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**Office of Waste Isolation
Union Carbide Corporation
Nuclear Division
Oak Ridge, Tennessee**

Update of 1972 Status Report on Deep Shaft Studies

Prepared by
Parsons Brinckerhoff Quade & Douglas, Inc.
New York

September 1976

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SHAFT SINKING STATE OF THE ART

S U M M A R Y

1. This report on the state of the art of shaft sinking considers the effects of the geology, discusses factors affecting shaft size, describes the conventional shaft sinking techniques and the newer mechanized methods, describes several representative or difficult shafts, and briefly comments of certain long term problems and solutions.
2. The best geological site for a shaft has competent, medium strength rock that does not overbreak during excavation nor creep under long term load, is overlain by a minimum of overburden, and has a minimum of water in both the overburden and bedrock strata.
3. The major factor in determining rock competence for shaft sinking is not the strength of the rock, but the discontinuities of the rock mass -- the bedding planes, foliation, and the joint and fracture systems.
4. Of the four geologies considered, the best for shafts are the massive granitics and non-cavernous and non-fissured limestones; the poorest are the shales, particularly those that overbreak badly or cave, and the salts, because of the potential for damaging differential settlements.
5. When a new development is planned and there are special project requirements such as an absolutely watertight lining and prevention of water migration along the outside of the shaft, the use of one of the smaller shafts as a pilot project to obtain detailed geotechnic and rock mechanics data and to demonstrate that project requirements can be met is strongly recommended.
6. Factors affecting the number and size of shafts include the need for alternate means of emergency egress, the amount of ventilation required, the daily rate of mining (tunnel) production, the separation of personnel and material travel routes in high production projects, the type and size of lining, shaft equipment, mining equipment, and the shaft construction method employed.
7. Ventilation must meet code requirements based on square footage of mine (tunnel) headings, on provision for men present and equipment used, on minimum velocities, and on the type of material being excavated. In addition, temperatures in work areas must be kept within reasonable limits, necessitating calculation of all the heat inputs from lighting, powered equipment, and other sources. Ambient rock temperature must be determined as it also affects the ventilation requirement. In extreme cases, it has even been necessary to install mechanical cooling equipment.

8. Costs of shafts vary widely. The major factor is the type of geology that must be traversed in reaching the working levels; the ground-water conditions are particularly important. Costs are not directly proportional to the amount of material excavated for the shaft. There are definite economic sizes for each excavation method; shafts slightly larger or smaller can be constructed for essentially the same costs, but as the deviation increases, so does the rate of cost increase until the ratio of costs of two sizes approximates the ratio of the diameters, rather than the areas, of the shafts. Similarly, shallow shafts are very expensive on a per foot of depth basis; as the depth increases, the per foot cost becomes proportionately less expensive until a constant per foot cost is reached. At very substantial depths, the costs of vertical transport becomes a major factor.
9. The only shaft construction method capable of meeting all geological conditions is the tried and true "conventional" system. The shaft is constructed to full diameter in a single pass, excavating below a multi-deck working stage, hauling muck to the surface in a pair of bucket skips on separate wire rope guides, placing forms and pouring concrete walls on intermediate decks, grouting as necessary and finally partially equipping (pipe lines) from the upper deck. Minimal to moderate ground water is handled by sumping, moderate to heavy quantities by grouting directly ahead of the excavation, and extreme quantities in non-groutable formations by a separate freezing system outside the shaft installed in advance, generally from the surface.
10. The shaft lining most frequently used is concrete; the minimum thickness is generally 12 inches. In sound rock, only a thin shotcrete lining may be applied to prevent spalling. Segmental cast iron linings formerly were the standard solution to serious groundwater conditions; today, welded steel linings, sometimes a double steel lining with a concrete infill, are standard where the concrete lining is unsuitable.
11. Major recent developments are all continuations of the trend to mechanization. The use of raise boring has increased dramatically in the last five years. Substantial advances in large hole blind drilling capability have also occurred. Down hole boring machines are nearer the time of practical application.
12. Conventional shaft sinking advance rates can be increased dramatically by using an advance pilot hole and an adjacent small diameter shaft -- both on the order of six to eight feet in diameter -- for muck handling. The muck is dropped down the "glory-hole" pilot, transferred through a cross drift to the other shaft and raised to the surface in that shaft. This permits both a full face advance and greater round lengths (depths); each significantly reduces sinking time.

13. The first shaft of a development project must be sunk either conventionally or by blind drilling. The combination procedure described in the previous paragraph is excellent for expediting and reducing the cost of the principal shaft. It also permits necessary pre-production development work at the mining level to proceed concurrently with the principal shaft excavation, shortening the time to the beginning of mine production.
14. Where the development requires several shafts and the geology is favorable (stable rock, not too much water) raise boring is the most effective method of expediting construction of all but the principal shaft.
15. Where the geology is both favorable -- rock in the lower strength range -- and unfavorable --unstable overburden and/or heavy water inflow in either overburden or rock -- and the shaft diameter moderate, the safest procedure is to blind drill the shaft using drilling mud to stabilize the walls and floating in a steel lining.
16. Where differential movements between shaft and overburden are anticipated, a carefully designed bitumen outer lining will protect the shaft. This also is an effective water sealing procedure.
17. If absolute prevention of water reaching the mining levels is essential to a project and if grouting cannot guarantee this condition, picotage or keilkranz must be used on one or more levels to provide a positive shutoff for water migrating down the outside wall of the shaft.
18. An observation chase cut in the bed rock outside the shaft and well below the lowest water infiltration area and above the highest storage level provides a safety check, assuring no vertically migrating fluid can reach the mining area as a result of shaft construction.
19. Unlike above ground structures in seismic zones, earthquake hazards to shafts are minimal (unless the shaft is in the rift zone) because the deformation of the shaft is constrained by the ground to very small movements in relation to shaft depth which create only minor stresses in the lining.

CHAPTER 1

INTRODUCTION

A. General

The current state-of-the-art in shaft sinking is described in this report. The intent is not to provide a technical compendium for those already knowledgeable but rather to set forth the various problems which may be encountered and construction methods which can be used in such a manner that those unfamiliar with shaft sinking will gain understanding and insight into the subject.

This report is not site specific since four bedrock geologies are to be considered. These are salt beds, shale deposits, non-cavernous limestone formations, and igneous (granite) rocks. Therefore, to keep the report within reasonable limits, the emphasis is on problems and procedures common to all, with differences due to specific geologies noted only where pertinent.

The excavation by mining of approximately 12,000 tons of rock per day is one facet of the study of which this report is a part. The tunnels being considered are horseshoe shaped, i.e., an inverted U in cross section, about 18 feet in height and width. The production rate and tunnel cross section combined require mining approximately 500 lineal feet of tunnel a day, a very substantial undertaking. The resulting ventilation and muck (excavated material) transport requirements call for large diameter principal shafts and smaller diameter auxiliary shafts.

B. Scope of Investigation

Originally, the scope of work indicated this report was to cover the following:

1. Review of an earlier report on deep shafts.
2. Developments since the date of that report (1972).
3. The effects of the four geologic bedrocks on shaft construction.
4. Problems in reaching the operating levels through the overburden.
5. The effects of earthquakes.
6. The types and spatial requirements of shafts for various functions.
7. Geotechnical investigations required prior to initiation of shaft design.

Later, informal discussions indicated a state-of-the-art and associated problems approach was desired.

It has been necessary, for conciseness, to confine the discussion of geology to generalization indicating favorable and unfavorable characteristics as they apply to shafts, citing the specific characteristics of the bedrocks only where major and pertinent differences occur among the four types considered. Unfavorable ground conditions, especially ground water, are discussed in detail under construction techniques rather than in the chapter on geology.

Among the factors affecting shaft sizing, ventilation and materials transport are the most important. The former is common to all underground developments and has been emphasized, while material transport, being a function that varies in effect with each project, has not been covered in detail. Since absolute costs are primarily a function of the specific geology encountered, the discussion of this subject has been limited to indicating how costs vary relatively with diameter and depth.

Non-routine procedures such as ground freezing and the use of bitumen linings have been considered in some detail, the former to illustrate how the worst of conditions may be handled, the second because of its importance in certain geologies where differential movements may occur during service life.

Geotechnic investigation must be tailored to the specific geology of the shaft site; hence it has not been discussed.

CHAPTER 2

GEOLOGY

Shaft sinking in salt, shale, non-cavernous limestone, and granite is considered here. Each has its own characteristics. Unfortunately, the range of these characteristics is sufficiently broad that the best of the worst classification may be better than the worst of the best classification. Furthermore, the presence or absence of water may drastically alter the desirability of a given classification. (For example, the mining characteristics of a fine sand may range from bad to good to terrible as the water content increases from dry to moist to saturated.) In addition, the overburden characteristics of a specific site may be of far greater importance to the shaft sinker than the bedrock characteristics. These considerations lead inevitably to the conclusion that making generalizations in a short discussion of alternative geologies may be misleading and a disservice while to treat the subject adequately to avoid misleading the reader would require a massive treatise entirely out of keeping with the purpose of this report. Having warned the reader, generalizations will now be made.

1. Any meaningful discussion of geology relating to constructability and economics of shafts must be site specific, i.e., based on a comparison of specific sites, and not generalized.
2. The more competent the material, the better the potential for shaft sinking.

Competence is considered here as meaning the ability to be self supporting. Competence decreases as the requirement for support increases and relates more to the mass rock characteristics than the intact rock (laboratory sample) properties. In other words, a massive, lower strength material that does not come apart is better than a high strength material that is in small pieces or is separated by openings that permit pieces to fall, and to continue to fall, into the shaft or tunnel. Competence is most commonly a function of the joint and fracture systems in the rock.

3. Moderate strength competent materials are frequently superior to high strength ones.

A greater range of excavation techniques are available in the moderate compressive strength rocks than in the very hard rocks. This will be apparent from the discussions on conventional and mechanical shaft sinking techniques.

4. The less the overbreak characteristics, the better the geology.

If an otherwise competent material overbreaks extensively, the excavation must proceed more carefully and much greater quantities of concrete will be required in the permanent concrete lining, thereby increasing costs. It is worth noting that, even in good material, the design requirement for a minimum 12-inch lining may actually require concrete equivalent to a 22-inch lining. Substantial overbreak could well double this. Unfortunately, one characteristic of many shales is the tendency to overbreak badly and to cave as well. Obviously, this influence the excavation technique to be selected.

5. The less water encountered, the better the geology.

Water is always a problem in shaft sinking. It can be handled, but it has a pronounced effect on costs. The cost may be nominal as in the case when only pumping from a sump in the shaft is required. When the water supply is unlimited in an unstable material, freezing may be the only solution; the cost then can approach the astronomical. In the middle is the geology wherein grouting can effect a sufficient shutoff; the cost similarly is in the mid-range. Unfortunately, few shafts do not encounter water problems somewhere; fortunately, very few encounter exceedingly severe situations.

6. The less the unconsolidated overburden, the better the geology.

Construction of a shaft is generally appreciably more difficult in unconsolidated material, i.e., soil or soft ground, than in rock. If the unconsolidated material is too deep or too unstable to permit excavating an open pit to the top of the rock, temporary support in the form of a ring of sheet piling or the use of segmental steel liner plate rings will be required. In particularly adverse conditions, concrete walls tremied in a circular, slurry filled trench or installation of a freeze ring may be required prior to beginning excavation; both are expensive operations.

7. The fewer adverse rock strata above the mining level, the better the geology.

Unfavorable rock strata such as heavily water bearing aquifers or unstable ground (crushed zones, "squeezing ground" that inexorably expands into the shaft after excavating, etc.) are difficult and expensive to penetrate. Pregroutng the troublesome zone is one procedure; sometimes, however, it sometimes is necessary to install a freeze ring more than 1,000 feet below the surface to stabilize the formation.

8. The better the long term characteristics, the better the geology.

If one characteristic of the bedrock geology is long term creep under low or moderate load, the gradual settlement of the shaft foundation can lead to distress in the shaft. Salt, unfortunately, has this characteristic. This general statement assumes the overlying geology is such that it does not permit an effective tying-off or intermediate founding of the shaft.

9. The higher the long term modulus of the formation being mined, the better the geology.

The lower the long term modulus is, the greater the areal settlement resulting from extensive mining operations. (Creep and long term modulus, while related, are considered separate characteristics here.) This areal settlement in turn can create differential settlement between the shaft and the surrounding materials, again leading to distress in the shaft. However, if too much material is removed from even a high modulus medium, i.e., if multiple tunnels are too closely spaced, areal settlement and resulting problems can occur.

10. For ease, safety, and performance of the shaft, the preferred geology consists of a minimum, relatively dry overburden overlying massive, non water-bearing limestones or moderately jointed to massive granitics with the joints and fractures closed or healed, both geologies to be in the moderate strength range. Certain of the shales will cause excessive problems. Salts also will create problems which must be reflected in the shaft design.

CHAPTER 3

SHAFT SIZING

The size of a shaft depends almost exclusively on its intended use, i.e., whether it is intended solely for ventilation, services, personnel, or materials haulout, or some combination of these. A secondary consideration is the construction method, and this may result either in some minor change of size or in revision of the arrangement of the shafts.

A. Research and Development

More than one shaft is normally required to develop a new mining project. However, the desire to begin production as early as possible with a minimum capital outlay generally results in the largest, or production, shaft being constructed first. If the geology is not well known or unfavorable, the shaft sinker must add a sizeable contingency in his bid, increasing costs. When early return on invested capital is not the primary criterion, a different approach may result in better and more economical construction.

All of the technical data required for confirmation (or alteration) of the main shaft design best can be obtained from a sizable pilot project. Since at least four shafts are required for the development being studied, this pilot project can and should be one of the smaller shafts. The in situ rock response to an excavation more than 15 feet in diameter cannot be determined with sufficient accuracy from a borehole core only a few inches in diameter. The time and distance-related effects of the elastic rock deformation, any inelastic deformation, and the excavation procedure minimizing and preferably eliminating even minute wall cracking, all must be ascertained. If the shaft passes through one or more strata containing water under high pressure, a demonstration of the ability to construct a sufficiently watertight lining might be a project requirement. If so, the ability to effectively seal off water migration from the aquifer downward alongside the shaft also should be demonstrated. All evidence to date indicates the demonstrations of construction ability will be successful using techniques already proven. These demonstrations obviously require time to perform.

B. Safety

One of the principal reasons for constructing at least two shafts in any sizeable underground project is safety. Practically all, if not all, mining laws require two shafts for safety considerations. The requirement is sometimes waived temporarily during development work on the basis of both cost and the fact that the existence of more than one development level provides safe refuges within climbing distance in the early stages of a project. Eventually, however, the second shaft must be constructed.

The requirement for more than one escape route, however, does not extend to the construction of individual mining-level tunnels. It is accepted that only one route is required from a working area (heading) to a trunk line tunnel; it is the collective egress from the deep working levels to the ground surface that must be duplicated.

C. Ventilation

The use of two or more ventilation shafts always should be considered for any sizeable mining project. Mines frequently add shafts for the sole purpose of increasing ventilation capacity. In fact, some of the present development in "big-hole" drilling can be traced to this need. Shafts having a lined diameter of six to eight feet normally suffice for this purpose. Ventilation of extensive underground workings is greatly simplified, improved and costs reduced by providing at least two shafts in the ventilation system.

The ambient rock temperature underground is related to the average annual surface temperature and increases with depth of shaft. About one degree increase per 100 feet of depth can be assumed if more definitive information is not available. This can greatly increase ventilation requirements if the project is located in certain areas in the United States. For example, ambient temperatures of 97° F have been recorded 2,000 feet below the surface in the South Carolina - Georgia area. Another ventilation factor is the type of material being mined. Ventilation in uranium mines recently has been increased drastically because of the silicosis problem.

Tunnel safety orders require an air supply of from 50 to 100 cubic feet per minute (cfm) per square foot of heading at the working face, or a minimum of 200 cfm per person plus (typically) 75 cfm per diesel horsepower used, or an overall air velocity of not less than 30 feet per minute (fpm) where drilling, blasting, or other activities produce dust, fumes, or similar pollutants in harmful quantities. Assuming for this discussion that two 80-horsepower muck hauling units will be required, for the 325 square foot tunnels considered, that only one heading at a time will be active, and that no more than 30 men will be underground at one time, these areal, personnel, and velocity criteria result in minimum theoretical volumes of from 10,000 to 32,000 cfm, without any consideration of heat buildup, which, as will be shown, could be intolerable.

If an open plenum exhaust velocity in the shaft of 20 miles per hour, or 1,750 feet per minute, is used, the required cross sectional area is 10 square feet for supply and the same for exhaust if the men and machines criteria apply and 19 square feet if the larger of the heading area criteria applies. If 500 feet of tunnel per day is required and the advance rate is 80 feet per day per tunnel, each shaft area must be increased six-fold. The required shaft diameter for combined supply and exhaust would be 12.5 for the smaller requirement, 19 for the larger. Using separate

shafts, the requirements would be 9 and 13.5 feet, respectively. If the tunnels are spread over a wide area, it might be more efficient and economical to use two exhaust shafts; these would have diameters of 6 or 9.5 feet. These sizes do not consider area reduction due to equipage, the 2000-foot elevation change, or heat inputs from tunnel lighting, equipment, transportation, or excavation, or from the supply fans themselves.

The ventilation requirements set forth by various agencies vary greatly. Therefore, the controlling codes and laws should be ascertained at an early stage in design. Factors other than code requirements, however, will undoubtedly govern.

There has been insufficient time in this study to make a detailed analysis of ventilation requirements. Such a study would have had little validity in any event considering this wide range of variables that could be encountered within the United States. Some idea of what could be required, however, can be obtained from such an analysis made for a similar facility in the 97° F ambient rock temperature cited previously. The equivalent of two of the 325 square-foot headings were to be advanced 40 feet per day. The distance from surface to furthest heading was limited to 6,000 feet. All construction heat load sources was considered in addition to the personnel and equipment ventilation requirements. It was considered necessary to install a chilled air system with a refrigeration capacity of 500 tons to chill the air to a 30° chill relative to the rock. Even with the substantial chilling, the ventilation requirements were approximately 180,000 cfm. Using air velocities of 1,750 fpm to permit shaft use for other functions--access and muck transport--required separate shaft areas in excess of 100 square feet for ventilation air alone.

If this requirement is applied to 6 simultaneous 80 feet per day headings, and if 100 square feet is added for equipage--materials skips, personnel hoists, partitions, ladders, pipelines, electrical conduits, etc.--cross sectional areas on the order of 400 square feet for separate shafts are required. The corresponding shaft lined diameters are 23 feet and 32 feet, respectively. Since the largest shaft now known is elliptical with diameters of 33'-6" and 35'-0", the advisability of multiple shafts become obvious.

An extreme case has been cited above, admittedly. The flow velocities could be doubled, halving the area requirements, but then neither shaft could be used for access. Elimination of the chilled air plant makes the analysis applicable to a greater geographic area.

D. Other Size Considerations

Continuing the minimum two-shaft concept, one shaft would supply air and also provide personnel access to permit maximum efficiency in the exhaust/materials shaft. The materials shaft must be sized for two spoil or materials skips or buckets operating simultaneously to handle the 500 tons per hour resulting from excavating 500 feet of tunnel per day, since 12 trips per hour is a practical maximum for a 2000-foot shaft.

A 4.2 cubic yard muck bucket 5.0 feet in diameter and 5.8 feet deep is a standard shaft sinking unit. Buckets mounted on wire rope guides should have at least 12 inches clear all around. This results in a minimum operating diameter of 14 feet, which is also considered by shaft sinking specialists to be the most economic shaft size in cost per lineal foot.

If a full face tunnel boring machine on the order of 22 feet in diameter is used for tunnel excavation to circumscribe the planned 18 x 18 horseshoe section in an effort to meet planned production notes the materials shaft must be capable of lowering large subassemblies. A 14-foot shaft can handle at most a unit 9.5 feet square or 6 by 12 feet in cross section, whereas units 11 feet square or 6 by 14 feet can be handled in a 16-foot shaft. While exact dimensions cannot be established without further investigation of both rock properties and equipment design, there is a possibility that a shaft even larger than 16 feet in diameter will be necessary to provide for such a machine.

In sound rock, a permanent concrete lining 12 inches thick will normally suffice. Thus two feet should be added to lined diameters to arrive at excavated diameters. If the water problem is severe or the ground bad, adding four feet (24-inch lining) is more appropriate.

When planning large shafts, it should be remembered that only conventional techniques, described in the following chapter, can be used.

Mechanized equipment, described in the second chapter following, has potential advantages in time and cost but, with a few exceptions is limited to the smaller diameters. Raise boring (excavating a shaft from the bottom up -- which requires an existing shaft and development level) has been used only to the 15-feet and under diameters; the maximum possible with existing units is 20 feet. Blind-hole drilling (blind, or "big-hole" drilling uses a large diameter bit or cutter to drill from the surface without benefit of a guide hole, using technique similar to oil-well drilling), if it is required, should be considered only for holes less than 12 feet in diameter in relatively soft rocks; a 16-foot diameter might be possible under certain conditions. Down-hole boring equipment is presently limited to 12 feet or less, although 1977 may see a doubling of that capability. These limitations of techniques should be borne in mind when sizing and costing shafts.

E. Costs

The costs of shafts vary widely. The principal variables affecting costs are geologic--the type and specifics of the bedrock, the type and amount of overburden, and the groundwater conditions. Poor geology can easily double or triple the cost of a shaft in favorable ground. Since most shafts are constructed for private businesses, cost information is rarely included in specific project descriptions in the technical press. Two such reports, however, indicate the great range in costs. A shaft completed in 1976 near Lafayette, La., was reported to cost \$3,000,000.⁽¹⁾ Driven into a salt dome, it was only 244 feet deep and had a lined diameter of only eight feet; see "Representative Projects" in the next chapter for further details. In contrast, a shaft for 500 feet deep and 28.5 feet in lined diameter constructed in 1964 for the National Aeronautics Space Agency in Cleveland, Ohio, cost only \$800,000.⁽²⁾

In contrast, the variables which man controls--size, depth, use--change the shaft costs only a small fraction as much as the geologic variables. A shaft with a lined diameter of 20 feet and depth of 2,000 feet will quite likely cost between, say, three and six million dollars at today's prices, depending on the geology. Adding or deducting 1,000 feet of depth quite likely would change the cost by only one to two millions. The high cost of setting up the shaft sinking operation is indicated by a cost of a shaft only 300 to 500 feet deep also costing say one to two million dollars.

The minimum economic diameter of shafts sunk by conventional methods has already been discussed. The above 20-foot diameter shaft is larger than this size. Increasing the diameter to say 30 feet would likely increase the cost by, say, 50 percent, i.e., the ratio of the diameters rather than the cross sectional areas. Diameter changes near the minimum economic diameter, which is roughly 15 feet, result in a considerably lesser cost change, probably less than half the percentage diameter change.

The use of the shaft, as reflected by its equipping, causes a very minor change in cost for conventional uses. The most pronounced effect of use, however, is if the shaft must be kept perfectly dry and migration of water on the outside of the shaft down to mining level prevented. Such a combination of requirements might increase the cost ten to twenty percent. The hidden factor here is that while the difficulty, and therefore the cost, of meeting these requirements increases rapidly as the groundwater conditions become more severe, these same rock conditions also increase the general shaft construction costs very rapidly. As a result the actual dollar range of the cost of positive water control is much greater than the small percentage range indicates.

It must be emphasized that the above discussion on cost according to diameter, depth, and use is very generalized and is intended only to provide a rough rule-of-thumb comparison for evaluating different diameters, sites, and ground conditions.

The cost of a shaft also depends on project status. Obviously, the first shaft on a project must be sunk conventionally or blind drilled. If blind drilled, the shaft must be relatively small, say less than ten feet in diameter. Subsequent shafts on the project could be raised bored if conditions were suitable. The economy possible and effect on project layout is substantial. A 1973 report on the state of the art in blind and raise drilling,⁽³⁾ indicated that a series of shallow shafts, 300 to 600 feet deep and about 12 feet in diameter, in Western Pennsylvania were sunk conventionally and lined but not grouted for \$500 to \$550 per foot while raise bored shafts were pregrouted, drilled, and lined for about \$400 to \$500 per foot in 50 to 75 percent of the time required for conventional methods. Presumably, multiple use of equipment and crews aided in holding costs down.

CHAPTER 4

CONVENTIONAL SHAFT CONSTRUCTION TECHNIQUES

"Conventional" refers here to the method of shaft sinking that has been practiced for years. It is a method that has proven capable of handling any shaft condition yet encountered. There are other techniques, also proven now, that can or should be used in special instances, and those are discussed in the following chapter.

A. General Procedure

The standard, or conventional, or traditional, method of shaft sinking involves "collaring the hole", erecting a headframe and sinking stage, and proceeding downward using a repetitive cycle of drilling, blasting, and mucking. During drilling and mucking portions of the cycle, concrete forms are set and the walls poured above the excavated area.

The first step in shaft sinking, collaring the hole, consists of removing the overburden to expose the bedrock. Procedures vary according to the specifics of the site. To the shaft sinker, the less overburden, the better. If the overburden is shallow, a large pit with sloping sides is excavated until the bedrock is exposed. Then a circular shaft, generally of reinforced concrete, is formed and poured back up to surface level. Meanwhile, the foundation for the sinking headframe is constructed. Where practical, the foundation will be the same as for the permanent headframe. In other cases, a circular sheet piling cofferdam is driven, the interior excavated, and the collar constructed within its protection. If the overburden is deep, the use of segmental steel liner plate rings may be the appropriate temporary support system; this requires a slow, ring by ring advance through the overburden. Grouting may be required to stabilize the ground. In certain adverse situations, slurry walls (walls with concrete placed by tremie in a circular trench filled with a special fluid to stabilize the trench walls) may be required. In others, a freeze ring is the only practical temporary support. In extreme cases, great depths of overburden today may dictate the shaft sinking method. For example, the shaft may be drilled blind and the casing floated in instead of slowly fighting down through the overburden using soft ground mining techniques. (This is discussed in the following chapter.) After the shaft is collared, the headframe is erected.

The collaring process will normally result in an excavation at least 50 feet deep for a major shaft, even though the overburden may be only five feet thick. This is to provide space within which to erect the multi-level excavating stage and to reeve the supporting wire ropes. Figure 4-1 illustrates one type of collar and excavating process; it, as well as the

Figure 4-1

other illustrations in the chapter, is taken from "Complex Equipment for Sinking and Drilling of Vertical Shafts," by N. A. Malevich.

The headframe will be from 100 to 200 feet high, again depending upon specifics. Sometimes the permanent headframe is constructed, sometimes the permanent housing with temporary hoisting gear installed, and sometimes only a temporary headframe is installed for the equipment. Figure 4-2 shows the rather complex arrangement of winches, hoists, etc. required at a shaft site. (Dimensions thereon are in metric.)

The excavating stage is a multideck affair designed to support several simultaneous operations. In this country, a four-deck Galloway stage is most often used. Figure 4-3 shows a five-deck platform, and most of the equipment required. Figure 4-4 shows the location of the major items of equipment; it is a composite, not all are located on the same deck. Figure 4-5 shows the sequence of operations; the equipment depicted is Russian. Use of a wall protection shield is not common in North American practice, although if concreting is closely following excavating, mats may be used to reduce fly rock and damage to forms.

The work cycle in sinking through rock is the basic mining cycle of drilling, shooting, and mucking. Holes are drilled into the rock at the heading (bottom of shaft) in a pattern calculated to produce the best breakage at least cost. The stage is then raised 50 feet or more, the holes charged with explosive, and the blast initiated. After "blowing smoke" (ventilating), the stage is lowered, and the muck (broken rock) excavated. Two general types of equipment are used, a Cryderman mucker (a clamshell on the end of an extendable arm) or a cactus grab (a relative of the orange peel bucket). The muck is hoisted to the surface in large buckets of 5 cubic yard capacity, or larger. The buckets travel at high speed (3000 fpm) on rope guides. After the bottom is cleaned, the cycle is repeated.

Even though the rock is of good quality, the shaft may be advanced in steps (half-faces) rather than full face (full shaft cross section). That is, there may be a step on the order of 4 feet high between the two benches, (faces), with work alternating between benches. This permits sumping if water is present, it also permits "blowing down" with air to clean the face of rock before drilling begins. This is an important safety feature that helps locate any holes containing unexploded dynamite cartridges. (A 9-foot deep round may be drilled, and 9 feet "pulled"; this lowers the bench, originally 4 feet above the other shaft bench to four feet below the other bench.)

Forming and pouring of the concrete walls proceed simultaneously with the drilling and mucking portions of the cycle, from the upper stage levels. This saves construction time, but, more importantly, if the rock quality is not good, or if the rock is water bearing, it stabilizes the wall and/or reduces water inflow.

N. A. MALEVICH

Fig. 74. Surface layout of equipment while sinking with the complex moael KS-2U.

1 - hoist; 2 - winch LPM-10 for rope guides; 3 - winch LPK-4/500 for supporting safety ladder; 4 - winch LP-35 for supporting sinking scaffold; 5 - winch for holding pipeline for lowering concrete mix; 6 - winch LP-5 for discharging concrete mix to bunker; 7 - winch LPM-10 for shuttering; 8 - winch 2LPM-10 for ventilation pipe; 9 - winch 2LPM-10 for compressed air pipe; 10 - winch LMP-10 for pump.

Figure 4-2

Figure 4-3

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Fig. 62. Diagrammatic sketch of equipment location in shaft while sinking with complex KB-1.

1 - bucket BPS-2; 2 - pump PPN-50x12; 3 - pipeline for lowering concrete mix; 4 - winch LPPG for suspending pneumatic loader KS-3F; 5 - pipeline for compressed air; 6 - safety ladder; 7 - pipeline for ventilation; 8 - rope for suspending shuttering; 9 - rope for suspending platform; 10 - guide ropes; 11 - cables for signalling, lighting, and connection; 12 - firing cable.

Figure 4-4

Fig. 77. Cyclic operations in the parallel method of shaft sinking using monolithic concrete lining.

phase I - rock loading, lowering of curb ring; phase II - lowering of shuttering, lining, loading of rocks; phase III - lowering of shield and drilling of holes; phase IV - raising of deck, charging of holes, lowering of shield, blasting, and face ventilation.

The shaft is partially equipped from the top deck of the stage as it descends. The concrete forms are first removed for reuse, then the various pipe lines installed. A separate trip of the stage is necessary to complete the equipping of the shaft -- placing buntons, brattice walls, etc.

The rate of sinking varies greatly with specifics. At least three months is typically required to collar the shaft, erect the headframe, and equip for sinking. A rate of sinking of 150 to 200 feet per month is considered a good advance in the United States, but new techniques may increase this by 50 percent. (As an item of interest, a Southern African shaft -- Western Reefs No. 4 -- was sunk 1,118 feet in October, 1961, but the daily down shaft crew totalled 450 men!)

Shaft sinking is rarely as simple as set forth above. Ground water conditions vary tremendously and dictate numerous control procedures. Required linings also vary widely. The following sections consider some of these factors; much of this material is taken directly from the October, 1972 "Status Report, Deep Shaft Studies" for the Bedrock Waste Storage Project for the Savannah River Plant of the United States Atomic Energy Commission.

B. Water Control Techniques

Other elements not controlling, a shaft location involving a minimum of ground water should be selected. Unfortunately, the underground workings and surface developments rarely permit this. The shaft sinker then must have a variety of water control methods at his command.

If ground water is minimal, the shaft may be advanced through both overburden and bedrock using only sumping in the bottom of the excavation. If greater control is required in the overburden and the overburden is granular, a few deep wells will control the water and permit excavation in the dry. If the soil is fine grained, one or more stages of well points can be used or, if space is restricted, eductors may be used.

Grouting has a longer historical record of ground stabilization in rock than any other method. It has proven most effective in porous or fissured rocks where, even though permeabilities are small, openings are large enough for cement particles to enter and hydrate or for chemical grouts to penetrate. It also has application in the overburden. However, it is not effective in very fine grained soils. The degree of water shutoff increases in some proportion to the time and money spent. Reduction of flows are rapid at first; complete shutoff, however, is normally unnecessary on most projects. Numerous types of grouts are used depending on the size of fissures to be grouted. Clay, cement, and chemical grouts are used for successively smaller pores; AM9 and TDM are two of the better known low viscosity grouts used to seal off very small channels. Structural capacity of the grouts varies as does the gel time. The problem in grouting is to intersect all of the flow channels. A "shoot and grout" technique has been

developed recently where small explosive charges are set off for the purpose of creating more, but intersecting, fissures; the method appears to improve grouting efficiency.

Grouting is used in preference to freezing whenever it is effective, for the savings in both cost and time is substantial. Analysis of complete soil samples is required to determine which method must be used. At least one Canadian potash shaft incurred considerable unwarranted cost because of the inadequacy of the sampling program. Grouting is not effective in unconsolidated materials having a significant percentage of fines. Freezing, on the other hand, introduces expansion stresses sufficient to crack concrete linings as the ground thaws and the shaft reaches operating temperature. Accordingly, it is standard procedure in frozen shafts to grout behind the lining as the ground thaws back from the shaft to counteract the effect of this cracking. Grouting is effective here where it would not be without freezing as grout travel is inhibited and therefore it can be forced into the smaller pores. Grouting has the advantage of being able to permanently improve the structural capacity of the formation whereas freezing cannot.

The freezing process has been used for nearly 90 years to stabilize water bearing sediments during shaft sinking operations. No other method is as versatile in handling as wide a range of soils and rock conditions. Many factors must be considered --the depth to be frozen, the quality, quantity, and velocity of water in the strata, the thermal and structural characteristics of the frozen soils, etc. In designing a freezing system, the required thickness of icewall must be calculated and, in turn, the spacing of the freeze holes and the diameter on which the holes are to be placed determined. Spacings will vary from 3.5 to 6.5 feet, and must take into account the drilling accuracy; the freeze ring diameter will be located from 6 to 12 feet outside the excavated diameter, with a rough rule of thumb being the lined diameter plus 20 feet in typical cases. The frozen distance on the inside of the ring typically is 50% greater than on the outside. A central observation and pressure relief hole is essential, fluid pressure increases as high as 400 psi having been developed as freezing progressed inside a closed ring. Calcium chloride brine is the freezing agent most frequently used. The freezing agent is circulated down a polyethylene pipe 2 to 3 inches in inside diameter and back up the annulus between it and a 5.5- to 6.5-inch diameter steel casing which is installed and grouted in a somewhat larger drill hole. These freeze holes must be accurately located; this was impossible before the advent of the turbo drill; now the drill holes are surveyed and when necessary a one- to two-degree angle correction made in a run of perhaps 30 feet with a turbo drill.

C. Excavation Techniques

Excavation is generally routine once the surrounding formations have been stabilized. Unconsolidated formations are excavated by pneumatic

tools as required -- air spaces, jackhammers, and pavement breakers. Small rippers may be employed. Drilling and blasting correspond with normal practice in rock. Controlled blasting techniques such as "smooth wall" or "presplitting" are used to minimize damage to the surrounding rock. These techniques all employ closely spaced holes drilled around the periphery of the shaft. The holes may be left unfilled or filled with special reduced strength charges which are exploded either just ahead of or following the charges in the primary holes. The trademark of this technique is a series of half round drill holes left on the wall of the shaft.

Unless a shaft is very large, the excavated material will be picked up by equipment mounted on the underside of the shaft sinking stage and deposited in materials skips which in turn are hoisted through openings in the stage and up to the surface. Normally two hoisting compartments are used to permit excavation to continue without interruption. In small shafts where only a single skip can be used, detachable buckets are employed. In most cases either a cactus grab or a Cryderman mucker is used to collect and place the excavated material in the skips.

D. Linings

Concrete, either plain or reinforced, is used whenever practical due to its pronounced cost advantage and ease of placement. Concrete linings are frequently placed concurrently with excavation because of the protection afforded the workmen below. When this is done, a multi-level stage is used so that forms may be removed, handled, and reset and concrete poured while excavation is continuing below. Approximately 20-foot lifts are poured. A kerb-ring is set accurately to line and grade for repositioning the forms. When minimal inflow is anticipated, the closure joint between a new pour and the prior pour above is drypacked; when more water or water under high pressure is anticipated, the new pour is placed against the old and the joint grouted. The formation outside of the shaft is also grouted whenever infiltration must be reduced.

When a concrete lining is used in ground stabilized by freezing, some shaft sinking specialists prefer to line permanently as they excavate, allowing a generous wall thickness to compensate for pouring against the frozen ground. The discussion of the Dutch State Mines later in this chapter gives insight into the temperature conditions encountered. Other specialists prefer pouring a minimal protective lining while excavating and "dry-walling" back up with the permanent lining which can then be somewhat thinner.

Very thick concrete walls are required when the anticipated water pressures are substantial. Even then, the pressures will enlarge very small leakage further into considerable leaks requiring repair after some years, or perhaps after only months. Under these conditions, cast iron segmental rings ("tubbing") have been employed in the past. The segments, typically

five feet high, 6.28 feet in length, and with wall thicknesses of four inches or more, are joined by bolting. Lead sheets in the joints and grommets around the bolts are used to minimize inflow. When used in frozen ground, repeated tightening of the bolts and caulking of the joints, particularly the horizontal ones, are necessary as the ground thaws.

When tubbing is used without a concrete lining, the void behind the tubbing, and the rock formation as well, must be grouted. More often, the tubbing is employed in conjunction with a concrete lining either inside or outside the tubbing. Again shaft sinking specialists differ. Some erect tubbing as they proceed downward; others prefer to start from a firm foundation at the bottom. The more unstable the ground, the more likely that the tubbing will be installed as excavation progresses. Leakage is rarely eliminated completely with cast iron linings: rather it is minimized to acceptable levels. Some cast iron lined shafts have an experience record of nearly 100 years.

German trained shaft sinkers insist that only a welded steel lining can provide adequate watertightness. While steel linings alone can be used, when the water pressure is great a composite lining composed of outer and inner welded steel cylinders with concrete sandwiched between is used. Except for floated-in linings used with the fluid filled big-hole drilling technique described in the next chapter, these linings are invariably constructed upwards from the bottom of the shaft. Grout pipes for contact (formation) grouting may be installed. The annular space between the outer steel sheel and the ground is filled with grout.

E. In-Service Leakage Control

It is considered axiomatic for some projects, e.g., potash mines or nuclear waste storage, that no water be allowed to enter the bedrock formation anywhere in the vicinity of the mining or storage areas. It follows that no liquid be allowed to enter the mining or storage formation through or around the shaft which undoubtedly will be passing through water bearing formations.

The amount of water permitted to infiltrate through a lining as defined by the term "satisfactory water shutoff" depends on the use for the shaft. A coal mine can tolerate considerable inflow; a potash mine practically none; a nuclear waste repository absolutely none. Grouting specialists maintain that any required infiltration minimum can be met in rock without recourse to a welded steel lining. Satisfactory conditions have been achieved even in the running sands of the Blairmore by grouting behind the lining as the surrounding ground thaws back from the walls. As many as three stages of grouting at successively higher pressures have been used. Obviously, a properly constructed welded steel lining should not require this ground treatment.

It is worth noting here that specific studies may demonstrate that an infiltration of a few gallons a day may be acceptable due to the large volume of air required for ventilation. It has been noted in some shafts for ventilation and other functions that damp spots on the wall do not create trickles of water at a lower elevation. This is probably because the infiltrating water is evaporated by the ventilation air. The allowable amount certainly is influenced by both combined temperatures and ventilation requirements. While the amount cannot be forecast, it is felt quantities as high as four to five gallons per hour range might be acceptable. The problem is what will be the situation when large air volumes are no longer required for mining purposes.

When major aquifers are contained between impermeable formations, as occurs with the Blairmore in the Saskatchewan potash fields, it is necessary to install seals outside the shaft proper to prevent the flow of this water up or down the exterior of the shaft to other permeable formations and then find its way into the mine by some other route. In recent years chemical seals have been developed which effectively prevent this migration. Inasmuch as the service records for such seals are short while the requirements of a project may be long term, they are not considered further here since a satisfactory sealing method exists which has a long and proven history of effectiveness.

Picotage rings (sometimes spelled 'pikotage), also known as "Keikranz" or wedging rings, are the time honored system for preventing migration along the exterior of shaft walls. These seals normally are used in conjunction with cast iron tubing, but special fabricated steel rings may also be used locally. A special ring is fabricated which is much deeper in the through wall dimension than the standard tubing. The seal is then constructed between the special cast iron ring and the impermeable rock formation. Use of these rings is described for specific shafts later in this chapter. Up to 15,000 dimensioned wooden wedges may be required to construct such a seal only 8-inches wide around a 16-foot diameter shaft.

Since picotage is a relatively little known technique outside the shaft sinking specialty, it merits a few words of explanation. Basically, it is a process of driving ever smaller wedges into an ever tightening ring until absolute refusal is encountered. The ring is laid originally with rectangular mating wedges typically about eight inches long, of a nominal one by eight inches in blunt cross section, and tapered to a feathered edge. Smaller wedges are driven into all spaces. These initial filling wedges vary from the original one by eight blunt cross section to 1/2 by 2 to 1/4 to 1.5 and then from one-inch square to 3/8-inch square. These rectangular wedges are 10 inches long and tapered to a feather edge. The square wedges have only the bottom inch or so tapered to a diamond point. One driving difficulty becomes too great, round, tapered, pointed dowels from 5/8 by 8 to 1/2 by 5 are driven. A wood such as a straight grained, clear oak is used; the grain

parallels the taper. When the wooden dowels can no longer be driven, tapered steel needles varying in blunt end diameter from one inch to 1/4 inch and nine to three inches long are driven until they too meet refusal. The needles have a hexagonal head and are rotated to improve penetration.

Bitumen also has a long history of use in waterproofing. A considerable thickness is required if it is to be effective. It has particular application for long term use where differential movements are anticipated since it retains a plastic characteristic which permits it to adapt to these movements. Its use of certain Dutch State Mines' shafts where areal settlements due to the extraction of coal were anticipated and where the value of coal left in the conventional shaft support pillar was sufficient to warrant reducing the pillar size by adding the protective ring of bitumen is described in the next section.

F. Representative Projects

Large diameter, deep circular shafts have been constructed in at least three widely separated geographical areas which are pertinent to this discussion. These include the potash shafts in Saskatchewan, Canada, shafts in the southwestern United States and coal mining shafts in the Netherlands and the German Ruhr. These are described in the following paragraphs or in the following chapter. Emphasis is given to the difficult projects to illustrate the troubles that can be encountered in shaft sinking.

The potash mines of Saskatchewan province in Canada are renowned to shaft sinkers for the difficulties encountered in shaft sinking. The source of the trouble is a geologic stratum named the Blairmore which is a fine, running or quick sand with grain sizes of less than 0.1 mm accompanied by silt and clay. The formation typically is 300 to 400 feet thick, generally begins some 1,400 to 1,800 feet below the ground surface, and has water pressures essentially equal to the depth below ground. The Nisqu formation, generally 200 feet thick, lies below the Blairmore and is a heavily water-bearing sandstone formation having water pressures as high as 940 psi, which makes for considerable difficulty in sealing shafts against infiltration without exceeding the strict values required to protect the potash workings. Freezing is the only method which has been used successfully to penetrate the Blairmore, but grouting is successful in the Nisqu.

A shaft begun in 1953 ran afoul of the Blairmore and was abandoned. The first shaft to be sunk successfully was for PCA's (Potash Company of America) No. 1 Shaft at Patience Lake, 15 miles east of Saskatoon. The shaft was 16 feet in lined diameter and was excavated by freezing the upper 3,000 feet of its 3,450-foot depth.

Freezing was accomplished by drilling 27 freeze holes at four-foot centers on a 35-foot diameter circle (plus one observation hole on the shaft centerline and another outside the freeze ring) and using an 800-ton

refrigeration plant. The shaft was constructed in 28 months with excavation and lining proceeding concurrently. The shaft lining was of reinforced concrete exclusively, 48 inches thick through the Blairmore (extending from 1,400 to 1,800 feet below the surface) and 18 inches thick in the shales and limestones. Pre- and post-grouting was used and at the time of shaft completion in June, 1958, 10 gallons per minute (gpm) were infiltrating the shaft. Mining began but had to be halted temporarily in late 1959 as leakage had increased to 40 gpm. Lining spalls up to 3 inches deep were found. By the time repair crews were assembled in November 1959, the rate had increased to 200 gpm. Before the corrective treatments became effective, the flow exceeded 300 gpm, graphic evidence of the deteriorating effect of minute flows at high pressure on concrete. Most shaft contractors consulted felt a steel inner lining was necessary, but the chosen repair contractor elected grouting. The trouble showed at the 2,000-foot level where natural water content was small. The water was found to be migrating down the outside of the original freeze tube casings, presumably from the Blairmore, following withdrawal of the inner freeze tubes to 2,000 feet to permit the lower freeze wall to melt naturally. Injection of nearly 1,000 sacks of cement dropped the inflow to 33 gpm. The Blairmore was then allowed to thaw. As the freeze retreated from the concrete lining, the back was sealed with chemical grout. Chemical grout was also injected outside the ice ring, which by then averaged 26 feet thick. After thawing was completed, the water was drained from between the two rings and the mass saturated with grout; all freeze casings were similarly grouted. The leakage having been reduced to satisfactory levels, the mine then reopened.

Another difficult shaft is at the Alwinsal mine of Guernsey some 50 miles east of Saskatoon. It has a lining composed of a double steel shell with an infill of concrete constructed within the protection of a temporary concrete lining. (In January, 1966, a record 570 feet of shaft was sunk and lined.) According to the shaft superintendent, who has been continuously engaged in shaft sinking since working on the completion of Yarbo No. 1 in 1957, this lining produced the driest shaft he has ever constructed (personal communication).

Two shafts constructed in 1965 and 1968 near the above shaft shed some interesting light on the matter of shaft costs. A shaft 18.5 feet in lined diameter was constructed at only four percent more cost than the adjacent shaft with a lined diameter of 16 feet, despite the 15 percent greater diameter and 33 percent greater cross sectional area. It should be noted, however, that these diameters are close to the optimum size for shaft sinking, and, therefore the variation is less than probably would occur with substantially greater diameters.

The Saskatchewan experiences show that freezing can be used to penetrate formations that cannot be controlled by grouting. (The success of grouting in unconsolidated strata can be forecast reliably only by

obtaining adequate samples covering the entire range of particle sizes and then analyzing the samples.) On the other hand, grouting can be used successfully in heavily water-bearing rocks to seal off the formations. A lining of concrete alone placed in a frozen shaft is unlikely to be successful, although in the case cited, the lining undoubtedly was too thin. Cast iron lining can be successfully employed, but the claim made by proponents of composite steel and concrete linings that such a lining is superior appears merited.

Two shafts recently completed for Cleveland Potash, Ltd. of Boulby mine between Teeside and Whitby near the northeast coast of Yorkshire, England, give further insight into the selection of excavation methods and lining types.⁽⁴⁾ Two shafts, 18 feet in diameter and 3,710 feet deep are only 300 feet apart. They are constructed through essentially dry and competent weak shales, siltstones, and mudstones for 2,076 feet. The Bunter formation, a fractured sandstone major aquifer containing saturated brine under full hydrostatic pressure (1,400 psi), occurs for the next 920 feet, followed by dry marls and then the potash deposits. Both shafts were excavated conventionally and simultaneously to the 2,000-foot level, using drilling and blasting to excavate, rock bolts for structural support and a finish lining of 3,750 psi concrete. Shaft sinking rates in excess of 400 feet per month were established in both shafts.

The upcast (ventilation return air) or "Rock Shaft" was excavated through the Bunter to the 3,100-foot level behind the protection of an ice wall. An 18-inch concrete lining was installed in 20-foot lifts concurrently with and about 60 feet above the excavation. The permanent lining was erected upward from a special foundation and consists of inner and outer steel cylinders (maximum steel thickness 1.625 inches) separated and joined by a high strength concrete infill 2.5 feet thick. The steel lining was delivered to working level in 12-foot high sections weighing up to seven tons with four sections forming the complete circle. Inner and outer shells were welded simultaneously in a single pass using inert gas automatic welding machines with laterally weaving electrodes and without preheat or stress relieving. The horizontal weld was made in several passes. Welds were ultrasonically tested. The lining is designed for only the hydrostatic pressure. The narrow space (about three inches) between the shell and the temporary lining was grouted. Also of interest is the fact that the 27 freeze holes were drilled from the surface on a 110-foot diameter circle and slanted to a 41-foot diameter at the 1,840-foot level and through the Bunter; a freeze chamber was excavated to a 50-foot diameter some 50 feet above the Bunter, where the average deviation of the drill holes was observed to be about eight inches.

The downcast (ventilation supply) or "Man Shaft" was excavated through the Bunter in 70-foot lifts using pregrouting with a specially formulated chemical grout. A curb ring was poured and a cast iron

segmental lining (tubbing) installed up to the previous lift and backfilled with a high strength sulfate resisting concrete. Ten four-ton, five-foot high segments with full width lead gasketing in all joints formed a ring. The segments were flanged, 10 inches deep, and had wall thicknesses of four inches. They were consistently furnished to tolerances of plus or minus .004 inches in height and well within plus 0.375 minus 0 inches in assembled diameter. Leakage through the completed lining totalled four gallons per minute and it was planned to reduce this by further grouting of the sandstone.

Reasons given for the different procedures and linings were that the rock shaft had to be absolutely watertight and constructed within a rigid schedule to meet a production deadline while the man shaft did not have to meet these conditions; also, the man shaft was estimated to cost a million dollars less, a savings which was not realized due to grouting difficulties and the high cost of the special grout. The consultants (a German firm) felt that the freezing technique was the only one predictable enough to meet the deadline and the only way the shaft could be made dry enough for welding. They considered that the tubbing could not be made sufficiently watertight at the high water pressures to be encountered; it was also felt that seasonal temperature variations would eventually produce new leaks through the tubbing.

Five shafts constructed in 1950-60 decade for the Dutch State Mines are of particular interest.⁽⁵⁾ All were constructed through a soft, water-bearing overburden from 700 to 1,600 feet thick. Two, each 22 feet in lined diameter, were constructed by freezing and had reinforced concrete linings. Three, with interior diameters of 13.7 and 18.4 feet, were drilled and had a lining which was floated into place while the hole was still filled with drilling mud; these are described in the next chapter. All shafts had a thick bituminous seal on the outside intended for waterproofing in the reinforced concrete lined shafts as well as for protection of all shafts against settlements and movements in the overburden caused by the mining.

The bituminous seal for the frozen shafts (Statamines Hendrik and Maurits) was five inches thick for the upper 350 feet and three inches below. The bitumen was placed in 20- by 24-inch blocks cooled to 23° F to maintain their shape during transport and handling. The bitumen type was Mexphalt 20-30; the mixture had a viscosity of 10⁹ poises at 77° F. Previously, bitumen only 0.50 to 0.75 inches thick had been used for shaft sealing. The reinforced concrete lining, increasing from 20 inches thick at the surface for one shaft and 23 inches for the other to 34 inches at the bottom, was then placed from the bottom up within the protection of a temporary lining of brick. Reinforcement varied from one percent at the surface to 1.8 percent at a depth of 700 feet.

The concrete lining at Hendrik Shaft IV contracted during freezing and expanded more than an inch during thawing. Nevertheless, the reinforced

concrete and bitumen combination effectively limited infiltration in the entire shaft through its 725 feet of water-bearing overburden to only four gallons per hour. More careful measurements were made at Maurits Shaft III, resulting in the discovery that the lining in a clay strata 500 feet below the surface contracted four inches in its 22-foot diameter during freezing although a contraction of two to three inches was more typical in most of the overburden. Temperature measurements at the concrete and frozen soil interface indicated concrete placed at 40° F rose to 70° F, then subsided in a few days to freezing and then more slowly to 15° F, remaining there for several months and then rising again to slightly above freezing. Measurements in the middle of a 16-inch lining reached freezing 80 hours after pouring; for a 23-inch lining, more than 300 hours elapsed before freezing occurred at the midpoint.

The Maurits shaft was not lined with reinforced concrete to full overburden depth (990 feet), although it would have, had the results of the Hendrik shaft been available. Below 750 feet, cast iron was again used to a maximum depth of 1,100 feet. Wall thicknesses of 3.35 to 4.75 inches were used in the 5-foot high, 12 segment rings which were manufactured to 0.004-inch vertical tolerance. Lead plates, 0.12 inches thick were placed in all joints and a minimum of 20 inches of concrete placed behind the lining. During thawing repeated caulking was necessary, the horizontal joints opening to a maximum of 0.28 inches.

Since the thick layer of bitumen prevented intermediate support, the entire linings had to be supported on a foundation cut into bearing strata of adequate strength. This occurred at the 800-foot level of Hendrik and the 750-foot level at Maurits where a stratum with a compressive strength of 1,400 psi was available.

The use of cast iron linings in the Dutch mines was discontinued subsequently because of the inability of such linings to take unequal radial loads, to work effectively compositely with concrete, or to work with thick bitumen layers; because the joints must be recaulked repeatedly as the shaft is thawed and because of cost.

Other difficult shafts using special techniques are described at the end of the following chapter.

CHAPTER 5

MECHANIZED SHAFT SINKING TECHNIQUES

The high and constantly increasing cost of conventional shaft sinking has led in recent years to intensive efforts to mechanize the procedures, an effort that is now paying dividends.

As used in this report, conventional shaft sinking refers specifically to the use of drill and blast procedures and the resulting intermittent advance of the face whereas mechanized shaft sinking involves the use of machines rather than explosives to break the rock and results in a nearly continuous advance of the face. The term "drilled shafts" is frequently used as the procedure is not unlike the use of giant drills.

Although there is a difference in manpower requirements, it is a mistake to consider the difference as being solely between labor intensive and machine intensive methods. A high degree of mechanization is employed in conventional sinking; for example, jumbo or gang drills used today are highly mechanized tools; the excavating equipment - both the Cryderman mucker and the cactus grabs are machines operated by a single man. There is no appreciable difference in pregrouting procedures between the systems, although blind drilling may attack the problem by drilling in a water-filled hole rather than using pregrouting. If grouting is required at depth, the conventional system permits greater flexibility in approach. If the shaft is to be lined with concrete, the conventional system is probably preferable because both excavation and lining can be done simultaneously.

Procedures used in mechanized shaft sinking can result in significant advances in sinking rates, and therefore in reduction of costs. Certain of these techniques can be combined with the conventional procedures to provide similar increases in rate and decreases in cost. For example, a large diameter shaft first may be raise bored to an intermediate diameter and the shaft then increased to final diameter by "breasting up" or "slashing down" using conventional, or drill and blast, procedures.

A. General

Mechanized shaft sinking will be considered here as divided into three general areas -- raise boring, blind drilling, and down hole boring machines. A fourth procedure, reaming up or down is really a modification of raise boring and is so considered here. The greatest advances in recent years in shaft sinking technology have been in the field of raise boring; however, this technique requires an existing mine, or at least some prior development at the proposed mining level. It is a highly developed technique and future advances will be in sophistication of equipment.

Blind drilling advances have not been as dramatic. It is felt the greatest potential for this method lies in handling certain very difficult ground conditions.

Down hole boring machines appear to hold the greatest potential. A Bureau of Mines funded demonstration project scheduled for the summer of 1977 may achieve the breakthrough, long discussed, that will make this procedure an effective, practical shaft sinker's tool.

B. Raise Boring

A vertical or inclined shaft interconnecting two or more levels of a mine without extending to the ground surface is generally called a raise. These frequently can be of rather small diameter. Sinking a shaft or raise always involves the problem of removing excavated material from the bottom of the shaft up through the excavated section. The need and the problem combined to result in the development of the technique called raise boring.

Raise boring involves drilling a pilot hole, about 12 inches in diameter, downward from the top level and then pulling up a reamer or raise boring tool, using the pilot hole for guidance and for transmitting pull (thrust) and torque via a drill stem connected to the motive power equipment at the top of the raise. The cuttings fall to the bottom of the hole where they are removed to another shaft for haul-out and disposal. Raises may be drilled vertically or inclined. The combination longest-largest raise at present is one 12 feet in diameter raised 2660 feet. The largest diameter shaft raise bored in the United States had an excavated diameter slightly over 15 feet. The Swedish Alimak raise climber, a completely different type of raise boring procedure, has cut 376 square foot raises (22-foot equivalent diameter) in Sweden.

Raise boring technology has expanded tremendously in the last decade. Prior to 1967, there were only ten machines in existence; records on raise bored shafts before that date are almost non-existent. By 1972, the number had increased to 100. Most were in the four-to-six-foot diameter range, the largest being for a hole 12 feet in excavated diameter. Today, the number has almost doubled, with the largest size actually bored being the aforementioned 15 feet. There are machines available now capable of raise boring to 20 feet in diameter; one, the Robbins 85R, is capable of a million-pound pull (thrust) a torque of approximately 500,000-foot-pounds, and a 1,000-foot raise.

While some consider raise boring in the 10 to 20 foot diameter range to be competitive pricewise with conventional shaft sinking, this would be true under only the most favorable conditions. A more generally held view, it is believed, is that raise boring, where practical, currently is more economic in the 12-foot and under range because of the penalty paid by the

conventional system due to lack of working space in the shaft. In the larger sizes and under suitable conditions, the combination method of raising an intermediate shaft, say 6 feet in diameter, followed by slashing to full size with conventional equipment, is probably more economic.

Raise borers do not appear to be limited by rock hardness, having operated successfully in rocks having compressive strengths in the 40,000 to 50,000 psi range. In this regard, they have a significant advantage over other mechanized shaft drilling methods.

Raise borers have two significant advantages over conventional equipment -- speed and safety. The advantage of not having workmen in the hole is obvious. Advance rates of 500 feet per month, while not commonplace, are no longer considered unusual. In contrast, 200 feet per month is considered an excellent rate for conventional shaft sinking in North America.

Raise borers also possess significant disadvantages relative to conventional procedures, limiting their use to the more favorable geologic conditions. Water bearing formations can be pregrouted from the surface or top level; this procedure does not always accomplish the desired shutoff, although it does effect a significant reduction in flow. If additional treatment is required, it can be accomplished readily at the face, or from an intermediate level with conventional equipment. With raise borers, however, the boring head has to be lowered for access to the face and a special stage raised from the bottom of the shaft. Also, the grouters are working from below the water rather than above, an additional disadvantage.

Caving ground (e.g. shale) also poses problems for raise borers. While a light lining can be suspended from the shaft of the boring machine, and brought up the hole with the head, it is difficult to control the ground in this fashion. In conventional procedures, the lining can be brought down to within a few feet of the face, and special procedures, backpacking, and grouting accomplished as required for control.

For smaller to moderate diameter shafts and favorable conditions, raise boring will probably be the preferred method for the foreseeable future. The economics of the large raise bored shafts (12 to 20 feet in diameter) remains to be proven by usage. The advantages of combining raise boring with conventional procedures has been dramatically proven in a shaft nearing completion at Bathurst, New Brunswick, which is 4,500 feet deep and 26 feet in diameter; a description is included in the last section of this chapter.

C. Blind Drilling

The predecessor to blind-hole drilling, calyx drilling, has been used on occasion in shaft work for more than 40 years, but essentially became

obsolete with the advent of blind-hole drilling. It is called "blind" because unlike raise boring, there is no guide-hole available and, therefore, sometimes wanders from a true vertical. Blind-hole drilling, as now known, has been used only since about 1960. It is frequently called "big-hole drilling" because it is basically an extension of oil-field drilling techniques. Most of the relatively few blind drilled mine shafts have been in the six to eight foot range, a very few in the 10 to 12-foot range. However, some 200 holes 14'-8" in diameter but only 65 feet deep were drilled in Wyoming in 1963 for missile silos. The largest diameter blind drilled shaft in North America is 16'-6" in diameter; further details are given in the last section of this chapter.

Three shafts for the Dutch State mines were drilled to a diameter of 25 feet by methods considerably different than North American practice. They were drilled to a depth of 1640 feet; further details on their construction are given in the last section of this chapter. The largest drilled shaft reported is 28.7 feet in drilled diameter and 2674 feet deep, constructed for a Russian coal mine. Four stage reaming was reportedly used. The deepest known blind-hole drilled shaft is 5,500 feet deep; it has a drilled diameter of 10 feet and was constructed for the (then) Atomic Energy Commission.⁽⁶⁾

Compared to raise boring, much more equipment and a considerably greater investment is required for blind-hole drilling. There has been nowhere near the development effort or coordination in this field that there has been in raise boring. Blind-hole drilling is also much more limited in application in one sense, being confined to the softer range of rocks.

The enormity of big-hole equipment can be grasped when it is realized that the derrick must be capable of lifting weights in excess of one million pounds. On the order of a million gallons of drilling mud must be circulated when drilling a 12-foot diameter hole at a depth of 1,000 feet. Despite the magnitude of these numbers, the technology and equipment are at hand. The weakest part of the chain appears to be in the amount of torque that can be applied; necessary weight can be applied to the cutting head, but the existing rotary tables have only limited capacity for delivering torque.

The major advantages of blind-hole drilling are safety and control of bad ground, i.e., unconsolidated or running ground, weak formations, and gassy ground. As in raise-boring, no men are in the hole until excavation and, generally, initial lining are complete.

Big-hole drilling in unconsolidated alluviums must be done with a slurry (mud) filled hole in order to prevent sloughing and failure of the walls. However, the effective control of such ground is probably the outstanding advantage of this method. The slurry is designed so that a thin cake forms on the shaft wall and, combined with the fluid pressure, holds the

unconsolidated material in place. Excellent results have been obtained in alluviums when the particle size is not too large. For example, 3-inch cobbles can be handled easily; but 6-inch ones are difficult to break down or to lift out, and they tend to cause serious wall caving. Large diameter holes must be drilled with reverse circulation, that is, the fluid enters the hole directly and returns up the hollow drill stem. This is necessary in order to achieve velocities sufficient to raise the cuttings. On deep holes, high pressure air ("reverse circulation, air-assist") is introduced part way down the stem to lighten the fluid column and increase the flow velocity. An advantage of the heavy drill string which is required both for support and to obtain necessary bit pressures is the plumb bob effect which tends to keep holes straight. Heavy annular weights called "donuts" are generally added to increase pressure on the drill bit. Ground water is not a problem since the greater specific gravity of the slurry prevents inflow; the mud cake prevents or reduces caving, and there are no harmful effects on the permanent lining requiring correction as there are with freezing.

Big-hole drilling can also be used in rock; it is most successful in the softer range of rocks. Such drilling has been used more in rock than in unconsolidated sediments, but it is limited normally to moderate diameters -- generally ten feet or less. The drawbacks to big-hole drilling in rock are the very heavy drill strings -- on the order of 800,000 pounds or more must be hoisted in deep shafts each time a bit is changed -- the quantities of power and circulating media -- air, water, or mud, as the formation dictates -- required and the inability to reduce the effect of destressing on the adjacent rock immediately after excavation.

When holes are drilled with fluid supporting the shaft walls, the linings are constructed on the surface where, incidentally, working conditions are far superior to those inside the shaft. Welded steel linings are invariably used, assuring complete watertightness. A single lining is most common, but sometimes a double steel lining with an in-fill of concrete is required. A temporary bottom is added to provide buoyancy. The lining floats in the fluid filled hole. Depending on the specifics of the situation, water may be added inside the completed lining, or completion of the interior of the lining delayed as required to assure the proper degree of buoyancy.

Several significant big-hole drilling projects are described in the last section of this chapter.

D. Down-Hole Boring Machines

The first use of a down-hole boring machine in this country apparently was in 1955 in the coal fields of West Virginia. More than 20 shafts, six feet in diameter and up to 700 feet deep, were bored in the first five years. The first 8-foot diameter hole was bored in 1961. By 1968, a total of more than 60 coal mine shafts had been excavated using down-hole boring machines.

In all cases, however, muck was disposed of by first drilling a 12-inch diameter pilot hole and then flushing the cuttings downhole into the existing mine.

The largest and deepest shaft excavated by a down-hole boring machine was constructed in 1969 for the White Pine Copper Mine in Michigan. It was 12 feet in excavated diameter and 1591 feet deep and was excavated in sandstone and shale formations. The largest known down-hole boring attempted in this country was 17'-8" in diameter and in a competent shale. Reasons for abandonment have not been verified.

Down-hole boring machines differ from blind-hole drilling in that the thrust and torque are developed near the face in the same frame housing the cutting head of the down-hole borer whereas in blind drilling the rotary effort must be applied through the rotary table to a drill kelly (a large, solid steel square bar as long as a length of drill stem pipe) at the surface and delivered to the face by the drill stem. Adequate force can be provided on the cutting bits in the big-hole equipment by steel donuts riding the drill stem but must be provided by thrusting off the wall in the down-hole equipment. The down-hole boring machine thus is essentially independent of surface equipment, while the blind-hole equipment is completely dependent on the surface motive power. At present both types are restricted to the rocks in the lower compressive range.

Down-hole boring machines appear to combine the advantages of both conventional and mechanized shaft sinking. Men need not be in the hole during excavation. On the other hand, there is direct access available at all times, unlike blind-hole drilling, so that effective remedial action can be taken when required. If desired, erection of the lining can follow closely behind the machine. Pregrouting from the surface is possible as with the other methods. If necessary, additional grouting can be performed from an intermediate level specially excavated for that purpose, unlike the other mechanized methods.

There has been much discussion about the development of satisfactory down-hole boring equipment over the years. In view of the tremendous developments in recent years of tunnel boring machines which are closely related to the down-hole equipment, it is strange that similar development has not occurred with the latter equipment. The problems appear to be two-fold. The muck removal problem is quite different with a horizontal face borer than it is with a vertical faced one. And, of course, the development costs are so high that private companies hesitate to underwrite them when other methods can be used.

It appears that both problems have been resolved. Work is scheduled to begin next April on a 24-foot excavated, 22-foot finished diameter shaft 1100 feet deep for the U. S. Steel Corp., coal mine near Birmingham, Alabama.

Development work is being funded by the Bureau of Mines. The boring machine will be manufactured by the Robbins Company. Shaft and associated work will be performed by the Cementation Company of America. Instrumentation and monitoring will be done by the Batelle Institute. The primary formation is a sandstone with a compressive strength of about 30,000 psi, plus some shale and coal horizons. The machine is designed to operate to a 2,000-foot depth. Disc cutters will probably be used with an electric drive through gear boxes. At present, a vacuum muck pickup is planned with bucket elevators leading into two muck skips. Thrust will be taken from the wall, probably through an expanding shield about 10 feet long. It is hoped that an advance rate of 2.5 feet per hour will be obtained. Pending core drilling exploration, pregrouting from the surface is planned. There will be probe drilling ahead of the machine. A 12-inch cast-in-place concrete lining is planned.

E. Representative Projects

The melding of raise boring and conventional techniques to an efficiency greater than achievable by either method alone has been demonstrated by work now underway by the J. S. Redpath organization at the Brunswick Mining and Smelting Company mine at Bathurst, New Brunswick.⁽⁷⁾ The first step was to raise bore and equip a small auxiliary shaft seven feet in diameter from a depth of 2,800 feet. Had this been a new development, the shaft presumably could have been blind drilled. The rock was of such quality that lining was unnecessary. Although the deviation of the shaft from vertical is as much as one percent in 800 feet, a 7-ton capacity skip is operating satisfactorily.

The next step was raise boring a pilot shaft of the same diameter from the 3,700-foot level for the principal shaft, which has a lined diameter of 26 feet. This shaft is now being slashed downward to the full excavated diameter. Muck from both the pilot and the slash is removed via the auxiliary shaft. Had an existing mine level not been available for the interconnection, a connecting cross drift could have been excavated for the same purpose.

There are a number of advantages resulting from this combined method. The mucking operation is simplified and safer. Instead of hoisting all muck to the surface, thereby imposing a constraint on sinking rate, it is dropped down the glory-hole pilot shaft, eliminating the need for filling bucket type skips. The danger inherent in undiscovered bootlegs is minimized in Canadian shaft sinking by blowing down the hole, that is, using air jets to thoroughly clean the face. This ordinarily necessitates a benched face, or advancing only half the face at one time. With the pilot hole available instead for both mucking and face cleaning the full face can be advanced. More efficient drilling is thus possible; longer rounds can be used, 15 feet being the standard in this shaft; and more efficient blasting is possible since the pilot acts as a gigantic "burn hole" (expansion chamber).

There is another distinct advantage to this auxiliary bore technique. Development work for crusher rooms and other activities is required at the base of the shaft before ore production can begin. Normally this work must wait until the shaft has been excavated and equipping is underway, complicating the latter operation. The advance bore gives early access to the development area so that shaft and development can be completed at nearly the same time. As a result, it is estimated that the shaft can be used for mine production nearly a year sooner than if only the conventional method were used.

The advance rates possible with raise borers was demonstrated within the past year in a shaft constructed for the Cayuga Rock Salt Mine in Lansing, New York.⁽⁸⁾ The 12-foot diameter shaft was completed to a depth of 2300 feet in 65 operating days for an average advance of nearly three feet per hour. The last 1000 feet were reamed at nearly twice that rate. The compressive strengths of the rocks bored ranged from 15,000 to 25,000 psi. The pilot hole was 14 inches in diameter and was bored to within 4 feet of true vertical for the 2000-foot length, a deviation of only 0.2%.

Two of the five shafts constructed in 1950-60 decade for the Dutch State Mines were described in the previous chapter. The other three, with interior diameters of 13.7 or 18.4 feet, were blind-drilled and had a lining which was floated into place while the hole was still filled with drilling mud. The lining was composed of two concentric structural steel shells and a structural concrete infill. All shafts also had a thick bituminous seal on the outside intended for waterproofing, for protection against settlements, and against movements in the overburden caused by the mining.

The two most interesting of the three drilled shafts (for Statemine Beatrix) were drilled to a diameter of 25 feet and a depth of 1,640 feet, the first 1,340 feet consisting of unconsolidated water-bearing strata of quick-sand and clay. After the lining was floated into place, the lowest 330 feet was grouted while the annulus remained filled with drilling mud. This annular space was 16.75 inches wide. As the lining sank, careful measurements were taken and it came no closer to the excavated shaft wall than 10 inches. The 1,640-foot long lining was installed in 44 days for the first shaft, an average advance of nearly 40 feet per day. Progress on the second was even better, requiring only 31 days, for an average of 54 feet per day.

After the bottom of the annulus was grouted, bitumen at a temperature of 300° F to 400° F was then pumped in to displace the mud. The two filling pipes were six inches in diameter and 1,315 feet long. Pouring of the bitumen mixture took 10 days with pumping pressures of from 45 to 450 psi being used. The specific gravity of the mixture was 1.3 and was obtained by adding finely ground marl. Bitumen type was Spramex 400-500 with a viscosity of 2×10^3 poises at 77° F. This method of hot placing had been thoroughly tested at a pilot installation. Subsequent to the above installation

the procedure of floating a lining into place and filling the annular space with hot bitumen was used at Shaft VII of the Auguste Victoria mine in Germany, that shaft was excavated by the freezing method.

The structural lining of the Beatrix shafts totalled 20 inches thick for the upper 920 feet and 23.5 inches for the remainder. At Statemine Emma (the first of the three drilled shafts) the steel shells were of welded mild steel plates, whereas at Beatrix they were formed of steel channels bent to curvature. Maximum steel plate thickness was 1.35 inches with a computed working stress of 25,600 psi. The channels used at Beatrix resulted in a flexible pipe capable of bending to a 3,280-foot radius while remaining watertight. The inner shell was bolted, was not designed to be watertight, and was not connected (by steel) to the outer shell. Shells were placed in units 45 feet high. Injection concrete was used in the annular space in order to provide the best quality of concrete possible and to insure a good bond to the steel shells.

The drilling method was quite different from United States practice. Beginning with a pilot hole 6.58 feet in diameter, reaming was used with successive passes limited to the removal of 65 square feet of shaft cross section. As a consequence more than 1,100 drilling days were required, since two hours were required to raise the bit to the surface from maximum depth, 20 percent of the time was spent in roundtrips and only 70 percent devoted to actual drilling, with the remainder spent in making measurements, repairs, etc. Some 8.3×10^6 kilowatt hours of energy were consumed, the bulk of it for the air lift compressors; 78,000 cubic feet of drilling mud with a specific gravity of 1.18 was required to fill the hole.

As a result of the experience with the five shafts, the engineer for the Dutch State Mines has concluded "mainly soft to medium hard strata and smaller diameters tend to the application of the drilling method, whereas hard, water-bearing strata and larger diameters tend to the application of the freezing method."

The largest diameter shaft drilled to date in the United States was drilled in 1970 near Grants, New Mexico. It was drilled full size, 16'-6" in diameter, in relatively dry shales, and to a depth of 800 feet with a maximum deviation from the vertical of six inches. However, the shales caved in some places to 30 feet in diameter. The ground had been pre-treated by a new "shoot and grout" technique designed to improve interception of ground fissures. The anticipated 350 gpm inflow was thus reduced to 50 gpm; even this small amount was difficult to handle with the reverse air circulation used. The hole was drilled dry under air pressure contained by an air deck at the surface designed for 10 psi; only 4 to 6 psi were necessary. Had the inflow been more than perhaps 100 gpm, water circulation would have been necessary, as was the case with an earlier 11-foot diameter shaft drilled by the same company near Carlsbad, New Mexico.

The Grants hole was lined to a diameter of 14 feet with a steel casing suspended in the hole while supported by attached drill pipes later used for grouting the annulus. The casing was fabricated in 33-foot increments, welded at the surface, and lowered as the next section was added. Thickness varied from 1/4 to 5/8 inches; total hung weight was 700,000 pounds.

Another significant "hung" casing was used on the first 10-foot diameter shaft drilled for the Atomic Energy Commission. A 144-inch diameter surface hole ("collar") was first drilled to 400 feet and cased in successive units 20 feet long at the surface while preceding units were supported in the hole by double elevators on the hoisting equipment. The following 120-inch diameter drilled hole was cased to 54 inches with successive units also surface welded and supported at the top of the hole by the elevators. Steel thicknesses ranged from 0.75 to 1.625 inches. The 5,600 feet of this casing weighed about 5,000,000 pounds. (A 16-foot diameter steel casing averaging one inch in thickness and 1,200 feet long would weight about 3,000,000 pounds.)

CHAPTER 6

LONG TERM PROBLEMS

The two previous chapters have considered construction techniques. This chapter considers the in-service problems. The materials used in construction must withstand the service environment if it is adverse. Proper construction must ensure water or fluid leakage is either eliminated or minimized, as appropriate; it must be demonstrated by observation that there is no hidden water migration. Differential settlement between the shaft and the surrounding ground, rather than water or bad ground conditions, may well be the principal shaft problem if shaft is founded in a salt bed or if areal settlement occurs as a result of excessive mining operations. Since earthquakes cause great damage above ground, it might be thought their effect on mines and shafts would be even greater. Such quite definitely is not the case, however. Finally, project layouts involving auxiliary shafts for each stage of mine expansion may reduce initial shaft and capital requirements; these should be given consideration.

A. Materials

Materials used in the shaft are affected by the shaft environment created by the shaft construction, the shaft usage, and the mining below. In essence, the shaft environment will be favorable (inert) or unfavorable (aggressive) as it affects the materials of construction. Actually, "design considerations" is a more appropriate name than "problems" for these items.

The nature of the water outside the shaft can dictate the choice of shaft lining. If it is aggressive in that it will attack either steel or concrete, appropriate precautions must be taken. Specific materials -- sulfate-resistant cement, non-expansive aggregate, and corrosion resistant steel -- can be selected once the specifics are known. If necessary, the lining can be of cast iron or ductile iron. A thick bitumen lining would provide the necessary outer protection so that standard materials could be used.

The amount of water outside the shaft also influences the design. In this regard, it is more a matter of pressure than of quantity. A veritable ocean of water can be stopped effectively by grouting if conditions are favorable. But a cup of water could impose very substantial pressure if it is in a small channel with a high effective head. If the pressure is not associated with an inexhaustible supply, it may be advisable to admit the water and relieve the pressure rather than to attempt to keep it out and resist the hydrostatic pressure.

If significant water inside the shaft is acceptable to the project operations, the shaft equipage must be corrosion resistant, a matter again of the proper choice of materials and coatings.

The more likely situation is that the material being mined will, as minute particles of it enter the ventilation air, develop a corrosive atmosphere. Galvanizing of all steel may be advisable. If so, every surface, including the holes for connections, must be treated. The solution may lie in the selection of corrosion resistant steels - copper bearing, Cor-Ten, Mayari-R, etc. or even stainless steel.

B. Outside Migration

The migration of fluid down the exterior of the shaft to the base of the shaft could occur even though impervious layers separate the foundation medium from the aquifer. Postgrouting should handle this condition. However, storage requirements may well require extreme precautions or a multiple protection approach. Picotage and its use have been described in an earlier chapter and may be necessary in addition to the postgrouting. Such an approach should guarantee that no water will reach the base of the shaft.

C. Observation Chases

Engineering guarantees notwithstanding, it must be proven that there is no downward migrating water, if this is a project requirement. The effectiveness of the water control can be demonstrated readily by the simple expedient of cutting an unlined chase perhaps four feet wide around the shaft well below the water shutoff point and above the operations elevation. If the methods were not as effective as expected, the chase would also provide a location and means for trapping any residual migration and pumping it to the surface.

D. Differential Settlements

Considerable attention has been paid in this and other chapters to the problems of water in shaft sinking. This is because it is a common problem. In salt, however, the most significant long term problems may be long term creep (settlement) under the shaft and differential settlements between shaft and surrounding formations due to areal settlements. This concern has prompted the considerable attention given elsewhere to bitumen protective linings. If a thick layer of properly designed bitumen surrounds the structural lining, it will absorb these movements without undue distress in the lining.

A shaft completed in July, 1976, near Lafayette, La., for the Diamond Crystal Salt Co. illustrates one problem and the solution. The shaft is 244 feet deep, constructed through unstable ground, and founded in a salt dome. The shaft was excavated within the protection of a freeze ring and first lined with a 12 gage outer corrugated iron liner 13 feet in diameter, and backed by concrete. A 0.375-inch thick steel liner 9.5 feet in diameter was then erected. Inside this liner, there is a precast concrete lining composed

of 69 rings each 40 inches high, with the final inside diameter being 8 feet. The 0.25-inch annulus between steel and concrete linings was grouted. The 18-inch annulus between the two steel liners was filled with a bituminous material to absorb the ground movements. The design considers that the two liners may eventually be in contact and makes allowance for the resulting bending stresses. The inner liners are supported on a foundation pad nine feet thick. Cost of the shaft was about \$3,000,000, or more than \$12,000 per lineal foot.

E. Earthquakes

The effect of an earthquake on an underground structure constructed by mining is entirely different than on an above ground structure. Above ground, the structure shakes and is damaged by its own response to the ground action. Below ground, the structure is constrained to the movements occurring in the ground. These movements are a small fraction of an inch in the bedrock, but may be amplified to several inches as they travel upward through a very deep alluvium or soil. So long as the shaft structure is flexible enough to deflect with the earthquake waves, there will be no basic damage. This assumes, of course, that there is no sharp cleavage or offset in the ground as occurs immediately across the fault. Obviously, no engineering structure should be located astride an active fault if it can be placed elsewhere.

Despite its apparent mass, the shaft is an extremely flexible structure insofar as seismic response is concerned. The deflection of the top of the shaft with respect to its base is a function of the fourth power of the ratio of its length to diameter. Since this length-diameter ratio is unlikely to be less than 60, the extreme flexibility of the shaft is apparent. The shaft, therefore, will be constrained to the limits of motion in the surrounding medium. These are unlikely to exceed 1/200th of the shaft depth. Therefore, the fact that a major earthquake will produce stresses of less than one psi in the shaft lining is readily understandable.

F. Alternative Layouts

One layout of the proposed storage facility has been investigated within the limited scope of the current overall assignment. Other layouts are possible. The intent here is to show that variations on that layout, or other layouts, are practical with different shaft configurations.

Ventilation is always a problem with underground mines. Intake and exhaust ventilation can be handled in the same shaft, or separate shafts, downcast (supply) and upcast (exhaust), can be used. If ventilation quantities and underground travel distances are limited, a single shaft probably is the answer. If the quantities are large, or the underground distance extensive, separate shafts are almost certain to be more economic. This is the reason ventilation shafts are continually being added to coal and other

mines. As an example, if it is 1.5 miles from a single ventilation shaft to the furthest heading, the air must travel (be propelled) a distance of 3.0 miles, with its quality constantly deteriorating en route. On the other hand, separate shafts would halve the travel distance, reducing the power required. It would also reduce the shaft size (beyond a certain diameter, shaft costs begin to increase rapidly) and eliminate the portion of shaft equipment required for separation of air flows.

Another facet of the project is the constantly increasing demand for muck transport over the years, ultimately on the order of 12,000 tons per day or more. Constructing the initial shaft to the ultimate capacity may result in excessive initial cost. It should be noted also that this high volume occurs for only a very few years, after which mining will cease completely. During these final few years of construction there will be a large demand for material to refill the caverns constructed earlier, on the order of 40 percent of the material excavated, apparently. Economy might be achieved if the reused material were not hoisted to the surface but rather stored temporarily until required. Muck to be reused would be hoisted to a reprocessing level via a separate relatively small diameter raise, reducing the size of the principal shaft accordingly. Suitable isolation of the reprocessing area could be achieved by a controlled air flows and a system of conveyor belts for delivery back to the principal level.

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