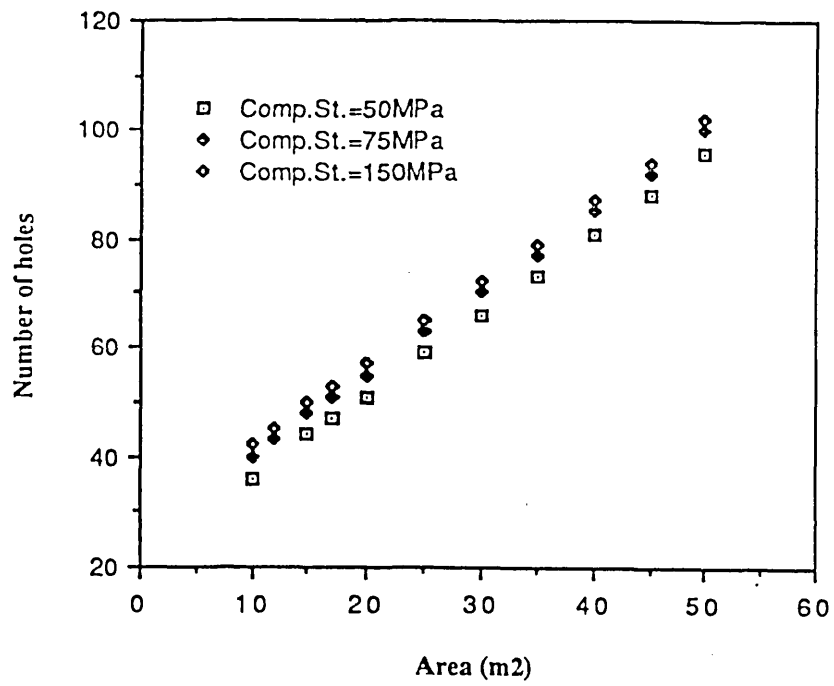


Figure 7.10 Graph of number of holes versus tunnel area

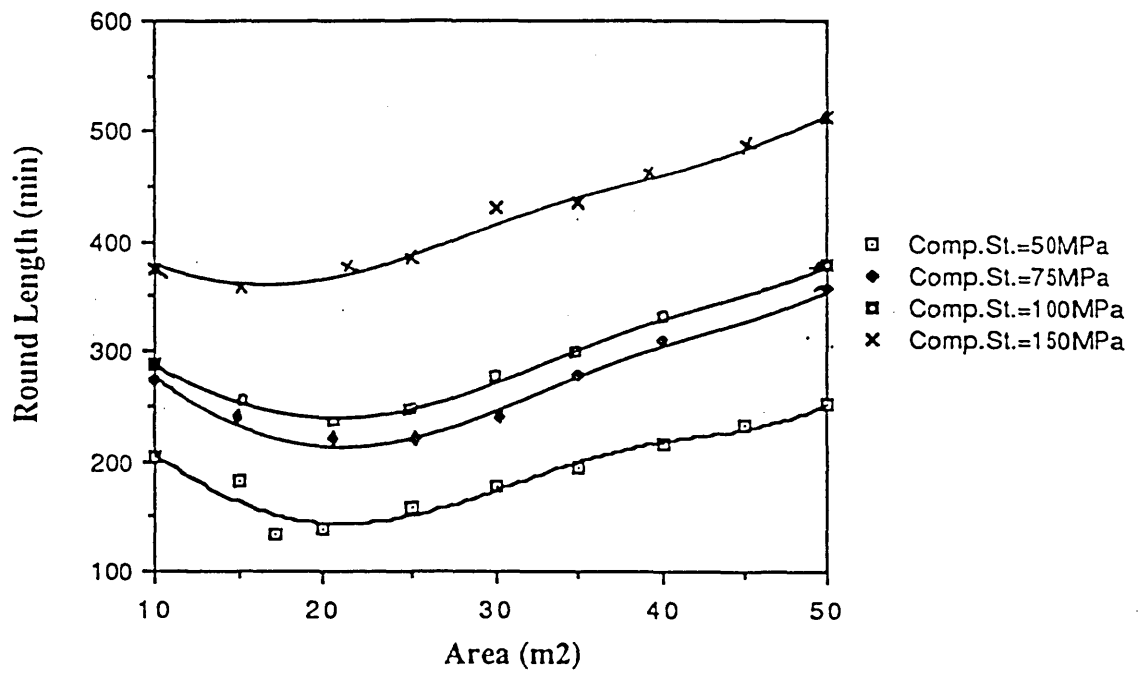


Figure 7.11 Graph of round length versus tunnel area

Also a decrease in the rock penetration rate due to higher compressive strength, will increase the total round length. Smaller areas mean short drilled metre per round therefore short round length. The graph illustrates that this relationship is nonlinear and the round length reaches a minimum value when the tunnel area is approximately 20 m^2 . This is due to the application of different sizes of equipment as a function of the tunnel area. When drilling and mucking systems are examined to find the best combination and to minimize the total cost, the equipment that cannot be used in the given area is eliminated. For example, Jumbo 3 can be used when the tunnel area is between $22 - 70 \text{ m}^2$ (See Figure 7.2), for areas less than 22 m^2 , Jumbo 3 is therefore out of the question. When the area is big enough the productivity of higher capacity machinery would bring the round length down. For the areas bigger than 22 m^2 , the best results were obtained with the combination of Jumbo 3 (3-boom) and LHD 5 (9.9 m^3 of bucket size).

Figure 7.12 illustrates the cost per round depending on the tunnel area and rock compressive strength. The graph is constructed for the same values as Figure 7.11. Here, the cost of the operation increases with the increase in tunnel area and rock compressive strength. The curves might take different forms for different costs of the equipment that are examined.

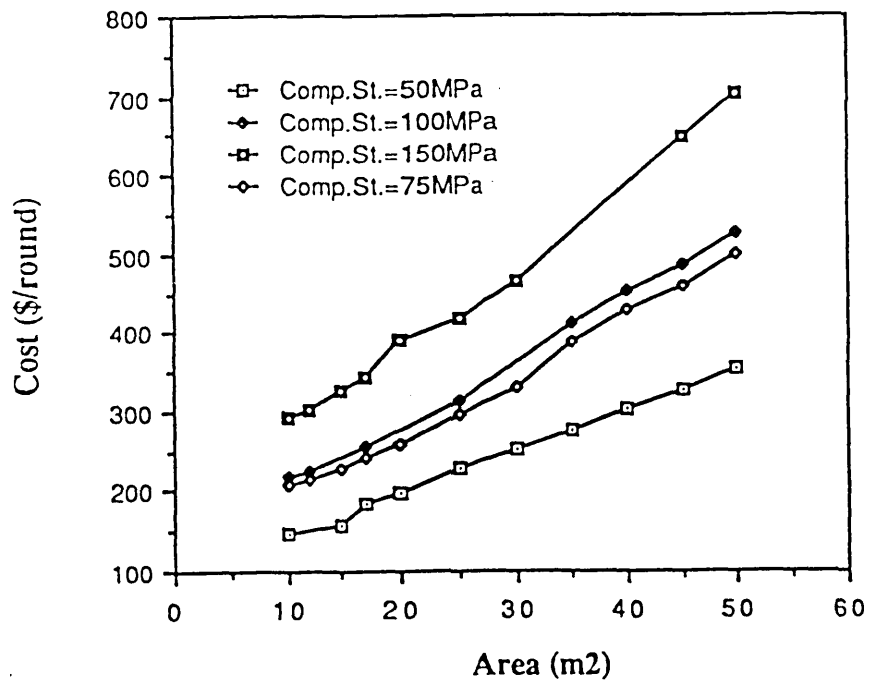


Figure 7.12 Graph of cost of a round versus tunnel area

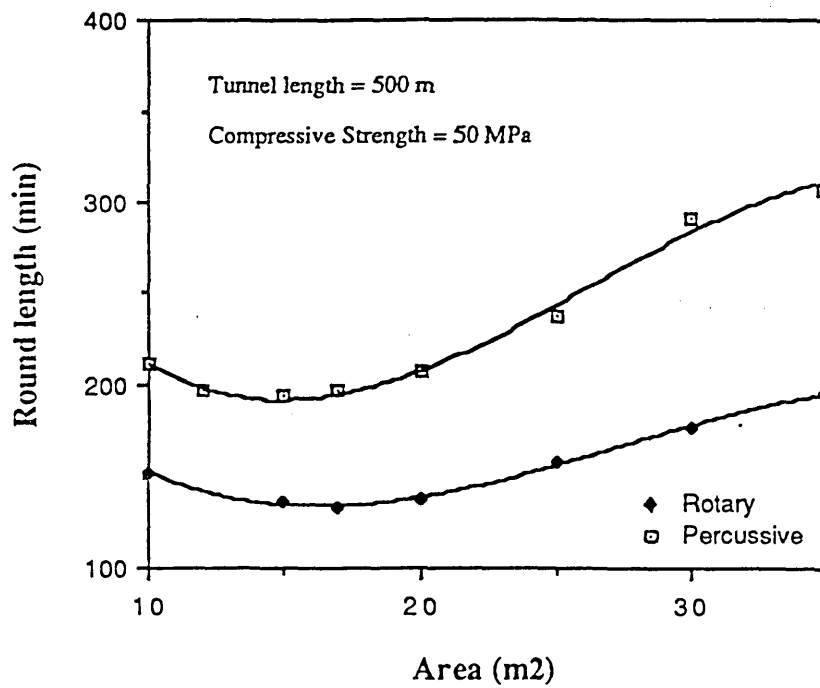


Figure 7.13 Graph of round length versus drilling type

Type of drilling and Rock properties

Figure 7.13 is a plot illustrating the operation performance for different tunnel areas and drill types. It was shown earlier (Table 2.2) that rotary drills have higher penetration ability in rock with low rock compressive strength compared to percussive drills. The graph shows the total round length as a function of different drill types for a constant compressive strength value of the rock. It can be seen from the graph that the round length obtained by the rotary drills is shorter than that of the percussive drills. The length of a round approaches a minimum for the tunnel areas between 15 - 20 m² because of the applications of higher capacity machines (Jumbo 3 and LHD 5). The round length shows a constant increase with tunnel area beyond 20 m².

Figure 7.14 illustrates cost of the operation depending on the drilling type.

Transportation, Ventilation costs and Tunnel area

Figure 7.15 is a graph of the effect of the tunnel length on transportation cost. The relationship is linear because in design calculations of rail transportation, it is assumed that the train accelerates until it reaches the maximum speed before decelerating towards the end of the tunnel. Therefore the variable times in one cycle of the train are the same for each value of tunnel length. This means that it is only the period when the train moves with a constant speed, that changes as a function of the tunnel

length.

Figure 7.16 illustrates the required ventilation power as a function of the tunnel length. This graph is constructed for a tunnel area of 15 m^2 by examining three types of fans (Figure 7.4) and three types of ventilation ducts of three different diameters (50, 75 and 100 cm). The power required is plotted against the tunnel length. The slope of the curve increases steeply by lengths in excess of 1250 m. As the tunnel length increases, the number of required fans increases and the optimum distance between the fans decreases along the tunnel towards the entrance. That means the quantity of leakage increases with the increase in air volume provided by the fan and the pressure developed at the fan.

Figure 7.17 is a plot of the ventilation cost versus the tunnel length. The cost of the ventilation shows the same relation as Figure 7.16.

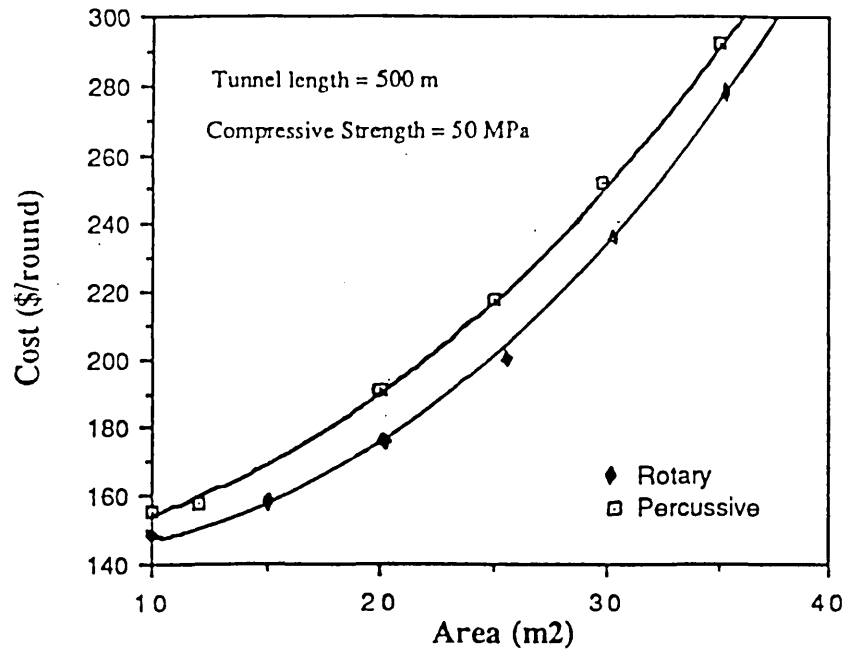


Figure 7.14 Graph of cost of a round versus drilling type

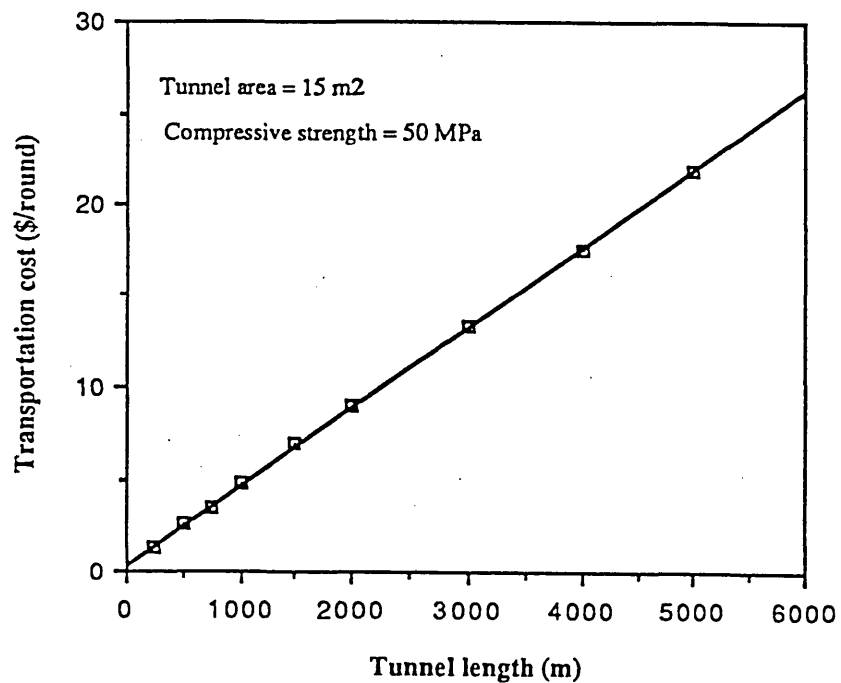


Figure 7.15 Graph of transportation cost versus tunnel length

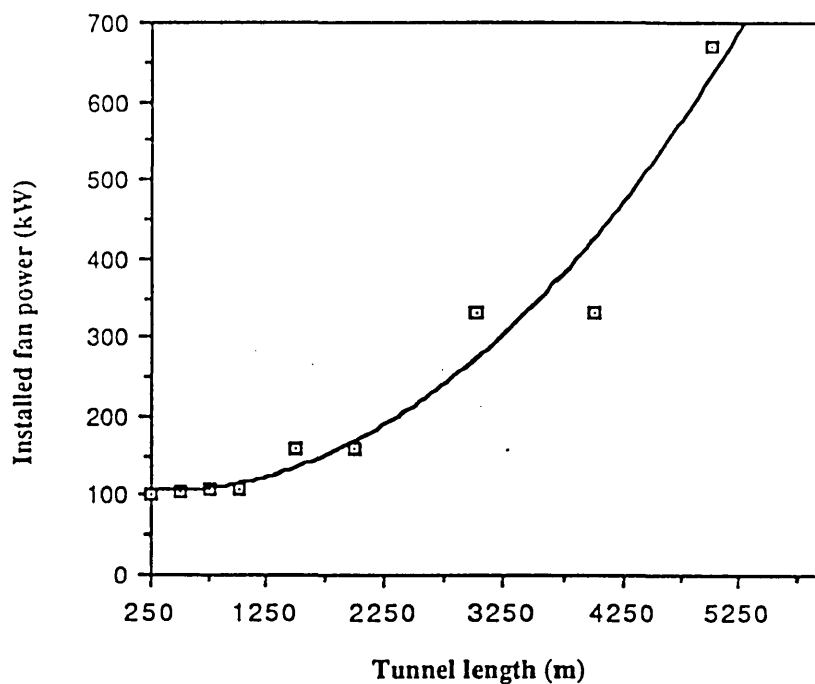


Figure 7.16 Graph of installed fan power versus tunnel length

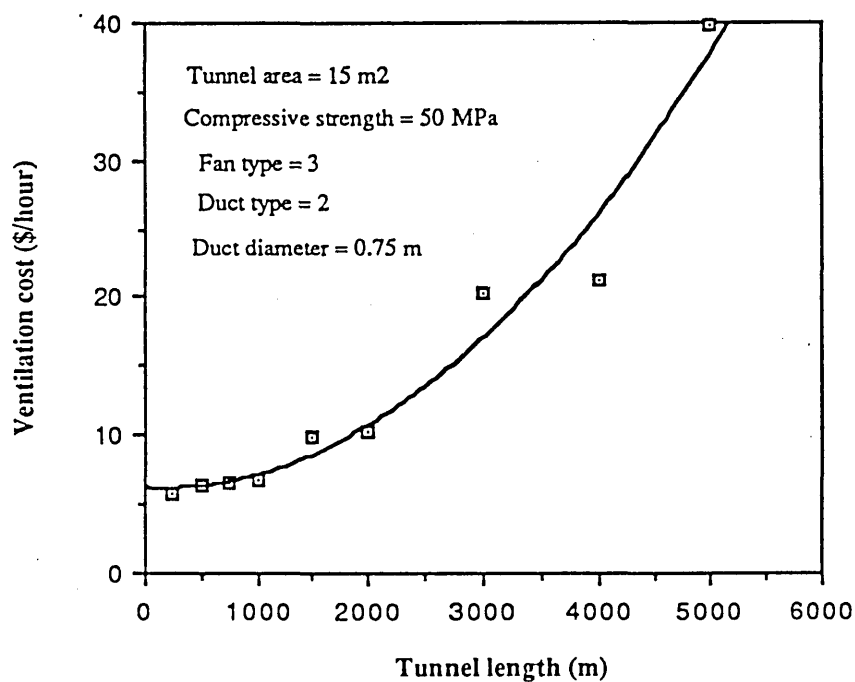


Figure 7.17 Graph of ventilation cost versus tunnel length

CHAPTER EIGHT

CONCLUSIONS AND SUGGESTIONS

Table 7.1 to 7.8 and Figure 7.5 to 7.9 give the output of TUNOPR for the lower power tunnel of the Tyee Lake Project. The results indicate that a drill jumbo with 2 booms produces the best performance in a drilling round. A Load-Haul-Dump Unit with a 3.3 m^3 bucket is the most suitable mucking system for obtaining the optimum results in the given area. In the actual operation, the type of mucker, Haggloader 8HR E1 did not ideally match the operation therefore, the mucking time formed approximately (100 min) 30% of the total round length. With the chosen LHD (LHD 3), mucking of the broken rock could be carried out in about 20 min. In ventilation of the face, one fan (fan 3) provides the required quantity of air over a length of 370 m where the actual ventilation column is 345 m long.

TUNOPR can be used to design tunnelling or drifting operations in underground mines where the tunnel area is greater than 9.83 m^2 . This is the area where the smallest size of LHD can work. In the model, definition of the compressive strength value of the rock is one of the important factors that affect estimation of the total cost, advance rate and life of the project. Therefore special emphasis should be placed on determination of this property.

The model can be used on any IBM or IBM compatible micro computer provided the optimization routine E04UAF is supplied to the program. The model offers an inexpensive tool for testing alternative design of conventional tunnelling method with Jumbos and LHDs for a proposed geological environment. Comparison of various parameter combinations in an existing mining operation can quickly be evaluated on a computer to rank the alternative changes based upon their potential economic benefit irrespective of the project location.

TUNOPR covers drilling, blasting, mucking, transportation and the ventilation phases of tunnel excavations. It would be beneficial if some further work examined the methods of ground support and model modification.

In the model development, the performance of the operation is heavily dependent upon the rock compressive strength, however, it is not the only factor that governs the efficiency of the operation. Other variables such as physical properties of rock, applied thrust, design of bits and rotary speeds should be examined.

It was observed that in tunnel ventilation, air temperatures at the face increase with tunnel length depending on the tunnel area, daily face advance, virgin rock temperature, barometric pressure, physical properties of rock, air quantity required at the face and the total installed power. Therefore, there must be a critical

distance beyond which either the employed fans cannot provide adequate air or refrigeration would be required.

It is recognized that wet-bulb temperatures below 29 °C are generally satisfactory, in that acclimatized personnel will be reasonably alert mentally and able to work under hot conditions at almost peak productivity and in complete safety from the physiological point of view (Whillier and Ramsden, 1975). For unacclimatized personnel wet-bulb temperatures below 28 °C are considered to be cool. The critical distance for a tunnelling operation where the face wet-bulb temperature exceeds 28 °C, can be determined by observing the output from the subroutine HEAT.

Further research could be directed to the refrigeration requirements so that the model could be applied to a wider range of tunnelling problems, particularly at depth and over long distances.

The computer program TUNOPR is available at the Department of Mineral Resources Engineering of the Royal School of Mines.

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APPENDIX-1

CONSTRAINED FLETCHER-POWELL (CONMIN ALGORITHM)

A. Method

This program finds the minimum of a multivariable, nonlinear function subject to nonlinear equality constraints:

$$\text{Minimize } F(x_1, x_2, \dots, x_n)$$

$$\text{Subject to } G_k(x_1, x_2, \dots, x_n) = 0, \quad k=1, 2, \dots, m$$

Method

The procedure has been described by Haarhoff and Buys (1970). The method incorporates the constraints into a modified, unconstrained objective function which is then optimized by the unconstrained optimization technique of Fletcher and Powell. Inequality variables are treated by use of slack variables and transformations. The algorithm proceeds as follows:

1. A new unconstrained objective function is formulated from the original function and constraints,

$$\Phi = F - \sum_{k=1}^m \lambda_k G_k + B \sum_{k=1}^m G_k^2$$

where λ_k and B are constants.

2. A starting point is selected (feasible or nonfeasible) and the value and derivatives F and G_k are determined. The derivatives can be either analytical or numerical approximations. The H matrix in the Fletcher and Powell procedure is set equal to the identity matrix for the first iteration. The λ_k values are determined from the

following system of equations,

$$\sum_{i=1}^n \sum_{j=1}^m \left\{ \lambda_j \frac{\partial G_i}{\partial x_i} \frac{\partial G_k}{\partial x_i} \right\} = \sum_{i=1}^n \left\{ \frac{\partial G_k}{\partial x_i} \frac{\partial F}{\partial x_i} \right\}, \quad k = 1, 2, \dots, m$$

The value of B is set equal to some positive number (The authors suggest the B value to be 30).

3. A series of search directions and one dimensional search steps are then determined per the unconstrained Fletcher and Powell general procedure for the modified objective function with updating of the λ_k values for each new iteration. When convergence is achieved, $G_k = 0$ and $F = \Phi$.

APPENDIX-2

INPUT VALUES FOR THE LOWER POWER TUNNEL OF THE TYEE LAKE PROJECT

Name of the project : The Tyee Lake Project
 Tunnel area : 14.28 m²
 Tunnel length : 370 m
 Compressive strength : 220 MPa
 Type of rock (abrasive or non-abrasive) : abrasive
 Type of drilling : Percussive
 Density of rock : 2.76 g/cm³
 Density of broken rock : 2.0 g/cm³
 Rate of advance aimed to achieve : 10.0 m/day

	NO OF BOOMS	PURCHASE PRICE(\$)	TYRE COSTS(\$)	SERVICE LIFE(YR)	OPERATING HOURS(YR)	RESALE VALUE(\$)
	-----	-----	-----	-----	-----	-----
JUMBO(1)	1	170000.00	3000.00	4	4500.00	10000.00
JUMBO(2)	2	240000.00	4500.00	4	4000.00	16000.00
JUMBO(3)	3	450000.00	6000.00	5	4000.00	17000.00

BUC.CAP. m³

LHD(1)	.80	56207.00	3000.00	5	4000.00	5000.00
LHD(2)	1.50	64718.00	3500.00	5	4200.00	7500.00
LHD(3)	3.80	119497.00	4500.00	6	4500.00	10000.00
LHD(4)	6.10	288943.00	6500.00	6	4700.00	13000.00
LHD(5)	9.90	569460.00	7000.00	7	5000.00	25000.00

Cost of jumbo operator : 9 (\$/hr)
 Cost of LHD operator : 9 (\$/hr)
 Cost of explosive : 2.5 (\$/kg)
 Cost of detonator : 0.2 (\$)
 Density of explosive : 1.505 (g/cm³)
 Interest rate (%) : 17

Cost of drill bit : \$ 12
 Cost of drill steel : \$ 10
 Cost of coupling : \$ 10
 Cost of shank adapter : \$ 10
 Cost of shank rod : \$ 10
 Cost of electricity : 0.05 (\$/kW.h)
 Cost of fuel : 0.25 (\$/litre)
 Cost of hydraulic fluid: 0.25 (\$/litre)

RAIL TRANSPORTATION

Number of different locomotives to examine : 3
Number of mine cars to examine : 3

	LOCO WEIGHT	PURCHASE PRICE	LIFE (YR)	SALVAGE VALUE	TIME WORKED (HR/YR)	GAUGE (mm)
	-----	-----	-----	-----	-----	-----
LOCOMOTIVE (1)	3.00	55000.00	3	2000.00	4000.00	609.60
LOCOMOTIVE (2)	7.00	70000.00	3	3000.00	4500.00	609.60
LOCOMOTIVE (3)	10.00	95000.00	4	3000.00	4000.00	609.60

	WEIGHT (TON)	CAPACITY (m ³)	PURCHASE PRICE (\$)	LIFE (YR)	SALVAGE VALUE (\$)	TIME WORKED (HR/YR)	GAUGE (mm)
	-----	-----	-----	-----	-----	-----	-----
CAR (1)	2.50	2.00	2390.00	4	.00	3000.00	609.60
CAR (2)	3.00	3.00	3390.00	4	.00	3000.00	609.60
CAR (3)	4.20	4.00	4100.00	4	.00	3500.00	609.60

Rolling resistance : 9.08 (kg/ton)
Cost of track per metre : 20 (\$/m)
Cost of ballast and track material : 50 (\$/m)

VENTILATION

Number of different fans to examine : 3
Number of different ducts : 3
Number of points on the characteristic
curve of each fan : 3

	<u>Purchase price (\$)</u>	<u>Life (yr)</u>	<u>Time worked (hr/yr)</u>
Fan (1)	2000.0	2	3000.0
Fan (2)	3000.0	2	3000.0
Fan (3)	4500.0	3	4500.0

Inlet volume and Pressure values on the fan
characteristic curves

	<u>Q (m³/s)</u>	<u>P (Pa)</u>
	-----	-----
Fan (1)	4.0	1100.0
	5.0	800.0
	6.0	500.0
Fan (2)	5.47	2115.2
	6.89	1224.0
	7.43	373.3

	Q (m ³ /s)	P (Pa)
	-----	-----
	18.0	6100.0
Fan (3)	22.0	5500.0
	26.0	4500.0

	Purchase Price (\$/m)	Life (yr)
	-----	-----
Duct (1) (metal pipe)	10.0	2
Duct (2) (flexible)	15.0	2
Duct (3) (flexible wire-reinforced)	16.0	2

Diameter of ducts : 50 , 75 and 100 cm.

APPENDIX-3

AN EXAMPLE INPUT FILE FOR THE MODEL TUNOPR

THE TYEE LAKE PROJECT
10.0
14.28,370.0,220.0,1,1,2.76,2.0
170000.0,240000.0,450000.0,56207.0,64718.0,119497.0,288943.0,569460.0
3000.0,4500.0,6000.0
3000.0,3500.0,4500.0,6500.0,7000.0
9.0,9.0
0.00250,0.2,1.505
0.16,0.16,4,4,5,5,5,6,6,7
10000.0,16000.0,17000.0,5000.0,7500.0,10000.0,13000.0,25000.0
4500.0,4000.0,4000.0
4000.0,4200.0,4500.0,4700.0,5000.0
12.0,10.0,10.0,10.0,10.0,0.05,0.25,0.25
3,3
3,55000.0,3,4000.0,10.0,2000.0
7.0,70000.0,3,4500.0,13.0,3000.0
10.0,95000.0,4,4000.0,15.0,3000.0
2.0,2.5,2390.0,4,3000.0,0.0
3.0,3.0,3390.0,4,3000.0,0.0
4.0,4.2,4100.0,4,3500.0,0.0
6.0
20.0,50.0,9.08
0.16,0.0,25.0
3,3
4,1100
5,800
6,500
5.47,2115.20
6.89,1244.0
7.93,373.26
18.0,6100
22.0,5500.0
26.0,4500.0
3
0.5
0.75
1.0
2000.0,2,3000.0
3000.0,2,3000.0
4500.0,3,4000.0
10.0,2
15.0,2
16.0,2