A COMPARISON OF DEEP SHAFT SINKING TECHNIQUES; A CASE STUDY USING THE CONVENTIONAL METHOD

by

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A COMPARISON OF DEEP SHAFT SINKING TECHNIQUES;
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Shaft is defined as an undeground cavity of an inclination 45 to 90 degrees to the horizontal line, of a straight course, and to be considered a connection to other underground cavities, of a minimum cross section of approximately $10m^2$, and of an essentially larger length in the vertical than in the horizontal direction.

There are many applications of shaft sinking in underground constructions such as access shafts for tunnel construction, ventilation shafts for long distance tunnels, hydro-electric power station pressure shafts, shafts to repositories of nuclear or chemical waste. It has to be recognized that worldwide 90 per cent and more of all shafts are sunk for mining purposes.

Most of the shafts required worldwide are constructed by conventional shaft sinking which is understood as breaking rock by blasting, loading and hoisting of the broken rock and installation of the lining. Alternative sinking methods to the conventional method are shaft drilling and boring where in shaft boring muck is removed to a lower level and there should be an existing underground connection, and in shaft drilling with muck removal to the surface or to the next higher level.

In this study, all these three sinking methods are examined and compared with each other. Conventional sinking method is examined in detail and a shaft, sunk in Çayeli Rize by a conventional method is taken as a case study. Construction steps are explained and sinking equipment alternatives and progress of the method is discussed.

DERİN ŞAFT AÇMA TEKNİKLERİNİN KARŞILAŞTIRILMASI; BİR ÖRNEK ÇALIŞMA - KONVANSİYONEL METOD İLE ŞAFT AÇILMASI

Şaft, yatayla 45 ila 90 derece arası eğimi olan, diğer yeraltı yapılarına bağlantılı, minimum 10m^2 kesit alanına sahip ve özellikle düşeydeki uzunluğu yataydaki uzunluğundan çok daha fazla olan bir yeraltı yapısıdır.

Şaftların, yeraltı yapıları olarak, tünel yapılarının yaklaşım şaftları, uzun mesafeli tünellerin havalandırma şaftları, hidroelektrik santrallerin basınç şaftları, nükleer veya kimyevi atıkları stoklama şaftları gibi bir çok kullanım alanı vardır. Bununla birlikte dünyada açılan şaftların yaklaşık yüzde 90'ınından fazlası madencilik maksadıyla açılmaktadır.

Dünyada açılan şaftların büyük çoğunluğu, zeminin patlatma marifetiyle parçalanması, parçalanan zeminin uygun taşıyıcılara yüklenerek taşınması ve şaft çeperlerinin kaplanması şeklinde kısaca tariflenebilen konvansiyonel metodla açılmaktadır. Konvansiyonel metoda alternatif olarak, hafriyatın daha aşağıdaki zemin seviyelerine iletildiği, dolayısıyla daha aşağıdaki bir seviyede önceden açılmış bir bağlantı bulunması gereken şaft boring metodu ile hafriyatın bir üst zemin seviyesine veya yer üstüne iletildiği şaft delme metodları da bulunmaktadır.

Bu çalışmada, her üç tür şaft açma teknikleri incelenerek karşılaştırmaları yapılmıştır. Konvansiyonel metod daha detaylı olarak araştırılmış ve Rize-Çayeli'nde konvansiyonel metodla açılan bir şaft inşaatı örnek çalışma olarak incelenmiştir. İnşaat aşamaları izah edilmiş, kullanılan ekipman alternatifleri ve inşaatın ilerleme hızı incelenmiştir.

TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

LIST OF SYMBOLS

B	Burden distance
Bo	Optimum burden distance
db	Blasthole diameter
dc	Diameter of the charge
di	Shaft inner diameter
do	Outher diameter
EF	Explosion energy yield per m3 of rock
et	Dynamic tensile breaking strength
F	Safety coefficient
f'c	Compressive strength of concrete
FL	Cutter force
H	Depth of the shaft
hb	Blasthole length and/or face height
ib	Basic penetration rate
K	Constant number
Kd	Correction factor of joint rating and frequency
Ks	Correction factor of cutter diameter
lc	Length of the charge
Ls	Stemming length
p	Penetration per revolution
Pe	Uniform external pressure
Pim	Impolsion pressure
Pl	Uniform lateral pressure acting to the lining
Pr	Uniform radial stress
q	Length to diameter ratio
r	Average radius
r1	Inside radius
r2	Outside radius

S	Blasthole spacing
S_R	Compressive strength
t	Wall thickness
x	Distance measured from the shaft center
ε	Explosion generated strain
σ	Tensile strength of the lining material
σ_c	Circumferential stress
σ_r	Radial stress
σ_r	Radial stress
σ_R	Rock tensile strength
σ_t	Tangent stress

1. INTRODUCTION

Shaft sinking is the most typical and in the same time the most dangerous work in mining, and all the following thoughts and comments refer to vertical underground cavities in mines intended as connections to other underground cavities, of a straight course, of a minimum cross section of approximately 10 m^2, and of an essentially larger length in the vertical than in the horizontal direction.

Even though there are many other applications for shaft sinking for other branches of underground constructions such as; access shafts for tunnel construction in urban areas, ventilation shafts for long distance tunnels, hydro-power-station pressure shafts, hydro-power-station surge tanks, pumping main shafts in sewage systems, shafts to repositories of nuclear or chemical waste, it has to be recognized that worldwide some 90 per cent and more of all shafts are sunk for mining purposes. The international mining industry is forced today to increase production at lower costs and to ensure the future of mines in operation as well as the successful development of new ones.

The ways to reach these targets in order to overcome the problems arising from the fluctuation of market prices and exchange rates, inflation and escalating working costs are the improvement of the traditional mining methods or, where this is not possible, the search for alternative technologies which are faster, safer and more economic.

One of these technologies is shaft boring and shaft drilling, employed for the first time in 1904, when the first shaft was drilled by the so-called "Honigmann-Verfahren " in Germany.

Actual Market Volume and Frequency of Applications

The whole shaft drilling industry that includes contractors as well as equipment manufacturers is still suffering from the run down of US Atomic Energy Commission drilling program and the recession in mining activities in specific areas (e.g. USA and FRG) or for specific minerals (e.g. iron, oil, shale).

But taking into consideration the constant increase in output of coal worldwide starting in the seventies, and foreseeable up to the year 2000 including the slight optimism taking place in the metal and mineral industry the estimation in Table 1.1 for shafts, needed up to the year 2000, is classified as conservative; especially as the economic climate appears to favor an increase of shaft sinking work by the reveal of formerly canceled or delayed projects for new mines of the extension of existing ones.

TABLE 1.1 : Forecast of numbers of shaft required worldwide up to the year 2000 [1].

1.	People's Republic of China	400
2.	Russia	300
3.	United States of America	200
4.	South Africa	150
5.	India	60
6.	Australia	20
7.	Western Europe	20
8.	Others	50
	TOTAL	1200

It was thought that the market potential for shaft drilling and boring techniques must be considerably less than this figure of 1200 shafts within one decade. Questions of dimensions, locations, and geological conditions would have reduced it at most to somewhere around 50per cent, which would still mean a potential of some 50 shafts per year to be drilled or bored.

It has to be pointed out as well that these shafts have to be subdivided essentially into two categories,

- Shafts where no connection between the surface and the level to mine is existing prior to the shaft being sunk to its total depth, under which circumstances the excavated rock has to be brought back through the excavation to the surface
- Shafts to existing underground workings whereby the removal of excavated rock can be effected by dropping it to the mined level via a pilot shaft established prior to the commencement of the shaft sinking work proper.

A private internal survey of shafts sunk by means of drilling or boring shows that starting in 1947 the number of drilled/bored shafts increased steadily, reaching an average level of 10 - 12 shafts drilled per year, taking the numbers of the last 10 years as a reference.

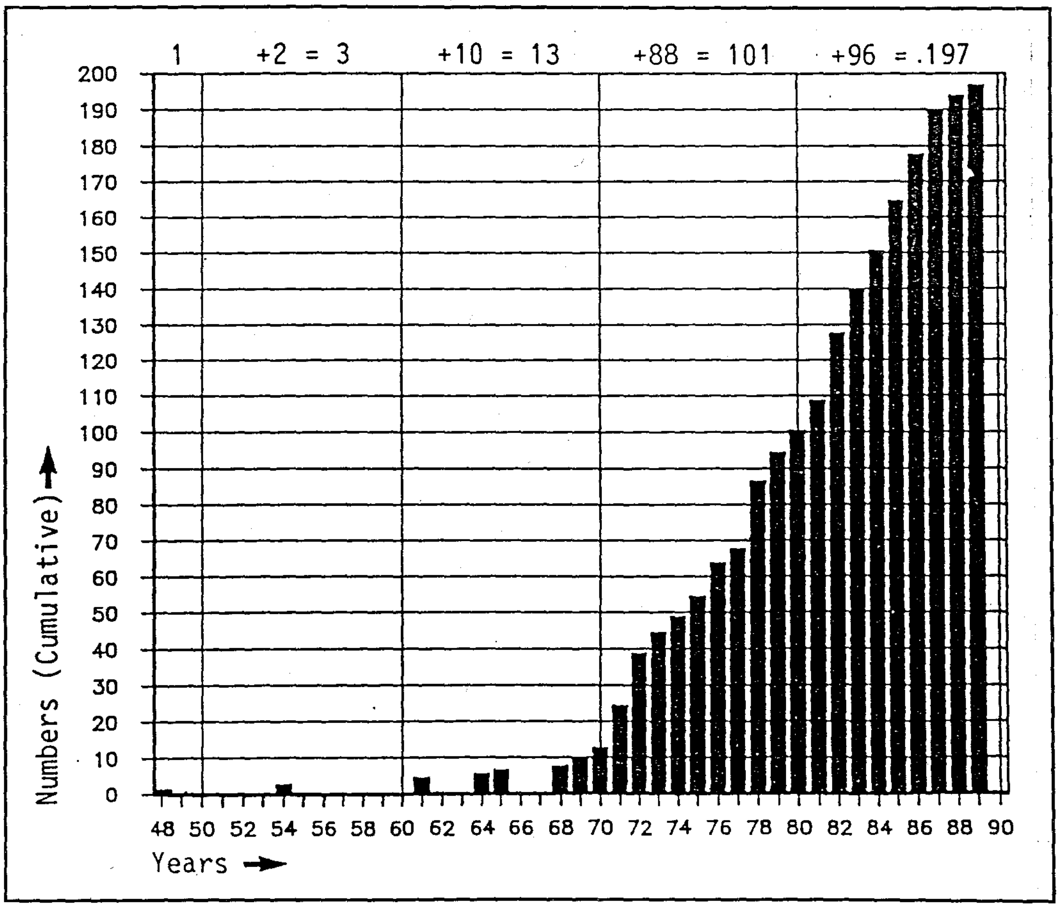


FIGURE 1.1. Frequency of application of shaft drilling/boring methods [1].

The tendency shown in Figure 1.1 indicates that a large part of some of the problems such as low penetration and high drilling costs must have been solved to some extent satisfactorily over the last 25 years to reach this level of application of the methods. But it has to be recognized as well that the frequency of application of drilling/boring in shaft sinking of an order of 20 per cent of the existing potential is still too small for shaft drilling in general to be considered “the standard alternative “ to the conventional shaft sinking method with drill and blast.

For the earlier mentioned two categories of shafts the following drilling and boring methods are available today besides the conventional shaft sinking method.

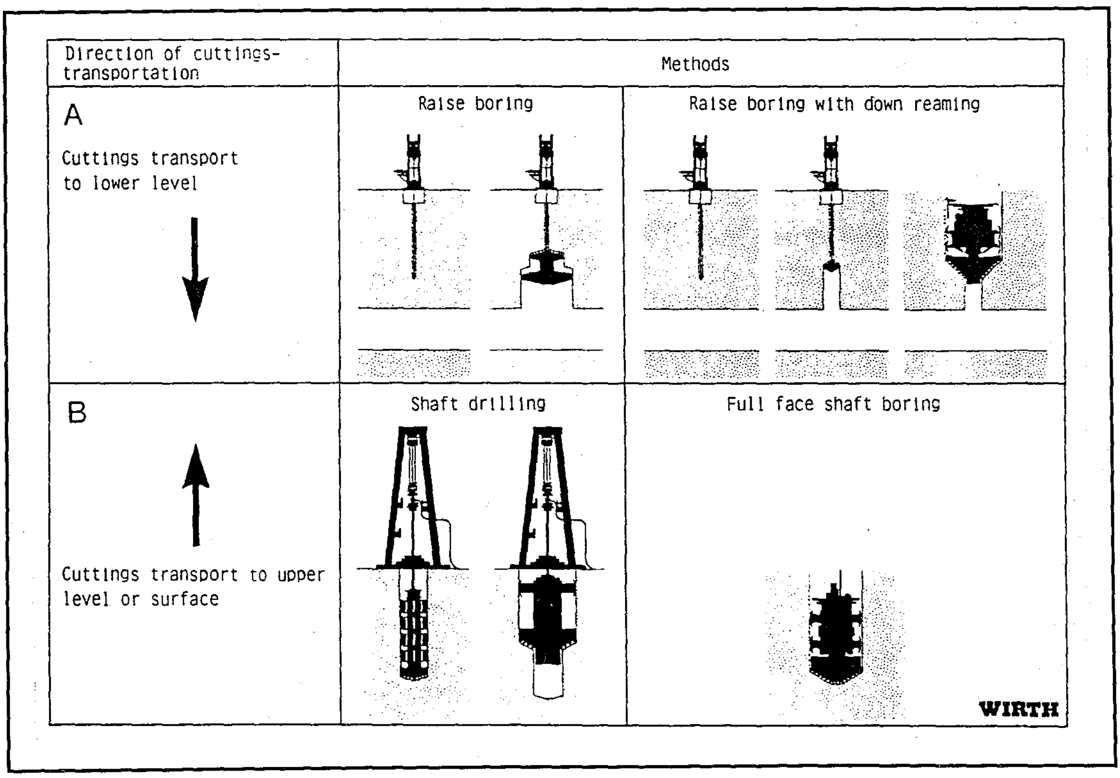


FIGURE 1.2. Systematic of shaft drilling and boring methods [2].

A) Shaft boring methods with continuous muck removal to a lower level
B) Shaft drilling or boring methods with muck removal to the surface or to the next higher level.

The methods indicated under A) include :

(a) raise boring method, and

(b) a combination of raise - boring and down reaming by shaft boring machines without drill pipe.

The shaft drilling or boring methods with muck removal to the surface or to higher levels, respectively, have to be subdivided into:

(a) full-diameter drilling methods using machines with drill pipe, and

(b) full-diameter boring methods using machines without drill pipe.

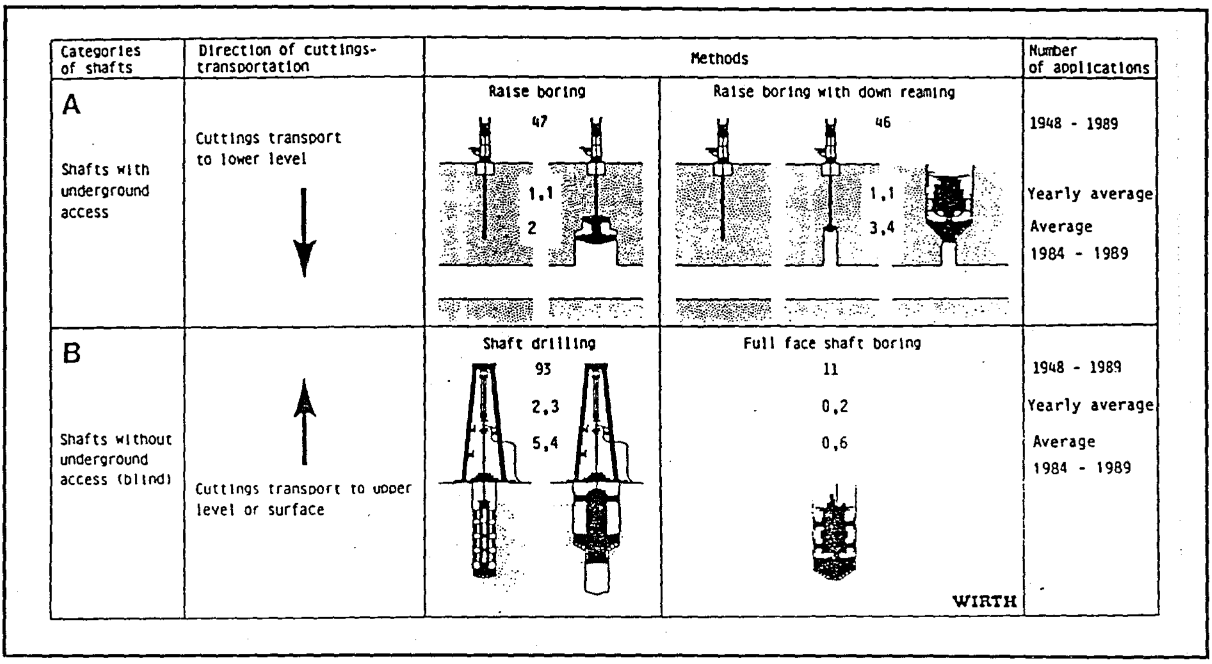


FIGURE 1.3. Number of applications with respect to the different methods [1].

Analyzing the number of shafts drilled or bored so far it is obvious that the full diameter boring method without drill pipe is still in an experimental state, far away from reaching the stage of industrial application.

Rotary shaft drilling was used on 93 shafts from 1948 to 1989. This represents the same numbers in total as for raise boring, and the combination of raise boring and down reaming as indicated. As both last mentioned methods started to be used very much later than rotary shaft drilling, the market share of each method can be measured more realistically by comparing the yearly average of use. This shows that all three methods have more or less the same frequency of application.

Using the frequency of application over the last five years only as an indicator for the future, it could be deduced that the use of raise boring is stagnating, the application of the combined raise boring and down reaming technique is increasing and the rotary shaft drilling technology will become the predominant method [1].

2. SHAFT SINKING METHODS

Despite the expectations, “Shaft drilling” or “Shaft boring” did not find a broad application during the last 5-10 years. However, there had been a general tendency to declare medium to large sized drill holes as shafts. The reason of that was to show ample acceptance of shaft drilling technology and increase the number of references.

Though this attitude was usual and partly understandable in the past, nowadays the state of art of shaft drilling or shaft boring has reached a level, which implies the expectation that in the near future mechanical shaft excavation will change worldwide into at least the first technological alternative to conventional shaft sinking and especially to the special shaft sinking methods still applied nowadays, without having revert to large-diameter drill holes as a reference for worldwide application of shaft drilling. But today the most widespread method in shaft sinking is still conventional shaft sinking.

From the results also the necessity to define the term “ shaft “ whereby a strict demarcation line between “shaft “ and “large-diameter drill hole” in terms of the cross-section does not appear to be purposeful as a single criterion.

As far as the difference of these terms is concerned, it has to be taken into account that the terms “shaft “ and drill holes excluded “ à priori “ mixing these two terms. That mutual exclusion applied over a long time however also to the different methods of construction (Table 2.1).

Though shaft drilling has now almost its 100th year anniversary, the essential development of the drilling technique, i.e. the possibility to economically drill holes of large diameters took place only during the last 40 years.

TABLE 2.1. Division between drill holes and shafts in respect of dimensions and methods of construction as used in the past [2].

	drill holes	shafts
Dimensions	Dia. < 1m	Cross-section 2x2 m^2
Method of Construction	Drilling (rock is excavated mechanically)	Conventional sinking (drilling, blasting, muck loading and hoisting incl. special methods)

During that period of development there also resulted a supplementary need for earth cavities, which as far as their dimensions are concerned cannot be categorized under the term "shaft " nor under the term "drill hole," i.e. the so-called "big holes," which are almost exclusively constructed using drilling methods.

Shaft is defined as an underground cavity of an inclination of 45 to 90 degrees to the horizontal line, of a straight course, and to be considered a connection to other underground cavities, of a minimum cross section of approximately $10m^2$, and of an essentially larger length in the vertical than in the horizontal direction. On the basis of this definition, it can be ascertained that most of the shafts required worldwide are nowadays still constructed by means of conventional shaft sinking.

Appropriate drilling methods predominate in the construction of big holes and shafts up to about four meters drilling diameter. In that drilling diameter range the frequency of application of the drilling methods drops quickly to less than 10 per cent. Subsequently, the frequency of use drops more or less linearly to zero at a cross-sectional area of approximately $64m^2$ (Figure 2.1).

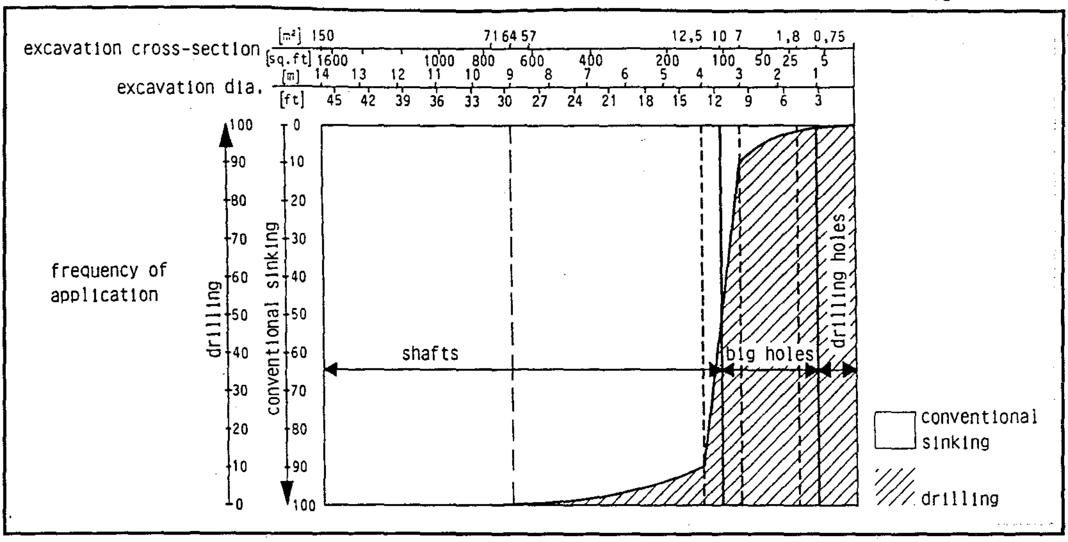


FIGURE 2.1. Frequency of application of drilling methods related to excavation cross-section and drilling diameter, respectively [2].

Under “usual shaft sinking,“ the conventional technology is understood as breaking rock by blasting, loading and hoisting of the broken rock and installation of the lining, including all pertinent special methods, such as; caisson sinking, coffer dams, ground water lowering, grouting, and freezing.

All these methods resulted in construction of shafts up to 14m diameter (Bath County, Virginia/USA, 100m depth) and up to 3300m depth (Western Deep Levels, S.A., and inner cross-section 75m^2) [2].

The very big shafts and the elliptical shafts are still the province of the drill and blast industry. If the mine design requires a very large, noncircular opening, the drilling option cannot be considered. However it must be acknowledged that equipment is now under development in the area of road-header and mobile-miner (disk-cutter-wheel) type shaft sinking, and it may, therefore, be theoretically possible to excavate by mechanical means any size, shape or depth of shaft. In a practical sense, however, most of the equipment fielded in this latter class is not ready for “prime time “ use and has yet to be proven consistently competitive with conventional construction. The maximum diameter for both blind and raise drilling is now somewhere between six and nine meters. Diameter and depth combine as a limiting factor in big hole drilling - the largest diameter cannot reach the greatest depth [3].

2.1. Shaft Boring

Shaft boring machines are available in three principal configurations: standard raise drills, reversible raise drills and blind-hole or box-hole drills. Standard drills are the most widely used. They are set up on one mine level or the surface to drill a pilot hole to a lower level, and the raise is then back-reamed. With reversible drills, the machine is established on a lower level, and a pilot hole is drilled to the upper level. The raise is then reamed from the top down. Usually, such machines can be reversed and used as a standard raise drills. Blind-hole or box-hole machines, which have found increasing use over the past five years, usually stand on a lower level and bore full-diameter raises up without the use of a pilot hole. However, the blind-hole concept has also been used to drill shafts downward vertically from the surface, as is currently being done at the Crownpoint uranium project in New Mexico [4]. If there is sufficient access at both an upper and lower level, the standard raise drilling method is usually recommended, because the equipment is less complex and easier to handle.

2.1.1. Shaft Boring Machine

The Shaft Boring Machine (SBM) is similar in concept to the tunnel-boring machine (TBM) except that it is directed vertically downward rather than horizontally. The equipment and the personnel are placed within the shaft opening. The rock is excavated using roller cutters. Systems are provided for removal of cuttings from the shaft for installation of ground support. So the shaft is excavated to its required diameter from the top to downwards using an in-shaft boring machines. Often this type of machine follows a pilot hole so that the cuttings may fall freely to an underground removal system. It may be used for blind boring when combined with systems for collecting and transporting muck to the surface.

Because of the speed of excavation, the ability to excavate a wide variety of materials, the inherent stability of the resultant circular opening, the labor saving potential and a safe working environment, SBM have gained wide acceptance in civil work projects. For the same reasons, they have been using in mining industry.

2.1.2. Components of Shaft Boring Machines

Shaft boring machine components can be divided in to two major groups; rotating components and non-rotating components.

A basic shaft-boring machine has the following rotating components;

- Cyclone Separators, Carousel Paddles, Elevator Discharge Chutes, Main Bearing and Seals, Disc Cutters, Scrapers, Bucket Elevators, Flight Conveyers, Cutter Head, Myno Pumps.

Non-rotating components of a basic shaft-boring machine are as follows;

- Hydraulic Swivel, Air Swivel, Exhaust Ventilation Duct, Carousel Gate, Measuring Pockets, Skip Load Chutes, Skips, Transfer Platform, Rear Shields, Gripper Shoes, Guide Rope Sheaves, Stabilizers, Torque Cylinders, Cutterhead Drivers, Gripper Cylinders, Propel Cylinders.

Description of the some important components of the machine is as follows.

<u>Cutter Head:</u> The cutter head support, which houses the non-rotating inner face and double-rowed tapered roller-bearing, is the principal load bearing structure of the shaft boring machine, providing both torque reaction from the rotating cutter head to the stationary gripper ring and the thrust of the stabilizer on the shaft wall, providing directional stability for the machine. The head consists of a central circular section with radial extensions each carrying roller cutters, including the

gauge cutters and scrapers. Oil for the various hydraulically driven auxiliary functions which rotate with the cutter head, such as the flight conveyors, bucket elevators and slurry pumps, etc. must pass through the rotary swivel at the top of the central column before being fed to the hydraulic motor in question.

Propel System: Hydraulic propel cylinders-in group are arranged at 120^0 intervals around the central section, mounted between the gripper ring and the cutter-head support which are capable of producing hydraulic thrust. During the propel cycle stabilizer shoes are extended against the wall, but not gripped, to provide stability. Gradual steering corrections may be affected by deliberately creating an imbalance between the stabilizer shoes.

Muck Removal System: Scraper blades on the cutter head move the cutting towards the lowest point of the cut face where they are collected by horizontally positioned single-chain flight conveyors and swept up ramps. The cuttings are deposited from each of the two ramps into independent hydraulically driven bucket elevators. Polyethylene buckets mounted on these chain-type elevators lift the muck vertically and drop it into an annular collection hopper or carousel. Paddles, which extend down from the rotating structure in the center into the non-rotating collection carousel, move the spoil around to either of two openings in the base and the trough, so that it falls into one of measuring pockets. Hydraulically controlled gates on the measuring pocket allows the spoil to drop into a skip capable of taking the full pocket load. The system is so arranged that when the measuring pocket is full, the gate under the carousel closes and a gate beneath the measuring pocket opens to allow the spoil to drop into a waiting skip. The skip carries the material to the top of the shaft.

Ground Support and Gripper System: Temporary ground support is provided by a full shield, which extends from immediately behind the cutter head above the muck transfer deck. The stabilizer shoes and the grippers are operated through the windows in the shield. The gripper system uses shoes, forming a ring mounted around the periphery of the machine. At each joint a pair of hydraulic cylinders, tangential to the circumference connects the shoes, which expand the ring against the shaft wall. Alternating protrusions and recesses are provided on the outer surface of the gripper ring. The protrusions extend through openings in the shield and contact the shaft wall.

Dewatering and Air-ducting System: Mono slurry pumps, which also rotate with the head, are available for dewatering the hole bottom. Their output may be directed through a cyclone or, if requested, dropped directly into a setting tank. To keep the manned areas free from the dust and the methane gas fresh air is ducted into the working section. The air passes down to the face of the machine through an annular clearance allowed between the upper cutter-head support deck and the shaft wall. From the hole bottom the air is drawn up through the machine central column and into a duct running to the surface [5].

2.1.3. Operation of SBM

Normal SBM Operation: Operation of the SBM will be described by reviewing a 0.6m boring cycle. At the start of the cycle, the walking cylinders are fully extended, both sets of grippers are clamped against the shaft walls and the machine has been properly aligned. The cutterwheel/carriage is fully retracted in relation to the main frame. The muck hopper is empty and is in the loading (down) position.

The cutterwheel is started and the plunge cylinders extended forcing the cutterwheel to penetrate 57mm into the rock. This is the depth of cut that will produce approximately 4.6m^3 of muck in 540^0 of slew rotation. When the plunge is complete, the slew drive and mucker systems are energized. The slew rate depends on borability of the rock and is adjusted by the operator to maximize power on the main drive motors. As slew rotation proceeds, cuttings are pushed into a pile in front of the scrappers, picked up by the hydraulic clam and discharged into the muck hopper. The mucker is controlled by computer and uses feedback signals from magnetic proximity detectors and hydraulic pressure sensors.

After 540^0 of slew rotation, the hopper will be filled to its 4.6m^3 capacity. The slew assembly has made 1 and 1/2 revolutions and is now aligned with the hoisting bucket opposite that which was dispatched up the shaft prior to the start of the cycle. The slew motion is stopped. The mucker is shut down and rotated to a storage position. The hopper is then raised to the dump elevation. Simultaneously, the hoisting bucket is lowered through the SBM to a position alongside and lower than the hopper. The extendible chute is activated and the hopper is discharged into the bucket. The bucket is then hoisted out of the SBM and the hopper returned to the loading elevation. The next plunge and 540^0 slew cycle are then performed, followed by muck transfer to the opposite hoisting bucket. This sequence of plunge and slew cycles is repeated until the shaft has been advanced 0.6m. At this point the machine must be reset.

The cutterwheel is stopped and raised using the plunge cylinders. The upper grippers are released and lowered by retracting the walking cylinders. The upper grippers are then extended, the lower set are released and the machine is lowered 0.6m by extension of the walking cylinders. Finally, the lower grippers are set and the next boring cycle may be started.

Directional Control : Directional corrections are made at the end of each 0.6m boring cycle while the SBM is supported by the upper grippers and is hanging from the walking cylinders. Two laser beams on opposite sides of the shaft establish coordinate position and inclinometers provide directional reference. Positioning is started by leveling the SBM using appropriate pairs of walking cylinders. Roll position is adjusted using the roll cylinders. Finally, coordinate location is established by adjustment of the lower grippers.

Ground Support and Lining: Shafts bored with the SBM will normally be lined with concrete. Conventional shaft concrete forming and placement methods will be used. This will be performed using a stage above the SBM. A concrete lining crew will work in parallel with operation of the SBM. Delays to the boring operation will be limited to movement and initial pouring of the curb ring when the SBM operators will assist the concrete crew. Other types of temporary or permanent ground support may also be installed from the stage.

When adverse ground is encountered, rock bolts, mesh or ring beams may be installed to within twometers of the bottom. This is slightly above the point of tangency of the cutters with the shaft wall. Platforms are provided on the lower part of the SBM from which ground support or probe hole drilling can be conducted. It would be necessary to suspend boring during these activities. When required, concrete lining can be placed to a point immediately above the upper set of grippers approximately 7.5m above the shaft bottom.

2.1.4. SBM Performance

SBM Operation and Boring Rate:

A two-man crew will operate the SBM. An operator will control basic machine functions and maintain alignment. An assistant will oversee mucker system operation and will handle bell signals related to bucket loading and dispatch. In-shaft labor will also include a three-man concrete crew. Surface employees will include hoistmen, operators engaged in materials handling and muck disposal, as well as maintenance, supervisory and engineering personnel. A typical crew for a 3-shift per day operation is shown in Table 2.2.

Predicted boring rates for various rock types are illustrated in Figure 2.2. These estimates are based on actual TBM performance in the various rocks with adjustments made for anticipated characteristics of the SBM cutterwheel. The mucker system has $0.5m^3$ bucket and a 20 second cycle rate providing a maximum design capacity of about $1.4m^3$ per minute. In soft rock this would limit instantaneous penetration rate to 1.73m/hr for a 6.7m diameter shaft. Penetration rate must be derated for muck bucket loading and for resetting following each 0.6m boring cycle. Maximum overall advance would then be 1.03m/hr and 0.82m/hr in Chicago limestone. Predicted operation in this rock, in which a large footage of tunnel and shaft boring has been performed, will be used as the "standard" rock type for performance and cost estimates presented in this study [6].

TABLE 2.2. SBM Operating Staff [6].

Job Description	Number of Employees			
	Day Shift	Afternoon Shift	Night Shift	Total
	Supervision and Engineering			
Project Manager	1	0	0	1
Mechanical Superintendent	1	0	0	1
Project Engineer	1	0	0	1
Engineer	1	0	0	1
Shift Supervisor	1	1	0	2
Total Indirect	**5**	**1**	**1**	**7**
	Maintenance and Indirect			
Mechanic	3	1	1	5
Electrician	1	0	0	1
Surface Labour	2	1	1	4
Total Staff	**6**	**2**	**2**	**10**
	Direct Operation			
SBM Operators	1	1	1	3
Mucker Operator	1	1	1	3
Ground Support (Miners)	3	3	3	9
Hoistman	1	1	1	3
Toplander	1	1	1	3
Total Direct	**7**	**7**	**7**	**21**
TOTAL EMPLOYEES	**18**	**10**	**10**	**38**

Reliable empirical approaches to estimating penetration rates for full-face machines are generally less complex than approaches for road headers. As reviewed seven published methods and concluded that one simplified method Farmer and Glossop developed a method that based on thrust per cutter and tensile rock strength, provided good correlation with actual penetration rate rates from 20 case histories. A complex model suggested by Lisleryd in 1983 was considered to have the potential for more accurate penetration prediction but was discount on the basis of cost [5].

These equations are also used for SBM applications but adjustments are made for anticipated characteristics of the SBM cutterwheel.

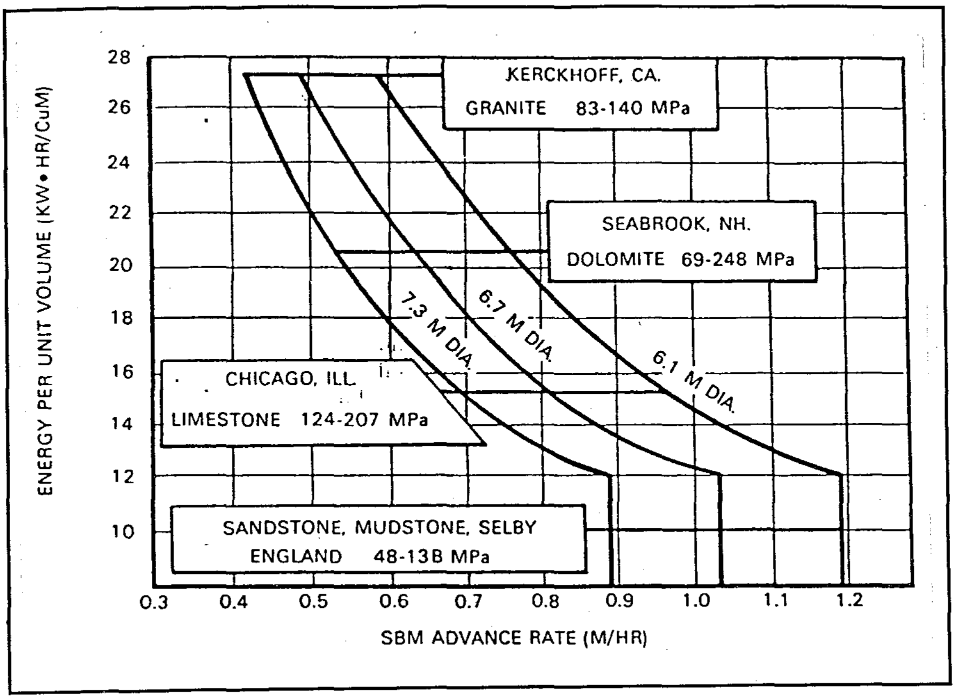


FIGURE 2.2. SBM Boring rate as function of rock type shaft diameter [6].

Method Described by Farmer and Glossop [5]:

Farmer and Glossop derived a relationship between the average cutter force F_L, the penetration per revolution P, and rock tensile strength s_R by equating the energy input per unit length of cut to the energy required to satisfy fracture surfaces in the rock:

$$P = K^*F_L / s_R \tag{2.1}$$

The value of K was obtained by least squares regression of eight cases where P, F_L and s_R were measured, so that in SI units:

$$P = 624^*F_L / s_R \tag{2.2}$$

s_R = (kPa), P=(mm/rev), F_L=(kN)

Graham has found a similar equation; it is based on use of the Robins TBM in hard rock with confined compressive strengths 140-200 Mpa.

$$P = 3940*F_L / s_R \qquad (2.3)$$

s_R : uniaxial compressive strength (kPa)

Method Described by Lislerud [5]:

Lislerud has developed a TBM performance prediction method on rock mass factor (rock mass jointing, intact rock strength, brittleness and abrasivity) and machine factors(thrust per cutter, cutter edge, bluntless, cutter spacing, cutter diameter, torque capacity and rpm and cutterhead curvature and diameter) Lislerud's equation for penetration is :

$$P = i_b*K_s*K_d \quad \text{(mm/rev)} \qquad (2.4)$$

i_b : basic penetration rate (mm/rev) (is a function of the thrust per disc and the drilling rate index.(DRI))

K_s : a correction factor of cutter diameter

K_d : a correction factor for joint rating and frequency

As noted this method requires a considerable amount of geotechnical and laboratory test data and is probably only suited to high-grade metamorphic rocks such as those found in Scandinavia. In less anisotropic rocks the relationships that are suggested by Farmer and Glossop and Graham is generally used [5].

Concrete Cycle : It is assumed that a 6.1m form is used. It is estimated that 120 min. would be required to move, set and pour the curb rings section using the full shaft crew. The remainder of the concrete cycle will be performed simultaneous with boring and does not affect overall rate of progress.

Concrete cycle : 120 min per 6.1m pour = 19.7 min/m

Utility Installation : Temporary in-shaft utilities are assumed to consist of :

Quantity	Description
1	1.07m dia ventilation pipe
1	150mm dia compressed air pipe
1	150mm dia concrete slick pipe
1	150mm dia pump discharge pipe
1	50mm dia water supply pipe
1	Electrical feeder cable
1	Lot communications and signal lines

The estimated time to install 18.3m of utility lines is 180 min using the full shaft crew.

Utility cycle : 180 min per 18.3m = 9.8 min/m

Cutter Change : Twenty-eight cutters are installed on the cutterwheel. An access door and handling tools are provided to expedite cutter inspection and change-out. The process is efficient and avoids the laborious handling and access problems typical of TBM cutter maintenance. Predicted cutter life for 6.7m SBM operated in Chicago limestone is 270 hours. It is assumed that 60 min are devoted to routine cutter inspection and random change-out each 10hr of operation approximately 12.2m of advance. It is further assumed that a full cutter change is made every 270hr or 330m of advance. This will require 16hr based on 2hr setup plus 30min per cutter. Time for cutter change is then as follows:

Inspection/random change : 60 min per 12.2m = 4.9 min/m
Cutter Change : 960 min per 330m = 2.9 min/m

Total Cutter Change 7.8 min/m

SBM Operation and Maintenance: After allowing for cutter maintenance, estimated mechanical/electrical availability of the SBM is 80 per cent. Adjustment of predicted overall boring rate for Chicago limestone, which is 0.82 m/hr, to provide for maintenance results in:

SBM operation/maintenance : 91.2 min/m

Plant Delays and Overall System Performance : In addition to time for in-shaft operations, surface plant maintenance and personnel related delays are estimated to average 1 hour per shift. Productive work time is then 21hr per day and overall performance would be as shown in Table 2.3.

Table 2.3. Overall system performance [6].

Description	Time / Foot	Time / Meter
Concrete Cycle	6.0 min.	19.7 min.
Utility Cycle	3.0 min.	9.8 min.
Cutter Change	2.4 min.	7.8 min.
SBM Operation / Maintenance	27.8 min	91.2 min.
TOTAL	39.2 min.	128.5 min.

Advance per day (based on 21 hr) is 9.81m

Similar estimates for different rock types and shaft diameters are presented in Figure 2.3.

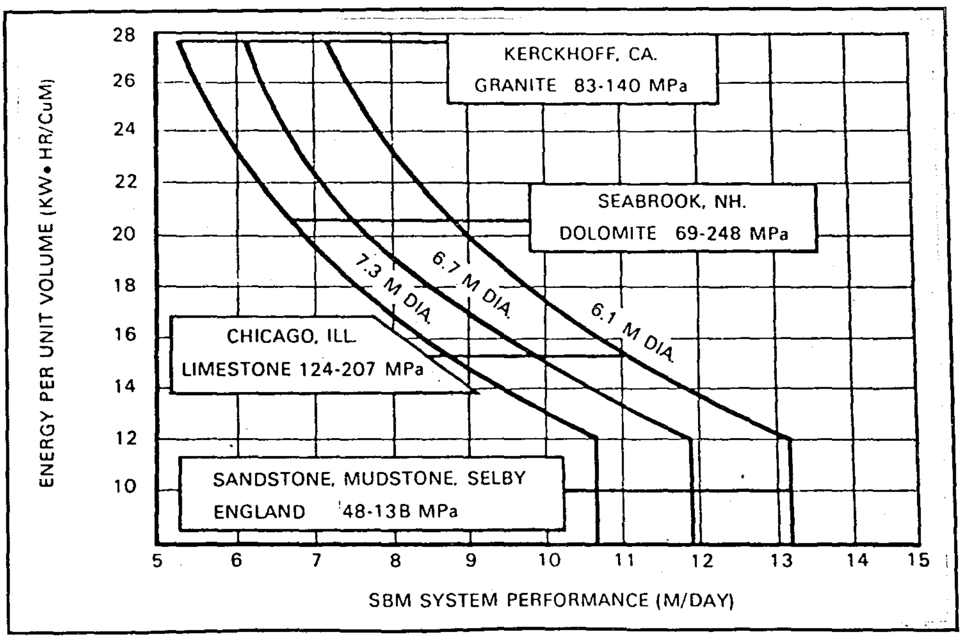


FIGURE 2.3. SBM Overall System Performance as Function of Rock Type and Shaft Diameter [6].

The predicted overall SBM performance is believed to be conservative. Thus, in the case of the 6.7m diameter shaft bored in Chicago limestone, an advance of 9.81m per day requires that the SBM operate approximately 50 per cent of total time. This is routinely achieved with TBMs. In comparison to tunnel operations, the hoisting and backup systems in a shaft should impose significantly less delay on SBM operation than is generally the case for TBM haulage systems. Maintenance and cutter change should be faster with TBM. Adverse ground conditions are easier to handle in shafts than in horizontal drives. A potential negative factor is the greater difficulty of disposal of water inflows from a shaft. Evaluation of the SBM design provides further support for the predicted performance. Some earlier shaft boring systems have cut effectively but have not been commercially viable due to problems in removal of cuttings from the bottom. In this case, while cutterwheel geometry is new, it has been evaluated by full-scale field tests and by operation of the prototype Mobile Miner. The mucker system, which is the key to success, is an adaptation of standard techniques and hardware.

2.1.5. Backup System Design

General Considerations Regarding Backup System :

Projected SBM performance is several times faster than is normally achieved with drill-and-blast shaft sinking. The 9.8m/day SBM rate in Chicago limestone compares to conventional sinking rates of 2.4 to 3.7m/day that would be expected in North America. Despite a significant speed advantage during actual sinking, most North American shafts are not deep enough to allow translation of this speed into a worthwhile reduction in overall project time unless efficiencies are also achieved in the mobilization/setup/take-down/demobilization phases. Construction of a typical 300m shaft requires about 235 days, of which only 90 days are used for actual sinking. The remaining 145 days are devoted to setup, collar excavation, station construction and demobilization.

Mobilization and removal of the SBM will be more complex than a conventional shaft set-up. Assuming, however, that this can be performed within the same period, then project time would be reduced by 60 days or only 25per cent for a 300m shaft, even though the boring rate is 3.2 times as fast as the 3m/day conventional sinking rate assumed for this example. This illustrates the importance of careful design of the backup equipment so that the plant used with the SBM will function as an efficient overall system. Some of the criteria and concepts applicable to this area are discussed in the following section of this study.

General Description of Support Plant : The plant will include a headframe supporting hoist and stage pores. Double-drum hoist and stage winches are required. Facilities for maintenance, change house, office and utility services will be required. A work stage will be provided for use during concrete lining or ground support operations.

Handling of the SBM : The 6.7m diameter SBM weights 180 t. While it is capable of walking up and down the unlined shaft, it is felt that for reasons of safety and time that this procedure should be confined to within a few feet of the shaft bottom. The SBM will not walk through stations or caving zones where significant removal of the circular section has occurred. Also, the grippers cannot be retracted far enough to clear normal concrete lining. The assembly or disassembly of the SBM is a major task, which would be easier and faster if performed on surface rather than within the shaft. It is concluded that provision should be made for hoisting the SBM in or out of the shaft as a reasonably complete assembly. This will be accomplished with an appropriate stage suspension system and gantry structure. A few minor components of the SBM must be removed or relocated during the hoisting or lowering operation. It will also be necessary to adjust center of gravity of the machine by relocation of the cutterwheel. This procedure will be aided by a system of pivot points, linkages and chain hoists that are integral components of the SBM.

<u>Stage Suspension System :</u> The upper deck of the shaft will be of heavy construction and will be easily detachable from the lower working platforms. An eight-rope suspension system using 40mm diameter high-strength ropes connected to either two or four stage winches (depending upon shaft depth) will be provided. Using winches with a rope pull of approximately 311000N this system will be capable of lifting the entire weight of the SBM plus the shaft stage.

<u>Shaft Gantry Structure :</u> A gantry structure will be used as a base on which assembly of the SBM will be performed. This will be supported on rails on which it may be rolled over the collar of the shaft. When assembly and checkout of the SBM have been completed, the gantry will be moved over the shaft. The SBM will be lifted off the gantry and lowered into the collar. This process will be reversed when the SBM is removed from the shaft.

<u>Shaft Stage :</u> A three-deck work stage will be provided. The upper deck will incorporate the sheaves for the eight-rope suspension system. This structure will be designed for the heavy loads imposed while handling the SBM. During normal boring operations, the stage will provide a platform from which installation of concrete lining, ground support and shaft services will be performed. It will be equipped with appropriate communications, signaling and lighting facilities.

<u>Headframe :</u> The headframe used during boring must support the heavy loads imposed while lifting the SBM. It must provide adequate clearance to permit handling the assembled SBM. These criteria would not be satisfied by most permanent headframes and a special headframe will be used on most shafts bored using SBM. Since the headframe will be used on number of shafts, the speed and simplicity of the setup/take-down process are important. A design, which satisfies the criteria for handling of the SBM and contributes, to rapid mobilization/demobilization is shown in Figure 2.4. The components of the headframe will be bolted together on the ground. A stiff leg will be installed near the back leg foundations. A winch and multi-part line will then be used to elevate the headframe to its erected position.

Main Hoist : A double-drum hoist will be required. The SBM hopper and operating sequence are based on a 4.6m^3 hopper system and production of this quantity of muck each 540^0 slew cycle. Ideally, the hoist should be sized to handle the 9t load produced by this mode operation. Hoisting speed and horsepower would then be dependent upon the anticipated boring rate and depth of the shaft. Use of a hoist with a smaller payload and/or lower speed and power would limit overall production rate.

2.1.6. Schedule and Sequence of Operations

The construction of a representative shaft using the SBM is described in the following sections.

Mobilization and Shaft Collar Construction : A collar must be completed to a minimum depth of 8m or far enough into competent rock to permit normal operation of the machine and its gripper system. The collar must be lined to the same diameter that will be bored.

When mobilization and site preparations have been completed, the first significant construction activity will be excavation and lining of the collar. The SBM will be assembled on its gantry structure while collar construction is in progress. Installation and commissioning of the hoists stage winches and remainder of the surface plant facilities will be performed in parallel with these operations. When the collar is complete, the headframe will be bolted together on the ground and erected. The upper deck of the stage will then be connected to the stage ropes. The SBM will be moved over the collar on its gantry. The SBM will be lifted off the gantry and lowered into the collar using the stage winches and upper deck of the stage. The SBM will then be used to deepen the collar to 30m. A crane and bucket will handle muck during this operation. Any temporary ground support that is required will be installed from the shaft bottom or the upper deck of the SBM.

At a depth of 30m, adequate clearance will be available for completion of the sinking setup. The forms and stage will be assembled in the shaft above the SBM. The collar door and dump structure will be erected. The hoist will be roped up and attached to the crossheads. Permanent concrete lining will then be placed through the collar section and routine SBM boring operations initiated.

Station Excavation : Drill and blast methods will be used during station excavation. The shaft will be bored 10m below the floor of the proposed station. The concrete lining will be stopped at least 22m above the brow of the station. Temporary ground support may be installed, if required, in this unlined section. The SBM will be prepared for lifting, attached to the stage using special slings and be raised so that its lower end is at least 10m above the stage. It will be clamped at that elevation using the gripper system. A 1.7 by 1.7m^2 opening is provided through the SBM for passage of equipment and muck buckets. The station will be mined and supported using equipment such as Eimco 630 muckers to load the buckets. When the station is complete, the SBM will be lowered to the bottom and boring resumed. When shaft station excavation is not required while sinking, or there are a large number of them, it will be probably proved more efficient to complete boring and removal of the SBM. Stations would then be mined using a portable steel bulkhead (Figure 2.5).

SBM Removal : Short moves of the SBM within the shaft, such as required during station excavation, will be performed by suspending the SBM below the full shaft stage. When the SBM is to be handled in the headframe, however, it will be necessary to remove the lower decks since the combined length of the stage and SBM would exceed headframe clearances. The stage will first be hoisted to the collar. While doing this, ventilation duct will be removed to provide clearance for the SBM. Other temporary utility lines may also be removed. When the stage reaches the collar, the dumps and collar door structure will be removed and the lower decks of the stage will be disconnected and taken out of the shaft. The upper deck will then be lowered down to the SBM and connected to it. The SBM will then be hoisted up the shaft using the stage winch system. When it reaches the collar, it will be placed on the gantry and moved off to the side for disassembly [6].

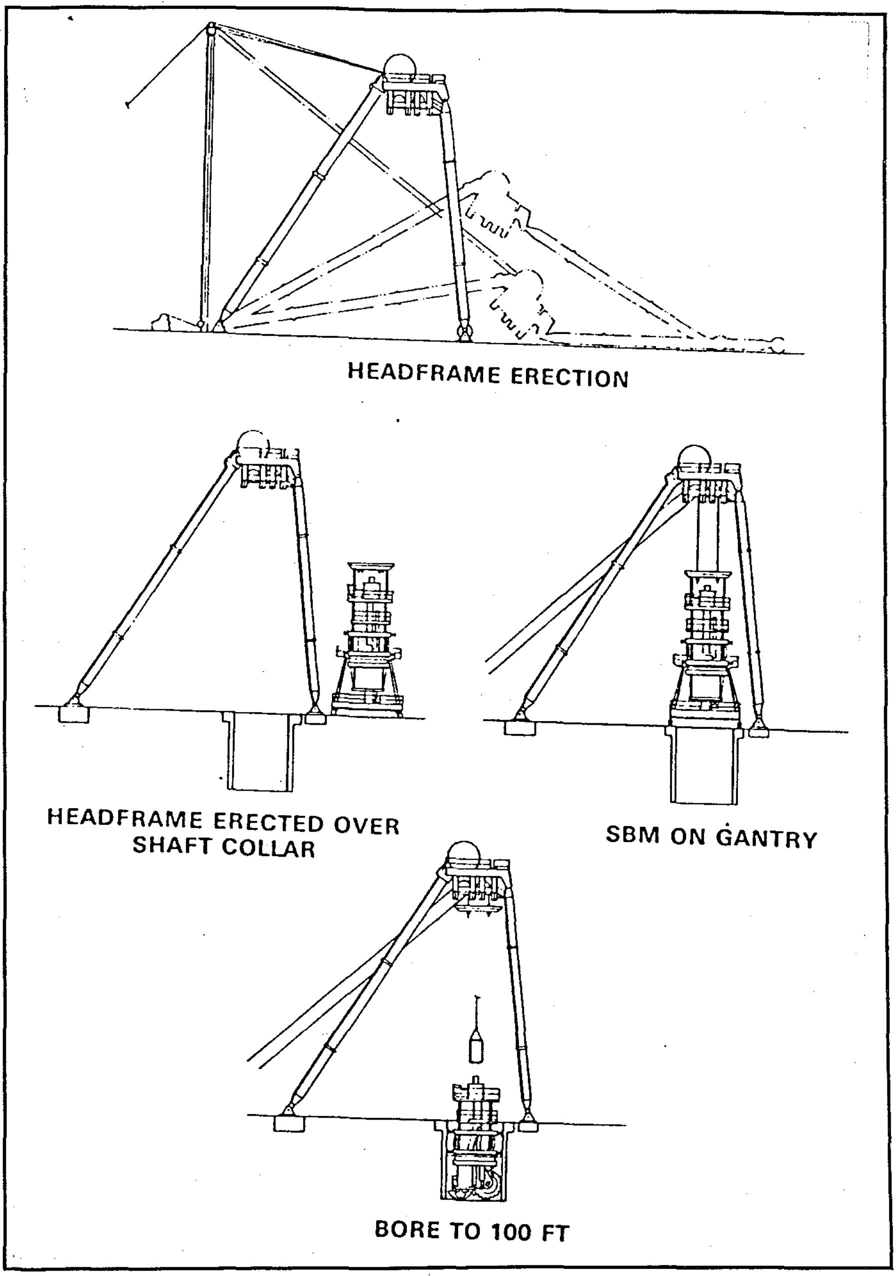


FIGURE 2.4. Shaft construction sequence using SBM set-up and collaring operations [6].

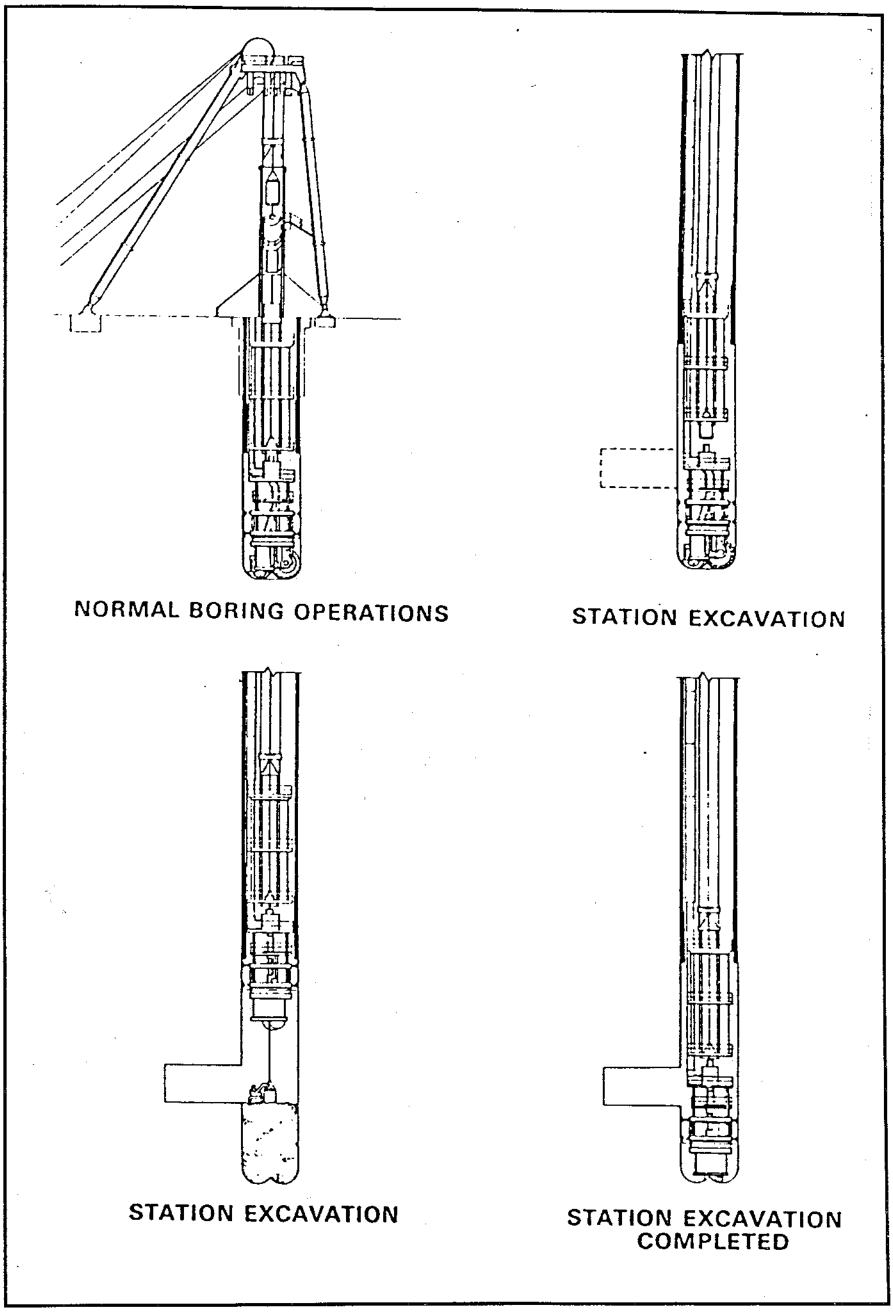


FIGURE 2.5. Shaft construction sequence using SBM normal boring and station excavation [6].

2.1.7. Safety on the Shaft Boring Machines

Sinking shafts is considered one of the most hazardous operations in the mining industry. From 1984 to the end of 1989, according to a study by the Mines Accident Prevention Association of Ontario (MAPAO) reported by Dursun, G and Algürkaplan, E [5], 13 per cent of all contractors' accidents occurred during shaft sinking projects.

The following safety interlock feathers are incorporated in the shaft boring machines:

- The muck-loading gate can not be opened unless a skip is present
- The upper gate under the carousel closes when the storage pocket is full
- The skip can not leave until the lower gauge is closed
- If the both measuring pockets are full, machine "shutdown" is activated
- If graper drops under a certain limit head rotation is interrupted
- Head rotation is also halted if low lubrication oil pressure is indicated
- If gas level of 2 per cent is recorded on any of continuous methane monitors located at the strategic position, power at the machine is automatically interrupted.

Other safety interlock feathers incorporated include stage separation, ground check circuits, short and overload protection, low air pressure, etc. [5].

2.1.8. Raise Boring Method

The raise boring method is the most widely used shaft boring method of all; so far, approximately 400 raise-boring machines were manufactured worldwide, of which almost 200 are believed to be in operation at a time. Employment of these units relies upon the existence of access below the future shaft; reaming being affected from the upwards after a pilot hole of up to approximately 380mm diameter has been drilled from above. The economic range of application of this method was over a long period of time restricted to three – four meters diameter and 500 to 600m depth, since the equipment available for larger drilling diameters and depths, permitted only low cutter loads which produced small depths of penetration and caused correspondingly higher cutter wear.

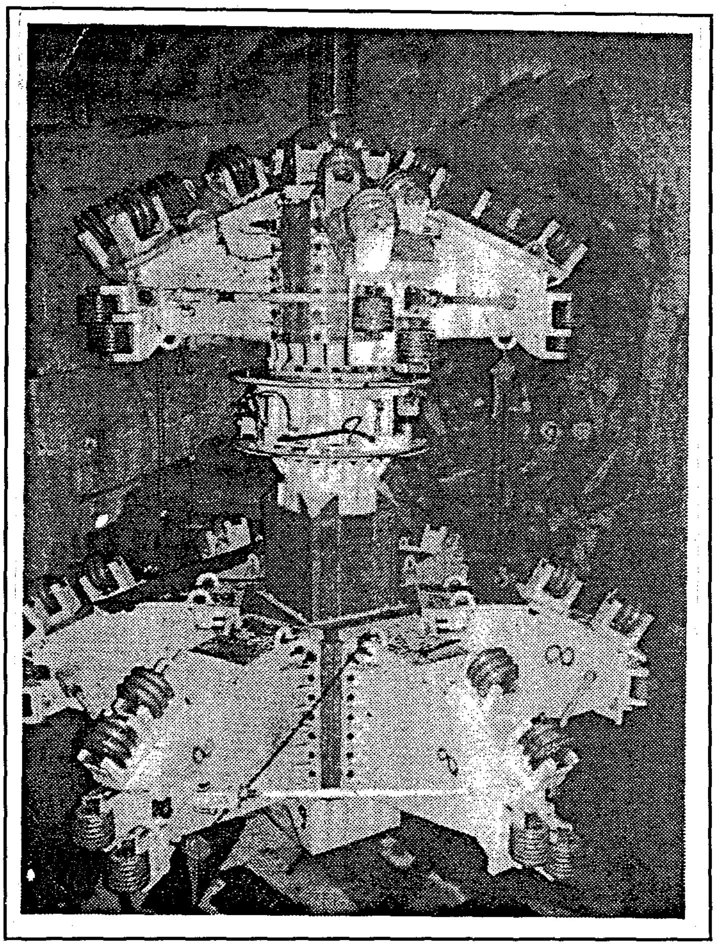


FIGURE 2.6. The WIRTH sequential head (dia. 4,2m/6,0m) [2].

Finally, the development of two new raise-boring machines and the sequential head from WIRTH (Figure 2.6) for diameters up to 6m, extended the range of economic application to the present state of technology, i.e. five – six meters drilling diameter and approximately 1000m depths.

This is correspondingly evidence by the 6m diameter shaft constructed in November 1984 at the Premium Diamond Mine S.A. (484m depth) and the Frank shaft of 1033m depth and 5.5m diameter.

2.1.9. The Raise-boring Method with Down Reaming

The other shaft boring method with continuous muck removal to a deeper level consists of a combination of raise-boring for drilling a pilot hole of 1.5m to approximately 2.0m diameter, which is reamed subsequently to the final diameter from the top downward by means of a shaft boring machine without drill pipe, frequently referred to as a V-mole [2].

The V-mole is a horizontal tunnel boring machine modified for vertical deployment by the German firm Wirth. First introduced to construct large diameter 4.88 to 6.55m shafts in Europe in early 1970's, it had been used in Alabama coal mine to construct 4.7m diameter shafts. This firm produced VSB and SB series.

The gripper assembly provides resistance to the thrust and torque required for rock boring. Rotary motion is transmitted from the gripper assembly to the cutter head through a kelly and up to six thrust cylinders are controlled by the operator to provide the required penetration rate. Muck is removed into a pilot hole by scrapers located on the cutter head. The shaft lining is placed from the work platforms located above the gripper assembly providing a continuous excavation cycle. Services and support equipment are deployed using techniques associated with conventional shaft sinking [5].

This shaft-boring machine is conditionally steerable, dependent upon the deviation predetermined of the pilot-hole. So far, this method was used principally in Germany, i.e. 36 times as a total there; three times in CSSR; and four shafts were drilled in Alabama/USA. An employment in S.A. will take place shortly.

The indicated range of application of this method is correspondingly evidenced by the bored shaft at "Lummerschied " of the Saarbergwerke/Germany, the intended final depth of 705m at a diameter of 8.20m having been reached in August 1985 [2].

2.2. Shaft Drilling

Blind shafts are holes drilled from the surface with no prior connection existing to any underground workings. Current surface mounted equipment and existing rotary drilling methods would enable holes up to 10m in diameter to be drilled to depths of more than 1000m.

Shaft drilling is performed using modified and scaled-up rotary drilling equipment. A drill rig at the collar of the shaft provides the forces to lift and rotate the bit and drill string. Cuttings are generally removed using mud and reverse circulation systems.

The large-diameter shaft drilling industry has its roots in both Europe and the United States. In the Dutch coal fields and Germany, 7.63m shafts were drilled in the 1950s by multiple-pass bits. In 1958, an in-hole-boring machine with the capability of drilling 7.32m diameter shafts was built in Russia. Also developed in Russia was a reaction-turbo drilling system that was used from 1958 to 1965 to drill a total of 84 shafts with diameters up to seven meters. This system is still being used in the Ukraine.

In the United States, the federal government gave shaft drilling a big push when the Atomic Energy Commission decided to test nuclear weapons underground. Modified oil-drilling rigs were used to drill shafts by direct circulation. This was followed by multipass reverse circulation techniques. The Nevada Test Site, north of Las Vegas, has a "classified" number of large-diameter shafts. These shafts range from 1.32m to 3.05m in diameter and range from 244 to 763m in depth. It is believed that, to date, over 1000 shafts have been used for testing nuclear warheads.

In 1953, independent of the federal government-controlled drilling programs, a 1.8m diameter core drill was manufactured and deployed in the West Virginia coal fields by the Zeni brothers. Since that time, there has been a steady improvement in the machinery fielded by the second and third generations of the Zeni family.

Shafts are now routinely drilled up to 6m in diameter, and thousands of drilled shafts have been successfully completed worldwide. Blind-drilled shafts for mining applications comprise nearly 10 per cent of the worlds total drilled shafts. This represents a significant body of proven technology that has been steadily gaining acceptance in mining applications [3].

2.2.1. Rotary Blind Drilling System

Large-diameter shaft drilling systems are an extension of conventional rotary drilling techniques used extensively for oil well boring. Systems components and operational considerations are described below.

<u>Shaft Collar and Foundation :</u> A shaft collar is typically excavated using either an auger rig or conventional drill-and-blast method during mobilization of the blind shaft drilling equipment. Collar depth depends on the overall length of the bottom-hole drilling assembly and is designed so that the assembly can be positioned below the drilling rigs rotary table. The collar may be lined with steel, shotcrete or concrete depending on ground conditions. A cast-in-place, reinforced concrete foundation will be designed to support the drill rig.

<u>Drilling Rig :</u> Major components of the drilling rig include a mast and substructure, drawworks and tugger hoists, rotary table, crown and traveling blocks, hook, swivel and kelly. The mast through a conventional crown and traveling block/hook assembly supports the drill pipe and down-hole drill tools. Static hook load capacities, for large-diameter blind-shaft drilling, may range from several hundred thousand kilograms to more than a half million kilogram requiring more substantial masts than typically used for conventional rotary drilling. The rotary drilling motion is transferred from the rotary table to the drill pipe using a square section kelly bar. Mud is pumped to the down-hole system via a swivel located above the kelly.

Downhole Drilling Tools : These include the drill pipe and bottom-hole drilling assembly. Drill pipe is selected based on maximum tensile and torsional loading conditions and consideration of mud circulation requirements. A common US drill pipe, used for large-diameter blind shaft drilling, has an outside diameter of 340mm, weights 134kg/m and requires in excess of 136 kN-m of makeup torque. The bottom-hole drilling assembly includes a drill bit, mandrel, stabilizers and donut weights. Cutters are mounted in cutter mounts or saddles and bolted to the underside of the flat-bottomed drill bit. The drill bit is in turn bolted to the mandrel that serves as a base for locating donut weights. Donut weights are added to provide the required normal force at each cutter (typically from 44.5 to 89kN) which is a function of the relative hardness of the formation to be drilled. Donut weights are secured to the drill pipe by a hold down clamp, which forms the top of the bottom-hole assembly. Stabilizers may be added directly above the mandrel and toward the top of the bottom-hole assembly to assist directional control.

Operations : An important element of blind shaft drilling is to maintain a straight, vertical alignment. The key to effective directional control involves minimizing the fraction of total effective bottom-hole assembly weight transferred to the bit while maintaining acceptable penetration rates. This maximizes the pendulum effect experienced by the bottom-hole assembly and in conjunction with stabilizers, provides a straighter shaft.

Most large-diameter drilled shafts use air-assisted reverse circulation. Drilling mud is added to the hole at ground level and circulated through the cutters and up the inside of the drill pipe. Air is added inside the drill pipe causing a density imbalance that induces flow rates sufficient to remove the drill cuttings. The "dual-string airlift reverse circulation" method incorporates a plenum chamber in the drill bit and dual string drill pipe. Mud and compressed air are pumped down the outer annulus of the drill pipe to the plenum chamber where the air separates from the mud. Mud flows through the plenum chamber and is forced through fluid jets located in the bit in order to clean the hole bottom. The air is routed through the top of the plenum chamber into the inside of the inner string and induces upward flow of mud and drill cuttings. Mud flows from the top of the drill pipe into the first mud pit where up to 90 per cent of the cutting drop out. Overflow from the first mud pit goes to the second and third where the remaining cuttings settle out.

Ground Support : A cake of mud is deposited on the shaft walls during drilling. The thickness and strength of this cake may be optimized based on cuttings removal and ground support requirements to prevent mud loss during drilling. Control of mud density and head (i.e. height of mud column) acting against this impermeable surface permits shaft excavation in poor ground.

Shaft Lining : The final lining for a blind drilled shaft typically consists of a ring-stiffened steel liner. This is equipped with external guides to familiate grout line deployment and is outfitted internally.

Liner sections are fabricated offside in lengths compatible with transportation and handling requirements; sections may be up to 18m in length. The liner is lowered into the mud filled hole using either jacks or the drill rig. Each liner section is aligned and welded to the one below to provide a completely watertight membrane. Loads on the casing jacks may be limited, in the case of deeper shafts, by capping the bottom liner section so that the liner can be "floated" into place. Water is pumped into the casting to control buoyancy as the capped liner is lowered into the shaft. When liner installation is complete, the annulus between the steel liner and the shaft wall is fitted with grout.

Other lining systems, compatible with the concept of rapid, remotely controlled placement have been developed but are not in common use. These include slip forming (both bottom-up and top-down) , jump forming, precast concrete cylinders and remotely placed shotcrete. Finally, if ground conditions permit, conventional lining placement techniques (e.g. involving slip forming, jump forming or shotcrete, placed using a galloway) may be used.

Blind-Drilling System Performance Prediction : The performance of a blind shaft drilling system is simply defined as a function of the operational penetration rate and the system utilization. Penetration rate is, in turn, a function of the geology (rock strength, fracture frequency, hardness, abrasivity); drill assembly (cutter type, size and spacing; cutter load and available torque) and cuttings removal system. Breeds, C.D. and Conway, J.J [7] reported that Moss et al. developed a drillability index to predict relative penetration rates in rock rated from exceptionally poor (Q=0.001) to fair (Q=10) with intact strengths ranging from 21 to 297Mpa in 1987. Variations in average penetration rates were smaller than expected and correlation with the index was poor. Several important observations were made as a result of this case study.

1. Lower than predicted rates of penetration in clay were thought to be due to plugging of the bit. Associated problems included reduced mud circulation rates and poor control of shaft verticality.

2. An increase in the rolling resistance when drilling in rock of lower rock mass quality was thought to result from fragments that were larger than those normally resulting from the cutting action.

The results serve to illustrate the potential shortcomings in generic performance prediction systems. The operational penetration rate for a blind-shaft drilling project can be estimated using available equipment specifications and the simple relationships suggested by Farmer and Glossop(1980) and Graham (1976) as reported by Breeds, C.D. and Conway, J.J [7]. Adjustments are required for available thrust, calculated as 30 per cent of the sum of the weights of downhole components corrected for buoyancy from the drilling mud and imperfect hole cleaning.

Blind-Drilling Costs : Table 2.4 provides a breakdown of blind-drilling cost components and their relative contribution to total project costs. It can readily be seen that the project costs are dominated by the acquisition and installation cost of the steel casing used for final lining [7].

TABLE 2.4. Breakdown of Blind-drilling Project Costs [7].

Component	per cent of Total Project Cost
Management	2
Engineering and Administration	8
Drilling	25
Casing	24
Welding casing	10
Cementing casing	7
Site construction	3
Drilling mud	7
Drill bit cutters and stabilizers	6
Drill hole surveys	1
Other (incl. Radiographic inspection, crane services, tools)	7

2.2.2. Breaking of Rock

The formation of a drilled hole requires only three basic functions.

1) Breaking of the rock at the base of the hole.

2) Removal of the broken rock from within the bore.

3) Support of the shaft wall.

Breaking of the rock at the base of the hole is achieved by cutters attached to a bit body. During the last three or four decades various types of cutter have been developed with a view to increasing the effectiveness of rock excavation. The underlying aim of these developments has been to increase the reliability and life of cutters while permitting increased loading to be applied to them.

Several different companies throughout the world manufacture cutters but there are only three basic types: Steel milled tooth, Tungsten Carbide insert, Disc or Kerf cutter.

All of these types of cutter have been in existence for many years and while manufacturers have extended the life of the cutter with improved metallurgy and bearings, they have not significantly improved its rate of cutting. Changes in cutter design that could lead to greater efficiency include:

1) Tooth shapes designed to help transportation of the rock chips to the pick-up point of the bit
2) Larger dia. cutters, which would be less prone to skidding when lightly, loaded.
3) Effective scrapers or other mechanisms that would clean the cutter.

Steel Milled Tooth cutters are most suitable for soft ground and they produce large cuttings. This give the potential for rapid rates of penetration but the performance of the cutter diminishes rapidly once the teeth begin to become blunt.

Tungsten Carbide insert cutters are usually used in harder rocks or in particularly abrasive formations. They require significantly larger bit loading than the other types of cutter and the rock chips produced are very small. The rate of penetration with these cutters is usually slow, typically a maximum of only a few hundred millimeters per hour, although similar performance is often obtained throughout the life of the cutter.

Disk or Kerf cutters are not normally used for blind hole drilling because they tend to produce large chips which cannot easily be transported across the bottom of the hole and up to the surface. In soft or plastic strata they can produce concentric furrows on the base of the bore with no excavation actually taking place. Because the relatively wide spacing of the discs on these cutters many more units are required to dress a bit than for tungsten carbide insert cutters when used for hard formation drilling.

In order to advance the hole downward, a force must be applied to the cutters so that the cutting structure (tooth) penetrates the formation. In soft strata the material is displaced by the tooth and subsequently dislodged by the movement of the tooth itself, whereas in harder strata the rock under the tooth is actually crushed and tensile forces are induced in the rock adjacent to the tooth thereby causing failure of the rock by splitting. The harder the rock, the more force is required to cause tooth penetration and the shorter the tooth penetration the smaller is the chip that is produced.

It will, therefore, be evident that for any particular rock strata the greater the tooth penetration that is achieved then the grater the rate of drilling will be. For many years the force (weight) that could be applied to the cutters was effectively limited because of the bearings in the cutters. In the last 10 years or so these difficulties have been substantially overcome and it is now possible to load individual cutters with forces up to 20 tones. The materials used in the manufacture of the cutting structure of the cutters has also been improved and this has resulted in much longer running times being achieved before cutter replacement becomes necessary.

2.2.3. Removal of the Broken Rock

Removal of the broken rock within the bore is the area in which the greatest improvements in efficiency can be made using existing technology and equipment. The greatest limitation in blind shaft drilling is the inefficiency of rock chip transportation across the bottom of the hole.

Ideally, the cutter should break a large chip of rock which is instantly dislodged from the bottom of the hole and transported to the pickup point of the bit and conveyed out of the hole with no secondary contact with ant other cutter.

This would enable the cutting structures to attack new pieces of rock on each revolution of the bit and for the maximum rate of penetration to be achieved. In practice, the rock chip, even if it is formed by a single pass of a cutter, does not immediately become dislodged. This means that the next cutter passing over that portion of the hole is not presented with fresh rock to cut but instead is running on old rock chip debris. This debris is ground into small pieces which may require several regrinding before they are small enough to be transported away by the circulation fluid.

The reasons why the chip is not initially dislodged are inadequacies in the cutter design and insufficient velocity of the circulation fluid over the bottom of the hole where the chip is lying.

In his study Pigott, C.P. [8] stated that Sundberg carried out work to determine the relationship between the horizontal flow velocity of water and the size of rock chip that could be transported. The relationship is found to be approximately linear on a log/log plot and an increase in water velocity by a factor of two from 100 cm/sec to 200 cm/sec caused chips that are approximately 3.6 times larger in diameter to be transported. Such a chip could be up to 46 times larger in volume than the chip capable of being transported by the fluid flowing at the original velocity.

In order to encourage the movement of larger cuttings across the bottom of the hole it is necessary to increase the carrying capacity of the fluid. This can be achieved in several ways:

1) Improve the circulation system to give larger fluid circulation rates.

2) Increase the velocity of the fluid across the very bottom of the hole where the rock chips are lying by restricting its path by skirts and shrouds.

3) Change the physical properties of the fluid by the addition of chemicals.

4) Change the angle of the bottom of the hole.

<u>Circulation Systems :</u>

Several types of circulation systems are available for blind shaft drilling applications. They include: Reverse liquid flush, Forward liquid flush, Reverse air or gas flush and Forward air or gas flush. It is, however, the reverse liquid flush system that is most commonly employed and which most often provides the best and easiest method of circulation.

the drill pipe; also rapid rates of penetration are not feasible at depth because the density of the "liquid" column inside the drill pipe increases as more rock cuttings are added to it. As the pressure exerted at the bottom of this column increases so as to approach the pressure at the bottom of the column of liquid in the annulus, the motive force diminishes and circulation rates decrease to unacceptably low levels.

A new form of reverse circulation has been developed and pioneered by Colin P. Pigott, Chairman and Managing Director of Pigott Shaft Drilling Limited Ormskirk, England, and is called "Pumped Reverse Circulation". With this system a submersible pump is installed down the shaft as part of the drill string. The pump is situated within the chamber of a closed vessel. The only entry to this vessel is from the drill pipe below and the output from the pump is connected to the drill pipe above.

When submerged in the fluid in the shaft the vessel fills with liquid and when the pump is activated it starts to pump liquid from within the chamber. This reduces the pressure in the chamber to less than the pressure outside the chamber and a flow is created. Because the vessel can be submerged to a depth which is only limited by the mechanical properties of the pump seals and the physical length of the power supply apparatus, the potential difference in pressure between the inside and the outside of the pump chamber could be many or even tens of atmospheres. Such a pressure difference would be sufficient to cause large flow rates through long lengths of drill pipe even when the fluid is laden with large quantities of cuttings.

Another advantage of this system is that the drill pipe is full of liquid and solids only, with no air or gas being required. Therefore, for any particular velocity in the pipe the maximum quantity of liquid is flowing, whereas with the air lift or air injection system the pipe is occupied by liquid and air, so that for the same fluid velocity less liquid is being moved. This is a disadvantage as it is usually the velocity of the drilling fluid across the face of the bit that is the limiting factor in rock chip transportation, not the upward velocity of the fluid in the drill pipe.

Large circulation volumes, however, require large diameter drill pipes. The largest, readily available size of pipe has a bore diameter of approximately 300mm. The maximum volume that can be circulated through this size of pipe using an air lift system with the ideal conditions of a deep air injection point is found in practice to be about 15 m^3 per minute. This is almost identical to the circulation volume achieved by Colin P. Pigott [8] at both shallow and deeper depths using the "pumped reverse circulation " system with 300m bore pipe. It has been calculated however that with different pump characteristics a volume of over 20m^3 per minute would be possible.

Changes to the physical properties of the circulation fluid may further improve the bottom hole cleaning. Sunberg's work was carried out on the horizontal transportation of rock chips by water explained in Colin P. Pigott [8]. If the viscosity of the circulating fluid is increased then the rock chip carrying capacity of the fluid is also increased. If the density of the fluid is increased then this improves the carrying capacity of the fluid but it becomes more difficult to dislodge the rock chip in the first place with fluids of higher density. Other problems may also arise with high weight drilling fluids and therefore, overall, it is thought to be better to use circulating fluid with low density but with a viscosity greater than water.

Changes to the angle of the bottom of the hole will assist in rock chip transportation if the surface over which the rock chips are to be moved is inclined downwards from the horizontal. It requires less applied force to move a particle down a slope than across a horizontal surface. Therefore for any given force, the steeper the angle of the inclined surface over which the particle is to be moved then the larger is the particle or rock chip, that may be moved across it.

2.2.4. Directional Control of a Blind Drilled Shaft

The drilling of large diameter holes at increased rates of penetration seems therefore to be an achievable goal. It is nonetheless essential that these holes be drilled on the line and in the direction required. Most large diameter drilled holes are specified as vertical holes. The methods by which blind drilled holes may be kept vertical fall into three categories:

1) Steering of the drill bit
2) Guiding of the bit by a pilot hole
3) Using the pendulum effect and stabilization.

Steering of the drill bit is widely used on small diameter oil, gas and geothermal wells where whipstocks or bent-sub mud motors are used to drill deviated or directional holes. No equipment is known to exist for use in large diameter drilling which would be equivalent to the bent-sub mud motor. Some attempts have been made over the years to move the drill pipe off the centerline of the shaft so as to cause the bit to drill at an angle slightly inclined to the centerline of the shaft. Such a system was used during drilling of a shaft at Schinnen for the Dutch State Mines in 1950 when an off-center stabilizer was used.

The use of these devices is hard to monitor and control and the rate of change of angle of the hole is also very slow. The system is not compatible with high rates of penetration and efforts have been directed more to keeping the shafts vertical rather than correcting them once they have deviated. Guidance of the bit by a pilot hole is the oldest method and one of the surest methods of directional control used in blind shaft drilling industry. The early holes were formed by drilling small holes, which were reamed to larger and larger sizes using multiple passes of hole openers of increasing diameters. A similar system is used today except that the pilot hole is often opened up to the final diameter required in a single pass. The type of bit used for this operation is very similar in design to other blind shaft drilling bits except that it is fitted with a central heavy duty stinger or guide pin which fits into the pilot hole and restrains the larger bit from wandering off center. This method of working allows for most of the weight of the bottom hole assembly to be applied to the drill bit, thus enabling rigs of any particular size to drill holes of a larger diameter than would otherwise be possible.

One of the disadvantages of the system is that it is difficult to drill the pilot hole to the degree of vertical accuracy normally associated with blind shaft drilling. If a specialist slim-hole rig is used to form the pilot hole these problems can be overcome and the economies associated with the use of a smaller size of shaft drilling rig than would otherwise be required may be sufficient to allow for the extra costs associated with the mobilization and use of the slim-hole rig. For very large diameter holes it may be necessary to use two or more passes with the shaft drilling rig. Alternatively it may be used to form a convenient size of pilot hole (say 2m) which could then be reamed to the full size in stages. The pendulum effect occurs when some of the weight of the bottom hole assembly is held back so that the drill string is in tension and thereby tends to hang vertically. If the bit is displaced from the vertical the gravitational or pendulum effect provides a force which acts to bring the bit back on line. It is usual to use large amounts of weight in the bottom hole assembly in the form of drill collars or drill weights situated above and near the bit. These weights are often interspersed between stabilizers, which may themselves be heavily weighted.

More weight is provided than is needed to be applied to the bit. This allows some of the weight to be held back by the rig which causes both the pendulum effect to occur and the drill string to operate in tension. This is a very desirable condition as it prevents stress reversals in the drill pipe, which could lead to fatigue failure of the pipe or pipe connections. The greater the weight of the bottom hole assembly the greater is the weight that can be held back and this results in a larger pendulum effect, which should enable a straighter hole to be drilled. The diameter of any drilled shaft is, however, always small in relationship to the depth of the shaft and the permitted tolerances of the shaft from the vertical are also very small. Because of the shallow angles involved, the resultant corrective force available from the pendulum effect is tiny in comparison to the excess weight employed and the deeper the shaft the smaller the angle of deviation will be for any particular horizontal displacement of the bit. It is therefore necessary to provide large amounts of excess weight in the bottom hole assembly in order for the pendulum effect to provide corrective forces of a significant magnitude.

Another benefit of having a heavy bottom hole assembly is that it acts as a flywheel. This tends to smooth out the bumps and jerks that result from the cutter running over different types of rock or dipping strata. The rotary table of the rig is thereby protected from most of these shock loads and this leads to a longer life for this unit. Notwithstanding the above it is doubtful that shafts could be drilled straight in non-homogenous materials using the pendulum effect alone [8].

2.2.5. Lining of Drilled Shafts

A technique by which precast concrete cylinders of convenient height are stacked in the drilled shaft has been developed to yield a dry and hydrostatic mine shaft liner. The base for the development of this technique has been that conventional sinking of large diameter mine shafts at mining properties with high-head aquifer zones and high water inflows has proven to be slow and costly.

Several alternatives to lining the shaft have been investigated. In one method, steel segments are welded together and floated into place. In another, concrete segments are attached at the surface and then floated in place. The approach incorporating steel liners is commonly used in smaller diameter mine-shafts: however, for a 4.25m ID hydrostatic steel line, the costs are nearly triple those for an equivalent concrete liner. Two problems encountered in floating a concrete liner in place, are the amount of time required to make connections between the segments at the surface and the possibility of losing the entire liner should a concrete segment fail while the liner is being floated. For overall cost effectiveness, as well as reliability in construction, the stacked-in-place concrete liner is superior.

Should hydrostatic pressure be low or absent in the shaft and water inflows minimal, water may be drained by placing drain lines between the liner and the shaft wall. The water will drain to the bottom of the shaft and then must be pumped out.

Design : The precast concrete sections are subject to many stresses. The major stress to which the liner is subjected is hoop stress caused by the outside hydrostatic pressure. Also the cylinder may experience tensile stress during lifting, compressive stress from its own weight when installed and possible bending stress due to shaft hole deviation.

The concrete liner cylinder may be designed by using the classical Lamé equations or by using empirical formula presented by H.H. Haynes of the US Naval Civil Engineering Laboratory. These equations, along with considerations for shell out-of-roundness, material strength and stability characteristics, will dictate wall thickness and the required steel reinforcement.

Proven results of concrete additives, along with proper pouring and curing techniques, will yield high strength concrete, which is necessary in order to minimize the wall thickness of the liner.

A concrete cylinder under uniform external pressure can be analyzed using several sets of equations and parameters. The effect of combined stress in the concrete has been neglected. The concrete actually shows an increase in strength during biaxial compression as compared to uniaxial compression. This will increase the established safety factor for the liner structure.

An empirical design to predict the implosion pressure of a concrete cylinder is:

$$P_{im} = q \ f'c \left(2.17 * t / d_0 - 0.04 \right) \tag{2.5}$$

where : P_{im} = implosion pressure (psi),
q = length to diameter ratio, equal to 1.0
$f'c$ = compressive strength of concrete (psi),
t = wall thickness (feet),
D_0 = outside diameter of cylinder (feet)

The major limitation to this equation is that t/d_0 ratios must be ranged between 0.031 and 0.188. Applying the design criteria to this equation yields a safety factor of 1.6 (safety factor = failure pressure / 1510 psi), if the effects of the grout behind the liner is ignored.

Another set of more conservative design formula is represented by conventional thickwall cylinder theory, using the Lamé equations:

$$\sigma_c = \frac{P_e r_2^2}{r_2^2 - r_1^2} \left(1 + \frac{r_1^2}{x^2}\right) \quad (2.6)$$

$$\sigma_r = \frac{P_e r_2^2}{r_2^2 - r_1^2} \left(1 - \frac{r_1^2}{x^2}\right) \quad (2.7)$$

where : σ_c = circumferential stress (psi)
σ_r = radial stress (psi)
P = uniform external pressure (psi)
r_1 = inside radius (feet)
r_2 = outside radius (feet)
x = (r1+ r2) / 2, r1 < x <r2

Analysis using these equations along with the design criteria yields a safety factor of 2.03, if there is an allowance for the grout behind the liner.

Because the major loading on the structure itself produces compression hoop stress, the concrete alone can carry these loads without steel structural reinforcement in the liner. However, minimum steel reinforcement is needed to cover volume changes and shrinkage in the concrete mass. Internal stresses due to expansion, contraction and shrinkage result in cracks, which may exceed acceptable limits. Steel reinforcement will also be needed for handling operations. Lifting of the sections will be from the bottom of the section, so this will not dictate substantial quantities of steel reinforcement.

The grout behind cylindrical liners causes a little increase in the strength of the structure. This largely due to the variables involved. The beneficial aspects of grout are in the distribution of non-uniform loads from the shaft walls and in carrying all of the vertical load through shear. In the conclusions from the Blake study which is reported by Richardson, P. and Thomas, W.E. [9], a cylindrical liner surrounded by a grout layer withstood pressure which was nine to 12 per cent higher than that calculated by the collapse pressure formula. The increase in the strength of the structure from the grout has not been considered, thus making the safety factors more conservative. Because of the high specific gravity of the grout, a full standing column of grout may cause the collapse of the liner. Grouting should be carried out in lifts or stages that are within acceptable pressure limits.

Installation : Liner installation will commence after drilling operations are complete. The shaft is drilled 30 cm beyond the desired depth and the drill bit is then pulled 30 cm off the bottom of the hole. A grout plug , 25 cm thick, is spotted on the bottom through the drill pipe. Once the grout plugs sets, it is dressed with the drill bit to ensure that the surface of the plug is perpendicular to the shaft wall.

The drill rig is disassembled and moved off location. A steel frame complete with overhead crane is installed over the unlined shaft. The steel frame is a pin and bolted structure made from standard wide flange shapes. The overhead crane is on rails to lift the concrete segments from a transporter and then move it over the shaft. A heavy-duty marine winch, with at least 945m of 2.5-inch (6.35cm) diameter wire rope rigged through the steel frame, is used as the hoist.

The first concrete liner segment (possibly a special breakout segment), having eight hard rubber-tread rollers, is transported to the shaft; these rollers act to centralize the segment in the shaft. The segment has a liner running tool (Figure 2.7) positioned in it. This tool has retractable rams, which mate with the steel ports in the interior of the segment. These rams are mechanically retractable once the segment is placed in the shaft, so the tool can be returned to the surface.

The segment and liner running tool are coupled to the overhead crane and then situated over the shaft (Figure 2.8). Three 1½ - inch (3.81cm) steel guide ropes are fastened to the outside of the first segment; these 1½ - inch (3.81cm) ropes will be used to properly seat the subsequent segments with those already situated in the shaft. Also six $^{5}/_{8}$ - inch (1.6cm) steel ropes are fastened symmetrically around the first segment, which will be used to guide the grout lines for grouting the liner in place.

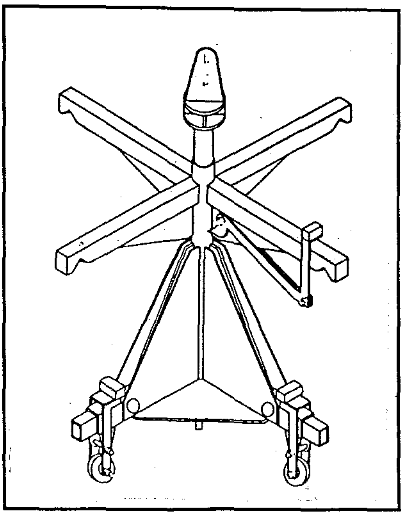


FIGURE 2.7. Running tool isometric [9].

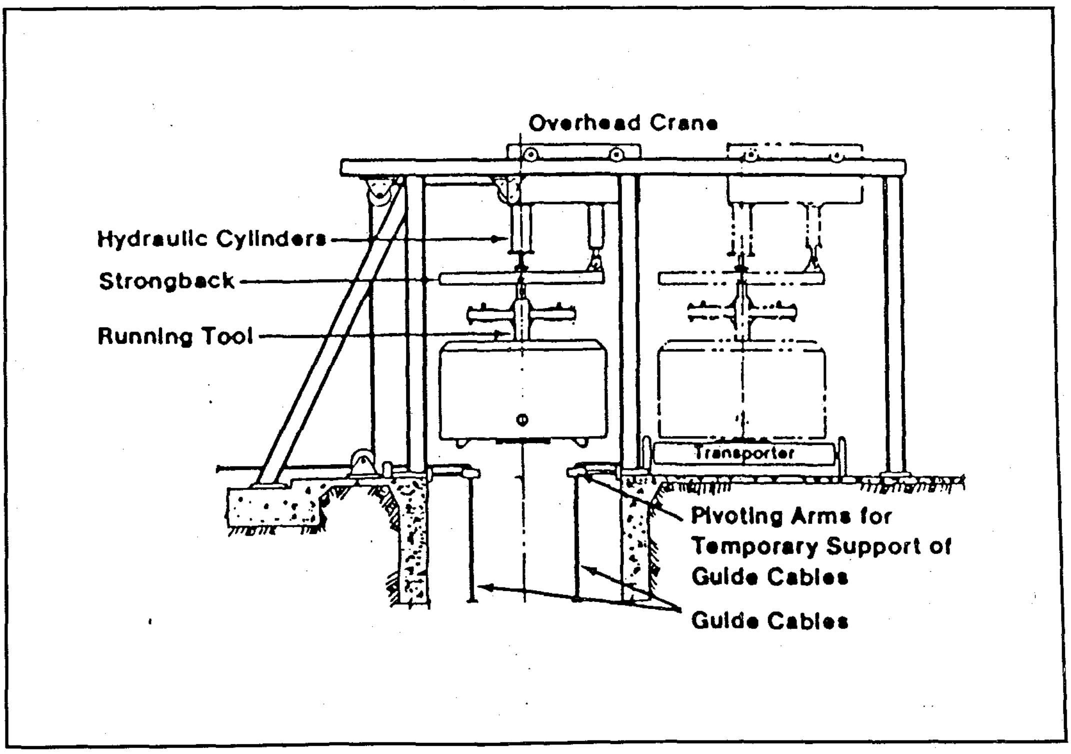


FIGURE 2.8. Picking-up liner and running tool and moving into position over the shaft [9].

The first segment is lowered into the hole on the $2^1/_2$ - inch (6.35cm) cable as the other cables are played out under slight tension. Once the segment is seated on the grout plug, the liner running tool is returned to the surface and all the periphery cables are tied off at the surface. Six 3-inch (7.62cm) slotted pipes are run to the bottom using the $^7/_8$ -inch (2.22cm) cables as a guide. These slottes pipes will guide the tubing used in placing the grout. The first segment is grouted in place.

Subsequent segments are positioned over the shaft, complete with the running tool. The $1^1/_2$ - inch (3.81cm) guide cables are clamped to the outside of the segment. The tool and segment are lowered into the shaft. A safety mechanism on the liner running tool prevents false release of the segment. As the segment mates with the prior segment, the rams on the liner running tool retract and the liner running tool is returned to the surface. Figure 2.9 shows a sequence of liners being stacked.

A small amount of rubble may settle on the bearing surface of the segments during installation. This rubble should be flushed down the slope of this surface by hydraulic turbulence created by the approaching segment. To aid this flushing action, part of the cylinder's top surface is beveled 45 degrees, which is above the angle of repose.

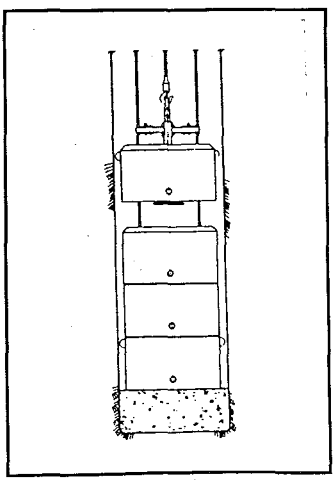


FIGURE 2.9. Sequence of liners being stacked [9].

As mentioned, grout is put behind the first segment once it is located and is then placed behind the liner after every ten segments are positioned. This helps to maintain centralization of the liner and also minimizes the vertical compression forces by transferring the liner weight to the shaft wall in shear. Once all segments are stacked in place and grouted, the drilling fluid is removed, and the shaft is commissioned.

In the case of a steel liner versus concrete liner, one should not attempt to make a general comparison. There are many conditions in which one liner will have advantages over the other. This is particularly true in small diameter (2.45m or less) shaft which are constructed in unstable ground conditions or areas prolific water-flows. 2.45m diameter shafts appear to be the crossover point. Smaller diameter shafts can be lined with fully hydrostatic steel at a cost less than a 15cm thick concrete liner with reinforced steel. To make this comparison, it is necessary to include the cost of material, the time required to install the liner, the safety of the installation crews and the early productivity or use of the shaft. There are instances, however, when ground conditions are such that it is unlikely that a concrete liner could be installed to produce a completely dry shaft. In these cases, the steel liner is the only method that can be used to produce a shaft that is completely dry, hydrostatically designed and has a reasonable long useful life.

In the case of shafts of 8 to 12 feet (2.45 to 3.65m) in diameter the difference between the cost of steel and concrete lining is much greater, and continues to increase rapidly as the diameter gets larger, with concrete being the least costly. Depending upon the design of the steel liner, it may be three or four times more costly than concrete. With the development of the precast- concrete segmented liner, the cost of the concrete liner has been further reduced to about one-fifth of an equivalent steel liner. Again, comparison takes into consideration the cost of materials, the length of time to install, the simplicity of the operation compared to the forming and pouring of a conventional liner, and the safety to personnel during the installation. On an overall project basis, the cost savings can be very significant for shafts 12 ft (3.65m) and larger in diameter [9].

2.2.6. Full-diameter Shaft Drilling Rigs with Drill Pipe

As far as shaft drilling methods including continuous muck transport to a higher level, or to the surface, is concerned, drilling with machines with drill pipe and - if required- with subsequent reaming is the oldest shaft drilling method of all, and certainly the most used large-diameter drilling method after the raise boring (Figure 2.11).

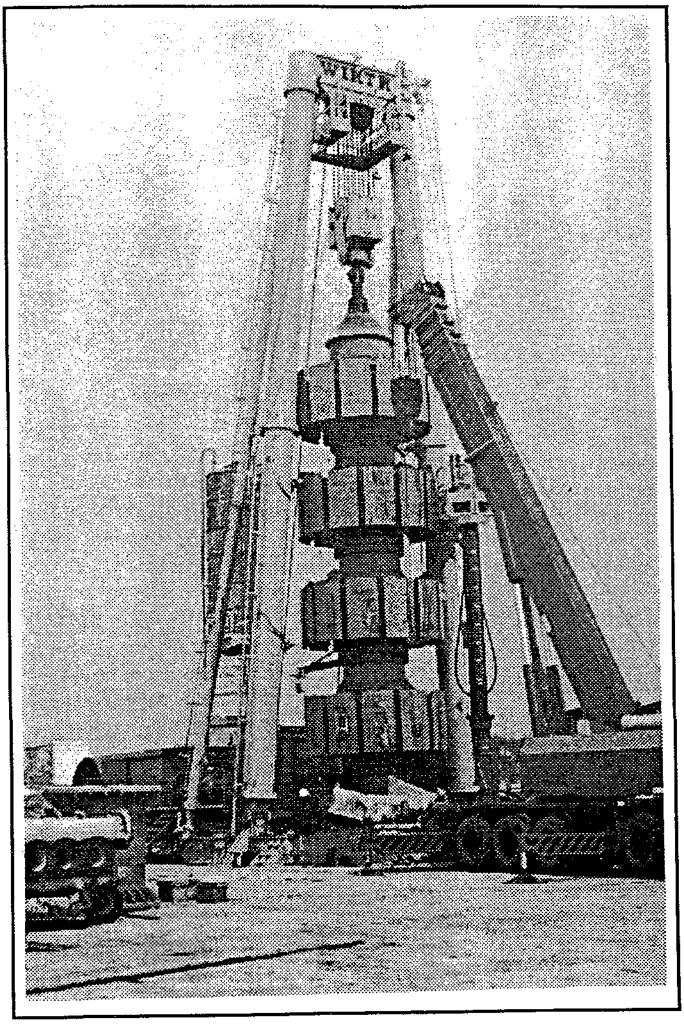


FIGURE 2.10. WIRTH L35 with bottom-hole-assembly on site in Donezk area [2].

After creation of the shaft drilling method by Honigman in 1892, this method was forgotten relatively quickly. Only in the fifties the shaft drilling method was then discovered again, i.e. in Europe (drilling of two Beatrix shafts in Holland) and in the USA, because of the nuclear test program, within the scope of which more than 300 shafts or big holes were sunk.

After much drilling with modified oil well drilling rigs, employing the cutters used in raise-boring and tunnel boring, the conviction of the industry prevailed worldwide to an increasing extent, that for drilling shafts machines are needed which are especially built for such a purpose. Nowadays, single purpose shaft drilling machines with drill pipe are known to be manufactured in PR of China, Russia, USA and in Germany.

The maximum shaft drilling diameter of nine meters reached so far, was achieved with a Chinese machine (500m depth), whereas the maximum depth of 754m was achieved with a shaft drilling rig, manufactured by the Hughes Tool Co. and used in Australia. In this case the drilling diameter was 4.40m.

2.2.7. Full-diameter Shaft Boring without Drill-pipe

Full-diameter shaft boring without drill pipe has quite evidently its origin in the development of the WIRTH shaft-reaming machine also working without drill pipe. Soviet manufacturers however first put in operation the full-diameter shaft boring machine, the corresponding prototype SK-1-U having been employed from 1976 on the Kanlin mine in the Donbass coal area, for drilling the shaft no.3, with a diameter of 7.7m (Figure 2.12).

The final depth of 1160m was reached in 1981. As far as this equipment is concerned, the shaft bottom is cut with center cutting hard-metal tipped disk-type steel plates, arranged on two rotating heads, carried on a rotating beam, differing from the face cutting of the German machine [2].

Modern shaft construction by blind shaft-drilling equipment has proven to be safe, effective and efficient, well deserving of consideration by the imaginative mining engineer. The early methods were slow and expensive. With the improvements and developments of the last two decades, the drilling of shafts has proven to be fast and relatively inexpensive. The equipment is available to drill shafts up to 20 feet (6.1m) in diameter to depths in excess of 3000 feet (914m). The hydraulic equipment available today can be utilized to construct shafts faster, safer and less affected by underground problems than conventional shaft sinking techniques

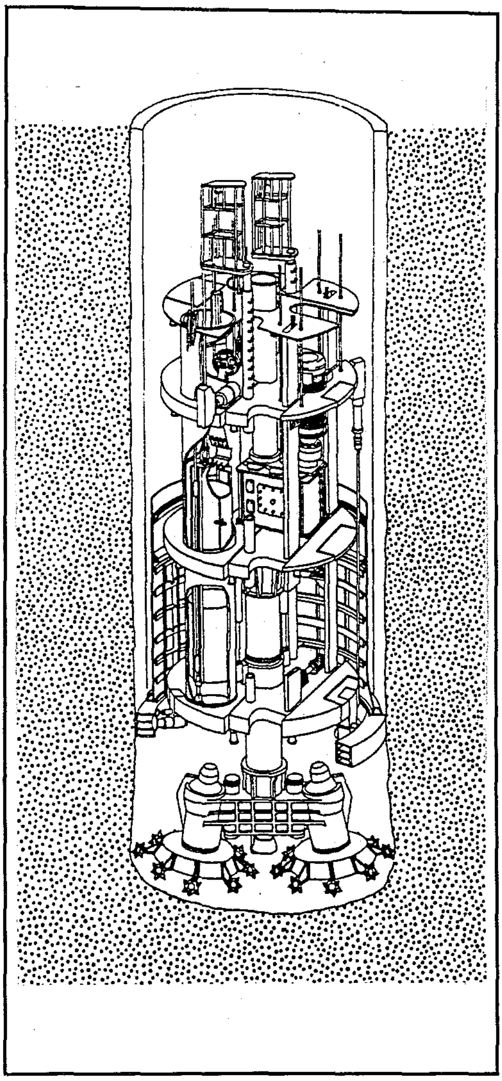


FIGURE 2.11. Full face shaft boring machine SK1-U [2].

2.3. Conventional Shaft Sinking

Conventional Shaft Sinking is the oldest sinking method. Although some other new sinking techniques, such as raise boring and blind drilling, have been developed; conventional sinking method is the most used sinking method. Its basic steps are the drilling, blasting, mucking and lining. According to the ground conditions temporary ground supports, such as shotcreting, can be needed before lining. Besides these water control is very important in shaft sinking. Typical arrangement of the conventional shaft sinking is shown in Figure 2.13.

First step in conventional shaft sinking is the drilling. Drilling can be achieved with either hand held machines or multi boom jumbo. Drilling is followed by blasting of the sump. After the blasting, the broken rock is loaded into muck buckets by a grab and hoisted to the surface where it is dumped via a chute in the twenty to thirty meters high Headframe. After this operation has been completed two or three times, depending upon the amount excavated each round, the in-shaft platform (stage) is lowered together with the formwork and the lining advanced about five or six meters. After the lining, the shaft services (pipes etc.) are extended. And this process is repeated and forms the basic cycle of excavation and lining.

2.3.1. Variations on Conventional Method

<u>Mucking Units :</u>

Mucking units, suspended from the underside of the stage, have been developed throughout the past twenty to thirty years and there are many variations. They all have the same aim however: to provide maximum coverage for grab operations in the shaft bottom. Mucking units fall into four main categories, defined by their action:

1) Transverse in one direction across shaft
2) Transverse in two directions, one at right angles to the other

3) Rotary; in which the grab moves at a fixed radius around the shaft
4) Radial; in which the rotary movement is combined with radial traverse

1) Figure 2.14 shows the simple Riddell Mucker in which the grab traverses back and forth across the shaft. A winch mounted on a trolley raises and lowers the grab and a separate, reversible, motor moves the trolley via a chain drive. Because it lacks sideways motion, the Riddell Mucker is usually limited to rectangular shafts.

2) Figure 2.15 shows a two way traversing mucking unit. This is similar to the one previously described but carries a secondary boom, normal to the main track, along with the trolley and grab travel. The main drive is made positive by the use of the rack and pinion principle. Transverse motion of the trolley along the boom is provided either by double acting pneumatic cylinders or by a tensioned endless chain system. This well-tried type of unit is relatively easy to fabricate and gives coverage of the shaft.

3) Rotary units, by definition give best coverage of circular shafts. However, they are more expensive to make because of the difficulties encountered in maintaining accuracy of the circular track. In small shafts it may not be necessary to have radial as well as circular motion and the grab can be cantilevered from the central pivot. In Figure 2.16, a unit designed around the ring gear from a crane allows the sinking bucket access to the sump on the centerline of the shaft. The grab winch and operator's cabs are located on opposite sides of a slewing frame providing the operator with good visibility of the shaft bottom.

4) In larger shafts it is essential to have radial as well as circular motion. In most cases peripheral track will be necessary to support the radial boom at its outer end since the weight can no longer be cantilevered. Figure 2.17 shows an example of this type of unit which circular motion is provided by a reversible motor driving a sprocket around the peripheral track. Radial movement of the trolley is, in this example, provided by a pneumatic cylinder.

With all underdeck mucking units, a few essential points must be taken into consideration:

(a) The unit must be designed as an integral part of the stage. The shock loads transmitted by the violent action of slewing a heavy grab full of muck, rapid changes of direction and the mass of the unit itself, can tear welds and twist the framework of the stage out of shape.

(b) Access to the main known wear areas. Bearings, sprockets, chains etc. should be as accessible as possible and/or replaceable as sub-assemblies to reduce breakdown delays. Unlike some other parts of the sinking cycle where partial failure may lead only to a slowing down the mucking unit is always on the critical path.

(c) Because the unit hangs below the stage, it is not possible to lower the bottom deck any closer than about six meters off the sump. In bad ground, where the lining must be kept close to the shaft bottom, the underdeck mucking unit can be a problem.

There are several alternatives in conventional sinking to the underdeck mucking unit and two are worthy of specific mention.

1) Eimco 630: This rockershovel mucker, which can be lowered into the sump through opening in the stage, has been used successfully on many contracts. Being independent of the stage it does not suffer the disadvantages mentioned previously for the underdeck unit, if it breaks down, can be hauled out and replaced by a spare thus cutting down on costly delay time. It is not at its best in very wet conditions however or in small shafts.

2) Cryderman: This wall mounted shaft mucker (Figure 2.18) invented and developed in Canada consists of an operator's cab and a grab on the end of an extendible boom. It is available in three sizes and with variety of boom lengths. The basic model is pneumatic but there is also a hydraulic version which, although more expensive, is easier to control. The pneumatic cryderman requires skillful

handling as the movements tend to be jerky. The whole unit is raised and lowered by a winch. The winch can be either remote from the cryderman or integral with it and working against a fixed point in the shaft. In larger shafts where the reach of the boom is insufficient, two machines can be used, each covering half the shaft.

Cryderman have also developed a machine based on the standard model which is attached to, and pivots about, the center of the stage giving radial coverage of the shaft bottom.

Stage:

The shaft sinking stage is multi-purpose work platform of from one to eight decks designed for the following main functions:

(a) pumping (pumps / tanks located on stage).
(b) lowering and setting formwork (winches or chain block brackets).
(c) concrete distribution to formwork.
(d) grouting - backwall injection.
(e) mucking - support for mucking unit where applicable.
(f) secondary means of egress for personnel from the shaft in emergency.

To minimize the number of times the stage is moved in the shaft, care is taken to ensure that the deck spacing correspond to the pour levels in the formwork. Figure 2.19 shows an example of a typical stage.

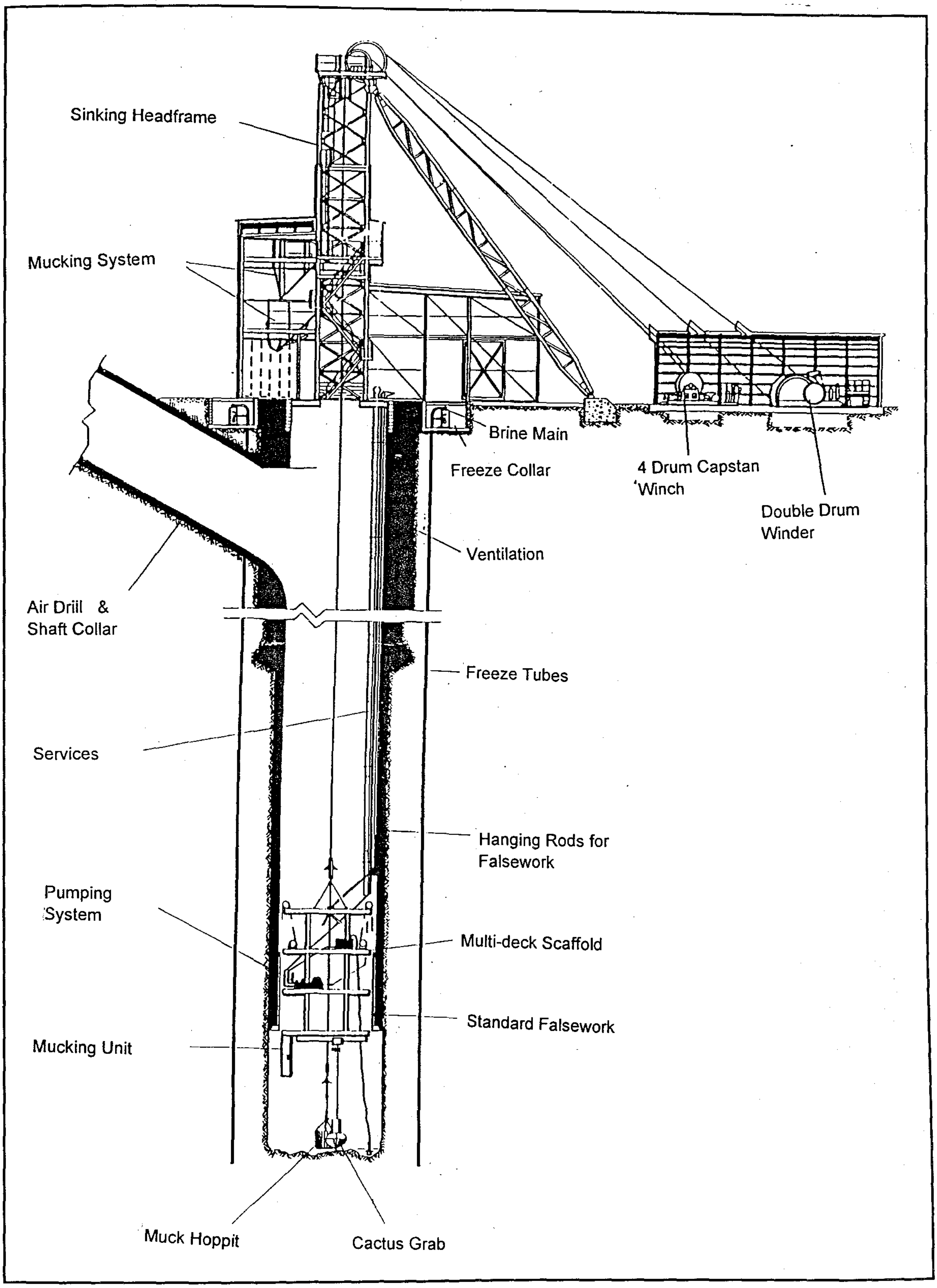


FIGURE 2.12. Typical shaft sinking arrangement.

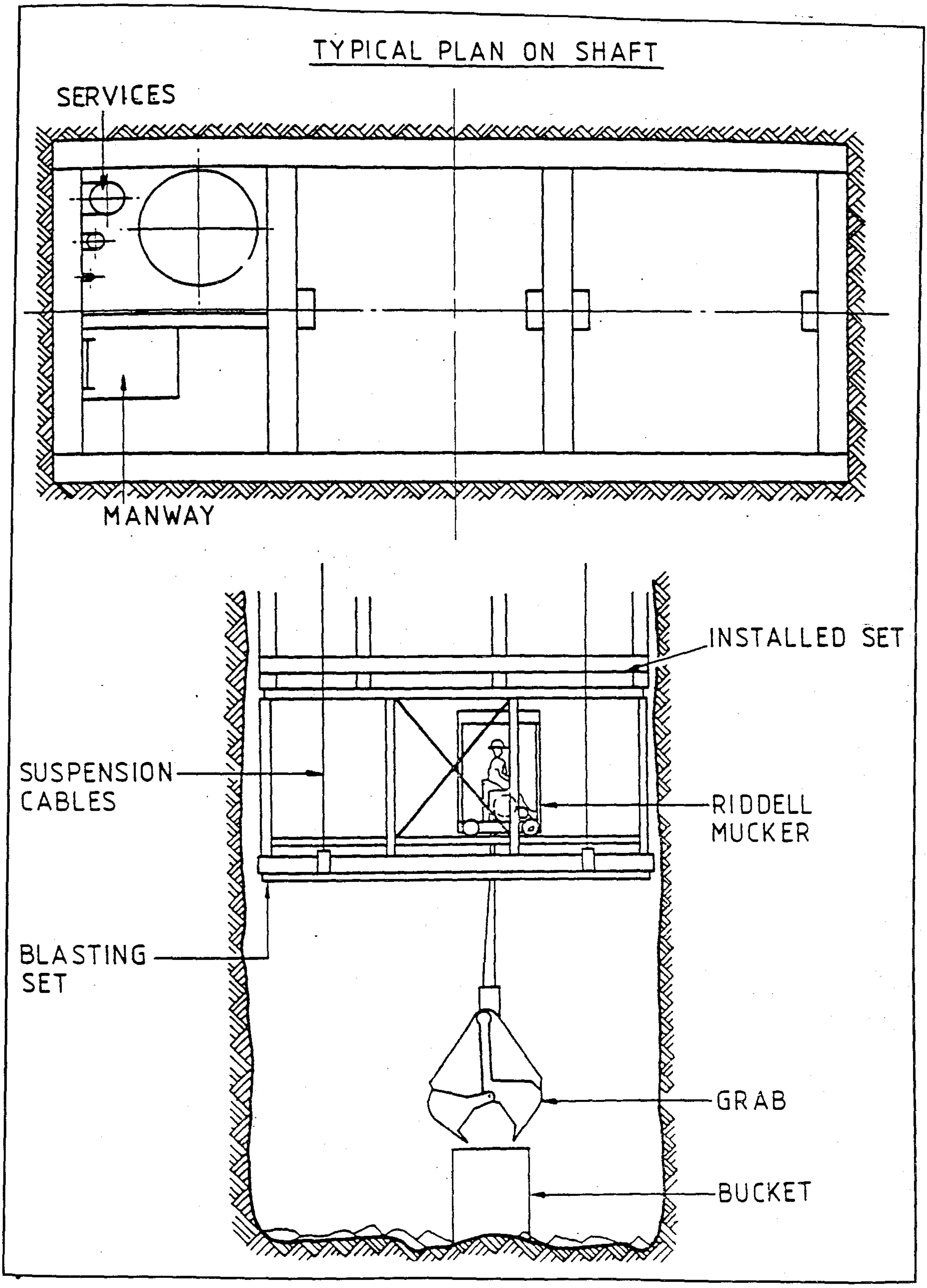


FIGURE 2.13. Simple Riddell type mucking unit (Single traverse) [10].

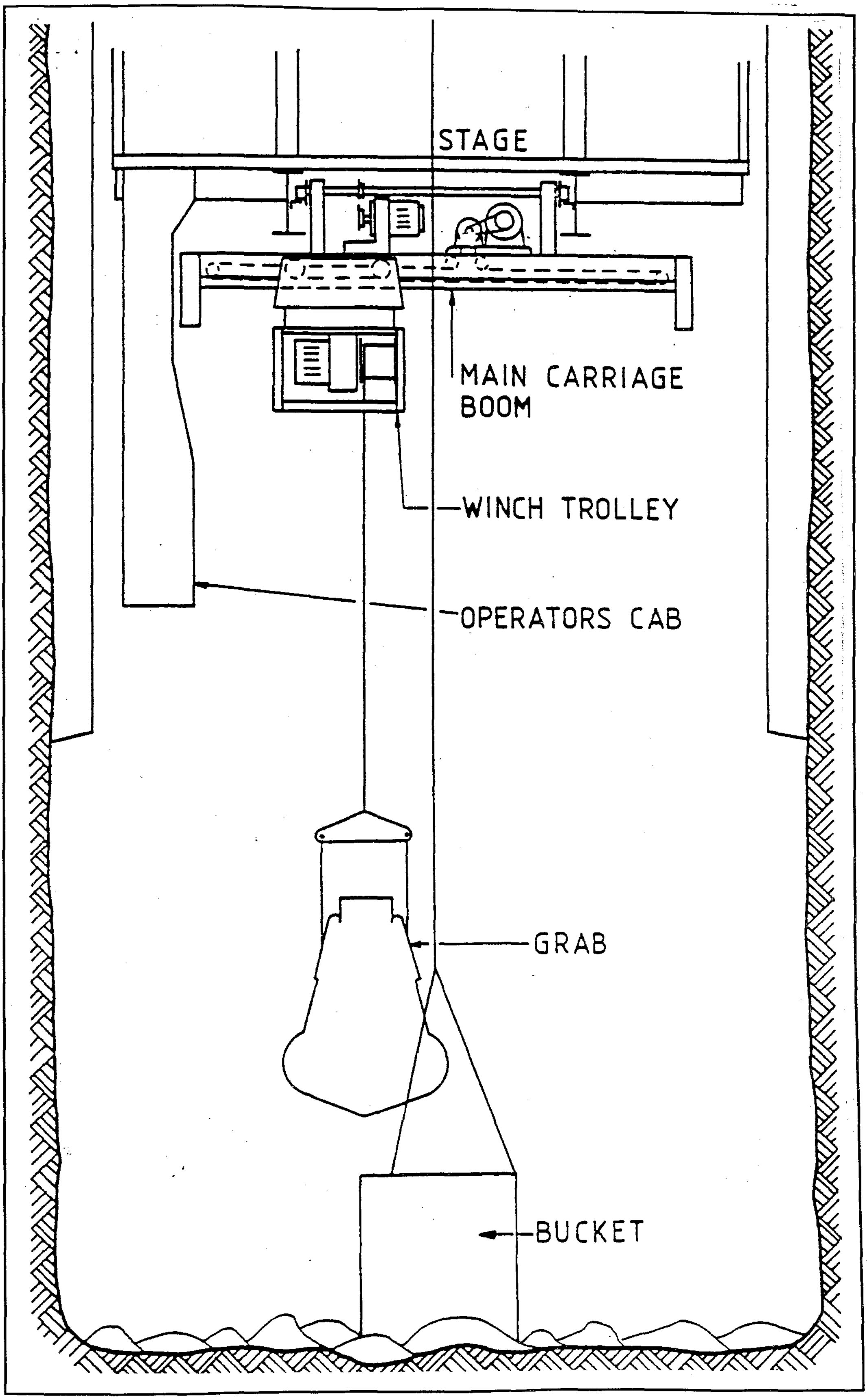


FIGURE 2.14. Two-way traveling mucking unit [10].

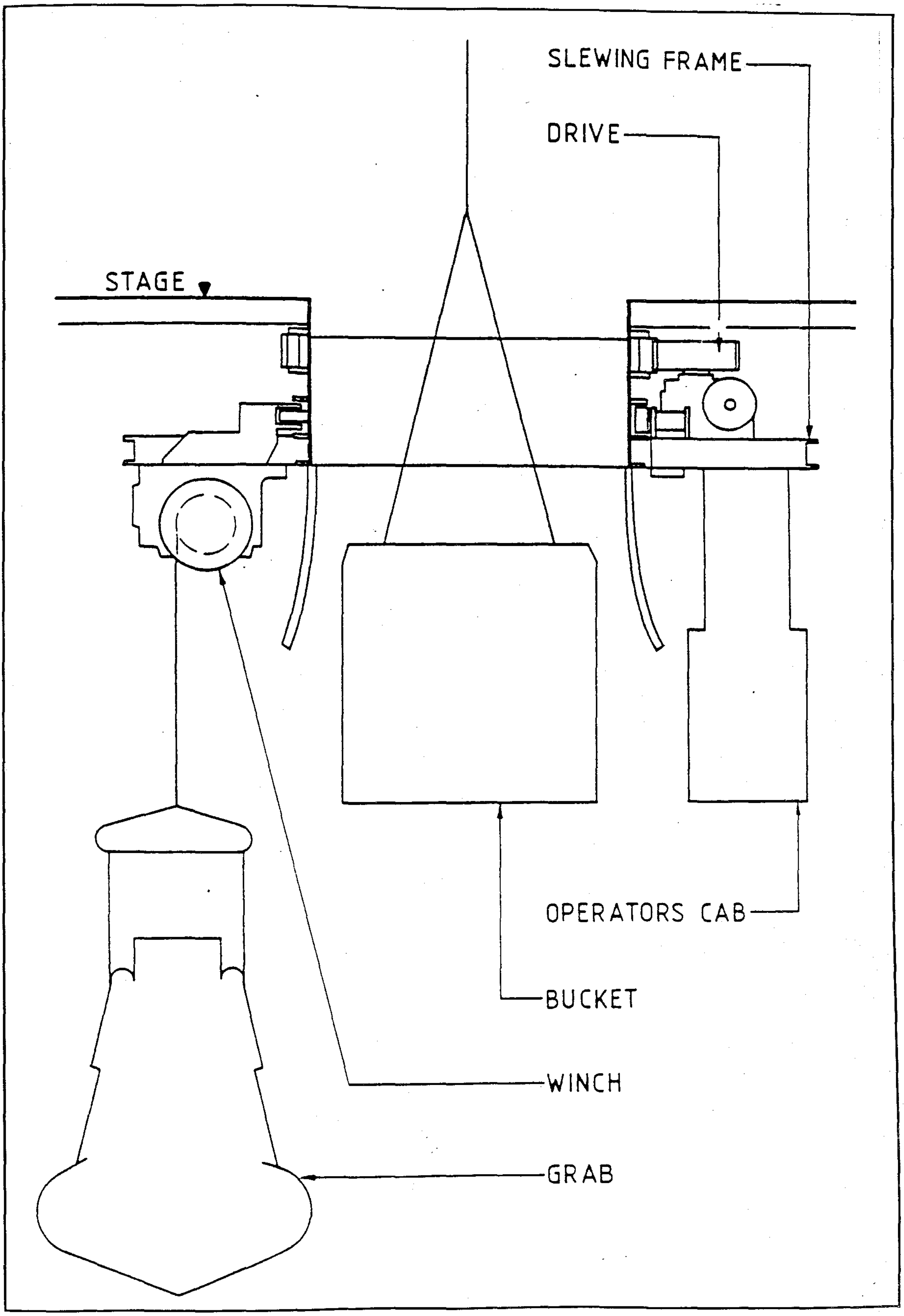


FIGURE 2.15. Rotary mucking unit [10].

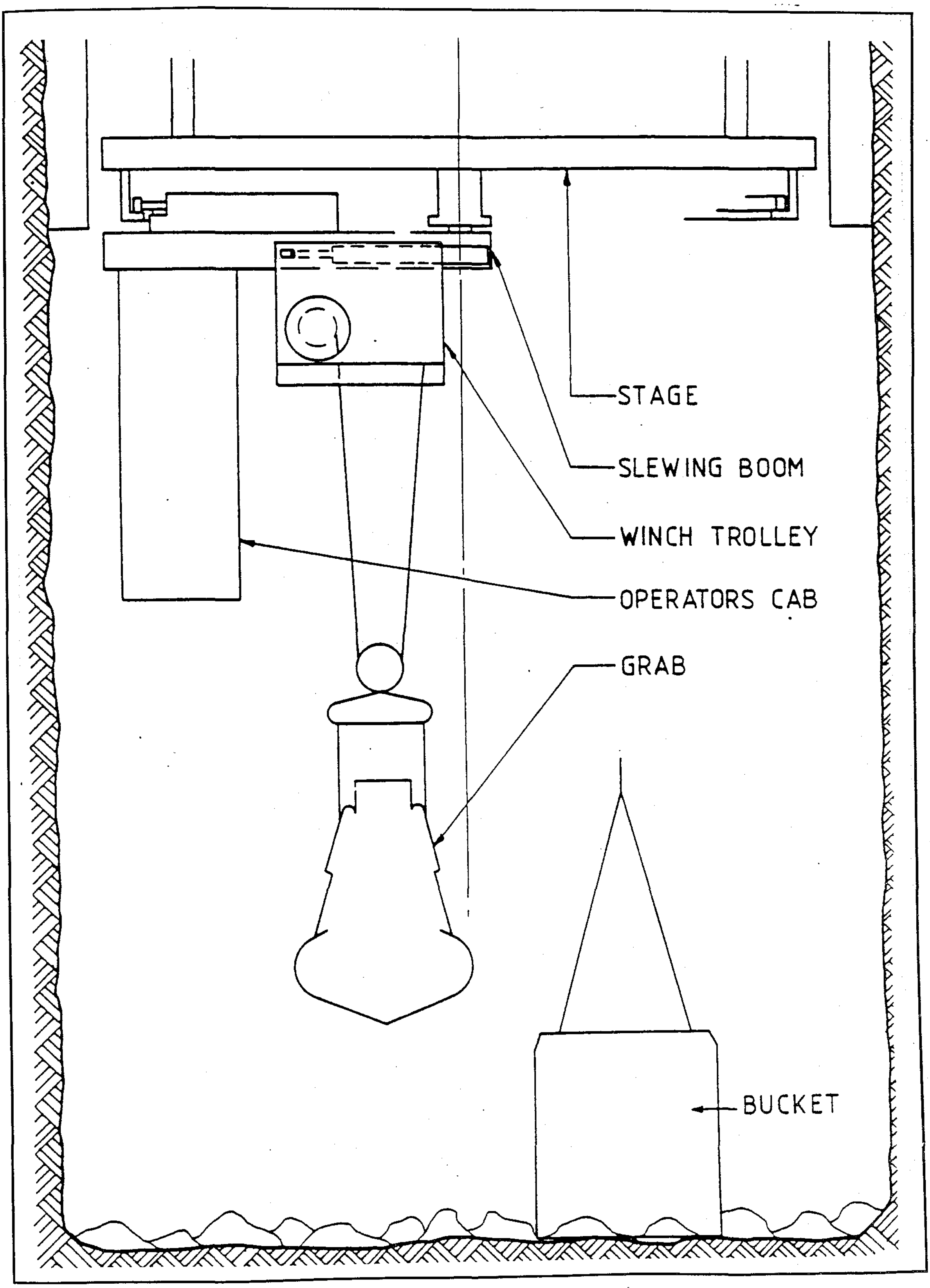


FIGURE 2.16. Radial mucking unit [10].

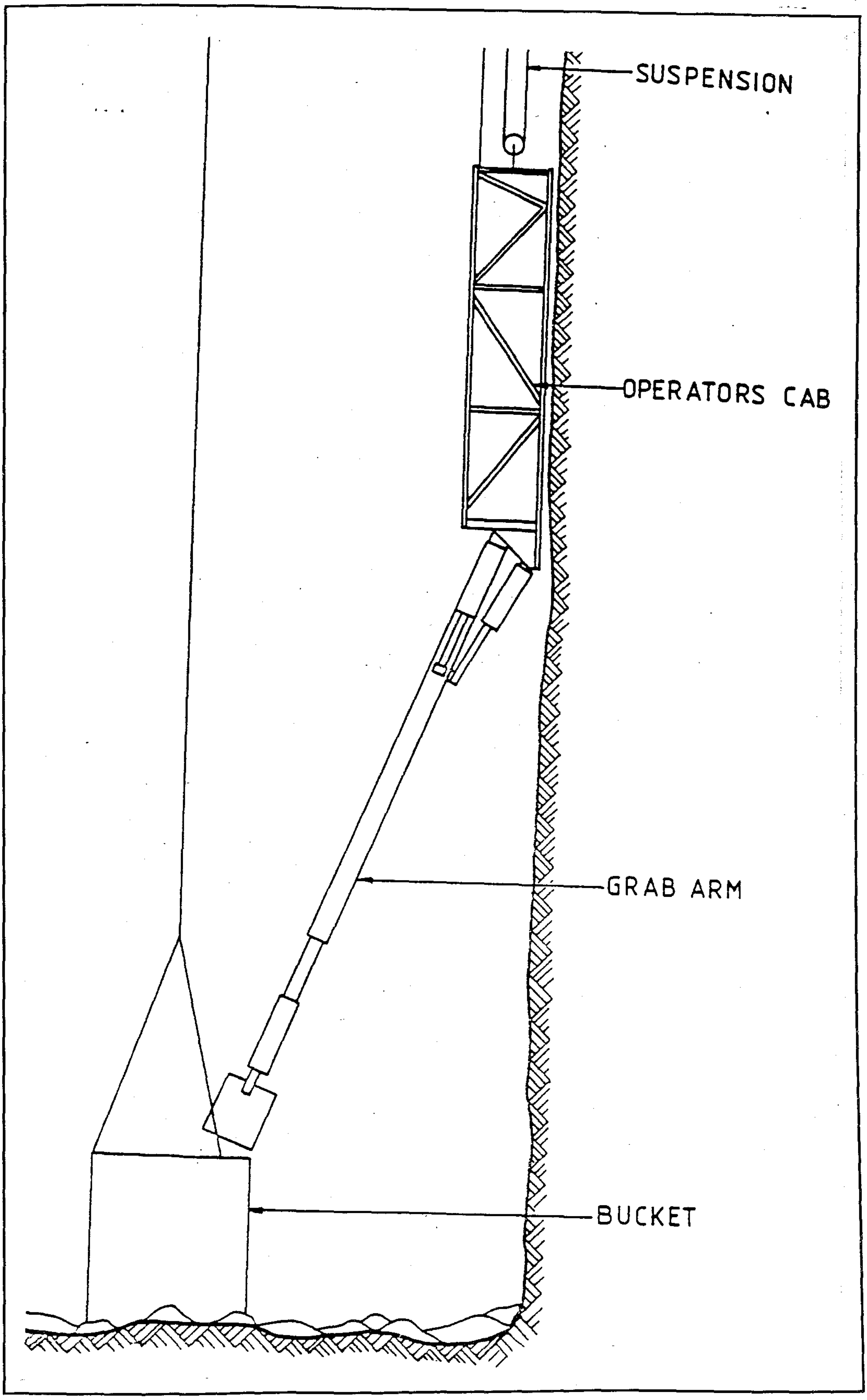


FIGURE 2.17. Cryderman mucking unit [10].

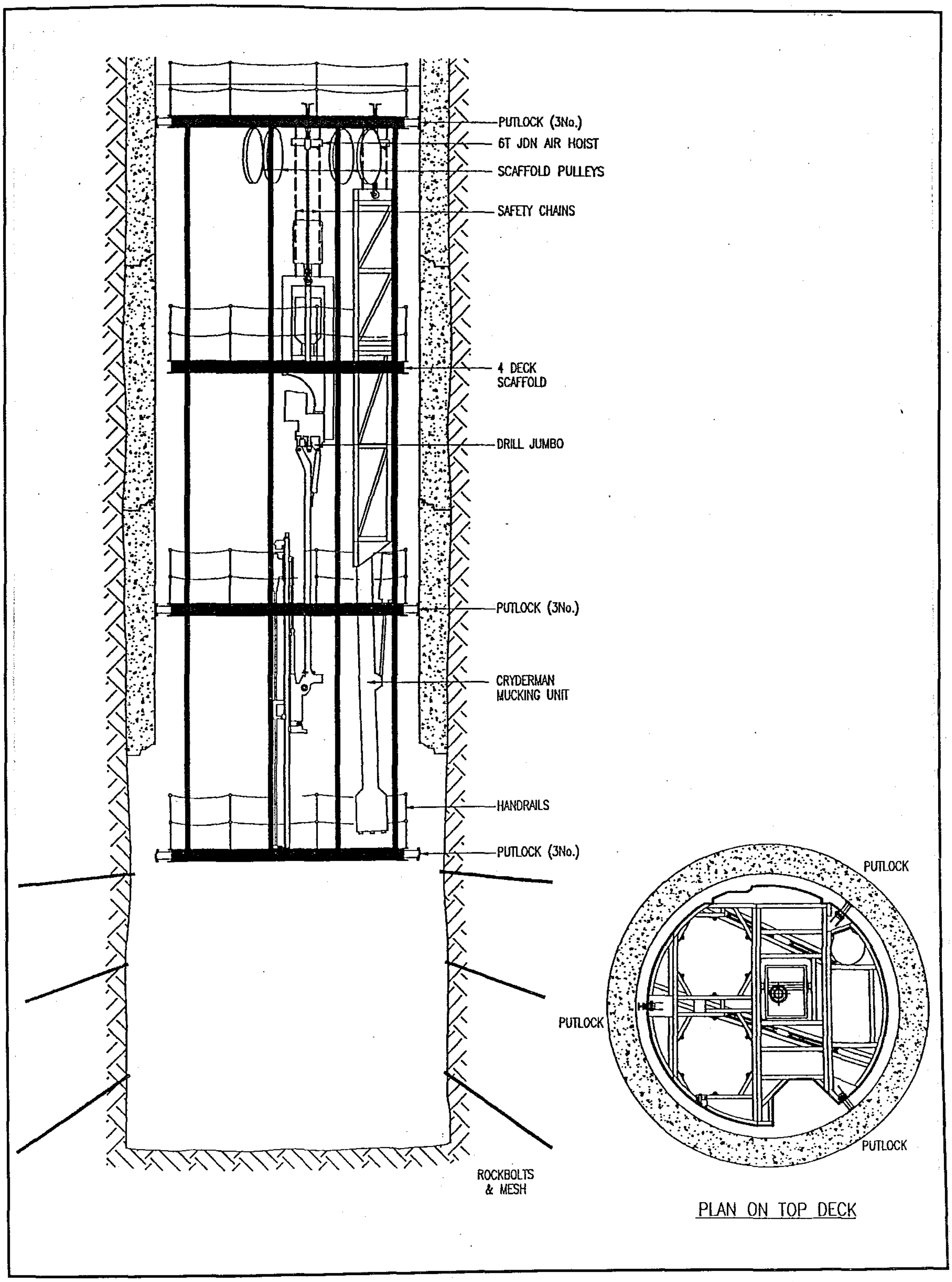


FIGURE 2.18. Example of a typical stage.

Normally sinking and lining are carried out consecutively because of the safety for men in the sump below the concreting operation and the crew size usually does not allow for simultaneous working [10].

2.3.2. Drilling of Rock

In shaft sinking, drilling is used either for blind drilling, which is explained before, or for blast hole drilling in conventional shaft sinking method.

In blast hole drilling, usually air leg drills, sinker drills or jumbo is used. The air leg drill is designed for drilling horizontal holes, but it is flexible and works well for other types of drilling. These are most used in drifts, crosscuts and stops. A sinker drill is designed for drilling down holes. The sinker is similar to an air leg drill without the leg. The sinker drill is hand held and is most used in shaft sinking.

For drilling blastholes, track-mounted bench drills and mechanical jumbos for tunneling and drifting underground have largely replaced hand-held drills. A jumbo is a self-propelled machine with several drills mounted on hydraulically operated booms. It gives a higher rate of production and lower operating cost. Development drilling is more rapid, which is particularly important in long-hole mining. It is very practical to drill deep holes with jumbo then hand held machines. Jumbos, however, require several headings if they are to be used effectively. Capital costs and maintenance costs are high. It has to be checked and cleaned after every drilling process [11].

2.3.2.1. Effects of Drilling to the Blast Performance

Effects of Blasthole Diameter: The blasthole diameter "d_b" is governed by; - the required degrees of fragmentation and overbreak control, -the blasthole length and/or face height (h_b) and very importantly, -the general reduction in drilling costs associated with increase in "d_b". Where "d_b" is small, the costs of drilling, charging, priming and stemming operations are high. If "d_b" is too small, these disadvantages outweigh the benefit of the slightly lower energy factor, which results from the superior energy distribution within the rock to be broken. Where "d_b" is too large, the correspondingly larger blasthole pattern may well lead to inadequate fragmentation, especially in rocks, which are strong and massive or contain widely spaced open discontinuities.

If the degree of fragmentation is to remain unchanged, an increase in "d_b" must be accompanied by an increase in the explosion energy yield per m^3 of rock (EF). The required increase in "EF" is greatest for blocky rocks and least for highly fissured rocks. In entirely massive rocks, an intermediate increment in EF is needed.

Effects of Blasthole Alignment : Where deep 150-200 mm down holes are drilled, the accuracy of blasthole alignment is most important in slot development, and especially for those earliest firing blastholes which are required to shoot to a raise (and particularly when the raise is bored rather than blasted).

When excavating caverns with downhole benches, the optimum blasthole inclination can vary between 0^0 (i.e., vertical) and about 25^0. For benches with heights of about 4m and less and blasts, which are always fired to a free face (i.e., a clean rock-air interface), satisfactory muck piles can often be obtained by drilling vertical blastholes. As face height "H" increases, vertical front-row blastholes become progressively overburdened at bench floor level. Therefore, the replacement of vertical by inclined blastholes maintains toe burdens at their design values. In benches higher than about 4m, blastholes are usually angled at $10\text{-}25^0$.

Effects of Blasthole Length: The face height or desired depth of pull should be such that the driller has a high degree of control over blasthole deviation and, hence, over both burden distance "B" and blasthole spacing "S" for the toes of charges. If blastholes are too long, both B and S will exhibit considerable variability. Where B and/or S is too small, fragmentation of an inadequate volume of rock will be excessive, and an appreciable proportion of the explosion energy will be manifested as air vibrations and flyrock. Where B and/or S is excessive, fragmentation will be sub-optimum. In those situations in which blastholes are tight and, therefore, drilled on close centers (e.g., in burn cuts in tunneling), blasthole deviation is a strong restricting influence upon blasthole length.

2.3.3. Blasting of Rock

In underground construction and mining, blasting is the dominant method of excavating rock and ore. Therefore, in order to maximize the cost-effectiveness of most of these operations, it is first necessary to optimize blasting. Any blast optimization program can show appreciable progress only after a clear understanding of the effects of the principal blast parameters has been developed and carefully applied.

2.3.3.1. Effects of Rock Properties. The design and results of blasts are affected by numerous factors, most of which are controllable. Unfortunately, the most influential single blast parameter, rock properties, can be effectively controlled only to the limited extent that the direction of firing relative to that of the dominant physical discontinuities within the rock mass can be varied somewhat by changing the initiation sequence.

Effect of Dynamic Tensile Breaking Strain (ε_t): If the fragmentation of blasted rock is to remain unchanged, any increase in ε_t (this being the effective resistance to breakage of a truly massive rock) necessitates; a) increases in energy factor "EF", and b) decreases in burden distance "B", blasthole spacing "S", stemming length "L_s" and, in some tunneling and shaft sinking operations depth of pull.

Where the strength of rock varies along the length of excavation, efforts should be made to progressively modify the blast design, so that any particular design is highly compatible with the rock strength at that location.

Effects of Structural Properties : Any increase in the mean spacing between physical discontinuities demands that a greater degree of (new) breakage is created in the blast. In massive rocks, therefore, EF values should be higher than B, S and L_s values lower then those in highly fissured rocks.

Highly fissured rocks often cause problems associated with overbreak and the stabilities of the back and/or walls of the excavation. The associated costs of support and/or linings tend to increase with the number of discontinuities unless this potential problem is given extra consideration during both the design and execution phases of blasting.

Perimeter blasting techniques are most successful in massive rocks and in formations in which tight discontinuities are normal to the axes of blastholes. In rocks which exhibit closely spaced discontinuities, some overbreak will occur (principally along the discontinuities), irrespective of the steps taken to prevent it. The very blasting technique that produces the desired effect in a massive rock may be quite unsuitable in a highly fissured rock. Because they need to change with rock properties, the spacing and charge concentrations for perimeter blastholes are site specific. The need for overbreak control increases with a decrease in effective strength of the rock [12].

2.3.3.2. Effects of Charge Properties and Types of Explosives

Types of Explosives :

Black Powder: Black powder is a mixture of potassium or sodium nitrate with sulfur and finely ground charcoal. It is still used in conventional munitions, sometimes in coal mining where lump size is to be retained, and in the quarrying of building stones such as granite, where the blocks must be separated along preexisting joints or planes of weakness rather than by shattering along newly created fracture surfaces.

High Explosives: In high-explosive dynamites used for rock blasting, the reaction front propagates supersonically, rather than subsonically as in black powder. Most of them have greater energy per unit weight of explosive. They are manufactured in the form of cartridges or "sticks" for use in small-diameter holes (down to 19mm), and have fair to excellent resistance to water. Nitroglycerin (NG), a major component of the dynamites, when in pure form, is relatively unsafe and sensitive to impact. "Straight dynamite" contains 20-60 per cent nitroglycerin together with sodium nitrate and a lesser amount of carbonaceous fuel. "Ammonia dynamites", which have largely replaced straight dynamites, contain ammonium nitrate and generally have a lower velocity of detonation, lower density, higher shock resistance, and better fume characteristics. "Gelatin dynamites" contain liquid nitroglycerin with nitrocellulose; 20 per cent and 60 per cent gelatin dynamites (the percentage refers to the content of NG) have very good fume characteristics and are often used underground; 75-90 per cent gelatin dynamites and 100 per cent gelatin and straight dynamite have very poor fume characteristics, are not used underground, and are mainly employed in underwater and geophysical blasting.

Initiating Explosives : Initiating explosives can detonate to produce an intense shock even when in the form of very small charges. They are used in small quantities to initiate detonation in larger and less sensitive high-explosive charges. They are supplied in copper or aluminum tubes to form detonators, or in the core of a detonating cord.

AN/FO and other Nitrocarbonitrate blasting agents: Although most rock is still excavated by dynamites, there has been a trend towards use of Nitrocarbonitrate (NCN) "blasting agents" such as AN/FO, which consists of an oxidizer and a fuel. The relatively inactive ammonium nitrate particles, coated with an inert absorbent material or treated with a surfactant to promote thorough mixing, are sensitized with fuel oil in the proportion of 94:6 by weight. The mixture detonates at a velocity of 3-4 km/s. AN/FO is the least expensive explosive available, giving an explosives cost one-third to one-quarter that of nitroglycerin-based high explosives. When correctly used, AN/FO performs as well or better than the dynamites and is safer to handle. However, it is soluble in water, so it can not be employed in wet conditions. Larger-diameter blastholes are drilled for economy because more AN/FO is needed to generate similar amount of energy. AN/FO is supplied either in bulk or in waterproof polyethylene bags. The separate components, delivered for bulk mixing on site, are not classified as explosives and so can be shipped without incurring the extra costs of transportation and storage precautions. For relatively dry holes, bulk free-running AN/FO is best because superior coupling is obtained.

Slurries: Specially designed ammonium nitrate emulsions and slurry-blasting agents have been developed for blasting in wet holes. A combustible fuel is mixed with granular AN dispersed with a sensitizer and thickener and with just enough of an aqueous solution of AN to give a semifluid mixture containing up to 20 per cent water. The combustible fuel component can be a material such as magneses, sugar, sawdust, sulfur, a heat-producing metal such as magnesium or aluminum, or even TNT. Thickeners include starch, water-soluble vegetable gum, or oil with an emulsifier. Sensitizers include microscopic glass bubbles, TNT, pigment-grade aluminum, and water-soluble organic nitrates. The mixture can be sold in final thickened form, or a gelling agent is added before or during loading so that it forms a thick gel after charging into the hole. This shields the slurry-blasting agent from external moisture. Sometimes slurries, and particularly emulsions, are blended with prilled AN or AN/FO to increase their energy density.

Slurries, gels, and emulsions have a higher velocity of detonation than AN/FO, ranging from 3.3 to 5.5 km/s. They also yield much higher detonation pressures, so they can be used with larger burdens and spacing to reduce drilling costs. Metallized slurries generate considerable heat on detonation and are used for blasting extremely hard rock. Slurries are denser than AN/FO, and so sink more readily in water-filled holes. They may be pumped, making loading quicker and easier, thereby avoiding some of the problems of underwater blasting. Slurry costs are comparable to those of dynamite and two-three times those of AN/FO [11].

2.3.3.3. Effects of Blast Geometry

Effects of Shape and Condition of Face(s) :

Good fragmentation and displacement are more difficult to achieve where the face; (a) is at an unfavorably large angle to the blasthole's axis,

(b) subtends a small angle at the blasthole,

(c) has not been cracked by one or more previous blasts, and/or

(d) is choked with previously broken rock.

A decrease in the angle between a blasthole and its face causes increases in fragmentation and muck pile looseness, the best results being obtained where blastholes are parallel to the face.

In benching-type operations, blasting is facilitated by both irregularities in the face and cracks in the burden rock created by the previous blast. Where smooth unfractured faces exist, fragmentation is achieved with greater difficulty, especially where the face curves away from the blasthole. When blasting to a relief hole commences in a burn cut, good initial fragmentation is discouraged for reasons, which include the following:

(a) Even where the relief hole has a diameter as large as 200mm, it provides a face which has a very restricted area and an unfavorable shape.

(b) The rock immediately around the relief hole contains few if any cracks created by previous blasts, especially near the base of each blasthole.

This explains why the earliest firing charge has to be located very close to the relief hole(s). Excessive burden distances tend to cause rifling and/or dislocation of (and possible ejection of charges from) adjacent later-firing blastholes. For the above reasons, the burden distance in burn cuts (and for the earliest-firing charges around bores raises) must be appreciably less than those which are employed when identical charges shoot to an extensive parallel planar (or biplanar) face. Because a relief hole represents a poor face, burn cuts should be designed so that each of the early firing blastholes can shoot to at least two equidistant relief holes. The combined cross-sectional area of relief holes should be increased when attempting to pull longer rounds.

Effects of Available Expansion Volume: When broken, all rocks expand. If the available expansion volume is less than about 15 per cent of the volume of the solid rock in a blast, all fragmentation mechanisms will proceed to completion, but the particles of broken rock will still be highly interlocked and, therefore, will not flow/ rill readily to the draw points. If the expansion volume is <<15 per cent, one or more of the later-acting breakage mechanisms may not even proceed to completion. Where mass pillar blasts are fired into slots, a minimum expansion volume of 25 per cent is usually required if consistently rapid mucking without hang-ups is to be achieved. If the void volume provided by the relief hole(s) in a burn cut is too small, recementation of the finely fragmented rock tends to take place. Wherever possible, the void volume should be 15 per cent of the volume of the cut. In rocks, which have a tendency to freeze, the void volume should be as great as is practicable if the probability of leaving long butts is to be minimized. Burn cuts with very small void volumes can still be pulled to full depth provided that the energy factor (in the cut) is increased sufficiently; but rounds are pulled more consistently and to greater depths through the application of finesse rather than brute force.

Effects of Type of Blasthole Pattern : Ideally, blastholes should be drilled on equilateral triangular grids, since these provide the optimum distribution of energy within the rock mass to be broken and, hence, the best fragmentation. Where blastholes are vertical, S= (1.15)B, but when they are angled at θ^0 to the vertical , S = (1.15)B $\cos\theta$.

Effects of Burden Distance "B" and Blasthole Spacing "S" : For a given set of blast conditions, there is an optimum burden "B_0" for which the volume of suitably fragmented and loosened rock is maximum and toe conditions are acceptable. Normally, "B_0"lies in the 30d-35d range; the coefficient of "d" tends to increase with a decrease in "d" and depends upon the properties of the explosive and, more particularly, the rock.

In benching and sub-level open stopping with parallel blastholes, "S" is necessarily a function of the width of the blast block. As this width decreases and/or "d" increases, the number of blastholes in each row decreases. In narrow blasts with large-diameter blastholes, there fore "S" may need to be considerably less than the (desired) optimum spacing "S_0" for those particular conditions. If $S<S_0$ then "B" should also be $<B_0$. The prevention of such inefficiencies should be a significant consideration when selecting "d" for a given situation. It should also be recognized that less overbreak can be achieved by selecting a spacing for perimeter blastholes which is 15-25 per cent less than that for other blastholes in the row or ring.

2.3.3.4. Effects of Initiation Sequence and Delay Timing. In any multi-row blast (including tunneling rounds), it is most important that charges detonate with the sequence and timing which maximize the successive development of free faces, which are as extensive as possible (and preferably concave) and reasonable near. When allocating delay numbers in initial designs, operators should construct theoretical lines of breakage for each charge. By doing this, any instances of poor sequencing are exposed, and alternative superior delay allocations can then be made.

The rock fragmented by the first one or few charges in a burn cut is ejected laterally into the void provided by the relief hole(s) before being swept outwards along the tunnel's axis. The time taken for these rock fragments to be completely swept from the cut is considerable (typically >= 100ms). It follows, then, that the delay between consecutive detonations should exceed 100ms if the probability of choking is to be minimized. Where charges are fired on consecutive numbers of a millisecond series of delay detonators, good progressive relief of burden is not achieved and, as a consequence, there is a higher risk of choking and a frozen cut.

In benching, initiation should commence at or close to the center of the first row. Delays should be allocated so as to maximize progressive relief and to minimize overbreak and ground vibrations. If possible, the delay allocation should ensure the successive development of biplanar free faces for the highest possible percentage of charges in the blast. Charges should not be fired in a square V or rectangular V formation, especially when downhole bench blasts with a length:width ratio <1 shoot into a buffer of broken rock [12].

2.3.4. Water Control

Many of the new mine shafts now being built penetrate deep aquifers, so it is essential to understand the hydrogeology of the new sites so that shaft wall designs have an in-built structural integrity against high pressures, and this low permeability enables dry conditions to be obtained. To obtain the hydrogeological data on which to base the shaft wall designs, it is necessary to bore a cored hole through the center of the proposed shaft site. This will show exactly the geological formation, through which the shaft will pass and permit all the tests required to give meaningful results. Ground water occurs in permeable geological formations having structures that permit appreciable quantities of water to move through them under normal field conditions. It is the distribution and size of these structures that is of prime importance when evaluating the hydrogeological characteristics of a rock formation. Water-flow through a rock formation, and hence the permeability, can be attributed to either intergranular flow, i.e. via pore spaces within the rock, or via fissure-flow, i.e. through natural joints in the structure.

There are essentially two methods of allowing an excavation to proceed efficiently through water-laden strata; water is either prevented from flowing into the excavation, or it is allowed in and immediately pumped out from the sump. It is considered that the present mining construction techniques can tolerate a flow up to 40-50 gpm into the excavation before action must be taken to reduce the flow.

There are three ways of excluding water from an underground excavation:

1) Pumping - using Well Points and Deep Walls: Well points are used to lower the water table in the immediate sub-surface. As they are operated by a vacuum pump (one pump working a number of wells via a manifold), they are limited to a

theoretical maximum depth of approximately 10m. This method has been successfully used to dewater glacial sands and silts to a maximum depth of 9m at Gascoigne Wood drifts in the new Selby complex.

The stability of the sides of the excavation must always be suspect in sandy formations; for this reason well points are often used in association with an open cut or a cofferdam. Where the ground has to be dewatered to depths exceeding 9m, then deeper individual pumping wells are needed. For a pumping well the hole has to be enlarged to contain a submersible pump which is positioned inside a perforated casing and gravel system.

2) Grouting: Grouting is a technique used in mining and civil engineering that not only reduces the permeability but also improves the stability of porous or fissured soils or rocks. There are two types of grout commonly in use, namely, cement and chemical grouts.

Cement Grouts: Cement-based grouts usually comprise of ordinary Portland cement, water and some form of filler such as sand. Cement grouts have been used very successfully in reducing flows in fissured and jointed strata whose fissure width is greater than 200 microns, e.g. Lower Magnesian Limestone and fractured Coal Measure Sand-stones. Cement grouts are primarily used in fissured strata, although they can be used to stem intergranular flows in very coarse sands and gravels. Cement grouts are virtually inert except in the presence of contaminated minewater enriched in sulfates and chlorides. Under such conditions there is the possibility of long-term chemical attack on the cement grout, especially when cement grouts are used in Permian strata containing evaporates. If a permanent grout curtain is required, a second grout phase should be planned some time after the final construction.

Chemical Grouts: Chemical grouts are normally used to stem intergranular flows when the coefficient of intergranular permeability is very low. Chemical grouts can readily penetrate medium grained sandstone whose pore size is greater than 20 microns, e.g. Basal Permian Sands. In many situations a combination of cement and chemicals is used. The range of cement grout covers fissures and joints of 200 microns gap and larger. Nevertheless, at depths of 1200m below the water-table a single hairline fissure (100 micron width) fully intersecting the shaft will produce, if recharged, an inflow in the region of 10 gpm and a number of such fissures will, in the absence of effective secondary chemical grouting, produce a significant inflow.

Some disadvantages in the use of grouting as a ground treatment are as follows:

(a) Although initial plant costs are low, costs in term of time and grout quantities can be difficult to predict in some types of strata.

(b) Severe difficulties can often be experienced when grouting in strata which exhibits both high primary and secondary permeabilities.

(c) Care must be taken in the application of grout pressures and the guard put against the possibility of hydro fracturing. Structures on the surface and within the zone of influence of the grout curtain should also be made secure against possible movement, which could be caused by grout pressure.

(d) Where portable water is extracted from the aquifer, toxic grouts should not be used.

(e) There is nearly always a residual flow of water in the excavation even after a successful grouting operation.

3) Freezing: Freezing is normally used when the permeability of a water-bearing formation indicated that a grout cannot be guaranteed to seal off effectively the water from the mining excavation. Freezing may also be used to stabilize a weakly cemented rock that may flow under full hydrostatic pressure and that, although penetrable, may not be sufficient stabilized by a chemical grout. It is also used where chemical grouts cannot be used because of their toxicity. The decision to freeze is not undertaken lightly due to a number of disadvantages:

(a) It is generally the most expensive method of ground treatment,

(b) It can cause complications of heave and settlement at the onset of freeze and at the cessation. This may not only damage structures and services but may cause mining works to be out of alignment due to movement in permanent survey stations.

(c) If any error is made with freeze hole abandonment, water could be communicated through impermeable strata which the holes have passed. A consequence of this can be to threaten the integrity of a shaft by applying a hydrostatic pressure on a non-hydrostatic designed lining. Where evapories are present, a serious problem can be created in the dissolving of these salts and can leave large voids behind the shaft wall.

(d) A successful and apparently perfect freeze hole abandonment can in time, and depending on the water characteristics in the strata penetrated, cause changes in the structure of the grout used for the abandonment, and consequentially will allow water-passage.

(e) There is no lasting reduction of permeability and there is no lasting additional strength given to the strata [13].

One of the first question asked when considering water control is whether any ground treatment will be required, and if so, whether grouting or freezing is the most appropriate method? These decisions require a proper hydrogeological survey of the ground conditions to have been made; assuming this to be available, there is usually no doubt about whether pumping from the sump alone will be sufficient or not. The arguments usually revolve around what type of ground treatment is most appropriate. There are, of course, circumstances when grouting is impossible, and freezing is the only viable technical solution - for example with silts or clays. In other cases both grouting and freezing are technically feasible.

When considering solely water control, and not strata strengthening, the best technique from the point of view of shaft security must be grouting - provided it is properly executed. This is because once the grout is in place, it does not require subsequent maintenance to remain effective. At the other end of the spectrum, dewatering rely on reliable pumping systems and wells; any prolonged equipment failure or bad silting up of the wells could result in, at best, conditions in the sump where working is impossible or, at worst, a flooded shaft.

Freezing is normally looked upon as a safe method of water control, and it is true that, once a proper frozen zone has been established, quite prolonged equipment breakdown can be accommodated without damage. However, if a leakage through the frozen ground does occur, the flowing ground water quickly melts the ice in the leakage vicinity, which can cause severe problems. The result is that it is necessary to apply a larger factor of safety to take account of this risk.

From the point of view of estimating time and cost for the three techniques, freezing is probably the easiest, followed by dewatering with grouting coming a poor third. This does not necessarily mean that the actual treatment cost is that order.

2.3.4.1. Grouting. In this section currently available grout systems and the selection of a suitable grout for a particular purpose during shaft sinking operations are explained. Environmental considerations prevented the use of a number of very effective grout systems. With the exception of Cemex D, none of the silicate-based systems described in this study should present any toxicity or other environmental problems in either the fluid or gelled state.

During the sinking of a shaft, probe drilling and/or borehole logs may indicate the presence of water and/or unstable ground conditions. Having decided that grouting is the method to be employed in order to sink the shaft through this particular strata, the following questions need to be answered:

1) What engineering properties are required from the grout cover?

2) What type of grout are we proposing to inject and under what conditions?

There are only three answers to question one. The properties required are either :
a) A water stopping grout; b) A consolidation grout; c) A combination of both.

If the requirement is for an increase in strength, the range of grouts, which can be used, depends on the final strength requirements of the grouted ground. The range of grouts available decreases as the strength requirement increases.

As the ground becomes tighter, if the number of grout holes required is not to be excessive (a process which can also have a weakening effect) then the viscosity of the grout which can be used will also limit the choice and may also limit the ultimate strength obtainable.

The sole "consolidation only" grout available is the urea-formaldehyde system. This grout is not suitable for water-stopping as its residual permeability is only 10^{-5} cm/sec. The urea-formaldehyde system is very strong and has a low viscosity, but it does present problems in use. The very high formaldehyde content makes it very unpleasant to handle, even with good ventilation.

Other readily available high strength systems are Cemex D3 and Cemex D4, the high solids silicate-ester system, the high solids resorcinol-formaldehyde systems and the isocyanate-based grouts. The isocyanate systems normally used where there is rapidly flowing water and flash gelation of the grout on contact with water is required. All the high strength systems with the exception of urea-formaldehyde have high viscosities. This high viscosity precludes their use in very tight ground.

Therefore, other criteria must be used in the selection of a suitable grout and obviously this selection will be influenced by the answers to the question "What type of ground are we trying to inject?" the following factors should influence this selection:

1. Permeability: The injectability of the grout is affected by its viscosity and cleanliness (i.e. the absence of particulate matter). The permeability of the ground will limit the grouts, which can be used. Thus, tight ground will preclude the use of high viscosity grouts.

2. High Back Pressure: High back pressure can cause the grout to extrude after they have set from cracks and wide pores. The low solids silicate grouts are very susceptible to this eg. Cemex DP, silicate-bicarbonate and silicate-aluminate. The acrylamide grouts will not extrude.

3. Brine Compatibility: The only grout systems compatible with brine are the acrylamide, resorcinaol-formaldehyde and urea-formaldehyde. All other systems precipitate when mixed with brine.

4. Ground Water into which the grout is injected: Ground water with a high salt content can cause the grout to form a fine precipitate which blocks the pores around the injection hole, preventing further injection of grout. The silicate grouts which gel on the alkali side (Cemex D, silicate-bicarbonate, Cemex F and Stabgel) are the most affected.

5. Temperature Limits: The lower temperature limit for all water based grouts is 0°C. However, as this temperature is approached all the grout systems increase in viscosity. For the high solids silicate systems this increase is very pronounced. The only grout system which it is possible to use at below 0°C is the modified Rocagil system which will remain fluid and gel at -30°C. Below 10°C Cemex D type grouts become very difficult to use and very sensitive to proportioning ratios due to the slope of the gel time/temperature curves.

The upper usable temperature is usually 40-50°C. However, some grout systems, particularly the resorcinol-formaldehyde systems may boil on gelation if the starting temperature is too high.

6. Temperature to which the grouted ground will be subjected: The acrylamide grouts will survive a freeze-thaw cycle. The silicate grouts, however, will decompose on freezing. There is no information available on any of the other grout systems.

7. Permanence: All grouts will eventually dissolve; the only variable is the rate and the type of dissolution. The rates of dissolution range from being fairly fast for the silicate-bicarbonate systems to exceedingly slow for systems such as TACSS. However, the silicate systems Cemex A2 and Cemex D and the acrylamide systems have an expected life of 10-30 years under the worst conditions. The life of the tannin-formaldehyde system is not known.

8. Gel Time Range: The majority of the grouts can be controlled to give gel times within the range 15 minutes to four hours, eg. Cemex A', Cemex D, Rocagil 1295 and BT. However, the gel time range obtainable for some of the other systems falls partly outside this desirable range, eg. Geoseal MQ5.

9. Strength: The low solids silicate systems, eg. Cemex A2, low solids Cemex D, silicate-bicarbonate, silicate-aluminate and low solids Stabgel systems are designed as water-stopping systems and do not impart much strength to the grouted ground. If these grouts are used to inject a sand layer the grout cover must be complete, as any flow through a grout window which may cause movement of the sand is liable to result in physical erosion of the grouted ground around such a window, causing failure of the grout curtain.

10. Toxicity: Toxic hazards have been responsible for the discontinuance from use of many very effective grout systems, particularly AM9 and TDM. Less toxic replacements have been developed for AM9, eg. Rocagil 1295 in which the DMAPN is replaced by an amine glycol mixture and Rocagil BT in which the acrylmide is methylated to reduce its toxicity. However, the methylation process does give rise to a small quantity of residual formaldehyde in the grout. This only apparent however, when the grout is used in a poorly ventilated confined space.

The grouts containing formaldehyde (Borden MQ4, MQ5 and MQ14 and urea-formaldehyde grouts) are very unpleasant to use unless the components are contained in closed vessels. The excavation of treated ground is also unpleasant. The disposal of spilt grout may also present a problem.

The most environmentally acceptable grout systems are those based on silicate. They are also, with the exception of Cemex D (where hydrolysis of the formamide produces ammonia) the most pleasant to use, being virtually fume-free. The production of ammonia by Cemex D, which does make subsequent excavation unpleasant, was the reason for the development of Cemex F. In this grout the formamide is replaced by glyoxal, which eliminates the problem, but there is a cost penalty.

11. Grout Cost: The most flexible grout systems (provided a small amount of residual toxicity is acceptable) are the acrylamide based systems, particularly the low toxicity variants Rocagil BT and BT/2. However, they are the most expensive of the grouts within the general consolidation category having strengths of 50-150 psi, with water-stopping capability. They are 20 times the cost of the cheapest grout (silicate-bicarbonate) and three to four times the cost of TDM, Cemex A2, Cemex D1 and Borden MQ5. Thus, there is little doubt that currently the silicate grouts are the most cost effective in satisfying most current grouting requirements.

12. Ease of Use: The plant used for chemical grouting ranges from a bucket and a stick plus a hand pump (used for some small leak sealing jobs), to a large grout production plant. Most of the silicate grout systems are sensitive to proportioning ratio and mixing order and, therefore, require more sophisticated plant which will proportion the two grout components accurately and mix them in the correct order. The acrylamides and TDM are less sensitive to mixing and proportioning but require accurate measurement of the catalyst concentration. The only grout system, which is insensitive to mixing, is Borden MQ5.

Having carefully considered all the criteria, which have a bearing on the selection of a suitable grout, the final choice should present no difficulty [14].

2.3.4.2. Freezing. Artificial Ground Freezing (AGF), used as an aid to excavation in many civil and ground engineering operations, involves the formation of a temporary ice wall thick enough to act as an impervious barrier to underground water inflow and strong enough to provide stability and mechanical strength in fine-grained water saturated soils/rocks. Examples include the construction of tunnels, large diameter cofferdam excavations and structural underpinning, but, by far the most important operation utilizing the AGF process is the sinking of deep-mine shafts.

In practice, the AGF process is carried out essentially by sinking a number of vertical concentric tubes equally spaced, about 0.92 - 1.37 m apart, around the perimeter of a circle or circles (freeze ring/rings) which may be up to six meters larger than the finished shaft (Figure 2.20). Each concentric freeze tube is composed of an open ended inner plastic pipe (pressure pipe) of about 50.8 - 63.5 mm in diameter and an outer ferrous-metal (fall pipe) with a diameter in the range 101.6 - 152.4 mm. To isolate the excavation from any underground water source, freezing tube usually terminate at a depth that is in impermeable bedrock. The exact location of the freeze holes depends on a number of factors including the design strength of the final ice-soil/rock material, the mechanical characteristics of the frozen ground, the duration of freeze time specified and the cost of the drilling operation. The coolant, usually 30 per cent calcium chloride at a temperature normally lying between -20°C and -30°C pumped continuously at a velocity of 0.457 - 0.762 m/s down the inner plastic pipe, extracts heat from the surrounding ground while traveling up through the annulus (formed by the two pipes) at a velocity of 0.31- 0.5 m/s. The coolant is subsequently collected at the ground surface via riser pipes, pumped to the refrigeration plant where it is cooled using a suitable refrigerant, commonly ammonia, and finally circulated back into the plastic pipes (Figure 2.21).

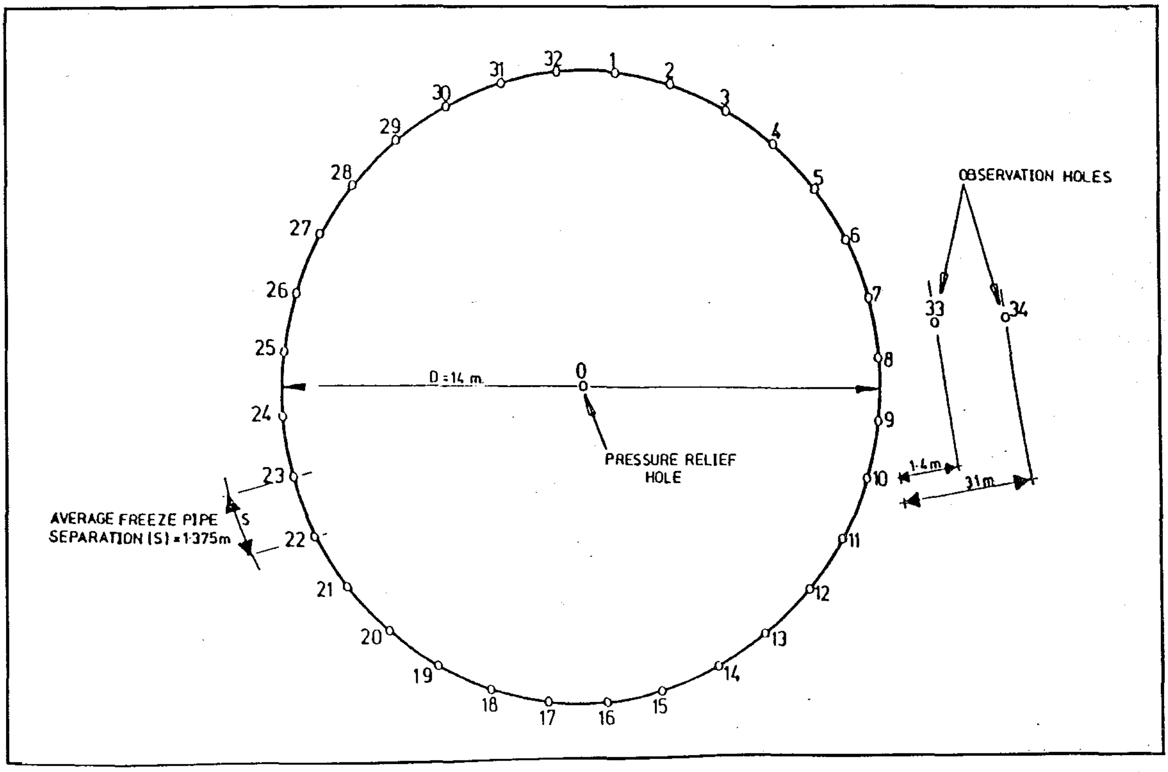


FIGURE 2.19. Example of a freeze pipe surface geometry – North Selby freeze pipe surface geometry [15].

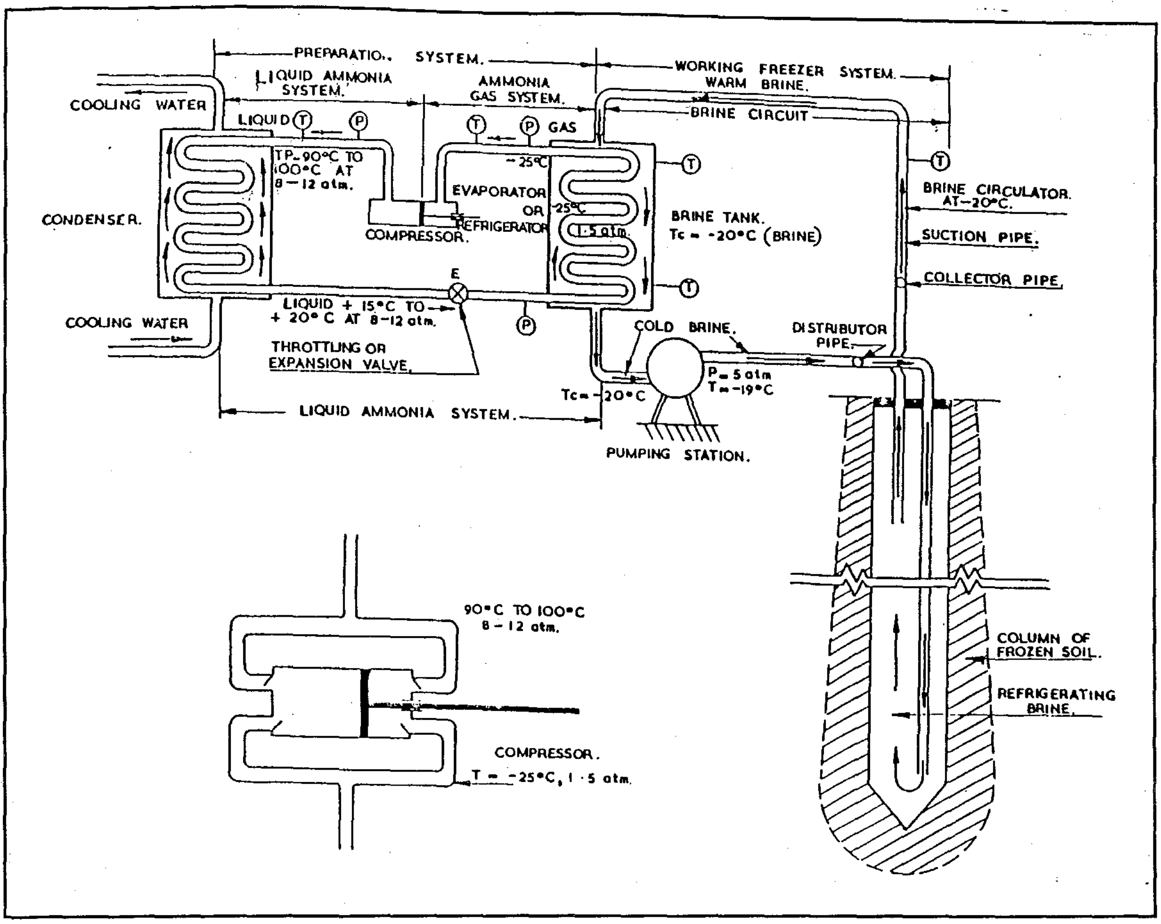


FIGURE 2.20. Schematic diagram of ammonia refrigeration system [15].

The rate of heat extraction and the final temperature distribution in the ground depends, for any given stratum, on a number of factors including heat flow in the ground, flow conditions inside the freeze pipes and brine temperature. Decreasing brine temperature in order to increase the heat extraction rate may lead to an unacceptably heavy refrigeration load. In practice, therefore, the refrigeration load and hence heat extraction rate is reduced at the start of the operation by using a higher brine temperature. The temperature is gradually dropped to the minimum specified value over a period of a few weeks as freezing progresses, hence, ensuring that the load throughout the operation is kept within practical limits.

The freezing of the ground initiates with the formation of a thin layer of ice-soil/rock, which grows gradually with time, usually over a three-four week period, to form a thick column about 2.3-3 m in diameter, around each freeze pipe. These individual frozen columns subsequently merge to establish a continuous ice-wall which then increases in cross-section as freezing continues. (Figure 2.22). The closure of the ice-wall could take up to four months to complete and the final established ice-wall is designed such that it is thicker near the bottom than at the top, so that it can withstand the increased stresses arising there during excavation. Furthermore, once the necessary ice-wall thickness has been developed, it is still important to continue the freezing process and maintain the ice-wall against thawing and externally applied loads. The freezing process is finally stopped when the foundation is laid or when the installation of the shaft lining has reached a suitable stage. The principle mode of heat transfer during the AGF process is by conduction with the rather low thermal properties of the soil or rock being a major rate controlling step. The operation is thus slow and expensive requiring up to six months at times. It is therefore important to ensure that alternative and often cheaper methods such as grouting are carefully considered and ruled out before undertaking the AGF technique. Further, ground freezing is often difficult to achieve when underground flow velocity exceeds some one - two m/day. Above this critical range, the rate of heat supplied by the flowing water is high enough to cause severe retardation of the freezing process and can lead to considerable openings (windows) in the final established ice-wall. Even with underground water velocities below this critical range, the rate of ice-wall build up is reduced and the time to develop the required thickness is increased.

From an engineering viewpoint, the mining engineer or the AGF designer who has to sensibly examine the cost and properly organize the development of the freezing operation needs to carry out the following:

1) Mechanical design leading to the required ice-wall thickness.

2) Thermal design leading to the specifications of :

   (a) location, spacing and sizes of the freeze pipes.

   (b) type, capacity and condition of operation of the refrigeration plant.

   (c) duration of freeze time.

   (d) the rate of ice growth, the final temperature distribution and the developed ice-wall thickness [15].

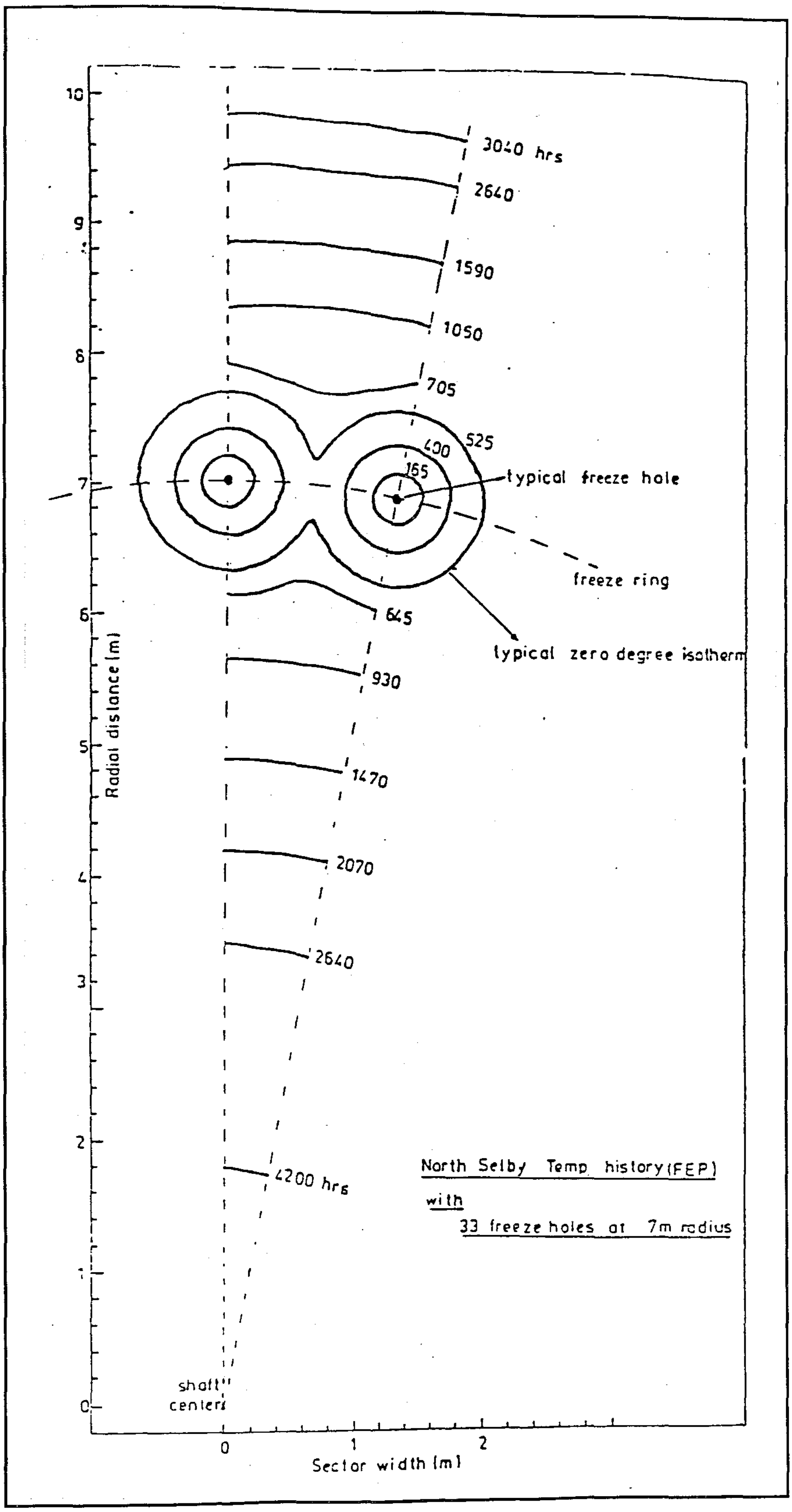


FIGURE 2.21. Developing of ice-wall thickness [15].

2.3.5. Shotcreting

The original process of spraying dry-mix mortar was given the proprietary name “Gunite”. The early 1930’s saw the generic term “shotcrete” introduced by the American Railway Engineering Association to describe the Gunite process.

In 1950’s, guns were developed which could handle coarse aggregate in the dry-mix process. In 1951 the America Concrete Institute (ACI) adopted the term “shotcrete” to describe the dry-mix process. In 1966, ACI applied the term “shotcrete” to all pneumatically applied mortar and concrete (both the dry-mix and wet mix processes). ACI 506R-85, “Guide to Shotcrete”, defines shotcrete as “Mortar or concrete pneumatically projected at high velocity onto a surface”.

Although various other terms have been used to describe the shotcrete process and different terms are still in use, the use of current ACI terminology “shotcrete” is, however, encouraged for both the wet-mix and dry-mix process.

When concrete is sprayed from a shotcreting gun onto a rough rock surface, it initially fills openings such as cracks, fissures and joint planes, binding together loose or partially supported fragments and preventing any further deterioration. The adhesion depends on a mechanical meshing of the particles in the concrete and those forming the rock surface. First a thin layer is formed by the cement grout and sand particles less than 0.2mm in size. This fine material penetrates the pores and cracks and provides a foundation for the placement and compaction of the total thickness. During the initial layer build up, most of the coarse aggregates rebound and fall to the floor.

Adhesion is obviously better on freshly broken thoroughly washed, rough rock than on a smooth surface such as metal or plastic. Weathered, spalling, friable, flaky, and muddy or shale rocks also have poor adhesion. Cleaning rock surfaces can be undertaken using a shotcreting nozzle to spray a water and compressed air mixture. Damage can be caused by seeping water and this has to be first channeled or diverted. Holes can be drilled and pipes attached and connected to drains. Shotcreting is often undertaken immediately following exposure of the new rock surface. To ensure a fast set, accelerating agents are added to the mix; the loss in final strength, which may result in, generally more than compensated by the increased safety and efficiency.

The main reasons for using shotcrete for lining are:

1) To prevent or minimize rock displacement in loosening ground by;

   (a) stiffening and strengthening the rock mass by filling open joints,
   (b) transferring the rock load to adjacent stable rock through adhesion or shear,
   (c) acting as a membrane in bending or tension when shotcrete bond is low and the shotcrete layer is continuous.

2) In oxidizing and slaking ground to seal the rock and prevent raveling and sloughing, which occur because of exposure of the rock to moist air and/or ground water.

3) To control water and ice formation by redirecting, draining or stopping water flow.

The decision as to whether plain or reinforced shotcrete should be used for one or more of the above functions is primarily a geological consideration and depends on the rock type, fracture and jointing, and water flow conditions as well as tunneling techniques.

2.3.5.1. <u>Shotcreting Systems.</u> Shotcrete has two primary application techniques; wet-mix and dry-mix. As both methods comply with common quality demands for the applied shotcrete, economic and practical considerations will be of high importance.

<u>Dry-shotcrete:</u> For the dry-shotcrete method cement, dry or slightly dampened sand and aggregates are thoroughly mixed. The mix is added to the delivery equipment, and compressed air conveys the shotcrete from the gun down the hose at high velocity. The moisture content of the sand and the aggregate should

be three-six per cent. If they are drier than this, dust could be problem. A mix with more than six per cent moisture would increase the chances of concrete prematurely setting and causing blockages. In the dry shotcreting system water is introduced under pressure through a water ring at or near the nozzle. After having passed the water cock, the water enters a water ring and is consequently injected into the passing dry mix through a number of specifically arranged holes. With the dry method of shotcreting it is important that the mix is fully wetted at the nozzle and the water pressure at the nozzle is high enough to penetrate into the mix passing through it.

The water/cement ratio of dry shotcrete lies between 0.40 and 0.45. accurate water control by the nozzleman is necessary to avoid excessive dust in case of an insufficient amount of water whereas too much water could cause the freshly sprayed shotcrete to flow off the surface.

Ease of use and greater productivity, reduced manpower requirements and uniformity of quality are the advantages of the dry-shotcrete.

<u>Wet-shotcrete :</u> In the wet-shotcrete process, all the components, including water, are thoroughly mixed. The mortar or concrete is fed into the hopper of the conveying equipment. With the dense-flow conveying method, the wet ready mixed material is pumped by worms or preferably double pistons to the nozzle from where it is sprayed onto the wall. This mix must be sufficient fluid to be pumped through the machine. The concrete thus contains more water than is necessary to hydrate the cement. The excess water evaporates from the shotcrete, making it more prone to shrinkage cracks. The highest quality concretes are produced when only enough water is used to cause hydration. However such a mix would not be pumpable. Therefore, the use of a water reducing agent, suitable for pumping concrete, is highly recommended. The high density on the wall is not achieved by an extreme spraying speed but by a well graded and thoroughly mixed concrete which is sprayed onto the surface in small drops, without any dust, giving a homogeneous lining. A comparison between dry and wet spraying is displayed in Table2.5.

Table 2.5. Comparison between dry and wet spraying [16].

FACTOR	DRY MIX	WET MIX
EQUIPMENT	* Lower total investment. * Maintenance relatively simple and infrequent.	* Less equipment at the job site. * Less wear rate in pump, hoses and nozzle. * Up to 60 per cent less air consumption.
MIXING	* At the jobsite or at the mixing/batching plant. * Premixed, dry ingredients can be used but cannot be left open in humid or wet environments. * Performance impaired by wet sand.	* Accurate mixing at mixing/batching plant. * Can use bulk ready mix. * Wet sands acceptable.
OUTPUT	* Rarely exceeds 5 m^3/h in place. * Can be conveyed over longer distances than wet mixes.	* Higher than similar dry mix machines, 2-10 m^3/h with hand-held nozzle up to $20 m^3/h$ with manipulator
REBOUND	* Can be 15-40 per cent from vertical walls, 20-50 per cent from overhead. * Forms rebound pockets. * Loss of aggregates makes compliance with mix specification difficult, and excess cement is usually added.	*Low rebound with correct mix, can be less than 10 per cent. * Rebound pockets do not occur. * Little loss of aggregate.
QUALITY	* Higher strength, due to lower W/C ratio. * Less homogeneous quality as the water addition is regulated by the operator and discontinuous material supply.	* More difficult to obtain high strength, due to higher W/C ratio. * More homogeneous quality.
IMPACT VELOCITY	* Higher, better adhesion, easier to use overhead.	* General adequate for tunnel and mining work.
ADDITIVES	* Powders added in mixer or before hopper. * Liquids at the nozzle.	* Generally as liquid
DUST	* High dust production can be reduced by prewetting with 5-15 per cent moisture (semi-wet method) or by moving the water ring back from the nozzle.	* Very little dust formed. * Better visibility. * No danger of lamination by dust
VERSATILITY	* Can be used for; sand-blasting, guniting, refractory materials, resurfacing	* Can be used as concrete pump for pouring in place

Dry method is more widely used. Investment cost for a wet mix shotcreting equipment is at least 3 times the price for a dry mix one. A dry machine should be more capable of spraying onto wet strata whereas the wet process has a limited ability to cope with water in such conditions. Regulating the water requirement in the dry system is vital. Too much or too little water will increase the rebound and adversely affect the strength. The rebound is much higher with the dry system than with the wet. However, for wet shotcrete, even at pressures which could be manually controlled, a robot is essential because the weight of the wet concrete in the hose makes it difficult to lift. At the other hand, cleaning time for wet spraying equipment is longer than that for the dry-shotcreting process. Cleaning should be carried out after every application.

2.3.5.2. Mix Design. Shotcrete mix design is governed by the same principles, which govern concrete mix design. The prime factors controlling strength and quality are the water/cement ratio, air content and degree of consolidation achieved. There are, however, a number of considerations in design in which shotcrete differs from conventional structural concrete. The major differences are in the aggregate gradation and cementitious content of shotcrete mix design.

The theoretical design, whereby sand and coarse aggregate are mixed in proportions which provide the minimum void volume to be filled with water and cement, is generally satisfactory for dry mix machines. When pumping wet mixes, however, additional amounts of finest size sand and cement are required to lubricate the flow and ensure that water cannot migrate through the mix. But, if the proportion of fine sizes is excessive, blockage will occur, especially in long pipes. It is now generally agreed that a suitable base mix contents about 20 per cent of cementitious material, together with 15-20 per cent coarse aggregate, and a sand content that lies normally between 60 en 65 per cent of the total aggregate weight. Natural washed sand will be preferred. To an extent, the proportion will depend on whether rounded gravels or granular stone chips are used as a coarse aggregate.

The combined aggregate should meet one of the gradations shown in Table 2.6.

TABLE 2.6. Gradation limits for combined aggregates [16].

Sieve size, U.S standard square mesh	Percent by weight passing individual sieves		
	Gradation No.1	Gradation No.2	Gradation No.3
19 mm	---	---	100
12 mm	---	100	80-95
10 mm	100	90-100	70-90
4.75 mm	95-100	70-85	50-70
2.40 mm	80-100	50-70	35-55
1.20 mm	50-85	35-55	20-40
600 μm	25-60	20-35	10-30
300 μm	10-30	8-20	5-17
150 μm	2-10	2-10	2-10

Gradation No 1 should be used for fine aggregate shotcrete. Sand for finish or flash coats may be finer than gradation No 1. However, the use of finer sands generally results in higher drying shrinkage. The use of coarser sands generally results in more rebound. As a base rule, the largest aggregate should not be over 16mm. Experience with aggregates over 16mm has shown that the rebound increases drastically with aggregates over 8mm, which would then make the economic value of spraying uncertain. Approximately 60-70 per cent of the aggregates above 8mm are contained in rebound.

For dry mixes the amount of Portland cement content should be kept within the following limits: 450-600 kg/m^3 for fine shotcrete 0-4mm, 350-450 kg/m^3 for shotcrete 0-8mm, 300-350 kg/m^3 for coarse shotcrete 0-15mm.

In wet mixes the water content usually aims at producing a test slump in excess of 50mm. When slumps are above 150 to 175mm, cohesion is lost and coarse aggregates may tend to separate out.

The composition or mix design of the sprayed concrete depends on various factors. For example; the composition of the initial mix, the design of the spray nozzle, the impact speed of the sprayed material, the capability of the operator, the surface to be sprayed, the dosage and the type of the additives, the reinforcement, the distance of the spray nozzle from the surface, the angle of spraying.

2.3.5.3. Additives

Accelerators: Accelerating chemical admixtures have traditionally been used in shotcrete to; cause the shotcrete to gain strength more quickly so that it is more competent to support the tunnel at an early stage, reduce the level of rebound of aggregate from shotcrete, enable thicker layers of shotcrete to be sprayed in one pass so that time is not wasted waiting for previous layers to set, enable shotcrete to be applied to a wet surface subject to slight infiltration, enable wet process shotcreting machines to be used for overhead work. Proprietary accelerators are available in both liquid and powder form. Careful dosing of the accelerators is important as they greatly influence the early and final strength of the shotcrete.

Silica Fume: Condensed silica fume and ferrosilicon dust (essentially the same material but with slightly different chemical compositions) are by-products of silicon metal and ferrosilicon alloy manufacturing. The light grey and extremely fine powder was once discarded as a waste material; research and testing have turned it into a valuable product, sometimes costing more than cement. Silica fume is highly pozzolanic and can be used in shotcrete, partly as cement replacement. When silica fume is added, a superplasticizer has to be added to achieve the required workability.

Water Reducing Agents : Water reducers are used to improve the wet-mix shotcrete workability, cohesiveness in the plastic state and pumpability.

Superplasticizers: They known as a high range water reducers, superplasticizers are chemically distinct from normal plasticizers or water reducers. They are used either to increase the strength or to increase the workability considerably without reducing in strength. Water reducing admixtures are normally no used in dry-mix shotcrete; they cannot be effectively activated in the short wetting time from mixing at the nozzle to impact on the wall. In fact, the addition of dry powdered superplasticizer can have disastrous consequences for dry-mix shotcrete; it can create a delayed fluidity, causing the plastic in place shotcrete to sag and slough. Water reducers and/or superplasticizers are commonly used in high quality wet-mix shotcrete.

Polymer Latex Additives: These additives could be added to the dry-mix shotcrete to impart special desired properties, like adhesion improvement, permeability reduction, resistance to chloride attack, increased durability in freezing and thawing conditions, impact resistance, steel production and strength improvement.

Air Entraining Admixtures : They should be used where the wet-mix shotcrete will be subjected to freeze/ thaw exposure.

2.3.5.4. Shotcreting Technique. Only a clean surface should be sprayed, as the adhesion of Gunite or shotcrete is very greatly impaired if sprayed onto contaminated substrates. The surface is cleaned simply by using the spraying nozzle with compressed air and water, or in severe cases by sand blasting. Care should be taken always to spray onto a well-dampened surface, otherwise there exists the danger that an excessively dry surface will draw too much water out of freshly sprayed concrete or mortar.

Successful concrete spraying not only depends machinery but also on the skill of the operator at the spray nozzle as he largely controls the rebound and keeps it to a minimum. Spray dust and considerable rebound have to be combated when dry spraying; when wet spraying, holding the heavy hose filled with concrete is a great physical exertion. The distance of the nozzle from the work, usually between 0.60m and 1.81m, should be such as to give the best results for work requirements: highest degree of compaction at the lowest rebound. The optimal distance between the nozzle and the surface to be sprayed is influenced by the; aggregate size of the mix, grading curve, required or desired surface finish of the Gunite or shotcrete and air pressure and speed of the conveyed material. As a general rule, the nozzle should be held perpendicular to the receiving surface, but never more than 45 degree from the surface. Otherwise, the shotcrete rolls or folds over, creating an uneven, wavy textured surface that can trap rebound and overspray. This technique is wasteful of material and may create porous and non-uniform shotcrete. To uniformly distribute the shotcrete and minimize the effect of slugging, the nozzle is directed perpendicular to the surface and rotated steadily in a series of small oval or circular patterns.

Shotcrete, like concrete, must be properly cured so that its potential strength and durability are fully developed. This is particularly true for thin sections, textured surfaces and low water-cement ratios associated with shotcrete. The best method for curing is keeping the shotcrete wet. Natural curing may be allowed if the relative humidity is at or above 85 per cent [16].

2.3.6. Concreting

Concrete is possibly the most common material used for the lining of shafts, because concrete, as a material is very suited to the lining of shafts. As concrete is such an important material for shaft linings, it is necessary to be aware of all the factors which contribute to its successful design, installation and performance. Care and attention should always be paid to the selection of the right mix ingredients, the correct proportioning of the same, proper batching and transportation, and protection (non-abuse) of the finished work in both the short and long term hardening states.

The advantages in using concrete for shaft linings are as follows:

1) Relatively small areas of wall can be constructed systematically in phase with the shaft sinking process.
2) Transport to any position in the shaft is simple by means of pipeline or skip.
3) Concrete is convenient material to handle and place within the restricted working space of a mine shaft.
4) When placed, concrete moulds itself to the excavated profile of the shaft providing an interlocking action with the surrounding rock.
5) Testing procedures for quality control are straightforward.
6) Resistance to sulfate attack is achieved easily by using sulfate resisting Portland cement, producing a structure, which requires little maintenance.

7) Unreinforced concrete can withstand most loading conditions normally encountered and it produces a dry shaft.

Concrete can never be considered to give an absolutely waterproof lining, although a very good apparent watertightness can be achieved if the lining is successfully backgrouted. Concrete has a finite permeability but this can be so low that it may only show as damp patches on a shaft wall. The main problem is making the concrete lining watertight is that of leaks through construction joints. This is reduced by the installation of water-bars at each joint during construction and is completed by grouting.

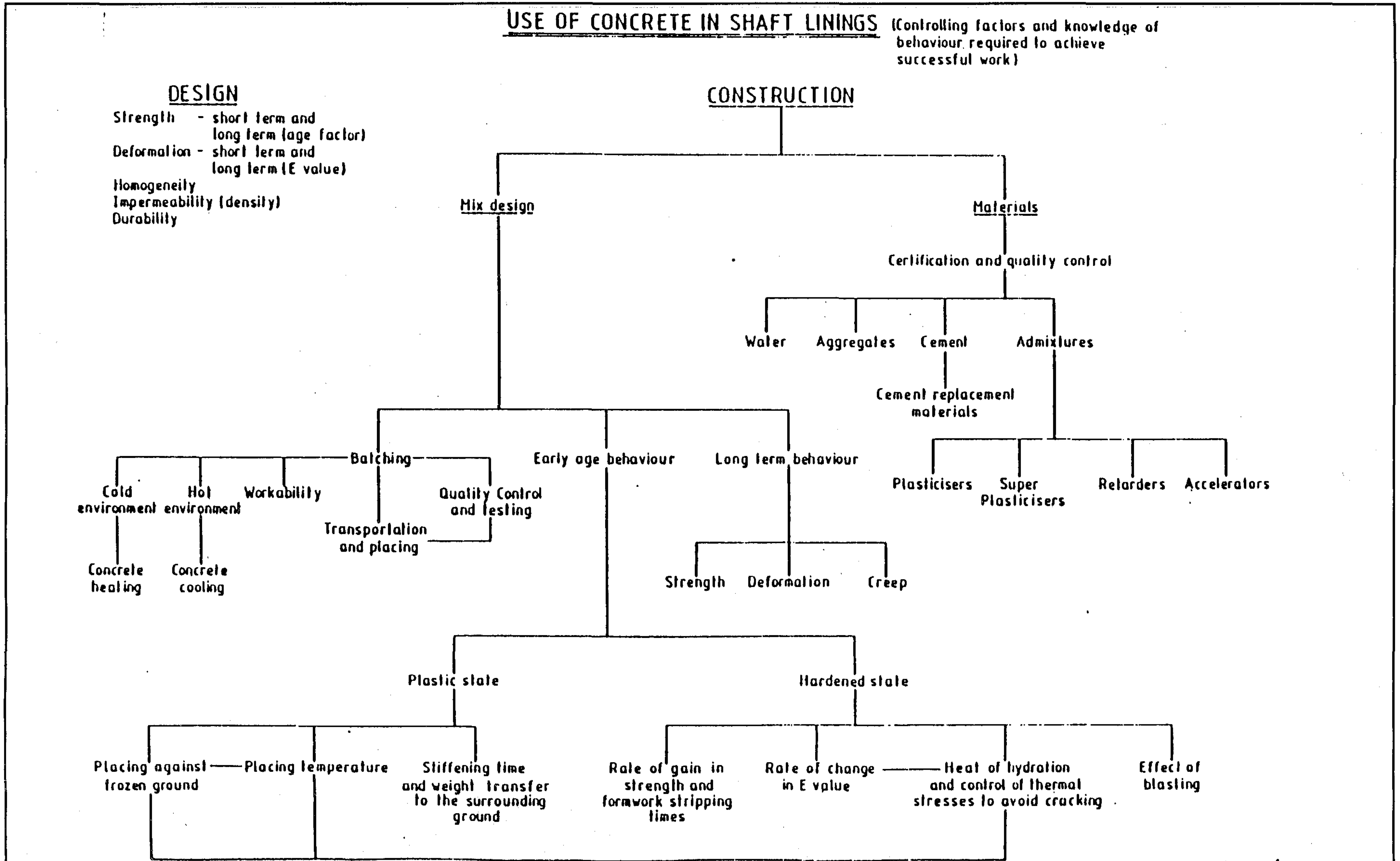
The Table 2.7 sets out the above parameters. Two major subjects are shown; one being Design and the other Construction.

What are shown under the Design heading are the desired properties for a sound lining:

1) Adequate strength;
2) Defined deformation;
3) A homogeneous material;
4) An impermeable material;
5) A durable material.

Allied to these properties is the section on Construction, whereby the parameters for achieving the above defined. The Construction section has been split into two sub-sections, one referring to mix design and the other materials. In the latter sub-section the choice, certification and quality control of the mix constituents is an independent area for study. The sub-section on mix design encompasses two separate subjects, one relating to the batching, transportation and placing of concrete in conjunction with quality control and testing, and the other being its behavior as a material in the plastic and early age hardened state and also long term. From the diagram, it can be seen that the subject of concrete in shaft linings covers a wide field of study [17].

Table 2.7. Use of concrete in shaft linings [17]



Most linings consist of unreinforced concrete, poured in thickness of from 150 mm to 1.5 m. In the days of hand mucking and early mechanized mucking a single ring of form, about 750 mm high would be built in the sump, centered, leveled and braced off the excavation. Longer walls were made up in multiples of 750 mm pours. Today, pours of about 6m are normal using a collapsible steel jump form lowered by winches or chain blocks. The weight of the form (about 13000 kg for a 6m diameter shaft) plus the fluid head of concrete must be supported until the concrete is sufficiently cured. Accelerators are frequently used to speed the process.

The temporary support for lining and form could be:

(a) a system of chains spaced evenly around the kerb and anchored to the wall two pours previous.

(b) hanging rods within the lining, joint by couplings every six meters pour to support the kerb from the underside.

Concrete is transported down the shaft either by:

(a) one to two m^3 bottom opening skips; distributed around the form using a long flexible hose, or

(b) slickline down the shaft wall; transferred by pipe to a collection bin at the center of the Stage and then distributed to the form via a multi-pipe system built into the Stage.

Linings other than concrete may be used where no great structural strength is required; perhaps merely to prevent air slackening or to create smooth internal surface for ventilation purposes. Shotcrete is good for this situation and GRC panels have also been used successfully.

<u>Calculation of the thickness of the lining :</u>

The thickness of the lining is very important. There are many factors, which affect the soil pressure acting to the lining, so it is difficult to form a full analytical modeling of the problem. In general the thickness of the lining is calculated from the results of the experiments.

In literature there are some different formulas about the calculation of the thickness of the lining. Some of the most widely used formulas are given in this study.

In their study Birön, C. and Arıoğlu, E. [18] reported that Prof. Protodjakonov calculated the thickness from the following formula:

$$t = ((P_r * r) / (\sigma - p) + (150 / \sigma)) \qquad (2.8)$$

Here; t : Thickness of the lining (cm)

d_i : Shaft inner diameter (cm)

P_r : Uniform radial pressure that acting to the lining (kg/cm^2)

σ : Tensile strength of the lining material. (This value is 1/10 of the compressive strength.) (kg/cm^2)

He gives an other formula for the formations whose hardness coefficients are between 3 to 8:

f = 3 - 8 ; Compressive strength 300 - 800 kg / cm^2

$$t = 0.007 \, SQR (d * H) + 14 \text{ (cm)} \qquad (2.9)$$

Here; d_i : Shaft inner diameter (cm)

H : Depth of the shaft (cm)

According to the two dimensional elasticity theory, tensile stress distribution around the thick cylindiric tube which is in an elastic state is;

$$\sigma_r = (d_o^2 / (d_o^2 - d_i^2)) (1 - (d_i^2 / x^2) \; P_I) \qquad (2.10)$$

$$\sigma_t = (d_o^2 / (d_o^2 - d_i^2)) (1 + (d_i^2 / x^2) \; P_I) \qquad (2.11)$$

Here; σ_r, σ_t : Radial and tangent stresses (kg/cm^2)

r_1 , r_2 : Inner and outer diameter (cm)

x : Distance measured from the shaft center (cm)

P_l : Uniform lateral pressure acting to the lining (kg/cm^2)

Sign of the σ_r, σ_t is always positive, which means that lining is always under the compressive stress.

If $x = d_i$, then the stresses on the inner face of the lining are;

$$\sigma_{r,\ min} = 0 \qquad (2.12)$$

$$\sigma_{t\ max} = (2 * P_l * d_o^2) / (d_o^2 - d_i^2) \qquad (2.13)$$

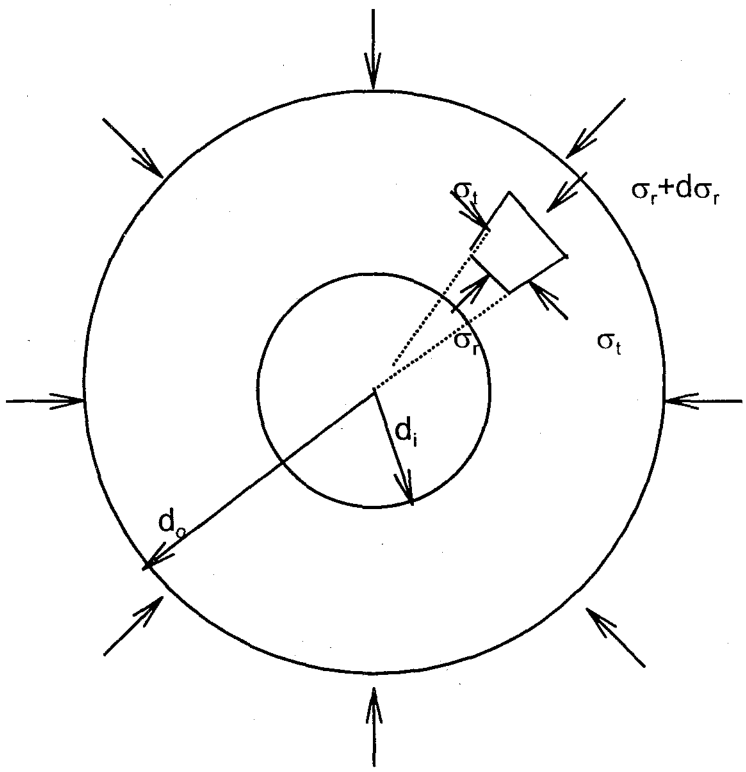


FIGURE 2.22. Two-dimensional stress analysis on a cylindiric tube. [18]

If $x = d_o$, then the stresses on the outer face of the lining are ;

$$\sigma_{r,\,max} = P_l \tag{2.14}$$

$$\sigma_{t,\,min} = ((d_i^2 + d_o^{\,2}) / (d_o^{\,2} - d_i^2)) * P_l \tag{2.15}$$

As observed from the above formulas, radial stress is less than the tangent stress. So, while calculating the thickness of the lining, the critical stress is the tangent stress in the inner face of the lining.

If we take $t = d_o - d_i$ and F as a safety coefficient the above formula ;

$$\sigma_t = (d_o^{\,2} / (d_o^{\,2} - d_i^2)) * (1 + (d_i^2 / x^2) * P_l \tag{2.16}$$

becomes like ;

$$\sigma_{t,\,max} / F = (2 * P_l * (t + d_i)^2) / (t * (t + 2\, d_i)) \tag{2.17}$$

From the last formula we can obtain "t ", the thickness of the lining, as;

$$t = (SQR ((\sigma_{t,\,max} / F) / (\sigma_{t,\,max} / F - 2P_l)) - 1)\, d_i \tag{2.18}$$

$\sigma_{t,\,max}$ is going to be taken as σ_b, 28th day strength of the concrete. Safety coefficient F is 2 - 3.

Formulas about the stress distribution are for ideal elastic states. In an other words to say, above formulas do not give the economic thickness in the shafts deeper than 300 - 400m, because at those depths formations act as in plastic state. In practical it is advised to use Heber's formula for the shafts that are going to sunk in really poor formations. [18]

$$t = (SQR ((\sigma_{t,\,max} / F) / (\sigma_{t,\,max} / F - SQR (3P_l))) - 1)\, r_1 \tag{2.19}$$

2.3.7. Sinking System

Conventional sinking method is the most dangerous sinking method. It has too many risks because of the methods complexity besides the actual risks of the underground working. Because it is labor intensive method and there are crews working both in the shaft at different stage levels and out of the shaft at the same time. So, for example there are risks of persons falling down the shaft and materials falling down the shaft. These hazards may cause death and serious physical injuries.

Besides these there is a conveyance, generally kibble, traveling in the shaft all the time. Sometimes it carries people, sometimes it carries materials or muck, and sometimes some equipment may be transported by slinging under the kibble. These conveyances are operated by hoist operator from the hoist room. As he can not see the things happening in the shaft or out of the shaft, he is given instructions by signals. There is onsetter in the shaft during the sinking operation and a banksman on the bank to give signals and only authorized personal can give signals to the hoist operator. All these operations have some important risks.

Although every sinking operation has its own conditions and its own system there are some common rules or procedures which are valid for all of the sinking operations. These procedures are determined in "Shafts and Winding Regulations" and "Approved Codes". The Approved Code is not intended to provide comprehensive guidance to safety in shafts. It aims to deal with the most important safety areas in shafts and winding but it cannot specify every possible detail or circumstances. Managers should remain alert to developments and hazards that may not be fully dealt with in the Code or other guidance material.

As every job has its own special conditions, every possible details or circumstances are covered in code of practices which is arranged specifically for that job. It contains details such as; general operations, code of signals, man riding procedures, mucking operation procedures, procedures about moving the stage, hoist procedures associated with blasting, hoist procedures associated with concreting, sinking operation procedures, procedures for issue, transport and use of explosives, safety procedures and etc. In this section some of the common general procedures are explained. In the case study section, special conditions and procedures for that job are explained.

In sinking operations two different hoist systems are used.

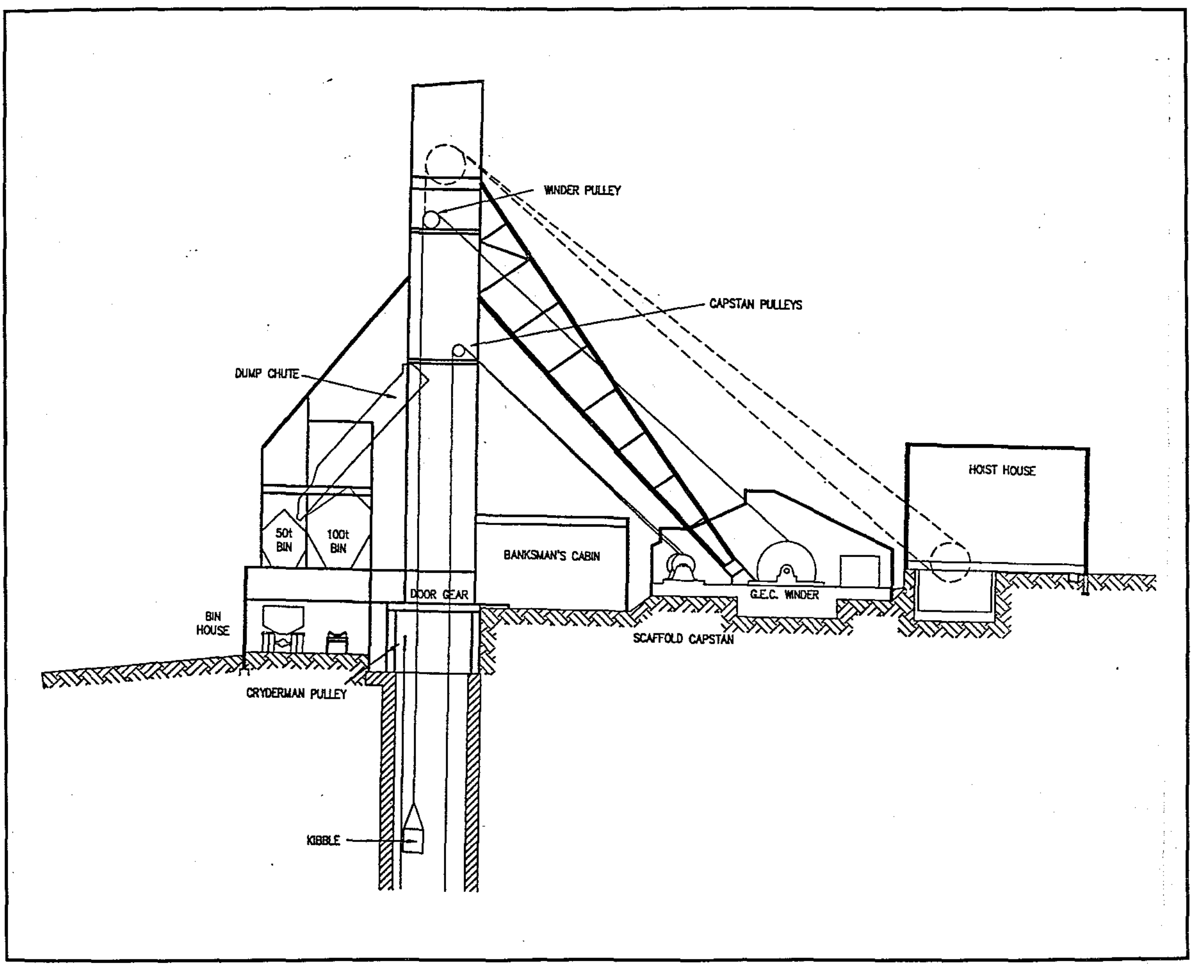


FIGURE 2.23. Two different hoist systems used in sinking operations.

First one is used for moving skips up and down. It is called "hoist" and operated from the hoist house. So that, man riding in the shaft, material traveling and muck traveling and tipping are all done by using this hoist system. In this hoist system there are two conveyances, so two different drums. They may work at the same time but in opposite directions. One goes down while the other goes up. But sometimes, for example during drilling and concreting, only one conveyance need to move. Beside this as shaft gets deeper, to adjust the winding ropes' length again only one conveyance needs to move. For these reasons it is possible to clutch out one of the drums and let the other conveyance move alone.

Second one is used for moving the stage up and down and is called "winder" and operated from the capstan house. Before blasting stage must be raised up to a secure level and after the blast lowered again for mucking. And during concreting, stage must be positioned so that formwork can be installed easily and concrete can be poured. It is connected to the stage by two different ropes so, it has also two different drums. In this hoist system both ropes can be lowered or raised at the same time but in some cases to adjust the level of the stage, they need to be operated individually. Here it is also possible to clutch out one of the drums and let the other drum move alone. Both hoist and the winder are operated by the same hoist operator and communicate by signals.

Winding is the main operation in sinking process. Because of the nature of the operations some deviation from the normal winding practice may be un-avoidable and precautions should be taken to ensure that danger does not arise as a result. First of all, all the chains and slinging material must be strong enough to carry the loads. Especially the hoist system, its ropes and the brake systems must be under control all the time. Stage ropes should not be used for a period exceeding 3.5 years unless the manager has satisfied himself that it is safe to do so. The static factor of safety of the scaffold rope should not be less than five. Stage ropes should be recapped at least every 12 months. The main load carrying members of the stage structure should have a factor of safety in tension of at least seven based on the ultimate tensile of the steel.

The movement of the conveyance should be controlled so as to minimize the danger of collision with the sinking stage. The automatic contrivance should be regularly adjusted to ensure that the speed of the conveyance cannot exceed two meters per second when it is at the same level as the stage or lower.

Interlocking should be provided so that;

(a) when winding is taking place, tipping chutes are proved clear of the path of the conveyance,

(b) during winding, the shaft top doors are proved open whenever a conveyance is in a zone extending from a safe stopping distance below the doors until it is above the doors, and

(c) before discharging conveyances into the tipping chutes, all shaft top doors are proved closed.

The banksman should not allow persons to enter or leave a conveyance or to load materials into or out of a conveyance unless the shaft top doors have been closed and the conveyance lowered onto them. The interlock system should allow the conveyance to be raised off the doors and the doors opened before the conveyance is wound down the shaft. When anything is being lowered through a shaft sinking, the hoist operator should stop the conveyance above a safe distance from the stage and should not lower it further until he has received a signal telling him that it is safe to continue winding. If anything is lowered to the sump, again the hoist operator should stop the conveyance above a safe distance from the sump and should wait for the signal. Similarly when raising anything through he shaft he should stop it about one meter above the point from which it is being raised and should not raise it further until he has received a signal to continue winding.

During the initial stages of sinking, before the sinking stage is in place, cranes may be used to transport men or materials through the shaft. The use of cranes should not continue longer than is necessary to allow the normal winding apparatus to be installed, and cranes should not be used to wind from depths exceeding 50 meters. All winding apparatus and equipment ancillary thereto shall be installed and used so as to prevent danger, so far as is reasonably practicable, and shall be suitable for the use to which it is put.

All winding apparatus shall be provided with the effective;

(a) brakes and brake locks,
(b) interlocking devices,
(c) means of controlling power to the winding apparatus,
(d) means of preventing an overwind,
(e) means of preventing a conveyance of counterweight traveling at an excessive speed,
(f) means of safely stopping and holding a conveyance or counterweight in the event of an overwind, and
(g) means of monitoring the movement of every conveyance in the shaft.

In designing winding apparatus for use in shafts, the following principles should be applied:

(a) The safe operation of winding engines should not so far as is reasonably practicable depend on single line components but where their use is unavoidable they should be designed so as to avoid danger, for example by failing to safety; and

(b) Facilities should be incorporated into the winding apparatus to enable routine maintenance tests to be carried out effectively.

All winding engines should be firmly anchored to secure and level foundations and suitably housed. Except at a shaft sinking, winding engines should be housed separately from other machinery, preferably each in its own engine house. Each winding engine should be fitted with mechanical brakes which act directly on the drum or sheave of the winding apparatus and which are capable of bringing the apparatus safely to rest at all times. Where the power supply is used for braking purposes, the mechanical brake should be automatically applied when the power supply is cut off. Where the mechanical brakes are applied wholly or partially by fluid, steam or air pressure, they should be backed up by a means of automatic application independent of such pressure. Brake systems should be fitted with pressure gauges to enable the braking force to be checked. The performance of winding engine mechanical brakes should be adequate for both service and emergency duties.

In drum winding systems to minimize the risk of injury to persons when the brake is applied following an emergency trip, the retardation of the conveyance should not exceed 1g. To achieve this the maximum retardation at the drum or sheave should not exceed five m/sec^2. The mechanical braking system should be able to bring the winding apparatus safely to rest and apply at least 50 per cent of the normal braking force in the event of a failure or malfunction of any one component of the system. The brakes should be automatically applied following any failure of operating fluid pressure in the system.

All electrically driven winding apparatus should be provided with a switch, placed within easy reach of the person operating the winding engine, which will cut off the supply of electricity to the engine other than for the purpose of braking it, and automatically apply the mechanical brakes.

Every main suspension member of a conveyance should have a breaking strength of not less than 7 times the maximum static load which is normally carried by that member. In determining the suitability of any ropes, account should be taken of the maximum load to be carried and the ratio of the diameter of the drum or pulley to that of the rope to ensure that operational stresses are kept within safe limits. Suspension ropes in lift installations should be rust resisting or galvanized. The winding ropes on drum winding engines should be properly anchored and, to minimize the loading on the anchorage, at least two laps of rope should remain on the drum when the conveyance is at the deepest part of the shaft to which it may travel. Spare rope stored in or on the drum should not be considered as reducing the load on the anchorage.

Every shaft equipped with winding apparatus shall be provided with a means of giving both audible and visual signals and a means of speech communication. All such means are specified above shall be so placed as to ensure the safety of persons in the shaft. Except in the case of lift installations, effective means of signaling should be provided from all entrances to the shaft to the winding engine room. Provisions should also be made for shaftsmen to transmit signals effectively between the conveyance and the surface from all positions in the shaft and, where necessary, to allow persons involved in headgear and shaft sump maintenance to transmit signals to the winding engine man; where other systems are inadequate, radio inductive loop or equivalent systems should be used. Signaling apparatus must give both audible and visual signals which should be displayed simultaneously at the shaft surface entrance, the winding engine room and those underground entrance normally used for man riding in such a position that they can be easily seen by the banksman, winding engineman and traveling onsetter respectively. Signals in the winding engine room should be automatically canceled on completion of the winding sequence by means of devices driven from the winding engine which are capable of distinguishing between stationary and creep speed conditions.

The code of signals to be adopted at the mine should be specified. The manager should ensure that all persons who are requires to use the code of signals are fully acquainted with it. A copy of the code should be posted at every entrance to a shaft and in every winding house.

All winding apparatus at a shaft, and all equipment ancillary to the winding apparatus shall be regularly and systematically inspected, tested and maintained in accordance with rules made by the manager for the purpose. The rules made by the manager should amongst other matters, clearly set out the nature and periodicity of the inspections, examinations, tests or maintenance to be carried out on each piece of apparatus and the safety procedures to be adopted in carrying out such work. The manager should draw up schedules detailing the inspections, examinations, tests or maintenance specific to individual items and ensure that they are kept up to date. Any person carrying out tests, maintenance, inspections or examinations should, at or before the end of his shift, make a written report of the work carried out, any defects found and remedial action taken. Where a defect is found likely to affect safe working immediate remedial action must be taken.

Before commencing any shaft sinking, the manager should arrange the working methods and safety precautions so as to minimize the risks to the health and safety of those employed. Every shaft through which persons travel in the ordinary course of their work shall be deigned, constructed and used and every fixture in a shaft shall be installed and used so as to prevent danger of injury to persons so far as is reasonably practicable. To prevent accidents due to falls, stage used in shaft sinking should be provided with safety rails and attachment points for safety harnesses. Stage should not be moved except under the direction of a competent person appointed to supervise the sinking. When stationary stage should be wedged or secured to the shaft wall as necessary to prevent danger.

There is particular danger to persons from falling objects in shaft sinking because the workings are unfinished and persons are working at different levels. No person should go into or across the uncovered space at the bottom of the shaft except for the purpose of working there; the manager should ensure that, except during shaft sinking, no person works in any uncovered space at the bottom of a shaft when the winding apparatus is in motion or when anything is suspended from the winding apparatus except for the purpose of maintenance operations or the handling of bulky equipment or material. Other measures taken by the manager should include:

(a) ensuring that banksmen and onsetters or other persons appointed by the manager, keep landings and insets clear of loose material;

(b) ensuring that any working platforms etc. are provided with toe boards to prevent tools etc. from falling off them;

(c) providing hand tools with wrist straps; and

(d) providing canopies and covered walkways at landings or insets where persons may be at risk; and

(e) doors are provided at the top of the shaft or deepening, except during the initial period of the sinking, which are capable of closing off the shaft, the doors should be kept clear and not opened except when free of debris.

Shaft-side barriers and gates should be designed and constructed so as to prevent persons and things from falling into the shaft or coming into contact with working parts of the winding apparatus. Except when loading and unloading, shaft gates should be kept closed. If any part of a shaft side barrier is removable, the banksman or onsetter should ensure that it is kept in place at all times except as necessary for the use of the shaft or for inspection or maintenance. Persons working in shafts must wear safety-harness at all times where it is necessary to prevent danger and ensure that the harness is secured to a suitable anchorage point and should only detach it to move a new working position. All conveyances and working platforms must be provided with sufficient anchorage points to enable persons to reach any part of the shaft on which it is necessary to work whilst making use of their safety harness [19].

3. CASE STUDY

3.1 Project Information

In this case study, the construction method of the new vertical ore handing system for the Çayeli Mine near Rize is examined. Shaft sinking and preliminary development was started in June 1996 and continue well into 1997.

At Çayeli Mine near Rize, CBI are to sink a new shaft from surface to a depth of about 500m. The vertical shaft has an expected life of 25 years and will be sunk in two phases. Initial production will be from about 269m and the shaft bottom will be about 320m. Development work from the shaft at the 269m level will provide the excavation for the installation of the mine underground ore handling facilities. At a later date the shaft will be deepened to full depth of about 500m. [20]

All design capacities and details are given in Appendix 1.

The shaft is designed as a 5.5m inside diameter concrete lined structure. and sinking is going to be realized by using "Conventional Sinking Method".

3.2 Shaft Design

The work which led to the present mine was started in 1967 by the Maden Tetkik ve Arama Enstitüsü (M.T.A) who carried out a geophysical survey and drilling programme and drove an adit into massive sulphite ore. Initially, five diamond drill holes were drilled to check the validity of data from the 22 M.T.A drill holes which intersected ore. Based on all available information a decline and cross-cut totalling 643.68 metres were driven to collect a bulk sample for metallugical testing.

Detailed drilling was then undertaken, both from surface and underground, on 12 sections spaced 40 metres apart to define the shape and composition of a continuous mineable orebody. From this work it became apparent that the initial bulk sample was not truly representative and a further cross-cut was developed to produce a more representative sample for metallurgical testing. Figure 3.1 shows the shape of the ore body and the location of the shaft [21].

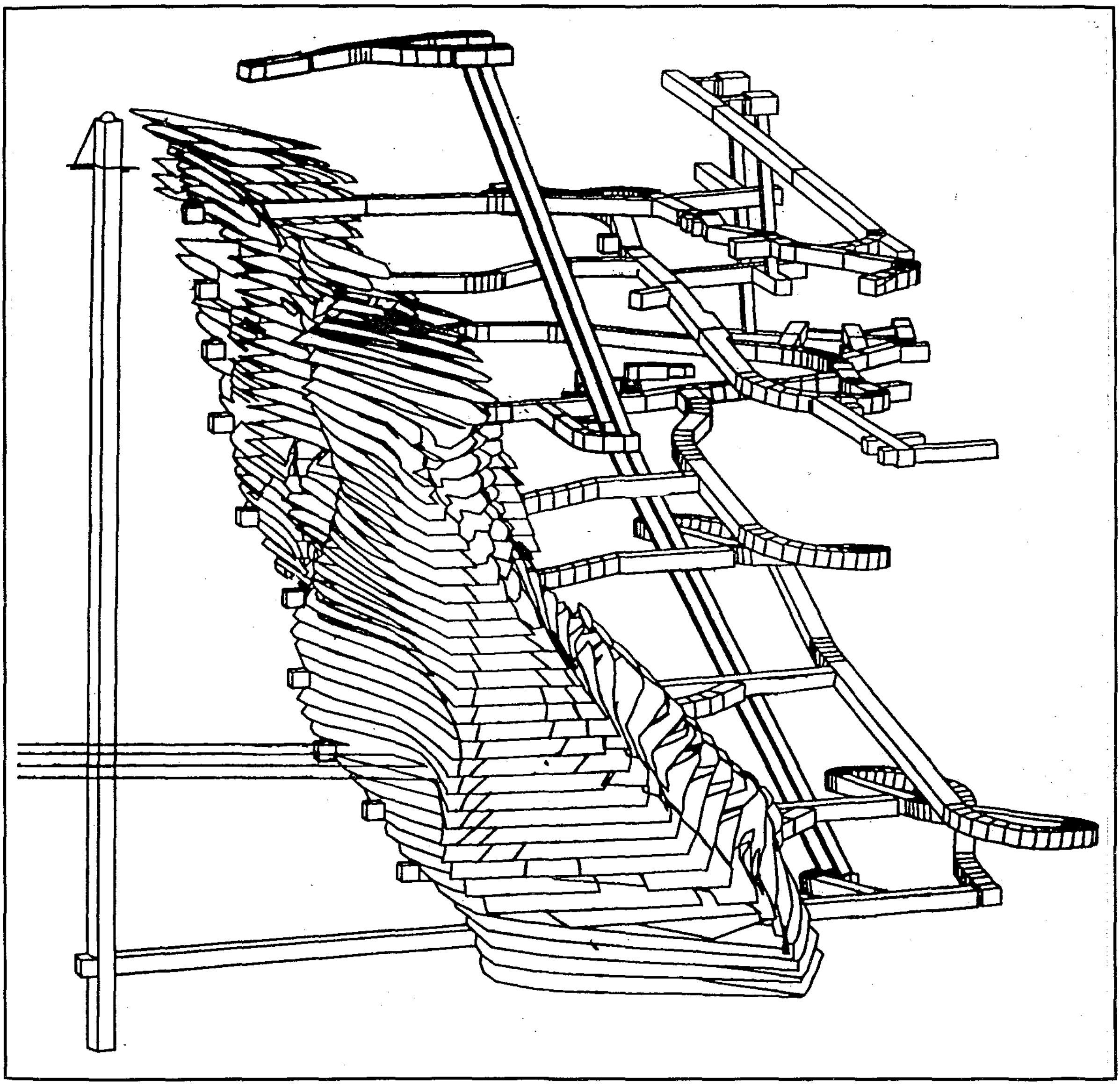


FIGURE 3.1. Shape of the ore body [21].

Before the construction, pre drilled cores were inspected and discussions held regarding the basic shaft design philosophy.

3.2.1 Basic Considerations

The rock conditions are generally variable with a random fracture pattern. Some fracture planes are clay infilled and water tight, others are open and contain ground water. There are a number of igneous dykes of varying thicknesses. These seem to act as a water barriers with potential aquifers above and below their location.

The ground water should be prevented from migrating behind the shaft lining and coming into contact with zones of particular high clay content. Also ground water should be prevented as far as practical from leaking into the shaft and making shaft and equipment maintenance more time consuming and costly.

3.2.2 Basic Recommendations

The freshly excavated shaft walls should be shotcreted in order to seal the rock from contact with water. This also has the benefit of temporarily securing the rock sides until the permanent concrete lining is placed. The shotcrete material cost is largely offset by the concrete it replaces. A much stronger bond is achieved between the irregular rock face and the shotcrete thus making ground water migration at its interface more difficult.

Two test holes should be kept a minimum of 6m ahead of the shaft bottom at all times. These test holes should be drilled some 30m ahead of the shaft and any groundwater encountered assessed and dealt with either by pressure grouting or allowed to drain depending upon its volume and pressure.

Arrangements should be made to backwall grout with cement, the entire shaft length. Areas of relatively large groundwater flows should be drained into a shaft collection system and piped to a pumping station. This will have the dual effect of draining and depressurising the aquifer, thus significantly reducing the pressure gradient required for water migration behind the shaft lining.

To achieve the above recommendations a shaft lining thicknesses of 400mm in the upper shaft and 600mm in the lowe shaft are proposed. A concrete design strength of 30 N/mm^2 should be used with a 25 N/mm^2 minimum strength being acceptable on the limited occasioning. This is a relatively high strength concrete which in addition to its strength gives reduced permeability and improved corrsion resistance. Additives should be considered to ensure that these properties are fully obtained.

The shaft sets are five meters centers, therefore it is logical and practical that the shaft walling lengths are also concreted in five meters increments. It is proposed that provisions be made in the construction joints in the shaft lining for backwall injection and draining of specific aquifers.

3.2.3 Technical Proposals

3.2.3.1 Geotechnical Assessment. The shaft centre borehole log emphasises the varied nature of the strata that will be encountered, as well as the rapid changes that occur between the strong and week formations, as we understand have been experienced in the existing mine. Although no test data is available the geological descriptions indicate that zones of strong, sometimes massive and extremely competent rock are frequently interspersed with lenses and bands of decomposed weathered strata, presumably due to hydrothermal action. Most of these lenses contain a weak clay matrix around gravel sized breccia. The additional presence of clay in the joint systems will inevitably lubricate the fracture of planes tending to mobilise the surrounding blocks of rock. These type of ground will require a careful and flexible excavation mehod, with immediate and appropriate sidewall support, and the facility to easily bring the permanent shaft lining down to sump level. The borehole log gives no hydrological information but if as expected there is some water ingress into the shaft, this will undoubtedly accelerate the deterioration of the excavated sidewall in the decomposed and weathered areas. In these circumstances, the need for immediate support, probably incorporating shotcrete, will be essential and lining design should incorporate the facility to pipe this water away from the sidewall interface as soon as possible.

3.2.3.2 General Comments on Sinking Methods. The various elements of the sinking operation are discussed below, and the preferred methods of working, materials, plant and equipment identified:

1) Drilling

The geological information predicts that a wide variety of rock strata will be encountered, rapidly changing from one type to another. These rapid changes indicate that little benefit will be gained from the use of a drill jumbo to drill rounds, as in the weaker broken rocks, it will be difficult to keep these holes open. Hence, it is proposed that all sump drilling will be undertaken using-held SIG machines.

2) Explosives

It is proposed to use Nonel detonators, initiated via four-bunch connectors by a single electric detonator. This electric detonator would be set off remotely from surface by a Beethoven exploder using a standart twin flex shaft cable clipped to one of the pipe ranges. It is preferred to have 25 and 32mm diameter gelatinous explosive cartridges available so that the round design can be modified for the prevailing ground being encountered; the smaller diameter cardridges would be mainly used in perimeter holes to control overbreak. It would be helpful if the explosives could be supplied in plastic connectable sheathes, as this can facilitate charging in difficult/broken ground conditions.

3) Mucking Unit

The ground conditions at Çayeli indicate that the mucking unit must have the flexibility to cope with well broken hard rock, as well as the highly decomposed and weathered, clayey strata that will inevitably be encountered. It is considered a wall mounted Cryderman mucker will provide more adaptable than either an Eimco loader in the sump, or a stage mounted grab unit, especially in the very weak rocks where its digging capability will be most advantageous. The use of Cryderman will result in a lighter simpler stage than if grab were installed, but more importantly, allows the stage to be dropped right down to the sump floor at short notice to concrete all the exposed sidewalls. If a stage mounted mucking unit were installed, the stage would be restricted to getting with three-four meters of the sump floor making formwork handling more difficult.

4) Sidewall Support

The random nature of many of the weaker zones, emphasised in the geotechnical assessment will result, in practice in most of the shaft depth needing to be meshed. Primarily this will be for safety reasons to prevent small pieces of rock falling into the shaft sump, especially in the weathered zones. However, it is not believed it will always be necessary to install two meters long rockbolts, considering that in the more competent strata 0.9m long rockbolts will be adequate to secure the mesh. The mesh would be best supplied in rolls, 1.8m wide and 10m long so that each roll will cover half the perimeter of the sump excavation.

5) Concrete Lining

It is proposed to construct the shaft lining in five metrs pours, so that the construction intervals match with the furnishing elevations. Longer pours (ie 10m), such as have been used in Europe, were considered and would indeed be with the provisions of an additional five meters steel formwork. Nevertheless these were rejected on the grounds that the permanent lining will for the most part need to be kept close to the sump floor to secure sidewalls in the weaker ground, and that under those conditions the additional formwork would present more of a hindrance than a help.

Concrete can be supplied at a lower slump, say 125mm. This placing method will reduce the possibilty of segregation. Quality can be best controlled by least interference with the supplied product.

Construction joints in the lining will be formed using a steel kerb ring suspended from 8 hanging rods, each capable of sustaining a S.W.L of eight-tonnes; the kerb ring will actually be designed to acommodate 16 hanging rods should either excessive overbreak be sustained, or the nominal concrete width be increased due to local ground conditions, i.e. squeezing ground. The falsework will comprise five rings of steel tubbing, with a filler ring to top off each pour.

3.3 Shaft Construction

The concrete lining is minimum 400mm to the first level and below that level will increase to 600mm wall thickness. The shaft steelwork will be galvanized for steel protection against corrosion for the main members and all conveyance guides will be timber, the timber being mine guide specification prepared and shipped from Canada.

The headframe is constructed to accept a shaft sinking arrangement with a temporary sinking sheave deck below the main sheave deck. The bucket dump to be used during sinking is arranged to discharge into the 200 tonne ore bin.

The shaft sinking system at Çayeli utilizes a stage mounted single boom electro hydraulic drill jumbo suspended from the top deck by JDN Hoist and a stage mounted "Herman" Cryderman Unit, suspended on a single winch rope from surface. The stage has four decks equally spaced at five meters apart, to suit the five meters concrete pours, and is suspended on guide ropes which are used to guide thesinking conveyances during shaft sinking. The kibbles have a capacity of $2.67m^3$ and are tipped using a Canadian system comprising a locator pin on the bottom of the Kibble that sits into a Keyway within the tipping chute, that rotates as the Kibble is lowered.

The shaft will be sunk to the first level and level development excavation will be completed prior to installing the shaft steelwork and timber guides in the shaft. After shaft equipping the conveyances will be installed and construction work will be completed on the development levels. The connection to the mine will be completed and the system will be ready for ore hoisting.

3.3.1 Training and Safety Management

Conventional shaft sinking method is the most dangerous sinking method. Besides this, all the local labour to be employed on this contract have little or no experience in shaft operations. Training these local employees is clearly one of the key elements towards a safe and successful shaft sinking.

In all cases, it is intended the training to comprise a balanced mixture of classroom sessions on methods of work and safety procedures, as well as practical sessions wherever possible to familiarise individuals with the plant, equipment and materials they will be handling. Everybody employed on the contract received an all embracing safety induction along with a booklet printed both in Turkish and English identifying general safe working practices. Safety training was initiated and given by the expatriate site management team and shift supervisors, using the Turkish Engineers and others as needed as interpreters.

The key duties to be identified during the training sessions are: Hoist operators, Banksmen, Onsetters and Sinking Crew. Experienced drivers from UK are used as a hoist operators. Three people were trained as banksmen to operate the shaft signals, communicate with the hoist driver and underground crews, sling equipment in and out of the shaft, and operate the kibble tipping system within the headgear. Only authorised banksmen is permitted to perform any of the above tasks as this role is vital to effective shaft safety.

Whilst all the underground crew were taught the shaft signals, only one person per shift is nominated as the onsetter and permitted/responsible for operation of these signals and communications with the banksman.

All the underground employees were taught the fundementals of shaft sinking, including practical sessions on sump drilling and good concreting practice. One member of each crew was trained on the Cryderman Mucking Unit as well as the Eimco loaders used for initial station development. The Cryderman unit was erected on a wall temporarily 4 weeks prior to sinking commencing, so that trainee drivers had ample oppurtunity to practice loading kibbles, and generally became confident in its use prior to installation underground.

Besides training about the technical matters, training about safety is also very important. All the shift supervisors was trained as first aiders. Once the Turkish chargehands had been selected they too were trained as first aiders. For an effective safety it is needed to form a safety management system and follow the results. In the scope of safety management following topics were evaluated.

1) Safety Induction : Using a pre-determined check sheet, each prospective employee received information on the company, on the work to be carried out (general outline), hazards of the site, risk assessments, use of personal protection equipment, emergency procedures, noise, and accident reporting. (Appendix 2)

2) Training Authorisations : Each man was authorised for specific work operations depending on his previous experience and qualifications to do that work, as well as on-site training.

3) Safety Meetings : Held with the site staff and the men to an agreed agenda, usually on a monthly basis.

4) Risk Assessments : All operations had written risk assessments (Appendix 2) and method statements (Appendix 3) which were explained to the operatives at the beginning of the shift by the supervisors.

5) Tool box Audits : Held on regular basis usually weekly with all the men taking part, the talk given by the foreman.

6) Safety Audit Training : All staff of a supervisory nature will receive audit training which is an on-site risk assessment that may target high risk operations or those of an unusual nature to agree a safe system of work. (Appendix 2)

3.3.2 General Operating Procedures

In this project there are three shifts each working eight hours per day. So every day, except sunday, sinking proceeds 24 hours. Day's shift is from 7:30 to 15:30; After's shift is from 15:30 to 23:30 and Night's shift is from 23:30 to 07:30. Every week shifts are changed and Day's shift becomes Night's, After's shift becomes Day's and Night's shift becomes After's. Sinking crew of this project is given in Table 3.1. Shaft sinking by conventional method is a dangerous job and all the hoisting process is controlled by the computer. So it has to be checked and controlled according to the related procedures. Besides these all the equipments used and critical sinking devices and ropes are to be checked periodically. Because of these, sundays are the general maintenance and testing day. According to the jobs and tests to be done an on duty crew works on sundays.

TABLE 3.1. Sinking crew.

Job Description	Number of Employees			
	Day Shift	Afternoon Shift	Night Shift	Total
	Supervision and Engineering			
Project Manager	1	0	0	1
Mechanical Superintendent	1	0	0	1
Engineer	1	1	1	3
Shaft General Foreman	1	0	0	1
Total Indirect	**4**	**1**	**1**	**6**
	Maintenance and Indirect			
Electrical Foreman	1	0	0	1
Fitter	1	2	2	5
Welder	2	0	0	2
Electrician	1	1	1	3
Surface Labour	1	1	1	3
Total Staff	**6**	**4**	**4**	**14**
	Direct Operation			
Hoist Operator	1	1	1	3
Shift Supervisor	1	1	1	3
Banksman	1	1	1	3
Onsetter	1	1	1	3
Cryderman Operator	1	1	1	3
Miner	3	3	3	9
Total Direct	**8**	**8**	**8**	**24**
TOTAL EMPLOYEES	**18**	**13**	**13**	**44**

All the hoisting operations are controlled by computer and operated by the hoist operator from the hoist house. The mimic panel on the drivers left hand side shows visually; Doors closed or open, Chute extended or retracted, Zones the conveyance are in, Winder in trip condition. Besides this from an other panel in front of the driver, the position of the stage in the shaft, the sump level, position of the travelling conveyance top and bottom levels of the stage, tipping level are all controlled. Doors and chutes are operated by air and also controlled by computer. The electricity in the shaft area is 110 V. Controlling is provided by a zoning system, operated by electrical contact switches and positional encoders plus U.H.M. logic system control, which, in turn, operate solenoid valves controlling supply of air.

Shaft is divided into four zones. Zone A is the unrestricted part of the shaft where the maximum winder speed can be attained. Zone B is the part of the shaft immediately below the door level. At the commencement of shaft sinking the zone extended 50cm below the doors which increased as the sinking progresses to a distance of 150cm when the winder attaind its full speed. This zone is within the decelaration cycle of the winder, and also provides protection from and ascending conveyance and its associated gear being driven into a closed door situation by tripping the winder. No recovery action is required by the winder, as the winder can be re-set, when the doors are proved open by the switches on the door rams. The open door situation will also be visible on the mimic panel. Zone C extends from the door level upwards to tip level. When the conveyance enters zone B with the doors open, the air supply to the doors is isolated by the solenoid valve, and is not available until the conveyance is above the door open level, ensuring the doors cannot be closed on an ascending or descending conveyance or its associated suspension. Zone D extends upwards from the tip level. The air has been made available to the doors above the door open level, this situation is retained until the conveyance is positioned at the tip level. No air is available to the chutes until both sets of doors are proved closed. When the chute is activated, the air is automatically removed from the doors, and when the chute is proved extended, tipping can commence. On the completion of the tip operation, the

Chute can be retracted. When the chute has been proved in the retracted position, air is again available to the doors which can be opened and the winding cycle resumed.

Safe and efficient work in the shafts can only be carried out if there is a clear and fool-proof method of communication between the Sinking and Linig crews, the Bank and the Winding Enginemen. In recent years loud speaker systems and walkie-talkie sets etc. have come into the foreground but the traditional method of signalling using pull-wires and bell-pushes is still the governing factor. It is in everyone's interest to learn all signals in use on the project, even one is not normally required to give or receive signals to do his job. Everybody should be prepared for an emergency. As all the instructions related to the hoisting operations given by signals using the signals are very important. All signals from underground will be returned by the hoist driver by flashing the stage lights the same number of times as the signal (called blinksink). Only banksmen on the bank, onsetter on the stage and man in charge on the sump is authorised to give signals. Some code of signals used mostly in this project are as follows :

Raise	1
Lower	2
Stop when in motion	1
Raise slowly	3
Lower slowly	5
Tip	2 - 1
Lock out secure	6 - 6
Man riding	4
Clear	2 - 2

All the hoisting operations have to be done according to the related procedures. Operations such as man riding, material sending to the shaft and all other hoisting operations during the construction steps, moving the stage etc. have strict rules and step by step instructions to be followed.

Some of general procedures about manriding are as follows:

- Before allowing any person to enter the kibble any bansman or onsetter shall give a signal of four to the hoist driver.
- The banksman shall not allow any person into the kibble unless they have first placed their tally on the tally board which is located alongside the banksman's cabin.
- All persons who go underground must be wearing a hard hat, safety glasses and safety boots, alongwith their lamp and self rescuer before entering the kibble. Other items of protective equipment thet must be carried are ear defenders, gloves, and safety harnesses dependant on the work they perform.
- When riding into the shaft the kibble shall stop just prior to the top deck of the stage and the onsetter shall signal and control the kibble whilst it is travelling through the stage.
- The maximum number of persons in a kibble shall be seven.
- Persons travelling the shafts are to be completely within the kibble and standing on the kibble floor.
- No person is to travel in a kibble full or partly loaded with rock or other material apart from hand-tools and surveying equipment.
- No person shall get out of a kibble unless it is completely stationary and the banksman or onsetter has indicated that it is safe to do so.

3.3.3 Construction Steps

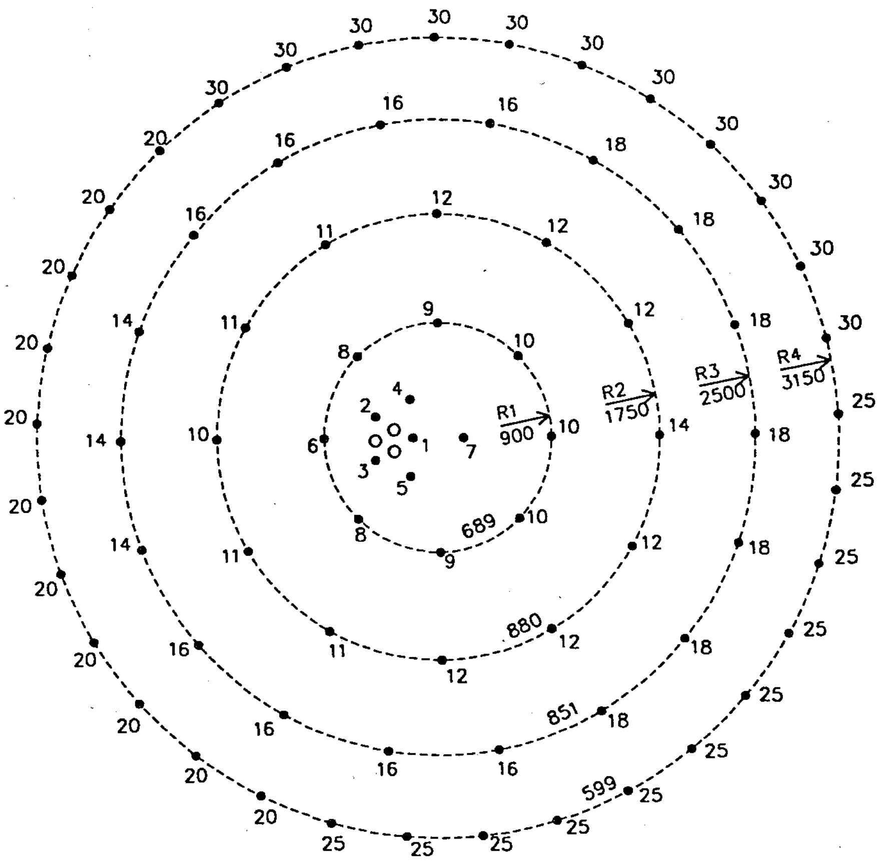
As it is mentioned before shaft construction is a cyclic process. General steps are drilling, charging, blasting, mucking, sidewall supporting, concreting and piping.

We take drilling as a first step. Holes to be drilled according to the blast pattern (Figure 3.2), approved by the Shaft Superintendent, are marked by the Shift Engineer and by using drill jumbo all the holes are drilled. Figure 3.3 shows the view of the stage during drilling. Normally there are 78 holes to be drilled and charged plus three big holes that are not charged at the center of the pattern. Center arrangement is given seperately in the blast pattern. All holes drilled 2.7m length and 35mm in diameter. 32x400mm and 25x200mm type dynamites are used for charging and nonel detonators are used for blasting. In each blast approximately 140 kg explosives is used. A wooden stopper is put in each drilled hole to keep holes open. After all the holes drilled drill jumbo is raised into the stage and explosives sent to the shaft with the supervision of the shift supervisor. Each hole checked that it is clear before charging, and properly stemmed immediately after charging. Any remaining explosives and detonators checked and brought out of the the shaft. In the explosive box both explosives and detonators are put in their respective containers. When all tubes of the nonel detonators have been bunched together and coupled up the shift supervisor signal the banksman to isolate power to the shaft. When this has been done, he then couple up the electric detonators and exit the shaft after the stage has been raised a safe distance from the sump. Once on the surface, the shift supervisor confirm with the banksman that all underground personnel have been accounted for on the tally board. Then shotfirer tests the exploder, stop the ventilation fan and fire the round. After the blast surface fan is switched on and it is waited at least 15 minutes for the smoke to clear.

Shift supervisor, taking with him a gas detector, a minimum of himself and the onsetter ride in the kibble to the top deck of the stage. The kibble stopped and a full examination made of the stage for damage or debris. The onsetter signal the kibble with the shift supervisor aboard to the shaft bottom. He then make a full examination for misfired shots or other safety issues, muckpile and sidewall, and bridge the shotfiring wire connections. On returning to the surface a continuity test made by the shift electrician on the shotfiring cable prior to starting on the next round. After every round of shots the shaft floor and any sockets throughly cleaned by blowing over and examined by a competent appointed person for mis-fires.

The shaft crew and Shift supervisor lower the stage to the pre-determined working position using the Codes for Lowering and Levelling the stage. When raising and lowering the stage, it is imperative that the following be under observation at all times:

- The signal cable, audio cable, power cable, and blasting cable, etc.
- That the stage does not “hook up” on pipe flanges, formwork etc.
- That the stage does not get out of level



NOTES:
ALL HOLES TO BE VERTICAL EXCEPT RING No.2 WHICH HAS A 2" LOOKOUT

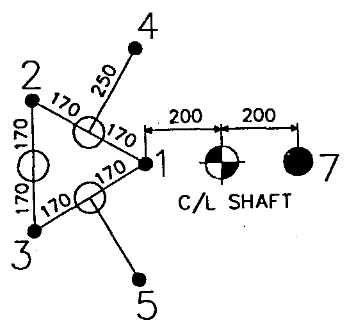
CHARGE WEIGHTS:

CUT EASERS AND RING No.1
6 No. x 32mm x 400 LG' DINAMIT

RING No.2
6 No. x 32mm x 400 LG' DINAMIT

RING No.3
9.No. x 25mm x 200 LG' DINAMIT
1.No. x 32mm x 400 LG' DINAMIT

RING No.4
11.No. x 25mm x 200 LG' DINAMIT



ENLARGED ARRANGEMENT AT PATTERN CENTRE
ALL HOLES DRILLED 2.7M

Figure 3.2. Shaft sump blast pattern.

No men is allowed on the shaft floor area when the stage is being moved. Kibbles are parked at the surface and at a safe distance above the stage on the fixed drum side before the stage is moved. Stage is lowered down on two winches to the required position. Individual winches are then used to level the stage by raising the desired ropes. After the levelling, stage is locked and the kibble is lowered to the where the following Top Deck position, Rider/Stage rope positon, Bottom Deck position and Waiting marks given to the Hoist operator. After the signalling of these marks Shift supervisor and two men transfer from Top deck of the stage to the kibble and proceed to the shaft bottom. Muck pile and side wall examinations are made and shaft floor mark is given to the Hoistman. On an "All Clear" comment from the Shift supervisor the men are lowered from the surface and stage to the sump floor. The cryderman mucking unit is then prepared for operation. To achieve this the put-locks and/or wedges secured at the top deck and bottom deck are positioned to centralize the stage, before the Cryderman is lowered to its working position (Figure 3.4). Normally drilling, charging and blasting takes eight or nine hours.

During mucking operations, manriding is generally forbidden unless special arrangements have been made by the shaft superintendent or shift supervisor. Mucking operations will usually utilise three kibbles; however occasionally only two kibbles are used. It is esential that a repetitative sequence of operation is quickly established for an efficient mucking cycle. All signals during the mucking cycle is given either by the stage onsetter, who is situated on the bottom deck of the stage, the shift supervisor or man in charge in the sump, or the banksman at surface.

Before commencing mucking two sumps are dug for the kibbles at the floor. During mucking kibbles will always be placed in these sumps prior to loading. The sump need to be continually dug out as mucking operation progress. When kibbles changed on the West rope, the cryderman mucking unit only operate in the East half of the shaft and when kibbles are changed on the East rope, the unit only operate in the West half of the shaft. Kibbles are not placed directly under their respective ropes for filling, but pushed North and South of this position towards the center of the shaft, so that a clear passage is afforded for steadying and raising the full kibble. All full kibbles are stopped when clear of the shaft floor, steadied and the under the bottom side cleaned by hand before being allowed to ascend. All descending kibbles are stopped at 5.5 metres (waiting mark) above the shaft bottom to await a signal from the shaft bottom before making the final descent.

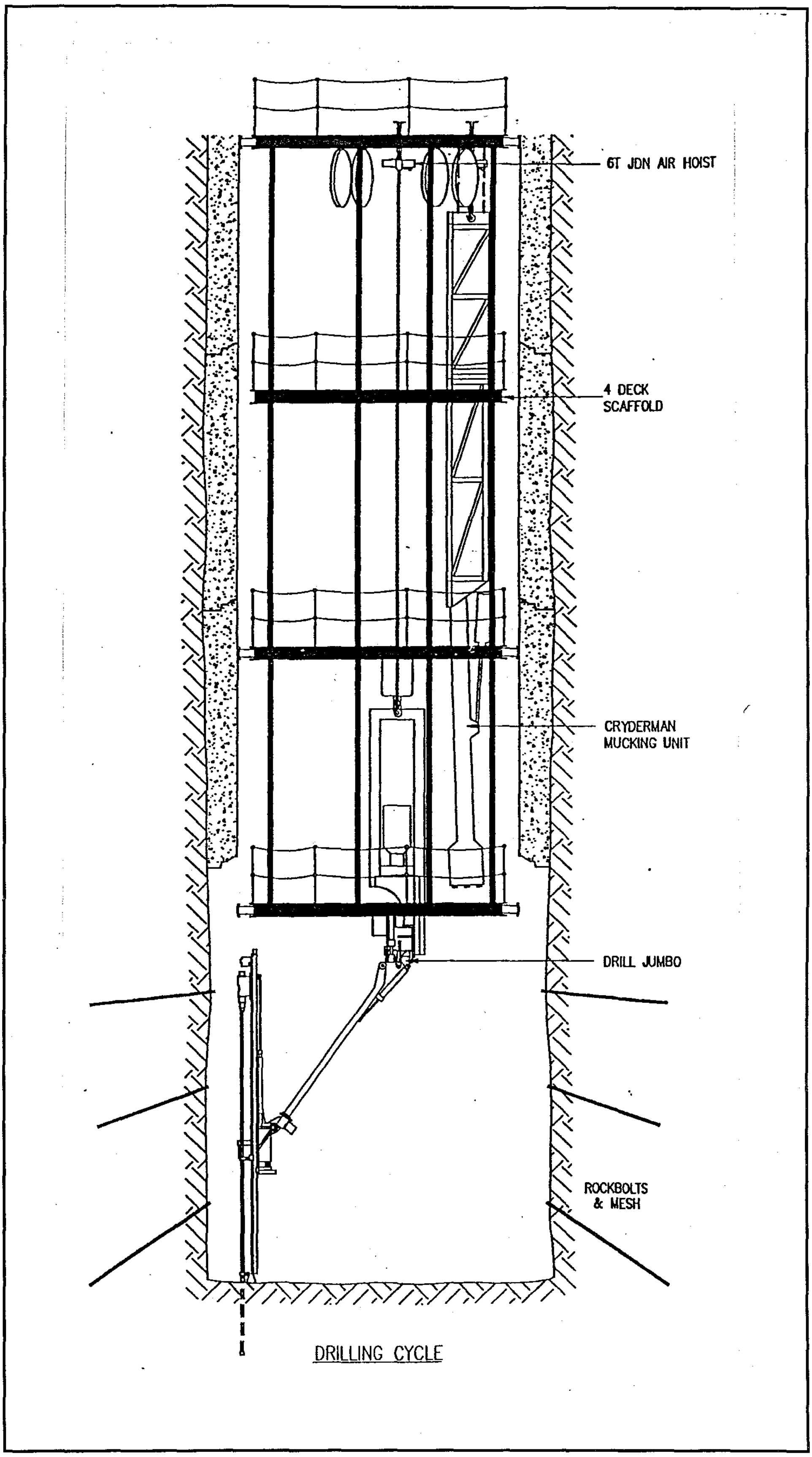


FIGURE 3.3. View of the stage during drilling.

During mucking operations when two meters of shaft side walls are exposed the following were applied in accordance with the Project Manager's Rules for temporary support:

The whole area of shaft wall was inspected and supported in accordance with the designated rock type between the lowest portion of permanent concrete lining and a depth of three metres above the shaft bottom at any stage of the sinking cycle. Rock type is classified according to the support type needed for supporting. Rock type classified according to this classification and the existing percentage of the easch rock type for 314m is given below.

Rock Type 1: Safety Bolting only as deemed necessary by the Shift Supervisor. Bolts put in specific areas where necessary (5 per cent).

2: Pattern bolting (max 2m x 1m). A full array of bolting as specified. Usual to type of ground hard rock (30 per cent).

3: Pattern bolting (max 3m x 1m) with wire mesh. Used where rock spoiling (40 per cent).

4: Pattern bolting with wire mesh and shotcrete. Really bad conditions (15 per cent)

5: Permanent concrete lining brought to Shaft bottom during sinking. Extreme ground conditions (10 per cent).

Ground is supported by wire mesh of adequate strength to withstand the weight of loose ground, in conjunction with rock bolts and shotcrete. Wire mesh is secured by 1000mm or 2000mm rock bolts. The horizontal and the vertical distances between the bolts do not exceed two meters. Nothing in these rules prevents a workman from setting supports at more frequent intervals when necessary for safety. Normally mucking takes one and a half shift time and average of 65 skips per round. In every round shaft becomes approximately 2.5 meters deeper. Our scaffold for concreting is five meters (Figure 3.5). So, before commencing to concreting process, drilling, blasting and mucking cycle is proceeded at least two times and when there is enough space under the last concreted section for scaffold erection it is passed to the concreting process.

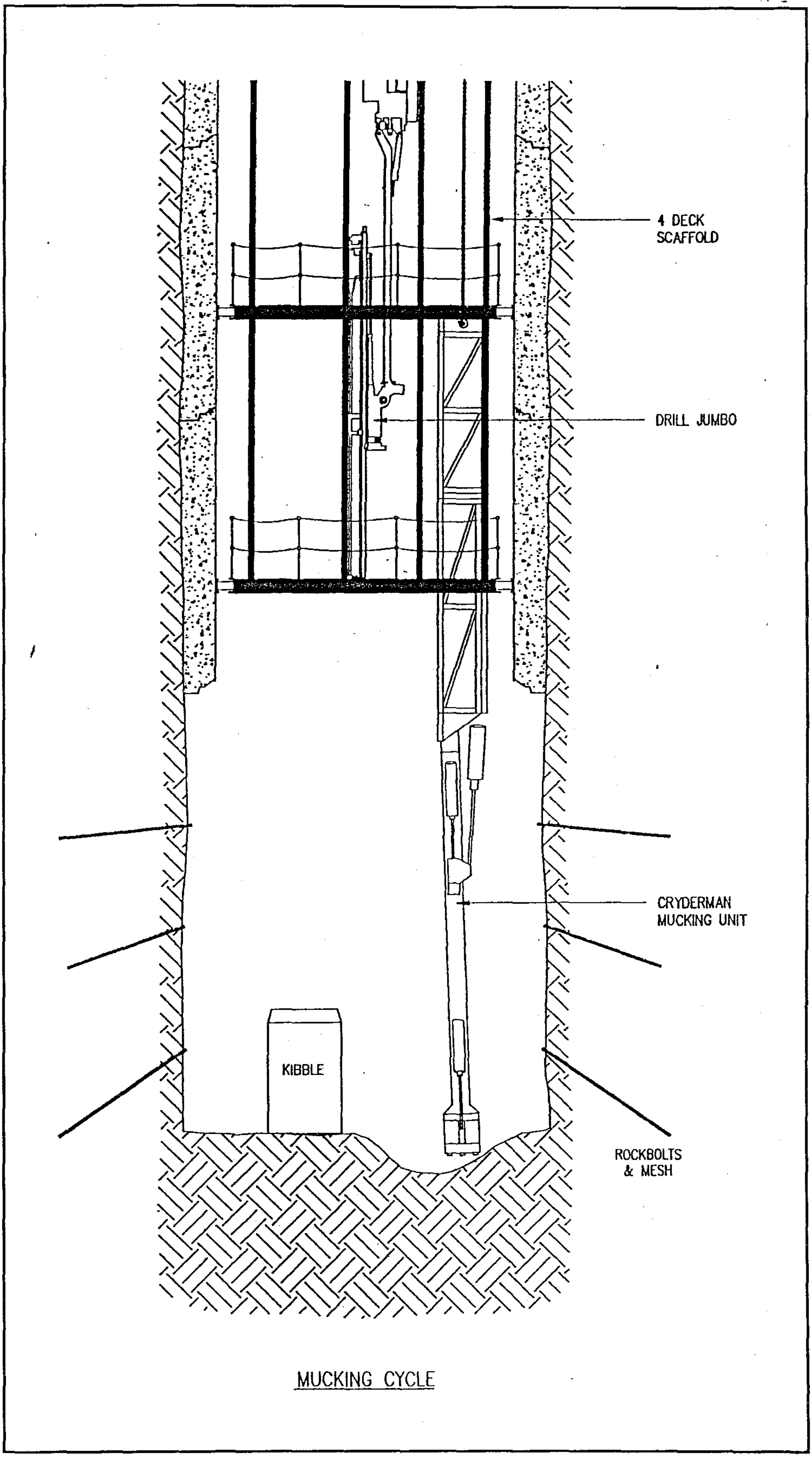


FIGURE 3.4. View of the stage during mucking.

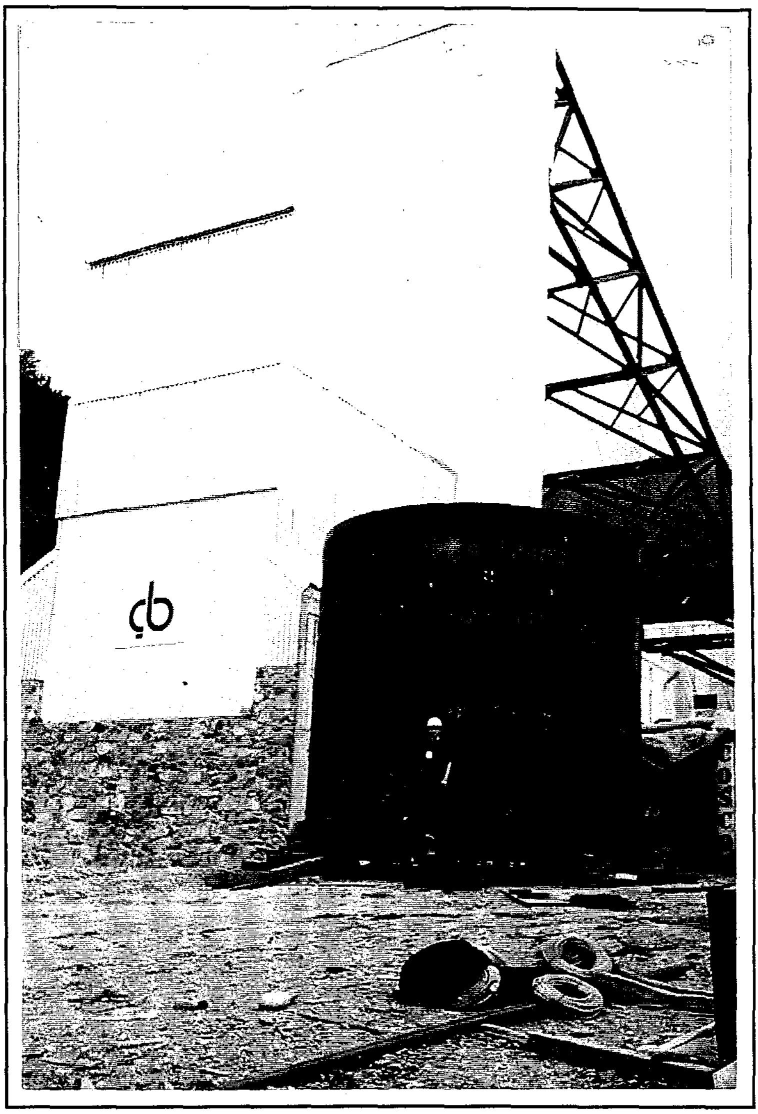


FIGURE 3.5. Photo of the formwork used for the shaft lining.

Before the concreting process sump is drilled, blasted and according to the depth needed under the previous concreted section maximum 25 per cent of a full round is mucked and rest is remained ready for mucking after the concreting. Figure 3.6 shows the view of the stage during concreting.

Concrete is poured directly from transmixer to the concrete box (Figure 3.7) and from there with gravity it is transported to the distribution box (Figure 3.8) which is positioned at the third deck of the stage via 15cm concrete pipe mounted to the shaft wall.

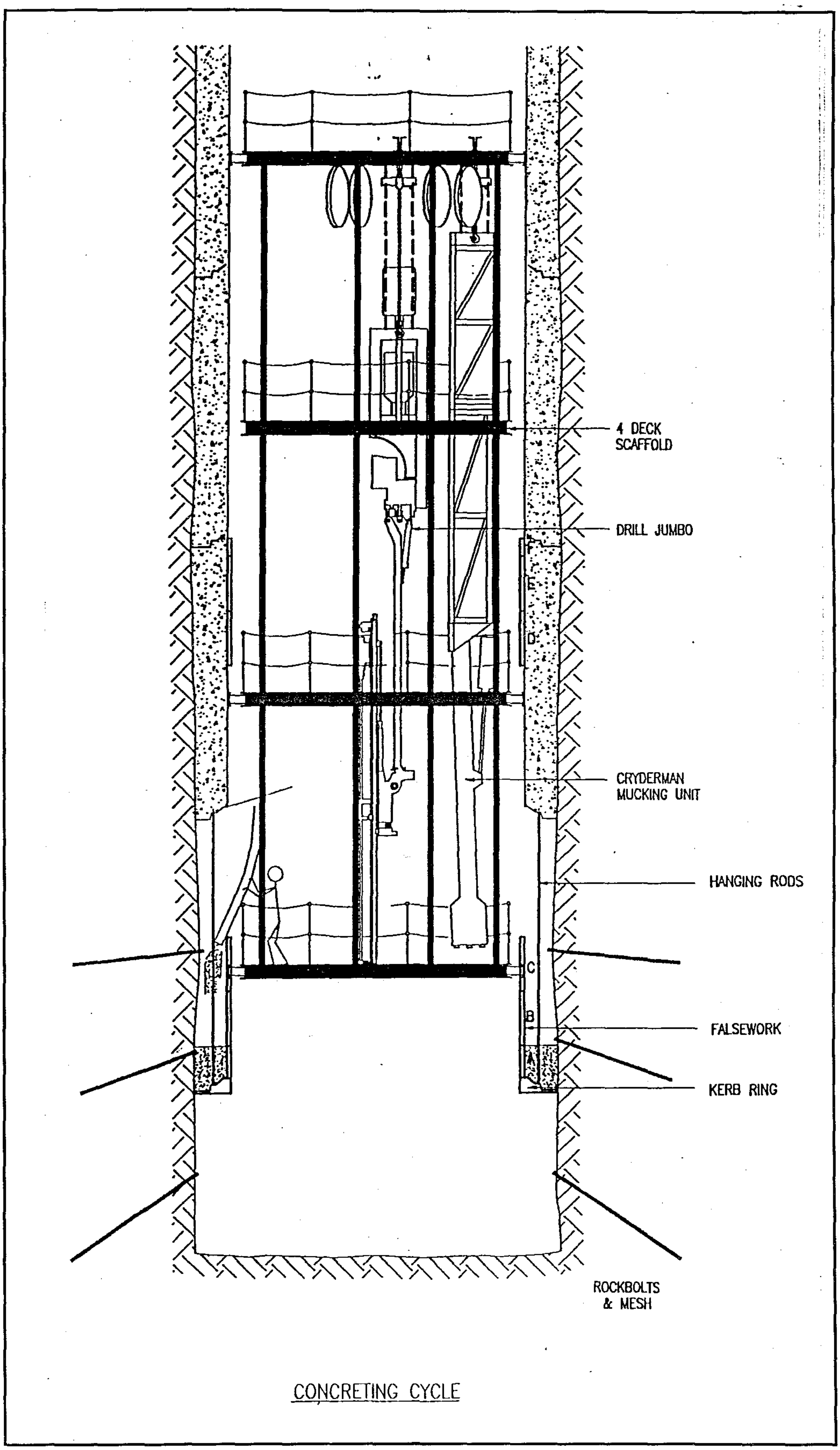


FIGURE 3.6. View of the stage during concreting.

During concreting the stage is positioned so that all concrete handling and vibration is done from the bottom deck to minimize spillage on the stage. This means that two stage movements are required during each pour. At the end of the 15cm concrete pipe there is a fitting called dashpot (Figure 3.9) to avoid segregation as concrete falls from a very high distance. Concrete falls vertically and first comes to the pocket of the dashpot. After concrete hits that pocket it pushes back and before falling out from the diagonal leg of the dashpot agregate and fine particles mix again. There are three flexible hoses mounted at the openings of the distribution box and each is used for pouring concrete to the different part of the shutter.

Before commencing to concreting, lowering and levelling of the scaffold is very important for the sake of shaft's verticality. There are 12 hanging rods (Figure 3.10) all around the shaft each one is coupled to the previous one. By these hanging rods bottom level of the scaffold and the alignment can be controlled. Lowering and levelling of the scaffold is explained in the shaft concreting procedure (see Appendix 3). Lowering and setting shutter takes one shift and pouring concrete normally takes five to six hours and average of 60-65 m^3 concrete is poured. As concrete is transported by pipes its slump is very important. Concrete's slump on site is normally 8-10 cm. By adding admixture its slump is increased upto 14-16cm. Test cubes are collected from each pour and tested. Concrete test cube results are given in Table 3.2. After concreting, all the equipments used for concreting are sent out of the shaft and cleaned, oiled and prepared ready for the next pour.

Table 3.2. Concrete test cube results.

Pour	3 days	7 days	14 days	28 days	56 days
1		21,33		28,44	
		19,11		31,11	
2		16,89	19,11	24,89	
		18,22	21,33	25,78	
3		14,22	18,67	20,00	
		11,56	15,11	17,78	
4		15,11	20,44	28,36	
		15,56	19,56	27,11	
5		17,78	22,22	21,67	
		19,11	24,00	28,80	
6					
7		17,78	24,00	23,60	
		18,67	23,56	27,43	
8		19,11	23,00	21,06	
		16,00	19,80	18,07	
9			19,04	27,08	
			27,06	30,61	
10		12,57	15,64	23,31	
		15,25	19,17	21,74	
11		21,71	21,86	25,91	
		24,41	21,72	25,73	
12		19,56	28,44	30,49	
		18,67		28,53	
		19,11			
13		19,56	27,56	30,31	
		17,78	16,00	32,18	
		17,78	25,33	33,60	
14		20,44	18,66	34,84	
		19,55	19,82	33,87	
15	13,77	16,44	24,30	26,93	31,56
		15,10		28,44	
		13,30		28,53	
				26,67	
				23,29	
				24,89	
16		17,06	21,60	25,33	
		18,76	22,22	27,55	
17	16,53	18,84	25,95	26,67	
	16,18	19,60	27,82	26,67	
	16,98	20,53	26,84		
18		16,89	20,71	23,11	
		16,71	21,42	20,00	
19		17,96	24,89	27,82	
		19,64	31,11	26,84	
20		16,09	20,00	20,00	
		16,00	18,22	19,60	
21		9,96	9,33	21,33	
		10,67	13,78	23,56	
22		18,67	25,69	27,91	
		15,55	20,62	24,01	
AVERAGE	15,87	17,38	21,64	25,99	31,56

FIGURE 3.7. Photo of the concrete box.

FIGURE 3.8. Photo of the distribution box.

FIGURE 3.9. Photo of the dashpot.

All the mechanical equipments used in the shaft is working by air. Drill jumbo works also by electricity and water is needed during drilling. Because of these there are some service pipes through the shaft wall. These are pipes for ventilation, concreting, water, pumping and to supply air (Figure 3.11). Ventilation is provided via a surface fan which delivers sufficient air to dilute and render harmless the fumes after the blast and provide fresh air to the shaft during sinking process. The ventilation from the fan is delivered through ducting which is kept in good condition as to prevent leakage. Concrete pipe is 15cm pipe and it is going to be dismounted after the shaft is completed and is seperately mounted by its special brackets. There are two pipes for air supply. One is 20cm main air pipe and the other is 10cm spare air pipe. Pipes for water and pumping are also 10cm in diameter.

FIGURE 3.10. Photo of the hanging rods just lowered to the stage

FIGURE 3.11. Photo of the service pipes mounted to the shaft wall

All pipes are 5m in length and all these four pipes (3x10cm ,1x20cm) mounted to the same brackets, which are screwed to the special inserts that are placed during the concreting to the shaft wall, by clamps.

3.3.4 Safety Measures and Instructions

Safety is very important in shaft sinking. Shaft sinking by conventional method is categorized as the most dangerous work in the world. Because of the high risks of the work's nature there are strict rules and detailed procedures that explaines every step to be taken by everybody, rules to be obeyed and the responsibilities. Safety Officer is responsible to the Project Manager for the safety precautions. He has followed up the safety records and take the preventive actions if necessary and needed. Some safety measures and rules stated for both general shaft sinking operations and for this project are given below.

TRAVELLING IN A CONVEYANCE:

- Persons travelling in a kibble must be completely within the kibble and standing on the kibble floor.
- Whilst travelling, hands are to be kept away from the kibble rim to avoid injury when passing through the stage.
- Leaning out of the kibble or travelling in a kibble full or partly loaded with rock stricktly forbidden

HOISTING OF PERSONS

- Kibbles containing men will not be rung away by the banksman or onsetter until all persons are completely within the conveyance.
- Kibbles hoisted from the bottom will be steadied and rung away by the men in the bottom.
- Men on stage will not steady the kibble as it reaches the bottom deck. If there is any tendency to swing, the man or the bottom deck will signal to the driver to stop and only resume travel when the kibble is steady.

- Kibbles will always stop at the waiting marks within the stage and above the sinking floor and will not move until the winding engine man receives a further signal.

HOISTING OF MATERIALS AND ROCK

- Kibbles must not be overloaded and persons who have steadied the kibble at the steady mark approximately 1.5m from the shaft bottom, will examine the underside for any adhering rock or material before ringing away and standing clear of the kibble.
- When loading or unloading material from the kibbles on the bank, all compartment must be closed.
- The bank doors and tip level are to be clean at all times and free of any spillage or equiment which could inadvertently fall into the shaft.
- No item of equipment shall be lowered through the shaft except by approved slings and chains, using D shackles to attach the chains to the equipment.
- All persons will wear protective clothing, including hard hats, safety boots, safety spectacles, gloves and ear defenders.

SAFETY INSTRUCTIONS

- In the event of there being thunder and lightning storm in the area, only nonel detonators and explosives will be transported from the underground magazine to charge the round in the shaft. The electric detonator will be transported seperately after the storm has passed to the fire round.
- All power supplies to the stage and shaft bottom will be switched off before the electric detonator is coupled and will not be switched on again until the round has been fired.
- Whenever the shift supervisor enters the shaft at the start of shift or after blasting to make a safety inspection, he shall take a gas detector. If men remain in the shaft bottom a gas detector must remain in vicinity. All shaft workmen must be familiarized with the operation of the gas detector and procedure should the gas detector go into an alarm state.

- The banksman shall not allow any man to go underground without wearing a cap lamp and self rescuer which every man must carry on his person at all times whilst underground. No man will be allowed underground with out the following safety equipment: helmet, cap lamp, overall, eye protection, ear defenders, gloves and harness where required.

The banksman shall receive from each person about to go underground his square tally. The round tally shall be retained on this person at all times whilst underground. When any person comes out of the shaft , he shall receive from the banksman his square tally even if that person is to go underground again later. This procedure is to ensure that the square tallies on the peg board represents all persons below ground. At the end of the shift both tallies shall returned to the lamp room by the person to whom they were issued [23].

3.3.5 Progress

Shaft sinking and preliminary development was started in June 1996 and continues well into 1997. Before that, an other company had sunk a fore shaft, about a depth of 28m. Installing sinking equipment to the shaft, training of the sinking crew has taken some time before commencing the sinking process. As most members of the sinking crew were local labors and they were not familiar with the shaft sinking and some problems encountered with the sinking equipment and sinking system there was understandable low progress on the first months, but as the crew get used to the sinking process and after the problems of the sinking system had solved an increase in the progress rate was provided.

An assumed operation cycle time table which was calculated before the operation is given in Table 3.3. In this table time needed for operations were calculated according to the rock types.

Following progress records for the month September are derived from the shift engineer's daily reports and summarizes the real sinking cycle. These records are also presented in a program format in Appendix 4.

Results from these reports and the assumed operation cycle are compared and the results are discussed and alternative operation time schedule is suggested by taking practical records into consideration (Table3.4).

Progress Records for the month September:

1 / 9 / 1996

The shaft was ready for concrete. As it was Sunday, only day shift worked and $66m^3$ of concrete was poured. Pouring started at 08:30 and finished at 14:00. Kerb level was 1024.2.

2 / 9 / 1996

DAY SHIFT first installed one 10cm water pipe, one 20cm and one 10cm air pipes and one 10cm pump pipe. Then lowered the jumbo started mucking. Between 12:20 to 15:30 22 kibbles of muck were tipped. AFTERS started mucking at 16:00 and finished mucking. They mucked 30 kibbles. There was a problem with cryderman between 21:10 and 22:30. NIGHTS were to drill but there was a problem with jumbo till 04:45. They started drilling at 04:45 and 20 holes were drilled.

3 / 9 / 1996

DAY SHIFT completed drilling. By 13:30 37 holes were drilled. At 12:50 charging was started and finished at 14:35. Totally 145 kg explosives were used in 57 holes. Shotfirer was waited till 15:50. At 15:50 sump was blasted. AFTERS lowered the stage and started mucking at 17:00. They mucked 37 kibbles and then started preparations for concreting. NIGHTS continued with concrete preparations.

4 / 9 / 96

DAY SHIFT started pouring at 8:55 and finished at 14:40. $59m^3$ of concrete was poured. Kerb level was 1019.2. After concreting, one 15cm concrete pipe was installed. AFTERS installed service pipes and started mucking. 29 kibbles of muck were tipped. NIGHTS continued with mucking. They mucked 11 kibbles then started drilling at 04:40 and drilled 27 holes.

<u>5 / 9 / 1996</u>

DAY SHIFT continued with drilling and drilled 50 holes. At 13:45 it was blasted. Totally 130kg of explosives were used in 77 holes. After the blast, it was shotcreted about two m^3. AFTERS also shotcreted about two m^3 and then started mucking. They mucked 24 kibbles by 21:40. Between 21:40 and 23:40 there was a problem in the cryderman. NIGHTS started mucking at 24:00 and mucked 45 kibbles by 7:30.

<u>6 / 9 / 1996</u>

DAY SHIFT mucked six kibbles between 7:30and 9:35 and then started drilling. They drilled 72 holes. AFTER charged the holes and used 160kg of explosives. Sump was ready to blast at 16:30 but shotfirer waited till 18:00 and blasted at 18:10. After the blast shift continued with mucking. 22 kibbles of mucked were tipped. NIGHTS also continue with mucking and mucked 39 kibbles. There was a problem in winder between 01:45 and 02:45.

<u>7 / 9 / 1996</u>

DAY SHIFT started preparations for concreting. Between 15:10 and 15:20 they poured five m^3 of concrete. AFTERS continue with concreting and poured $75m^3$ of concrete between 15:30 and 21:09. Kerb level was 1014.2. 15cm concrete pipe was installed after concreting and started mucking. By the end of the shift four kibbles of muck were tipped. NIGHTS continue with mucking and mucked nine kibbles. Finished mucking at 2:30 and started drilling. 36 holes were drilled by the end of the shift.

<u>8 / 9 / 1996</u>

As it was Sunday only day shift worked. Finished drilling and blasted at 12:30. Totally 81 holes were drilled and 160kg explosives were used.

Between 1/9/1996 to 8/9/1996 shaft were sunken about nine meters and lined 10 meters.

<u>9 / 9 / 1996</u>

DAY SHIFT installed service pipes and started mucking. 24 kibbles of muck were tipped. AFTERS continued with mucking and mucked 44 kibbles. NIGHTS mucked only one kibble and then started drilling. 54 holes drilled. Jumbo was broken down about 3.5 hours.

<u>10 / 10 / 1996</u>

DAY SHIFT finished drilling. Drilled 27 hole and fired at 11:40. Totally 140kg explosives were used in 78 holes. After the blast shift started preparations for concreting. AFTES finished concrete preparations and started concreting at 18:10. Shift poured 65m^3 of concrete between 18:10 and 23:30. NIGHTS finished concreting. Poured 6m^3 concrete by 23:45. Kerb level was 1009.2. After concreting a 15cm concrete pipe was installed and started mucking. 22 kibbles of muck were tipped by the end of the shift. There were 1:45 hours of delay because of the cryderman's break down.

<u>11 / 9 / 1996</u>

There was a problem with cryderman so, DAY SHIFT could started mucking at 9:00 and mucked 37 kibbles. AFTERS mucked four kibbles and finished mucking. They installed service pipes. NIGHTS grouted standpipes, cleaned and repaired the stage. Also they cleaned the bin house

<u>12 / 9 / 1996</u>

DAY SHIFT started drilling and drilled 71 holes. Besides these, shift drilled two probe holes of totally 18m, which took four hours. AFTERS finished drilling the round. Drilled three more holes, charged and fired at 18:30. 160kg explosives were used in 74 holes. After the blast shift continued with mucking and 23 kibbles were mucked. 30 minutes delayed because of the problem in the cryderman. NIGHTS continued with mucking and mucked 41 kibbles.

<u>13 / 9 / 1996</u>

DAY SHIFT finished mucking and mucked two kibbles. At 9:10 drilling was started and sump was ready to blast at 15:30. AFTERS waited shotfirer about an hour. At 16:30 sump was blasted. Shift started mucking at 17:00 and mucked 43 kibbles. NIGHTS mucked 21 kibbles till 02:54 and started preparations for concreting.

14 / 9 / 1996

DAY SHIFT finished concrete preparations and started concreting at 13:15 and poured $35m^3$ concrete. AFTERS finished concreted at 17:45 and poured $31m^3$. Kerb level was 1004.2. Installed a 15cm concrete pipe and continued with mucking. Mucked eight kibbles and finished mucking. At 22:00 shift started drilling and drilled 18 holes. NIGHTS continued with drilling and drilled 59 holes. Water was found in inner rings of holes about one meter down from sump floor. Slow drilling there after. Water flow reduced after initial surge, it was probably a standing water. By the end of the shift, sump was completely drilled and charged, ready to blast. 155kg explosives were used in 77 holes.

15 / 9 / 1996

DAY SHIFT waited for shotfirer and blasted at 08:15. This Sunday was maintenance day. Cleaning and repairing works were done during the shift.

Between 9/9/1996 to 15/9/1996 shaft were sunken about 12m and lined 10 meters.

16 / 9 / 1996

DAY SHIFT installed service pipes and mucked eight kibbles. Then shotcreted upto five meters below to last pour. AFTERS mucked 13 kibbles between 15:30 to 17:10 then rider was broken down. Shift continued with cleaning and repairing works. Shutters were taken out of the shaft and started cleaning. NIGHTS continued with rider's repairing. Shutters were cleaning.

17 / 9 / 1996

DAY SHIFT continued with rider's repairing and shutter's cleaning and remounted cleaned shutters. AFTERS continued with shutter cleaning and remounting. They installed a ventilation pipe. NIGHTS continued with shutter cleaning and remounting. Rider's repairing was finished and assembled .

18 / 9 / 1996

DAY SHIFT checked the shutters alignment and rider. At 11:05 shift started mucking and mucked 27 kibbles. AFTERS mucked 14 kibbles and shotcreted about four m^3. NIGHTS mucked till 02:40 and mucked nine kibbles. Then started drilling and drilled 78 holes. Started charging.

19 / 9 / 1996

DAY SHIFT finished charging and blasted at 09:40. 162kg explosives were used in 78 holes. Shift then started preparations for concreting. AFTERS finished preparations for concreting by 18:00 and started pouring concrete at 18:06 and finished at 23:25. Totally $68m^3$ concrete was poured. Kerb level was 999.2. NIGHTS installed service pipes and started mucking at 04:30. 17 kibbles were mucked.

20 / 9 / 1996

DAY SHIFT continued with mucking and mucked 43 kibbles. AFTERS mucked 16 kibbles and finished mucking. Then shotcreted about $4m^3$. Shift started drilling between 22:00 and 23:30. NIGHTS were finished drilling and charging. 144kg explosives were used in 79 holes and blasted at 07:24.

21 / 9 / 1996

DAY SHIFT started mucking and mucked 43 kibbles. AFTERS continued with mucking and mucked 12 kibbles. Installed service pipes and started preparations for concreting. NIGHTS continued with concrete preparations.

22 / 9 / 1996

Sunday was a maintenance day. Cleaning and repairing works were done during the DAY SHIFT.

Between 16/9/1996 to 22/9/1996 shaft were sunken about eight meters and lined five meters.

23 / 9 / 1996

DAY SHIFT poured $74m^3$ concrete between 10:00 and 14:50. Kerb level was 994.2. AFTERS mucked 14 kibbles and shotcreted about $4m^3$. NIGHTS continued with mucking and mucked 11 kibbles by 03:15. Then started drilling at 03:45.

24 / 9 / 1996

DAY SHIFT finished drilling by 9:00 and charged. 138kg explosives were used in 76 holes and blasted at 11:40. Shift continued with mucking and mucked six kibbles. AFTERS continued mucking and mucked 43 kibbles and shotcreted about four m^3. NIGHTS continued with mucking and mucked 21 kibbles. At 5:40 drilling started.

25 /9 / 1996

DAY SHIFT finished drilling ant blasted at 15:30. 170kg explosives were used in 83 holes. AFTERS mucked 22 kibles and shotcreted about five m^3. NIGHTS continued mucking till 01:45 and mucked 13 kibbles and then started concrete preparations.

26 / 9 / 1996

DAY SHIFT finished concrete preparations and started pouring concrete. Between 11:10 and 11:45 12m^3 concrete was poured. Then shutters were cleaned and bin house was emptied. AFTERS continued with pouring concrete and poured 72 m^3 concrete. Kerb level was 989.2. After concreting service pipes were installed. Ground was loose so NIGHTS worked for ground supporting. They installed wire meshes, put rock bolts and shotcreted about four m^3.

27 / 9 / 1996

DAY SHIFT mucked 30 kibbles. AFTERS continued with mucking and mucked 15 kibbles and then started drilling at 22:00. 42 holes were drilled by 23:30. NIGHTS charged and blasted. 30kg of explosives were used in 42 holes. Only one part of the sump was blasted. After the blast, shift continued with mucking and mucked nine kibbles and then started preparations for concreting.

28 / 9 / 1996

DAY SHIFT finished concrete preparations and poured 15m^3 concrete. AFTERS continued with concreting and poured 54m^3 concrete. Kerb level was 984.2. After concreting service pipes were installed. NIGHTS mucked eight kibbles and then started drilling.

29 / 9 / 1996

Sunday was a maintenance day. Cleaning and repairing works were done during the DAY SHIFT.

Between 23/9/1996 to 29/9/1996 shaft were sunken about 10m and lined 15 meters.

30 / 9 / 1996

DAY SHIFT finished drilling and blasted at 11:50. 138kg explosives were used in 71 holes. Shift started mucking at 13:30 at mucked 10 kibbles. AFTERS continued with mucking and mucked 24 kibbles. NIGHTS mucked two kibbles and finished mucking. Then drilled, charged and blasted at 07:30. 125kg explosives were used in 69 holes.

Comparison of assumed operation cycle and report results:

Times needed per linear metre excavation is taken from the Table 3.3.

Ground Type-1:	361 min	20.6 m/week
Ground Type-2:	419 min	17.7 m/week
Ground Type-3:	453 min	16.4 m/week
Ground Type-4:	685 min	10.8 m/week
Ground Type-5:	516 min	14.4 m/week

Percentages of ground types were given for the 314m depth. There is five per cent of Type-1 ground, 30 per cent of Type-2 ground, 40 per cent of Type-3 ground, 15 per cent of Type-4 ground, and 10 per cent of Type-5 ground.

By using these informations we can calculate average sinking and lining rate as;

Ground Type-1	5 per cent	16m / 20.6m/wk	= 0.78 wk
Ground Type-2	30 per cent	94m / 17.7m/wk	= 5.31 wk
Ground Type-3	40 per cent	126m / 16.4m/wk	= 7.68 wk
Ground Type-4	15 per cent	47m / 10.8m/wk	= 4.35 wk
Ground Type-5	10 per cent	31m / 14.4m/wk	= 2.15 wk
	100 per cent	314m	20.27 wk

Average Advance = Sink and Line: 15.49m/wk

Average advance derived from the monthly reports are as follows:

(Total working hours were 22.5 hr/day and 6 days/wk, that makes 135 hr/wk)

In the month August progress was 35m in 31 days and Total delays were 44.25 hr.

31 days = 4.43 wk (52/364 x 31)
35 / 4.43 = 7.9 m/wk

44.25 hr = 0.33 wk (44.25/135)
Then, the revised progress was 35 / (4.43 - 0.33) = 8.54 m/wk.
Total time per linear metre of shaft = 948 min. (135 / 8.64 x 60)

In the month September progress was 40m in 30 days and Total delays were 66.25 hr.

30 days = 4.29 wk (52/364 x 30)
40 / 4.29 = 9.32 m/wk

66.25 hr = 0.49 wk (66.25/135)
Then, the revised progress was 40 / (4.29-0.49) = 10.53 m/wk
Total time per linear metre of shaft = 770 min. (135 / 10.53 x 60)

According to the assumed operation cycle time table, average advance is calculated as 15.49 m/wk but actual advance was 7.9 m/wk in the month August and 9.32 m/wk in the month September. This progress may be developed but not as much as proposed in the assumed operation cycle time table.

Assumed time table was prepared according to the previous experiences but in Çayeli, sinking system was slightly changed from the proposed one, such as instead of 4 drill machines only one drill jumbo was used and instead of 3m3 kibbles 2.67 m3 kibbles were used, and most of the sinking crew were local labors and they were not familiar with the job and the system. Beside this there was a language and culture differences between the Turkish labors and the English labors, foremen and lead miners. They had to communicate each other by the help of Turkish shift engineers. The job was also new for the Turkish shift engineers and it took time for them to get used to the job. This language and culture difference effected productivity very much especially in the first months. As the sinking crew get used to the job, progress increased in the following months. Besides crews' productivity, system and equipment faults caused some delays in the first months. Especially because of the tipping system, tipping operation took longer time than it had to. Also, because of the cryderman operators, mucking operation was very slow in the first months but in the following months there was a noticeable increase in the progress.

Cautions to be taken to increase the productivity of the system and the necessary points that must be taken into consideration before assuming a operation cycle schedule are discussed below and an alternative operation cycle schedule is given in Table 3.4.

The main points that effected the progress rate were the performance of the operations and delays. Delays were mainly caused because of the faults of the sinking equipment. Beside this, concrete supplier and waiting for the shotfirer caused some delays.

Causes of Delays:

As the job is a cyclic job, one job follows the other in the "finish to start" order. Because of the limited working area and the nature of the job, only one job can be done at a time. And when a delay occurred because of any type of problem all the sinking process is effected from this problem and every operation is shifted.

According to the laws only authorized shotfirers who have a shotfiring certificate can do the blasting. Because of this, every time after the sump was charged an authorized shotfirer from the mine had to come and do the blasting. Sometimes shotfirers had their own businesses to do in the mine and could not arrive to the shaft on time for blasting and this caused some delays up to two hours. This problem could be eliminated by employing a shotfirer or some of the workers can be trained for to take a shotfiring certificate. As there were 24 hours working in three shifts second alternative is the better solution for this problem.

To eliminate the problem of waiting for a concrete, a serious organization and coordination had to be done with the concrete supplier.

The main reasons of the delays were the problems with the sinking equipment. Especially the cryderman and, at the early stages of the operation, tipping caused long delays. Cryderman is operated by air. Because the high air pressure and the friction of the hoses with the moving parts of the cryderman hoses were burst very often. To eliminate this problem, a special design could be made for the cryderman hoses or they must be kept away from the moving parts of the cryderman. Chutes also caused some problems. In Canadian system there is comprising a locator pin on the bottom of the Kibble that sits into a Keyway within the tipping chute, that rotates as the Kibble is lowered. There had been a problem with the locator pin and the Keyway. Most of the time locator pin caught in the keyway because of the design and the mud. To minimize this problem bottom of the kibble should be kept clean all the time. By increasing tipping efficiency mucking time can be decreased.

To eliminate or minimize these types of delays will cause at least 10 per cent increase in the progress. For example in August monthly progress was 7.9 m/wk. After the delays were deducted then the revised progress was 8.54 m/wk which means 8.1 per cent increase in the progress. In September, monthly progress was 9.32 m/wk. After the delays were deducted then the revised progress was 10.53 m/wk, which means 13 per cent increase in the progress.

Performance of the operation was an other factor influencing the rate of progress. The only solution to increase the performance of the job is time. Within the time as sinking crew get used to the job and each other, progress rate will increase successively. Progress rate increase from August to September gives the same result. Actual progress was increased from 7.9 m/wk to 9.32 m/wk from August to September which means 18 per cent increase and revised progress was increased from 8.54 m/wk to 10.53 m/wk during the same period which means 23 per cent increase. As job has a cyclic nature and the operations are repeated all the time first months can be evaluated as a training period and in the following months job become automated. In every operation where to stay, what to do, and what is coming next. Some type of operations especially during the concrete preparations and the concrete pouring these type of automation will speed up the job. Because of this it will be better to employ the same man for the same job every time. As mentioned before language and culture difference caused some communication problems which effected the performance negatively. Before making an assumption about the performance of the job beside the technical data such as drill rate of the jumbo of tipping capacity of the kibble, social factors, such as cultural differences, working in an other country, working in a new job with new equipment and new system, must be taken into consideration.

By taking all these above mentioned factors into consideration and with the light of the six months experience an alternative operation cycle schedule was calculated for the future period of the job. Table 3.4.

From an alternative operation cycle schedule we can calculate average sinking and lining rate as;

Ground Type-1	5 per cent	16m / 13.7m/wk	= 1.17 wk
Ground Type-2	30 per cent	94m / 12.9m/wk	= 7.29 wk
Ground Type-3	40 per cent	126m / 11.5m/wk	= 10.96 wk
Ground Type-4	15 per cent	47m / 9.6m/wk	= 4.90 wk
Ground Type-5	10 per cent	31m / 15.5m/wk	= 2.00 wk
	100 per cent	314m	26.32 wk

<u>Average Advance</u> = Sink and Line: 11.93m/wk

Table 3.3. Assumed Operation Cycle

	ASSUMED OPERATION CYCLE					
		Cycle - for each Support Type				
	Operation - Excavation	Type 1	Type 2	Type 3	Type 4	Type 5
1	Drill basket in - mark up	20 min	20 min	20 min	20 min	n/a
	Number of Holes	74	74	74	74	n/a
	Depth of Hole	2.4m	2.4m	2.4m	1.8m	
	No of Drill Machines	4	4	4	4	n/a
	Average Drill speed per machine/min.	300mm	300mm	250mm	200mm	n/a
2	Drill Time	148min	148min	178min	156min	n/a
3	Drill basket Out - Explosives in	15 min	15 min	15 min	15 min	n/a
4	Charge up - clear sump	75 min	75 min	90 min	90 min	n/a
5	Raise stage	20 min	20 min	20 min	15 min	n/a
6	Blast and clear smoke	30 min	30 min	30 min	30 min	n/a
7	Lower Stage / Cryderman - prepare to muck	60 min	60 min	60 min	60 min	60 min
	Depth of Pull	2.2m	2.2m	2.2m	1.6m	1.0m
	Average Dia. (incl overbreak)	6.5m	6.5m	6.5m	6.7m	6.7m
	Mucking Rate - 3 m3 kibbles	8/hr	8/hr	8/hr	7/hr	4/hr
8	Muck Out Time	310 min	310 min	310 min	274 min	300 min
9	Support Rockbolts & Mesh	Concurrent	120 min	120 min	120 min	--
10	Support Shotcrete	--	--	--	120 min	120 min
11	Clean Up and Blow Over	60 min	60 min	90 min	120 min	n/a
	Cycle Time	738 min	858 min	928 min	1020 min	480 min
	Contingency - 7,5%	55 min	64 min	70 min	77 min	36 min
	TOTAL	793 min	922 min	998 min	1097 min	516 min
	Length of Pull	2.2 m	2.2 m	2.2 m	1.6 m	1.0 m
	Times per linear metre excavated	361 min	419 min	453 min	685 min	516 min
	Shaft Advance in week of 123.75 hours	20,6	17,7	16,4	10,8	14,4

Table 3.4. Suggested Alternative Operation Cycle

SUGGESTED OPERATION CYCLE						
		Cycle - for each Support Type				
Operation - Excavation		Type 1	Type 2	Type 3	Type 4	Type 5
1	Drill basket in - mark up	20 min	20 min	20 min	20 min	n/a
	Number of Holes	78	78	78	78	n/a
	Depth of Hole	2,7	2,7	2,7	2,0	
	No of Drill Machines	1	1	1	1	n/a
	Average Drill speed per machine/min.	540mm	540mm	450mm	360mm	n/a
2	Drill Time	390 min	390 min	470 min	430 min	n/a
3	Drill basket Out - Explosives in	15 min	15 min	15 min	15 min	n/a
4	Charge up - clear sump	75 min	75 min	90 min	90 min	n/a
5	Raise stage	20 min	20 min	20 min	15 min	n/a
6	Blast and clear smoke	30 min	30 min	30 min	30 min	n/a
7	Lower Stage / Cryderman - prepare to muck	60 min	60 min	60 min	60 min	60 min
	Depth of Pull	2,7	2,7	2,7	2,4	2,1
	Average Dia. (incl overbreak)	6.7m	6.7m	6.7m	6.85m	6.9m
	Mucking Rate - 2,67 m3 kibbles	5/hr	5/hr	5/hr	4/hr	3/hr
8	Muck Out Time	820 min	820 min	820 min	810 min	800 min
9	Support Rockbolts & Mesh	Concurrent	90 min	90 min	90 min	--
10	Support Shotcrete	--	--	--	120 min	120 min
11	Clean Up and Blow Over	60 min	60 min	90 min	120 min	n/a
	Cycle Time	1430 min	1520 min	1705 min	1800 min	980 min
	Contingency - 12%	172 min	182 min	205 min	216 min	118 min
	TOTAL	1602 min	1702 min	1910 min	2016 min	1098 min
	Length of Pull	2,7m	2,7m	2,7m	2,4m	2,1m
	Times per linear metre excavated	593 min	630 min	707 min	840 min	523 min
	Shaft Advance in week of 135 hours	13,7m	12,9m	11,5m	9,6m	15,5

4. DISCUSSION

In the first part of this section all three sinking systems are compared with each other and their advantages and disadvantages are discussed. In the second part sinking system of the case study is discussed. Reason of the selection of the conventional method and sinking equipment alternatives within the conventional method are compared.

4.1 Comparison of Sinking Systems

If we evaluate sinking methods individually all methods have some advantages and disadvantages and some limitations for practical cases. Such as for shaft boring method there must be an underground access. For the cases where there is no underground access this method can not be evaluated as an alternative method.

4.1.1 Shaft Drilling Methods

The most important advantage of the shaft drilling method is the highest degree of safety. The other major advantage is a considerable reduction in highly skilled personnel for executing the work. At the same time working conditions are much more attractive when compared with the situation on the bottom of the shaft.

Shaft drilling methods are faster than conventional method but up to diameters approximately five meters only, and under the condition that the variation of geological strata is not too great and the danger of drilling fluid losses is limited. Under such circumstances, the drilling rates should always be ahead of the rates which results from the conventional shaft sinking. This advantage is often increased by the higher speed of the lining operation

Drilling "Blind shafts" is especially advantageous when shafts have to be sunk in unstable, running ground formations, in which the conventional sinking technology has to be aided by the costly procedure of ground freezing, increasing the price per meter of shaft three times. Using the drilling method, most of these problems are easily kept under control by an appropriate mud mixture exerting a pressure on the shaft wall sufficient to assure its stability and to prevent inflow of water. The labor intensive conventional shaft sinking technique is more costly to support, especially in remote areas where the crew must live in a camp and fly in and out on a rotational basis. Blind shaft drilling involves only three or four crew members per shift, including supervisors and maintenance personnel. In most cases, the shorter duration of the drilling project cases the need to rotate the crews, which considerably decreasing the overall project cost.

The essential disadvantages of "Full-diameter shaft drilling " have been until now the lack of accuracy of alignment, the execution of lining operations after completion of the drilling operations, and high excavating costs. The last are due to too low a cutter load - which for reasons of adherence to direction could not be increased in the past - and the method-inherent higher wear of cutters, since the muck is crushed several times, and fines are dispersed in the drilling fluid. Furthermore, there are still other disadvantages with respect to drilling large diameters and depths, a lower degree of flexibility with respect to adaptation to frequent rock formation variations, and equipment- and method- inherent limitations of thrust which cause less economic penetration rates in hard rock formations.

"Full-diameter shaft boring machine without drill pipe " has only limited possibilities of employment considering the variation of the drilling diameter. It is not appropriate for employment in unstable, unconsolidated formations, and implies very high investment costs. Furthermore, the excavating costs have to be classified as disadvantageous since the cutter costs, higher muck transport costs have to be added, when the requirement for a continuous conveyance of muck is maintained. Vice versa, in case of a discontinuous removal of the muck, for example, in connection with mechanical hoisting, the degree of utilization of the equipment is too low to justify from an economic point of view the high investment costs for the drilling machine.

4.1.2 Shaft Boring Methods

The safety of personnel, improved ground stability, faster rates of advance, and cost savings are among the reasons raise boring methods has found wide acceptance in the mining industry.

Shaft boring methods takes the men out of the hole and only the drill rods, stabilizers, and raise head work in the hole. The personnel involved the sinking operation under safe conditions.

The second major advantage of the boring methods is the improved stability of the raise. The completed raise is a round, structurally stable opening. It is a smooth, clean bore that, in competent formations, and normally does not require lining. There is no overbreak, such as usually occurs with drill-and-blast techniques. In ventilation raises, air flow is more efficient, because the turbulence is reduced. In ore passes, the smooth sides allow passage of materials without bridging. In man ways and service raises, the danger of loosened, falling rock injuring personnel or destroying utilities is greatly reduced.

Another major advantage is the time requirement. Raise boring is normally a continuous operation, with an overall rate of progress that is faster than is possible with the conventional drill-and-blast techniques.

A forth major advantage is cost saving. Obviously, the first three advantages mentioned must be included in this category. In addition, the much lower noise levels, less direct labor and supervision, and timesavings associated with personnel removal for blasting translate to cost saving.

Raise boring methods show disadvantages with respect to its application over the entire diameter range, the entire depth, the lack of suitability for use in unconsolidated rock formations and the restriction that the location of the shaft to be drilled must in any case be accessible from the bottom. Increase in rock strength or abrasiveness will favor use of conventional methods, due to the decrease in boring rate and higher cutter cost. Beside these common disadvantages, the combination of the raise boring method with the shaft reaming machine has a reduced flexibility of use with respect to different shaft diameters, and investments costs are high.

4.1.3 Conventional Sinking Method

On conventional shaft sinking no essential performance increase can be expected any longer. The so-called "learning curve" has flattened out and experience of the method over centuries has resulted in such an optimization that expectations of further important improvements appear to be improbable.

Nevertheless, conventional shaft sinking offers a wide range of advantages, in fact; the highest flexibility with regard to application over the cross section and depth range, the highest reliability as a proven method of long term experience, the highest flexibility for reacting when unpredictable changes in the geological formations arise, and, equipment investment costs reduced by a spread over a broad range of possible applications. Drill-and-blast method is used for station, loading pocket or other type of constructions. This will be slower and more expensive with the mechanical sinking methods since the sinking equipment must be moved up the shaft to avoid blast damage. Operations will also be less effective when they are conducted with the sinking machines in the shaft and the labor skills are different from those required to operate mechanical sinking equipment. Increasing requirements for station openings will tend to favor the selection of conventional sinking methods.

The most important disadvantages are; the cyclic nature of excavating and lining operations, the low degree of safety, the very low accuracy of the excavated profile, the need of highly qualified personnel in very large numbers, and, very low potential for important improvements in increasing sinking rates and in costs. Besides these in the case of the application of special methods to conventional shaft sinking the total excavating costs are an essential disadvantage.

There are some basic requirements to be complied with shaft sinking methods. These basic requirements are economy of the method, safety and accuracy of direction.

Except for the basic requirement "safety ", the degree of compliance of usual shaft sinking with respect to accuracy of direction, suitability for application and sinking costs is identical or greater, respectively, than that of the drilling or boring methods. This however changes essentially if within the scope of usual shaft sinking methods become necessary (for example freezing) and the degree of compliance drops correspondingly as far as the costs are concerned. This effects immediately an increase of frequency of application of shaft drilling with drill pipe, if a sufficiently high number of meters to be drilled seems to justify initial investment.

Table 4.1 gives the comparison of factors influencing the selection of construction methods.

Table 4.1 Comparison of Factors influencing the selection of construction methods.

Comparison of Factors Influencing the Selection of Shaft Drilling, Shaft Boring and Conventional Shaft Construction Methods			
Factors ifluencing Selection	Shaft Drilling	Shaft Boring	Conventional Shaft Sinking
Safety	High Does not require miners to work underground.	Medium Only requires underground crew set-up and mucking. Operators are located remote from the working face	Low Requires miners to work under ground in all stages of the sinking.
Need for skilled personnel	Low	Medium	High
Working Conditions	High	Medium	Low
Sinking Speed	High Upto diameters approximately 5 meters and under the condition that the variation of geological strata is not too great and the danger of drilling fluid losses is limited.	High	Low
Accuracy of alignment	Low	Medium	High
Shaft Size	Low Limited by required depth, available equipment, and cost.	Low Limited by available equipment.	High Required to be larger than 3,5-4m for most applications. Upper limit is not controlled by method.
Shaft Depth	Low	Low	High
Flexibility	Low	Low	High
Miscellaneous		Requires existing underground access	
Costs			
Initial	High	Medium	Low
Operating	High	Medium Incerase in rock strength or abrasiveness will favor use of conventional methods, due to the decrease in boring rate and	High Especiall where there is need for costly procedure of ground freezing.

4.2 Discussion of Conventional Method in Rize

Conventional sinking method is the most appropriate method that can be used in Rize when compared with the other methods.

For example shaft boring methods are available if there is an underground access to the shaft to be drilled. In practical it is usually used underground from level to level and usually 100 m maximum or very short shafts where center 1.5m maximum cored hole can be drilled accurate and vertical. In Rize as there is not underground access for the shaft, shaft boring method is not available in Rize.

And shaft drilling is only successful in small shafts less than five meters and ground has to be composite. In Rize inner shaft diameter is 5.5 meter and proposed outer diameter is about 6.3 meter. Beside this there is Felcite strata in Çayeli which is not composite and there is a danger of wall collapse. Because of these conditions shaft drilling is not a suitable solution for Çayeli shaft.

Conventional method is most successful method for its use of various options i.e. with latest shaft drilling rigs etc. On the other hand, loading pockets in Çayeli shaft makes the conventional method appropriate.

The shaft sinking system at Çayeli utilizes a stage mounted single boom drill jumbo suspended from the top deck by JDN Hoist and a stage mounted "Herman" Cryderman Unit, suspended on a single winch rope from surface. The stage has four decks equally spaced at five meter apart, to suit the five meter concrete pours. The Kibbles are tipped using a Canadian system comprising a locator pin on the bottom of the Kibble that sits into a Keyway within the tipping chute, that rotates as the Kibble is lowered.

The advantages and disadvantages of the equipment and tipping system used in Çayeli project and their alternatives are discussed below.

Stage Mounted Drill Jumbo :

Advantages:

It is easy to set up the drill jumbo (especially if floor is uneven / sloping which has zero effect.). there is no slinging into / out of shaft. And drill jumbo is ideal for this type of ground where extracting rods over 1.8m long is very difficult if drilling by hand. It is easy to drill all the sump including cut / easers and there is lots of space on sump floor whilst drilling. And it makes job effective with less man to drill.

Disadvantages :

Beside of these advantages, stage mounted drill jumbo has some disadvantages such as access for maintenance within stage can be difficult for some parts as there is a limited space in the stage. Initial set up of the drill jumbo is slower than the hand drilling. Before commencing to drilling cryderman must be raised into the stage to provide space for jumbo and drilling, stage must be lowered to a drilling distance and then jumbo must be lowered for drilling. It takes nearly 45 minutes from last kibble to first hole drilled. As drill jumbo is mounted in the stage it take up space on stage and restricting access for concrete pours.

Stage Mounted Cryderman as opposed to Wall Mounted :

Advantages :

It is much quicker to set up for mucking and no delays during mucking for bolting / unbolting Cryderman to the wall. Before commencing to mucking stage must be lowered to a mucking level and then cryderman must be lowered for mucking. As it is not mounted to the wall there is no need for the insets to cast into the shaft lining and ideally if stage is balanced there is no need for rope from the surface. And cryderman does not have to be lifted above formwork prior to a concrete pour.

Disadvantages :

At Çayeli there is a lack of space on bottom deck when concreting. However this can be easily avoided in future by ensuring the Cryderman can be lifted within the stage clear of bottom deck, and having fold down doors to cover the opening. The stage design at Çayeli does not permit the "Yoke" of the Cryderman to be lifted above second deck, thus the bucket does not clear bottom deck.

<u>Stage with 5m Between Decks as opposed to 3 or 4m :</u>

Advantages :

Five meter distance between the decks suits the five meter concrete pours so concrete is always poured from same deck (bottom deck) - therefore no change to hoses from distribution box during pour and limits concrete spillage to bottom deck. Hoses kept permanently on stage. There are space under the second and the third decks which can be utilized for equipment, tanks etc.

Disadvantages :

Every time concrete is poured from the bottom deck at top of shutter, tendency for men not to clean down shutter properly afterwards. And again because of the limited space on the stage inter deck access to equipment / shaft wall / formwork is more limited.

<u>Canadian Tipping System :</u>

Advantages :

Canadian Tipping system is a safe method for tipping as nobody is close by. Good for small Kibbles in tight headgear with not much space. It allows a gentle tip. There is no need for bumping bar and once the system works properly tipping goes fast. It can be automated, thereby saving personnel and time.

Disadvantages :

In this system geometry is very important. Locator pin at the bottom of the kibble must properly sit into a key way within the chute and button under kibble needs to be regularly cleaned and oiled. There is a big learning curve to get working properly. As it is gentle tip muck inside the kibble sticks easily and it is impossible to belt Kibble to get sticky muck out. Opposed to the other tipping systems its installation is more expensive, with regards to time and materials. It needs a rider catch system, monitoring etc. All complications can lead to problems. High sides required to chutes as muck is launched from kibble from about one meter above rotating key plate. There is a potential for rope wear as rope passes through rider during tip.

5. CONCLUSION

After evaluating the shaft sinking methods it is understood that the conventional method is still the most widely used sinking method. Although the mechanical sinking techniques offer some important advantages to the shaft sinking market some of the restrictions of these methods are limiting the employment of these methods. Such as, if there is not an underground access for the shaft to be drilled, the shaft boring methods can not be evaluated as an alternative sinking method under any circumstances. And the shaft to be drilled to be drilled is bigger in diameter and depth e.g. more than six meters in diameter shaft drilling method is not a suitable solution.

As mentioned earlier, for conventional shaft sinking, an essential performance increase cannot any longer be expected. There may be some improvements in the sinking steps such as increasing the drilling time and efficiency by developing drilling machines and drill bits, speeding the mucking time with more effective mucking units and developing fast and safety tipping systems. But most important disadvantage of this method is the safety and the bad working conditions. Conventional method must be replaced by the other mechanical sinking methods as soon as possible. In case of necessity for the conventional method great care must be taken for the safety.

The studies are concentrated on to find a solution to the disadvantages of the other mechanical shaft sinking methods for them to replace the conventional method.

Like-wise conventional method, "raise-boring " and the "combination of raise-boring and down reaming " with a V-mole have reached a level on the basis of which further technical progress with respect to a considerable performance increase cannot be expected from these systems either.

Without any doubt shaft drilling with drill pipe has the most important shortterm realizable potential benefit with respect to a drilling speed increase. This opinion is based on the fact that

- the drilling speed of this method experienced so far is the lowest of all drilling methods, and
- an increase of the drilling speed was precluded in the past since the necessary additional load to be exerted onto the cutter heads would have caused in most cases deviations from the intended shaft axis, which could not have been corrected at an economically justifiable expenditure.

Since recent experience indicates that adherence to direction does not impose a problem any longer and, thus, as the cutter loads can be higher, a considerable reduction of cutter costs can also be taken into consideration.

It should be indicated however that - additionally - further, probably long lasting, still hazardous and intensive engineering will yet be required;

- to widen the limits of the existing diameter and depth restrictions,
- to reduce the cutter costs and the costs of their exchange in such a way that they will not be higher than the costs for drilling, blasting and loading of the conventional shaft sinking method and last but not least ,
- to conform to increasingly aggravating environmental restrictions.

Nevertheless, the potential for further developments in full face boring should not be taken out of mind in total, especially as the different new steerable down the mole machines and devices for shaft drilling have more and more the appearance of shaft boring machines. Therefore, a further combination of both types of equipment and methods could be the appropriate answer to the existing obstacles hindering shaft drilling/boring from becoming the standart method for the construction of shafts.

In case of the full face shaft boring machine, this benefit potential has to be similarly evaluated, but it has to be presupposed that the development of an effective system, suitable for mining, for continuous muck removal will require more time than will be the case in the near future for achieving the potential increases in performance of shaft drilling rigs with drill pipe.

APPENDIX 1

Design Capacities and Details

Design Capacities and Details	
1.0 MINING RATES	
Ore Production - tonnes / year	1000000
Ore Production - tonnes / day	3630
Waste Production - tonnes / year	%10
Waste Production - tonnes / day	363
Hoisting Rate : Days / week	5,5
Hoisting Rate : Tonnes / day	3993
Hours per Day	18
Days per Year	275
In situ ore t/cu m (-300 mm)	4
Broken Ore wt. t/cu m (-300 mm)	3
In situ waste t/cu m	2,6
Broken waste wt. t/cu m (-300 mm)	1,6
Hoisting Rate t/hr	222
2.0 SURFACE STRUCTURES	
2.1 Headframe, Sheaves and Ropes	
H/F Height	34 m to Sheave
Skip Sheave Diameter	3048
Skip Rope Diameter	32
Skip Rope Wt. (max. loading)	5.77 Kg/m
Breaking Strength (max. loading)	918.6 kN
Type (max. loading)	Full Lock Coil
Cage Sheave Diameter	2438
Cage Rope Diameter	31,7
Cage Rope Wt. (max. loading)	3.3 Kg/m
Breaking Strength (max. loading)	622.8 kN
Type (max. loading)	6 x 27 Strand
2.2 Collar House	
Basic Size	7.2m W x 12m L x 6m H
Rail Size	None
Overhead crane (Conv. removal)	8 Tonne

Design Capacities and Details	
2.3 Bin House	
Ore bin capacity	200 Tonnes
Waste bin capacity	50 Tonnes
Waste Chute	Yes
Waste Chute Angle	50 Deg.
Ore flow area heated	Yes
Floor Heating	No
Loadout	For CAT 769 4.7x8.2x3.85 or Conveyor
2.4 Collar	
Elevation	1109.3 m
Plenum	Rectangular
Plenum Resistance	59 Pa. max.
Mine air heater	Provisions allowed for future
Capacity	160 cu m/s
2.5 Structural Design Loads	
Snow Load	200 Kg/sq m
Wind Load	100 Kg/sq m
Earthquake Load	1 in 100 yrs. 0.06g
2.6 Hoist / Compressor House	
Structural Steel Building or Concrete	13m x 27m
Overhead Crane	13.6 Tonne
Basement	None
Skip Hoist	3050mm Dia. min.
Hoist Drum Width	1524
Hoisting Speed	9.3 m/s
Hoist Horsepower	863
Skip Compartment Size	1700 x 1520
Skip F/F Guides	1550
Skipc C/C Compartments	2060
Shaft Steel Design Life	25 years

Design Capacities and Details	
3.0 SHAFT EQUIPMENT	
3.1 Conveyances	
Skips - Kimberly --Quantity	2 + spare
Skip Capacity - Payload Initial	5216 Kg
Skip Capacity - Payload Final	5670 Kg
Skip Capacity - Volume	1.89 cu m ore
Skip Dry Weight	3522 Kg
Skip/Cage - overall length	7.3m
Tralling Cage	Yes - 718 Kg
Seperate Cage Quantity	1
Cage Capacity - Number of men	27
Cage Capacity - Max. sling load	5 t
Cage overall length	2565 mm
Skip / Trailing Cage Weight	4240 Kg
3.2 Stations	
Levels	840, 650
3.3 Loading Pocket	
Level	840, 650
Capacity Tonnes	5.67 / Double
3.4 Spill Pocket	
Level	810
Capacity	55 Tonnes
3.5 Services	
Compressed Air	200 mm
Drain Line	100 mm
Potable Water	No
Service Water	100 mm
Discharge Line	2 - 100 mm
Mine Signals	Yes

Design Capacities and Details	
4.0 SURFACE CONVEYOR SYSTEM	
Conveyor Feed Rate	250 t/hr
Material Lump Size	300 x 300 x 450
Direct Mill Feed	No
Mill Stockpile Area	Provisions for 7 bins
Conveyor to stockpile transfer	Overhead tripper
Stockpile to mill transfer	Mobile Equipment
Transfer system	Two intermediate towers

APPENDIX 2

Safety Measures

INDUCTION CHECK LIST

Trainee :

Job :

Department :

Date Started :

ITEM	INITIALS	ITEM	INITIALS
1) Conditions of Employment Hours of work Holidays Sickness/absence procedure Contract of employment Rates of Pay		**6) Fire Precautions** Causes Location of equipment Use of extinguishers Exits Alarm system	
2) Site Rules Do`s and Don`ts PPE rules Safety meetings Disciplinary procedure		**7) First Aid** First aid boxes First aiders Accident book Accident reporting	
3) Health and Safety Information on hazards Risk assessment summary Safety rules Working methods Responsible staff		**8) Welfare** Canteen and catering Cloakroom and toilets Lunch/tea breaks No smoking rules Notice boards Safety clothing facilities Health and hygiene facilities	
4) Training Training programme Promotion Development		**9) Management** Who`s who in management Safety policy	
5) Emergency Procedure Evacuation of tunnel Use of self rescuer Tally system		**10) General** Reporting of defects Safety suggestions	

Induction checked by:

Signature :

Position :

Signature of trainee :

Date :

RISK ASSESSMENT SHEET					ACTIVITY : SHAFT WORK	
HAZARD	HAZARD EFFECT	SEVERITY	PROBABILITY	RISK	MINIMIZE RISK BY :	RES. RISK
Persons falling down shaft	Death, serious physical injury	High	High	High	Hand rails around exposed openings Limit access to only authorised personnel Use of experienced shaft operators or persons under instruction Compliance with written method state. Use of shaft harnesses	Low
Materials falling down shaft	Death, serious physical injury	High	High	High	Covers over shaft provided at bank level All materials lowered into shaft attached to approved and tested lifting chains Operations personally supervised by shift supervisor Adhere to written method statements and procudures	Low
Lowering or raising materials by hoist	Swinging load, crush injury	Medium	Medium	Medium	When lowering or raising swinging load, signal given to stop hoist until load is steady. Keep hands well away of moving load Do not put handsd in nip points Maintain good communication & clear signals	Low
Use of pnuematic tools	Eye injury, vibration white finger, noise induced hearing loss	Medium	Medium	Medium	Use of eye & hand protection Alternate use of percussive tools to prevent single employee receiving excessive exposure Use of hearing protection	Low
Debris falling down the shaft	Serious physical injury	Medium	Medium	Medium	Clean off staging prior to work in shaft bottom Dress and secure shaft walls Clean off the bottom of kibble prior to raising on mucking operation	Low
FINAL ASSESSMENT :	Operation safe to continue providing all control measures remain in place				OVERALL ASSESSMENT : LOW	

OBSERVATIONS SAFETY AUDIT FORM

PLACE : DATE : DURATION :

TYPE OF WORK : No. of Men :

HAZARDS IDENTIFIED

Persons Falling		Collapsing / Overturning	
Struck by Falling Objects		Contact with Machinery	
Struck by Falls and Grounds		Contact with Harmful Substance	
Struck by Moving Equipment		Tripping or Slipping	
Trapping / Crushing		Manual Handling Injury	
Struck by Flying Particles		Electric Shock	
Dust		Vibration	
Noise		Fire	

CATEGORY A

REACTION OF PEOPLE

adjusting p.p.e
change position
rearranging job
stopping job
hiding/dodging
changing tools
applying lockout

CATEGORY B

POSITION OF PEOPLE

striking against
struck by
caught between
falling
electricution
inhaling
absorbing

CATEGORY C

P.P.E

eyes and face
ears
head
hands and arms
legs and feet
respiratory

CATEGORY D

TOOLS AND EQUIPMENT

right for job
used correctly
safe condition
seat belts in use
warning lights/notices
safety devices

CATEGORY E

PROCEDURES

is standard practice
adequate for job
is it maintained
is the procedure
written down
are people authorised

CATEGORY F

ORDERLINES/TIDINESS

standards established
standarts maintained
walkway clear
materials tidy
general housekeeping

CATEGORY	STANDARD DEVIATIONS OBSERVED
CATEGORY A	
CATEGORY B	
CATEGORY C	
CATEGORY D	
CATEGORY E	
CATEGORY F	

NAMES OF MEN CARRYING OUT AUDIT	

NAMES OF MEN AUDITED	

COMMITMENTS MADE	FOLLOW UP ACTIONS REQUIRED	DATE

REMEMBER THE RULES
At least 2 auditors.
Conscious decision to audit.
Stand and observe people problems.
Talk to the team.
Use a questioning attitude .
What do men know, rules, procedures, policy ?
Are they aware of th hazards and risks ?
What are their ideas ?
Make commitments.
Act on people problems.
Follow up action.

APPENDIX 3

Methof of Statement

METHOD STATEMENT

SHAFT CONCRETING PROCEDURE

1. Never commence concrete preparations unless there is at least 75% of a full round of muck in the sump.
2. Install dashpot along with 4m flexible hose on outlet. Anchor dashpot to concrete pipe bracket with 2 No 3T chain blocks.
3. Mean while fit temporary covers over both Kibble openings on bottom deck.
4. Lower stage until scribing rails are just above handrail on bottom deck. Attach 4 JDN Hoists to A ring and take weight of Kerb and A. Pull back scribing rails and stack any long ones on the decking over the Kibble openings.
5. Remove cone nuts, open door in A ring and pull A ring in using steam boat jack, to strip it off the wall.
6. Lower Kerb and A ring with JDN hoists until Kerb is clear of hanging rods in last pour. Then lower stage until A ring is about 300mm above its new position. On the way down strip all the the old scribing plw away from the underside of the previous pour (save as much as possible) and claen up the construction joint removing any poly thene or sandbags.
7. Whilst stripping Kerb sling Hanging Rod Basket in on the West side and lower into vent opening. When Kerb is lowered install 12 No Hanging rods - walk them on by one around the edge of the stage, insert through Kerb and screw on until the couplers are tight. Meanwhile blow out joint between Kerb & A ring to ensure it is clean.
8. Screw on 4 No Cone Nuts flush with the end of the hanging rods,(N,S,E & W) and lower Kerb with JDN's onto them.
9. Shut door in A ring and bolt A ring securely to Kerb.
10. Level up top of A ring to the 4 shaft tapes setting at the elevation given by the Shift Engineer. When set put on the reminder of the cone nuts, hand tight.

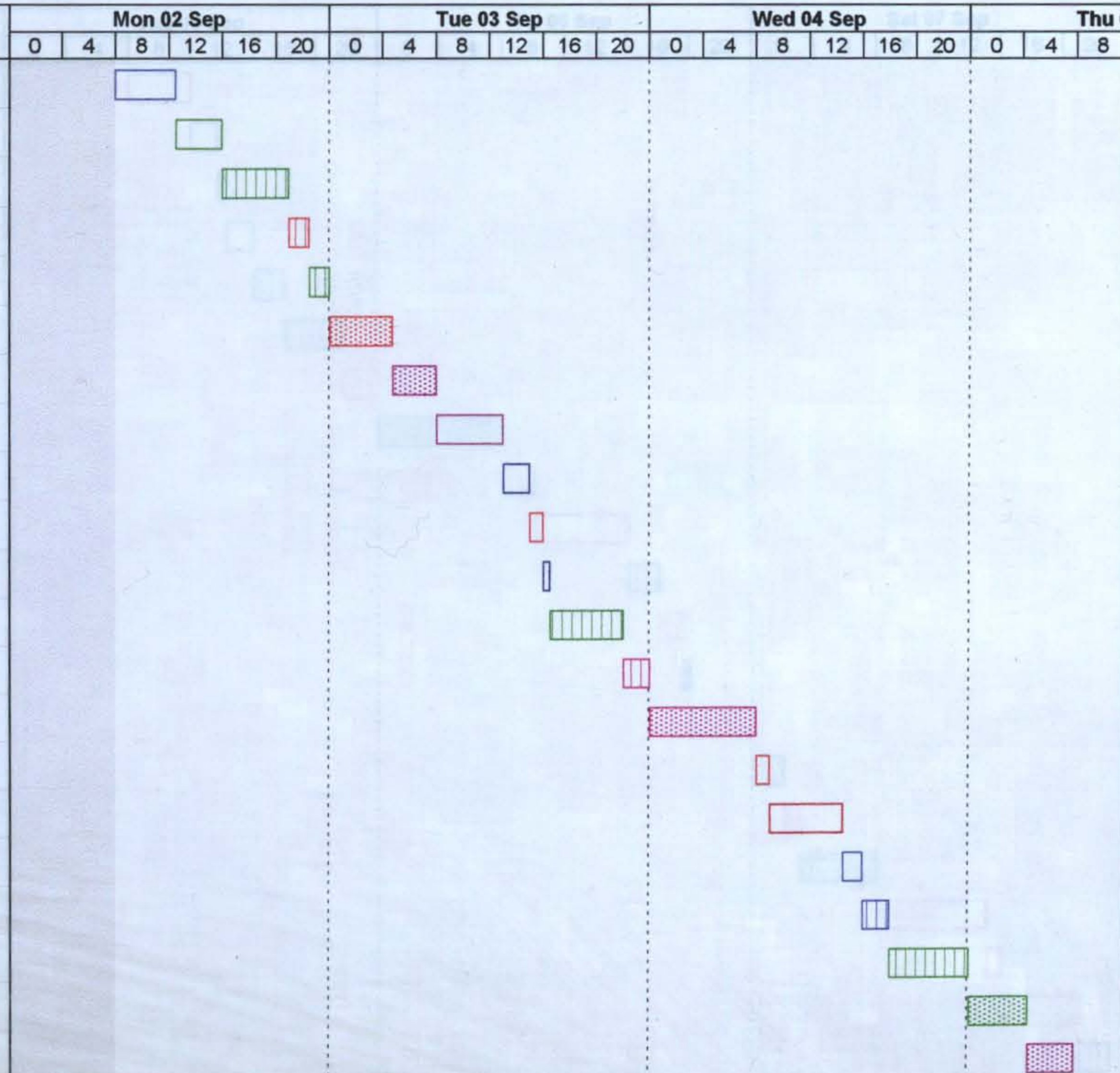
11. Fix 2 rock bolts one 1m either side of West Sideline. Level with the top of A ring. Lower sidelines and attach shaft plumb-bobs. Set Kerb for alignment (150mm from sidelines) and orientation (align sidelines with saw cuts in top of A ring). Secure top of a ring with a minimum of 8 tubbing jacks and 2 pull lifts off the rock bolts to hold it for spin." A ring" is permanently connected to the Kerb by 10 No M16x90 bolts which are set to allow 40mm of vertical movement between them.
12. Install scribing rails setting chamfered end about 25mm from excavation. Cover rails with scribing plywood and fill any small holes between plywood and excavation with polythene sandbags.
13. Fix 28 inserts and 8 grout pipes in A ring.
14. Put denso tape around hanging rods at Kerb level and brush Kerb and A ring with soap oil.
15. At the same time as items 12,13 & 14, lower in distribution box on the East side complete with 3 No 4,5m flexible hoses attached, and position on 3rd deck. Attach these flexible hoses to those kept on the stage. Extend dashpot hoses into distribution box.
16. Pour A ring (call for concrete hour before you are ready). Make sure the concrete is well vibrated.
17. Whilst pouring A ring attach JDN hoists to B ring. Undo doors on rings B, C and D and use steamboat jacks to ease shutter off the wall.
18. Lower B, C & D ring, bolt to A ring and align top of D with the previous pour using a plumb line.
19. Pour B, C & D
20. Whilst pouring B, C & D, attach JDN's to E ring, strip E & F rings off the wall as above.
21. Lower E & F and fill with concrete level with the top of f ring.

22. Clean up. Wash down all hoses & equipment. Wash down inside face of shutter with blow pipe & brush stage clean. Send out Dashpot and Distribution Box.
23. Whilst drilling next round in sump slacken off all the cone nuts by 10-20mm, remove all short bolts between Kerb and A ring and jack Kerb down to sit on the cone nuts. Thus breaking bond with concrete.

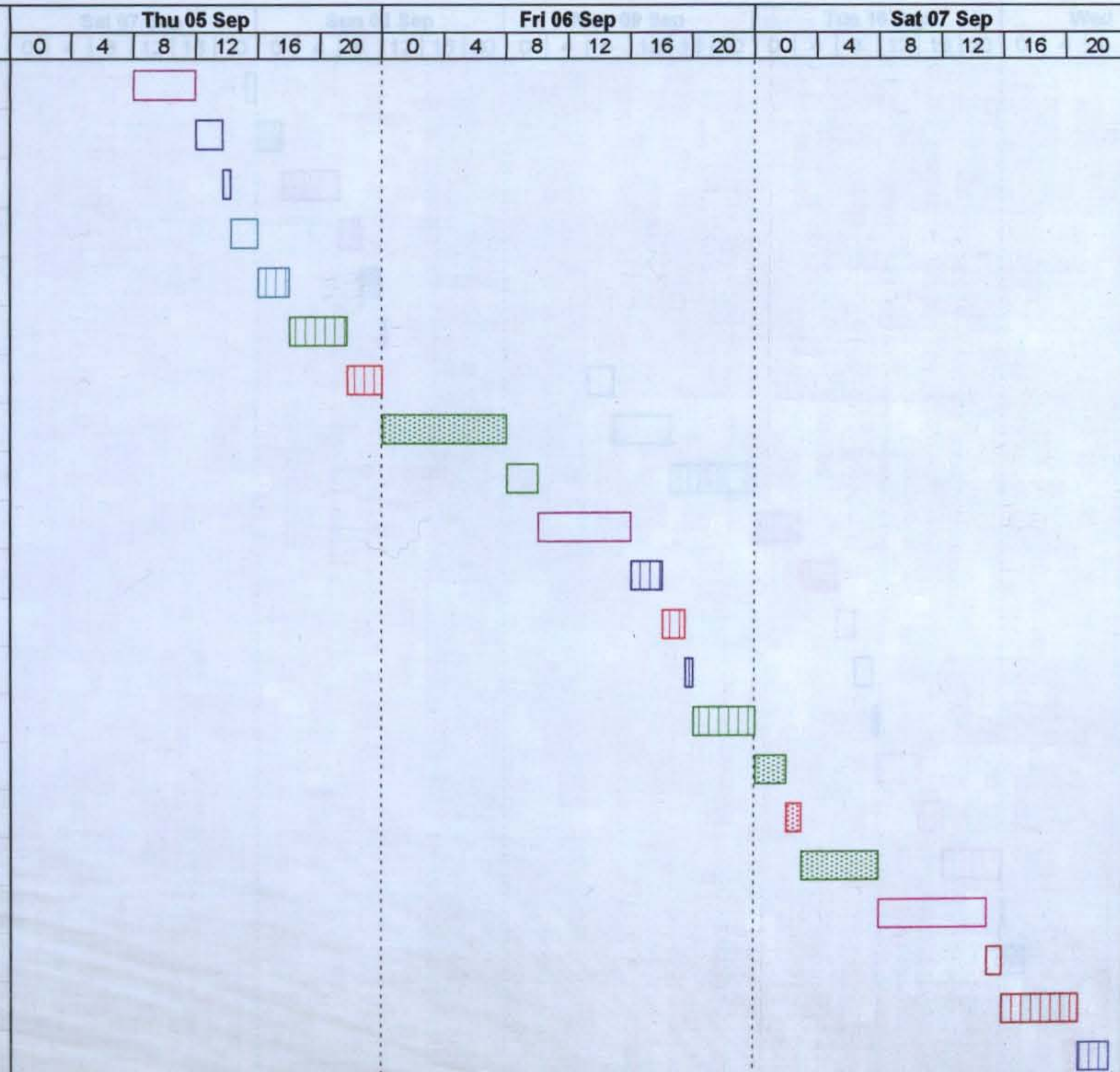
APPENDIX 4

Progress Schedule of Month September

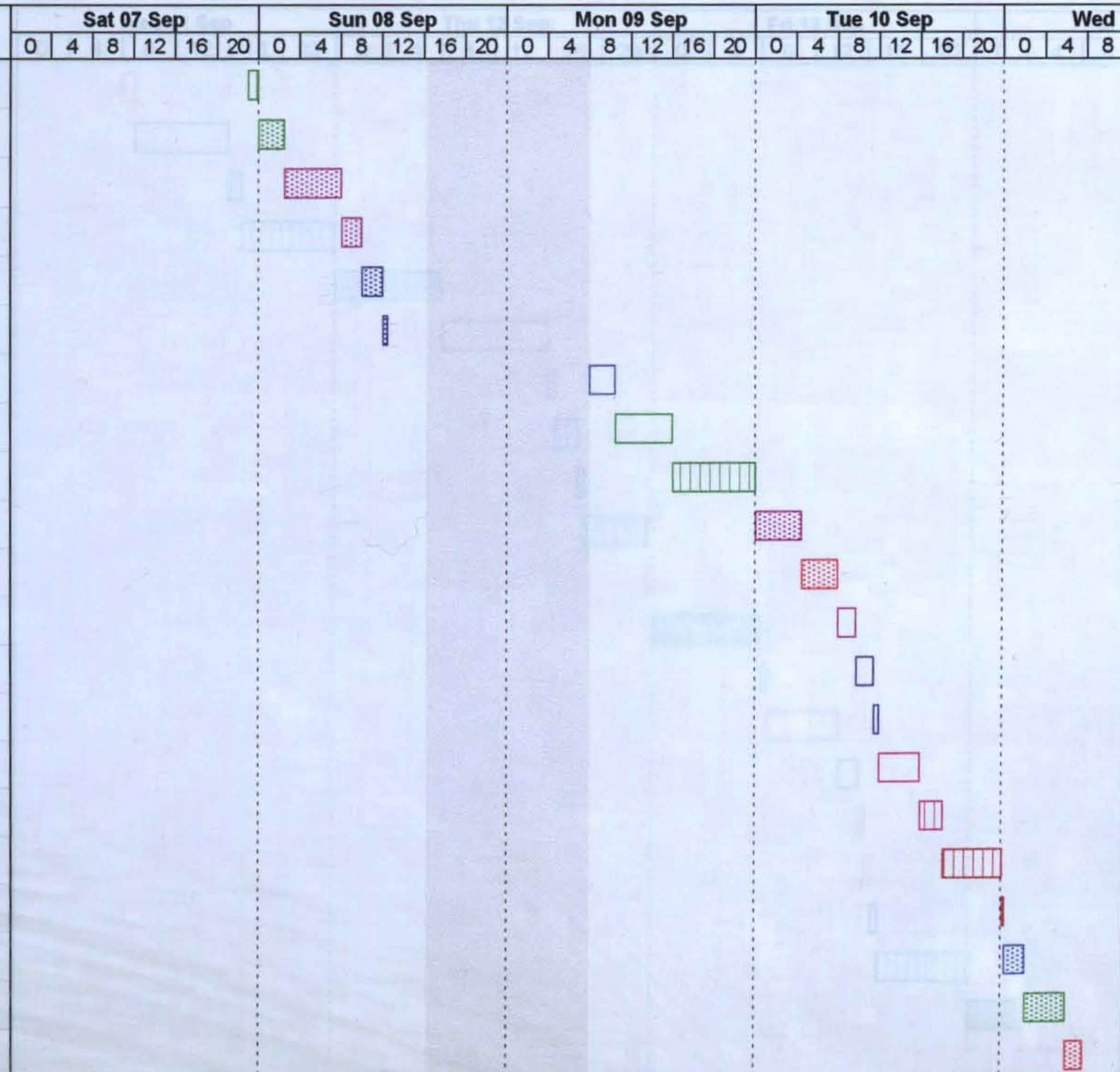
ID	Task Name	Duration	Shift
1	Service pipe installation	4,5h	D
2	Mucking	3,5h	D
3	Mucking	5h	A
4	Problem with cryderman	1,5h	A
5	Mucking	1,5h	A
6	Problem with drill jumbo	4,75h	N
7	Drilling	3,25h	N
8	Drilling	5h	D
9	Charging	2h	D
10	Waited for shotfirer	1h	D
11	Blasting	0,5h	A
12	Mucking	5,5h	A
13	Preparations for concreting	2h	A
14	Preparations for concreting	8h	N
15	Waited for concrete	1h	D
16	Concreting - Pour no:12, L:1019.2	5,5h	D
17	Service pipe installation	1,5h	D
18	Service pipe installation	2h	A
19	Mucking	6h	A
20	Mucking	4,5h	N
21	Drilling	3,5h	N



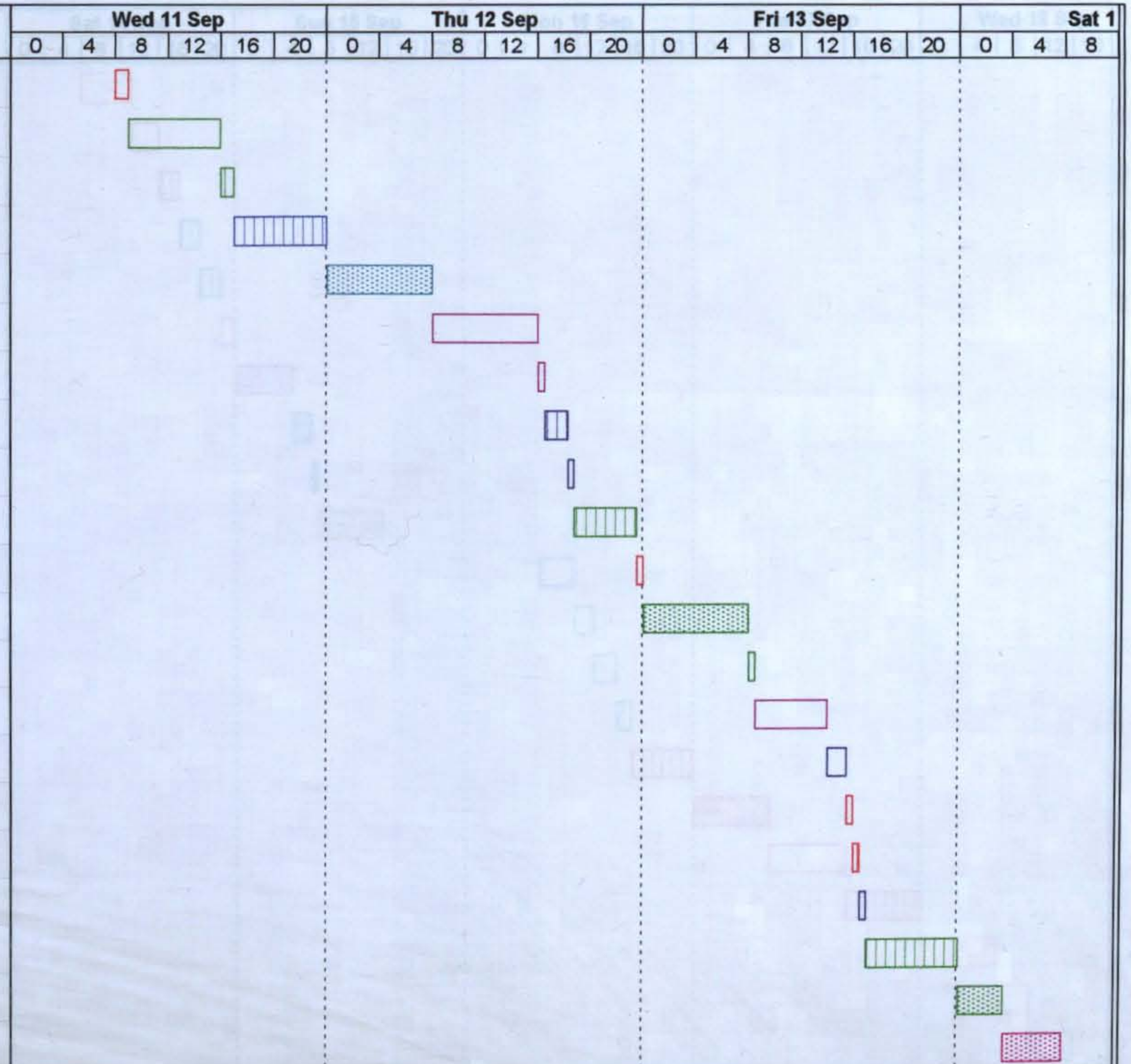
ID	Task Name	Duration	Shift
1	Drilling	4h	D
2	Charging	1,75h	D
3	Blasting	0,5h	D
4	Shotcreting	1,75h	D
5	Shotcreting	2h	A
6	Mucking	3,75h	A
7	Problem with cryderman	2,25h	A
8	Mucking	8h	N
9	Mucking	2h	D
10	Drilling	6h	D
11	Charging	2h	A
12	Waited for shotfirer	1,5h	A
13	Blasting	0,5h	A
14	Mucking	4h	A
15	Mucking	2h	N
16	Problem with rider	1h	N
17	Mucking	5h	N
18	Preparations for concreting	7h	D
19	Concreting - Pour no:13, L:1014.2	1h	D
20	Concreting	5h	A
21	Service pipe installation	2h	A



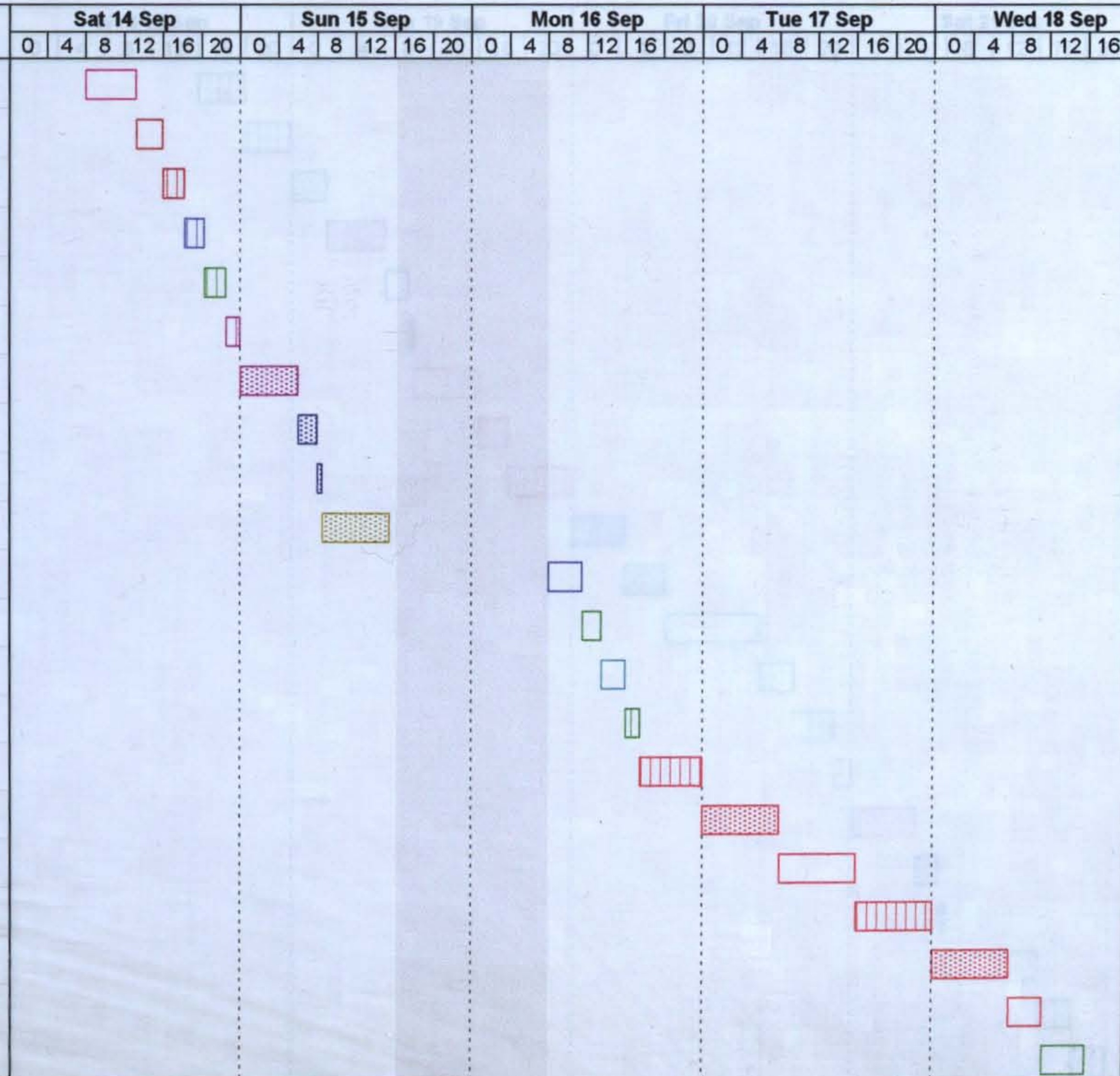
ID	Task Name	Duration	Shift
1	Mucking	1h	A
2	Mucking	2,5h	N
3	Drilling	5,5h	N
4	Drilling	2h	S
5	Charging	2h	S
6	Blasting	0,5h	S
7	Service pipe installation	2,5h	D
8	Mucking	5,5h	D
9	Mucking	8h	A
10	Drilling	4,5h	N
11	Problem with drill jumbo	3,5h	N
12	Drilling	1,75h	D
13	Charging	1,75h	D
14	Blasting	0,5h	D
15	Preparations for concreting	4h	D
16	Preparations for concreting	2,25h	A
17	Concreting - Pour no:14, L:1009.2	5,75h	A
18	Concreting	0,25h	N
19	Service pipe installation	2h	N
20	Mucking	4h	N
21	Problem with cryderman	1,75h	N



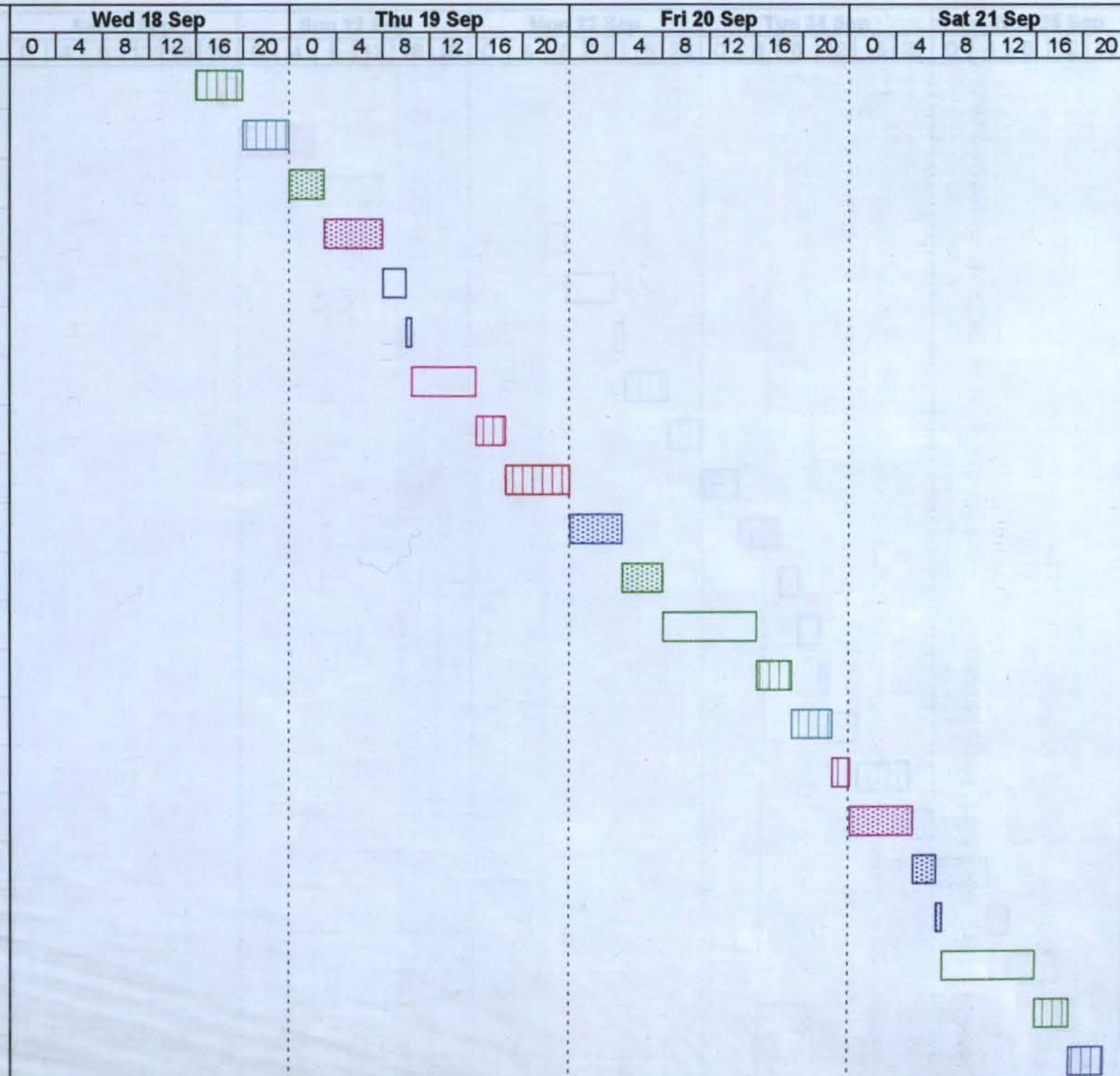
ID	Task Name	Duration	Shift
1	Problem with cryderman	1h	D
2	Mucking	7h	D
3	Mucking	1h	A
4	Service pipe installation	7h	A
5	Grouting and cleaning	8h	N
6	Drilling	8h	D
7	Drilling	0,5h	A
8	Charging	1,75h	A
9	Blasting	0,5h	A
10	Mucking	4,75h	A
11	Problem with cryderman	0,5h	A
12	Mucking	8h	N
13	Mucking	0,5h	D
14	Drilling	5,5h	D
15	Charging	1,5h	D
16	Waiting for shotfirer	0,5h	D
17	Waiting for shotfirer	0,5h	A
18	Blasting	0,5h	A
19	Mucking	7h	A
20	Mucking	3,5h	N
21	Preparations for concreting	4,5h	N



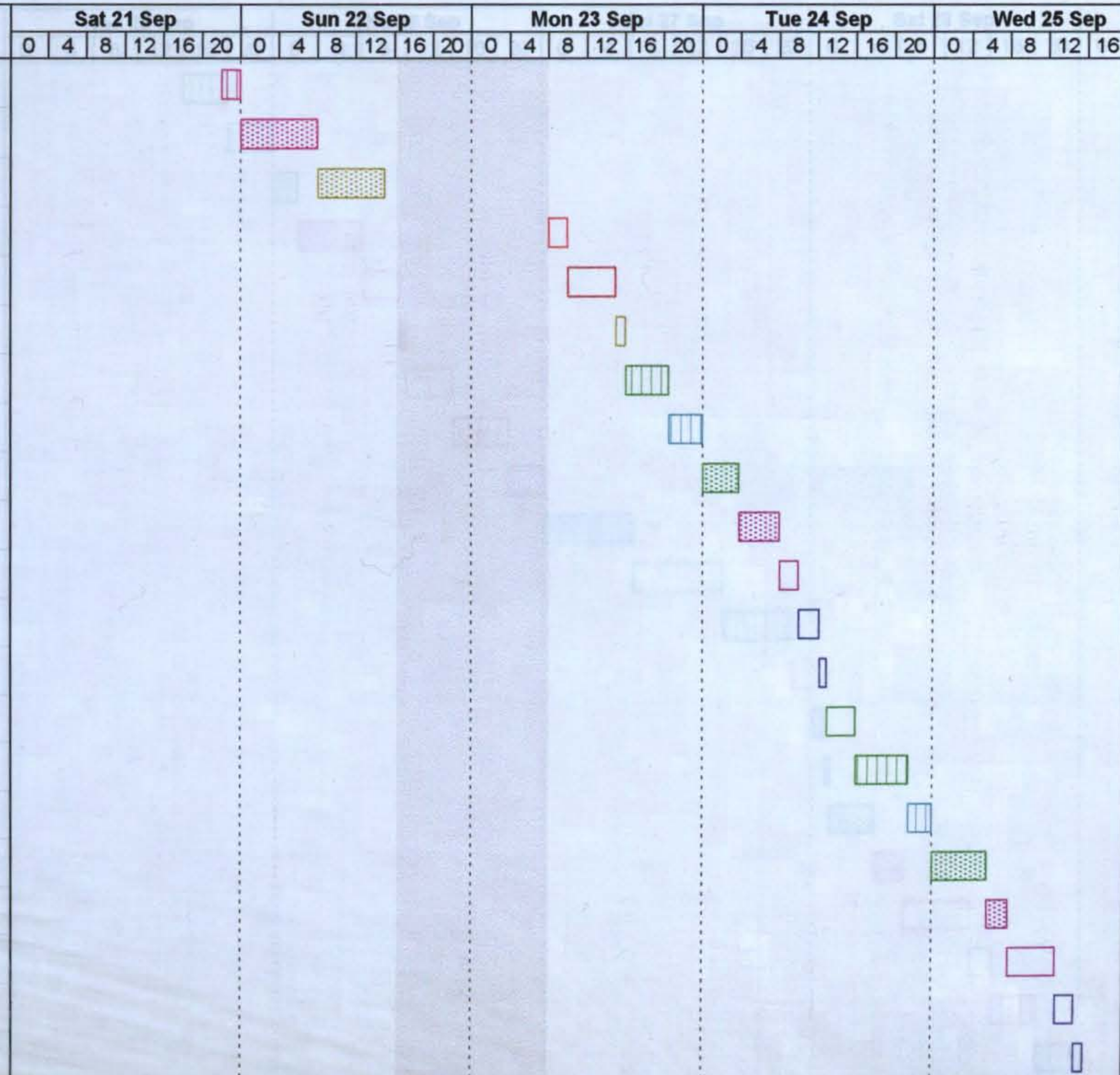
ID	Task Name	Duration	Shift
1	Preparations for concreting	5,25h	D
2	Concreting - Pour no:15, L:1004.2	2,75h	D
3	Concreting	2,25h	A
4	Service pipe installation	2h	A
5	Mucking	2,25h	A
6	Drilling	1,5h	A
7	Drilling	6h	N
8	Charging	2h	N
9	Blasting	0,5h	S
10	Maintenance and cleaning	7h	S
11	Service pipe installation	3,5h	D
12	Mucking	2h	D
13	Shotcreting	2,5h	D
14	Mucking	1,5h	A
15	Problem with rider	6,5h	A
16	Cleaning and repairing works	8h	N
17	Cleaning and repairing works	8h	D
18	Cleaning and repairing works	8h	A
19	Cleaning and repairing works	8h	N
20	Repairing works	3,5h	D
21	Mucking	4,5h	D



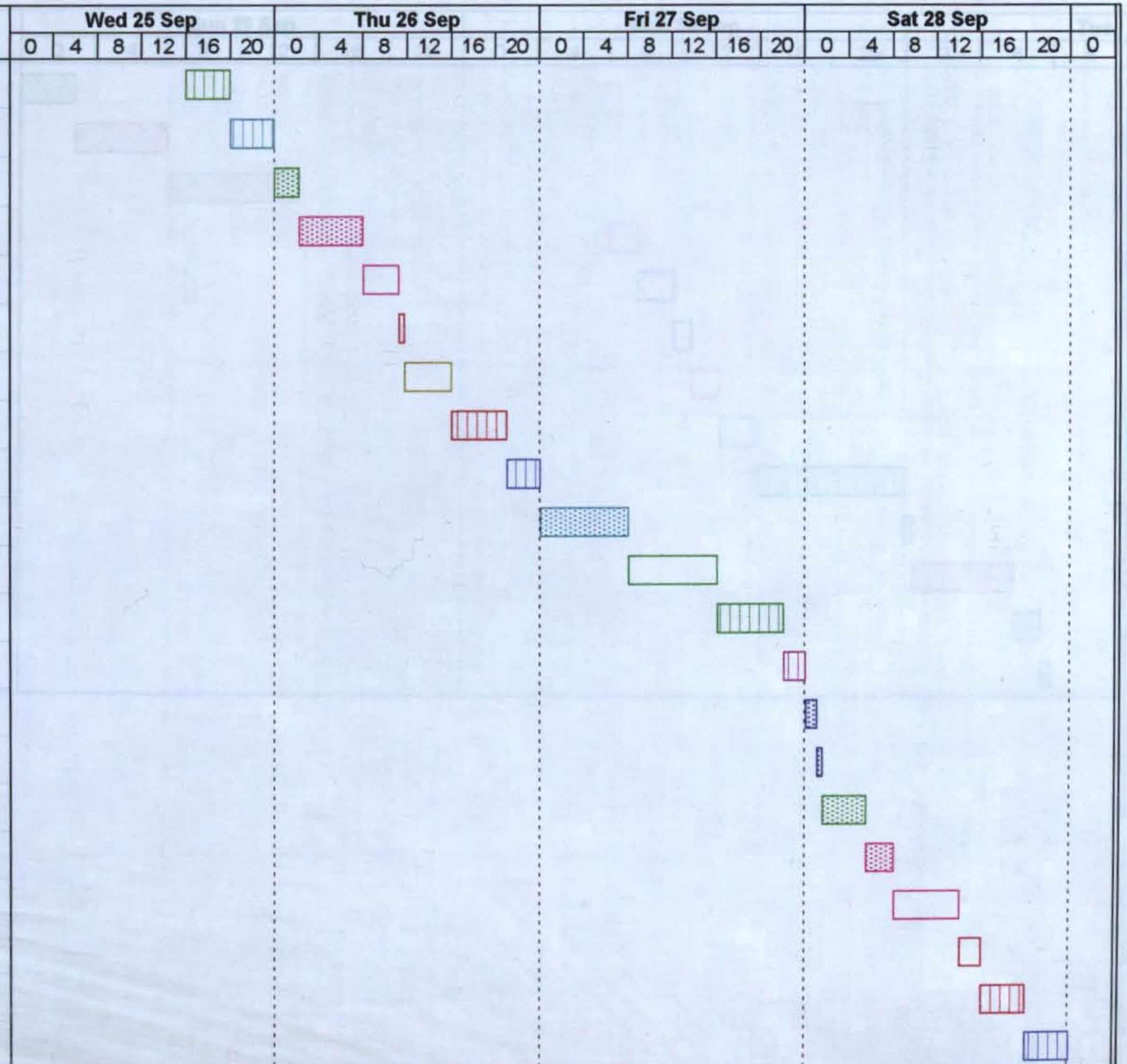
ID	Task Name	Duration	Shift
1	Mucking	4h	A
2	Shotcreting	4h	A
3	Mucking	3h	N
4	Drilling	5h	N
5	Charging	2h	D
6	Blasting	0,5h	D
7	Preparations for concreting	5,5h	D
8	Preparations for concreting	2,5h	A
9	Concreting - Pour no:16, L:999.2	5,5h	A
10	Service pipe installation	4,5h	N
11	Mucking	3,5h	N
12	Mucking	8h	D
13	Mucking	3h	A
14	Shotcreting	3,5h	A
15	Drilling	1,5h	A
16	Drilling	5,5h	N
17	Charging	2h	N
18	Blasting	0,5h	N
19	Mucking	8h	D
20	Mucking	3h	A
21	Service pipe installation	3h	A



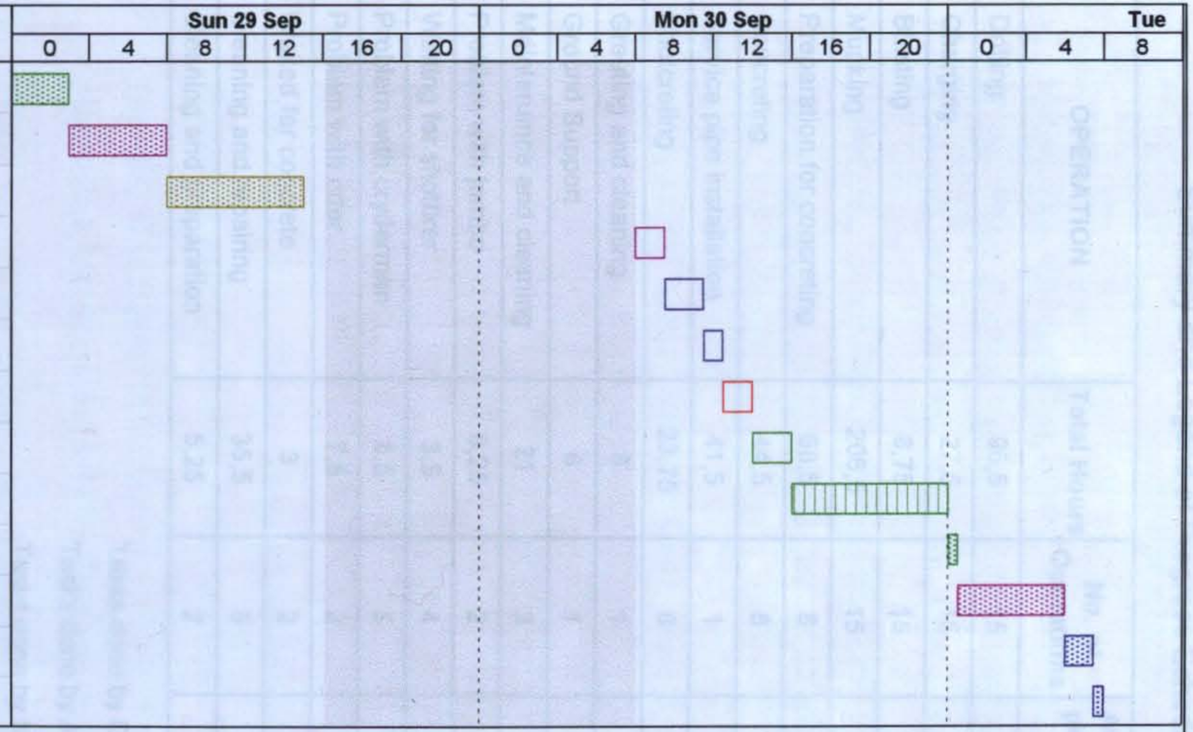
ID	Task Name	Duration	Shift
1	Preparations for concreting	2h	A
2	Preparations for concreting	8h	N
3	Maintenance and cleaning	7h	S
4	Waited for concrete	2h	D
5	Concreting - Pour no:17, L:994,2	5h	D
6	Cleaning and preparations	1h	D
7	Mucking	4,5h	A
8	Shotcreting	3,5h	A
9	Mucking	3,75h	N
10	Drilling	4,25h	N
11	Drilling	2h	D
12	Charging	2,15h	D
13	Blasting	0,75h	D
14	Mucking	3h	D
15	Mucking	5,5h	A
16	Shotcreting	2,5h	A
17	Mucking	5,75h	N
18	Drilling	2,25h	N
19	Drilling	5h	D
20	Charging	2h	D
21	Blasting	1h	D



ID	Task Name	Duration	Shift
1	Mucking	4h	A
2	Shotcreting	4h	A
3	Mucking	2,25h	N
4	Preparations for concreting	5,75h	N
5	Preparations for concreting	3,25h	D
6	Concreting - Pour no:18, L:989.2	0,5h	D
7	Cleaning and preparations	4,25h	D
8	Concreting	5h	A
9	Service pipe installation	3h	A
10	Ground supporting	8h	N
11	Mucking	8h	D
12	Mucking	6h	A
13	Drilling	2h	A
14	Charging	1h	N
15	Blasting	0,5h	N
16	Mucking	4h	N
17	Preparations for concreting	2,5h	N
18	Preparations for concreting	6h	D
19	Concreting - Pour no:19, L:984.2	2h	D
20	Concreting	4h	A
21	Service pipe installation	4h	A



ID	Task Name	Duration	Shift
1	Mucking	3h	N
2	Drilling	5h	N
3	Maintenance and cleaning	7h	S
4	Drilling	1,5h	D
5	Charging	2h	D
6	Blasting	1h	D
7	Problem with cryderman	1,5h	D
8	Mucking	2h	D
9	Mucking	8h	A
10	Mucking	0,5h	N
11	Drilling	5,5h	N
12	Charging	1,5h	N
13	Blasting	0,5h	N



Summary and Legend of Progress Schedule of September

OPERATION	Total Hours	No. of Operations	Average Time per Operation (hr)	Color represents the task
Drilling	96,5	15	6,43	Violet
Charging	27,5	15	1,83	Dark blue
Blasting	8,75	15	0,58	Dark Blue
Mucking	206,5	15	13,77	Green
Preparation for concreting	68,5	8	8,56	Pink
Concreting	44,5	8	5,56	Brown
Service pipe installation	41,5	1	41,50	Blue
Shotcreting	23,75	8	2,97	Light blue
Grouting and cleaning	8	1	8,00	Light blue
Ground Support	8	1	8,00	Light blue
Maintenance and cleaning	21	3	7,00	Yellow
Problem with jumbo	8,25	2	4,13	Red
Waiting for shotfirer	3,5	4	0,88	Red
Problem with cryderman	8,5	5	1,70	Red
Problem with rider	7,5	2	3,75	Red
Waited for concrete	3	2	1,50	Red
Cleaning and repairing	35,5	5	7,10	Red
Cleaning and preparation	5,25	2	2,63	Red

Tasks done by Day shift

Tasks done by Afters shift

Tasks done by Night shift

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