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On Recommendations for Abandonment of the Wills-Weaver mine and mine shafts, Carlsbad, New Mexico

By

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This report has not been edited for conformity with U.S. Geological Survey editorial standards or stratigraphic nomenclature.

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ABSTRACT

The technical question of principal interest to the abandonment of the Wills-Weaver mine shafts is whether fluids should be allowed to migrate down-shaft into the mine workings. The possible requirements for inhibiting fluid ingress can be evaluated by considering the potential impacts of entry of water into the workings.

It is recognized that it is not practicable to construct a perfect shaft seal and it is considered that eventually the mine void will be filled with saturated brine. It might therefore be argued that elaborate measures to achieve limited rates of water inflow will simply delay the inevitable. The reason that this is not the case is related to the flow behavior exhibited by the salt-like rocks of the mining horizon. Continued creep of the supporting pillars results in an ever decreasing mine void volume and reducing the rate of water ingress will result in a lower volume of brine within the underground void when full saturation finally occurs.

The question to be addressed therefore is what are the desirable consequences of achieving a smaller rather than a larger volume of brine located within an underground reservoir at the mine horizon level. In addition, it is also necessary to understand what inflow rates should be achieved in order to be effective and for how long it is necessary to maintain these low levels of inflow.

Impacts

Ingressing water from the aquifers overlying the Salado Formation is relatively fresh in terms of its dissolution

capacity and large volumes of salts will be dissolved. Hence, the volume of brine within the mine void will be substantially larger than the volume of fresh water which enters the mine as a result of saline minerals going into solution. An important question is whether or not the brine will remain contained within the present limits of the mine workings. There are two mechanisms by which lateral migration of the brine might take place:

- Flow through the salt (e.g., along bedding planes, etc.) as a result of the natural permeability of the material.

- Lateral propagation by salt dissolution.

Significant migration of brine from the mine reservoir by the former process is considered unlikely because of the low in situ permeability of the formation. Lateral propagation of a dissolution front, however, is a real probability at Wills-Weaver if significant volumes of water enter the mine workings.

The critical element responsible for this predicted behavior is the occurrence of the highly soluble mineral carnallite which exists at the mine level and extends for substantial distances external to the mine limits. This material is very soluble even within an halite/sylvite saturated brine and its distribution in relation to the mine void implies that it will continue to be exposed to carnallite unsaturated brine until substantially all of the mine fluid is saturated in magnesium chloride. It is estimated that the total volume of carnallite available for dissolution within the present mine void is sufficient to saturate only a few million cubic feet of water. If the volume of water which enters the workings is substantially greater than this, its dissolution capacity can be satisfied only by carnallite located external to the present mine limits. Since

the equivalent thickness of pure carnallite available for dissolution is small (1.5 to 3.0 inches), a dissolution front may extend considerable distances in order to dissolve a sufficient volume of carnallite to saturate the mine water. For example, it is estimated that based on a mine void closure rate of 150,000 cubic feet per year, an inflow rate of approximately 130 gph will result in about 30 million cubic feet of water entering the mine void over a period of 200 years. The average dissolution front would have to extend several thousand feet from the present mine limits in order to saturate this volume of water, and in specific directions the dissolution front may extend considerable distances (e.g., several miles).

It is evident that such significant lateral growth of a dissolution front is undesirable in terms of the potential impacts on future mining or other activities in the region of Wills-Weaver. A method of mitigating the impacts of such a scenario is to limit the amount of brine available by keeping inflow rates sufficiently low that the majority of the mine void is lost by natural closure of the horizon.

A region that is particularly susceptible to dissolution damage is the point of water ingress into the workings, i.e., the shafts. If water is permitted to migrate directly into the mine station, dissolution in this area can be expected to ultimately result in substantial movements within the overlying strata through which the shafts are constructed. This behavior may cause the eventual exposure of the Salado salts behind the liners and dissolution of these salts will occur. The lower portions of the shafts will eventually be lost and a considerable dissolution void may develop in the Salado salts surrounding the shafts. The size of this void may become significant enough that local subsidence through to the surface will occur. In addition, of

course, the water producing aquifers which are located a relatively short distance above the Rustler/Salado contact would be intersected and inflow rates could be expected to increase markedly.

The above impacts can be mitigated by minimizing the total quantity of water which enters the mine voids through the shafts. In addition, of course, it is necessary to ensure that dissolution damage to the mine station area does not occur. If this latter condition is not satisfied, the resulting sequence of events will make it highly improbable that the former requirement (i.e., limiting water ingress) can be satisfied.

Requirements

A minimum rate of mine void closure as a result of creep of the mine pillars is estimated to be 100,000 cubic feet per year. In order that the volume of water ultimately existing within the mine void be limited at saturation, it is necessary that the rate of loss of void space caused by water inflow be small in relation to the rate of loss of void space by mine closure. For example, an inflow rate of 10,000 cubic feet per year (8.5 gph) will result in approximately 5 million cubic feet of water entering the mine before it is fully saturated after a period of about 500 years. It would therefore appear appropriate that the decommissioning method should limit water inflow rates to a few gallons per hour and that it be capable of maintaining the low inflow rates for a period of several hundred years.

Potentially, the overlying aquifers can provide very large rates of flow, e.g., during grouting of the Culebra aquifer in sinking the No. 3 shaft, one grout hole made in excess of 130 gpm. Flow rates of this magnitude would fill the mine void within a few years.

Shaft Abandonment Procedures

A variety of potential abandonment alternatives are shown in schematic form on Figure 1. These procedures are briefly discussed in terms of their ability to satisfy the hydrologic criteria discussed above.

Procedure A - Clay Plug on Backfill Support

The use of a compacted clay shaft plug supported by compacted insoluble backfill is considered to provide the technically optimum solution to the problem of permanent abandonment of the Wills-Weaver mine shafts. This abandonment procedure would be capable of minimizing inflow into the mine void for the time periods considered appropriate.

Insoluble fill used to support the clay plug must be placed to maximum density in order to limit settlements, and the clay plug must be of sufficient dimensions and composition to withstand estimated settlements without compromising the seal. The fill should be adequately contained within the shaft mine stations and ingressing water should be drained away so as not to cause mine station dissolution which would compromise the containment of the fill within the shafts.

The placement of such fill within the mine shafts is expensive, and the estimated decommissioning costs for both shafts is $730,000. Although this abandonment procedure is considered to be technically optimum, it is unlikely to be acceptable because of the substantial costs.

Procedure B - Mass Concrete Plug

Although a mass concrete plug could be constructed which would perform satisfactorily in the short-term, the presence of high sulfate content groundwater indicates that the integrity of the plug could not be guaranteed for the time period of interest (several hundred years). High pressure water (100 psi) on the upper surface of the plug would result in flow through the plug and the great majority of the volume of concrete would be exposed to sulfate attack. The estimated decommissioning costs for both shafts is $245,000 but this abandonment procedure is not recommended.

Procedure C - Hybrid Clay/Concrete Plug

The use of mass concrete to support a clay shaft plug would offer substantial cost economies compared with backfill support. The estimated cost of decommissioning both shafts with clay plugs, supported by mass concrete bulkheads, is approximately $360,000.

The long-term performance of the concrete could not be guaranteed. In order to maximize the expected life of the concrete plug, the design should ensure that a minimum volume of the concrete is exposed to the high sulfate content groundwater. This could be achieved by incorporating a drainage blanket at the base of the clay plug and including drainage holes through the concrete bulkhead. High pressure water would not then develop on the upper surface of the bulkhead and fluid passing through the clay material would be drained away.

Although there are doubts about the longevity of this abandonment procedure, it is recommended as a cost-effective

method of permanently decommissioning the Wills-Weaver mine shafts so as to minimize water entry into the mine horizon. The concrete plug should, of course, be designed to offer maximum resistance to sulfate attack, i.e., low water-cement ratio, rich mix, low C_3A content cement, etc.

For a number of other shafts in the potash mining area, this method of abandonment would be considered technically preferable to the procedure which utilizes backfill support. A number of shafts are not concrete lined though the salt section and any water passing through the shaft plug would cause dissolution of the walls of the shafts within these regions. Such action could result in settlement of the shaft backfill and subsequently cause the clay plug to fail. A clay/concrete hybrid plug would tend to be isolated from such problems unless the dissolution void became so large as to cause disturbance to the overlying sediments in which the plug was located.

Procedure D - Clay Plug Through Aquifer Zone

This abandonment procedure is a modification of either Procedure A or Procedure C discussed previously, and involves the extension of the clay hydraulic plug through the entire aquifer zone, rather than the use of a limited volume plug positioned beneath the aquifers. Such action will tend to inhibit water mixing within the shafts but the impacts of this are considered minor because of the uniformly low water quality of the region.

Extension of the clay plug through the aquifer zone will not add appreciably to the efficacy of the seal, even if the fill is compacted to the optimum condition. Placement of such material is an expensive item and is not considered to be cost-effective. It is not, therefore, a recommended abandonment procedure.

Procedure E - Cap Shaft and Grout Aquifers

If grouting of the aquifers can limit inflow rates to a few gallons per hour and the shaft caps are constructed so as to permit the natural air convection cycle to exist, this abandonment method is the preferred temporary decommissioning scheme. It is considered that this procedure would provide satisfactory abandonment for a number of decades before deterioration of the aquifer seals resulted in excessive rates of shaft water inflow. This procedure is not recommended for permanent abandonment of the mine shafts.

The estimated decommissioning cost for both shafts is $245,000. The technical advantage of this abandonment procedure is that it is considered to provide maximum protection to the shaft liners from the effects of sulfate attack, and this is an important consideration for temporary abandonment where it is presumed that the shafts will be utilized again at some time in the future.

The above costs are substantial if it is considered that the shafts would still require permanent decommissioning at a later date. Unless the probability of future utilization of the shafts was high, it would be more appropriate to permanently abandon the shafts.

Procedure F - Backfill Shafts with Uncompacted Sand

The placement of noncemented fill of any kind will not contribute significantly to the structural integrity of the shafts, and loose sandfill dumped from the surface would not add appreciably to surface security as a result of the expected development of a substantial settlement void beneath the cap.

This method of abandonment would not provide any hydrologic control of shaft water as the permeability of the in-place material would be sufficiently high so as to allow infiltration of flow rates in excess of those likely to be encountered from the overlying aquifers. The estimated decommissioning cost is $48,000 and the method of abandonment is not recommended. The above cost does not include measures taken to protect the mine station from dissolution damage.

Procedure G - Backfill Shafts with Uncompacted Clay Fill

As discussed above, such fill will have a minor impact on the structural integrity of the shafts. In addition, the quality of the hydrologic control which would be exhibited by such an abandonment procedure is considered suspect. Although the increased plug thickness (compared to Procedures A or C) would reduce hydraulic gradients, the in situ permeability of uncompacted clay fill could be several orders of magnitude greater than that of compacted material. The overall hydraulic conductivity of a shaft abandoned in this manner might therefore be two to three orders of magnitude greater than that of a shaft decommissioned by use of a discrete compacted clay plug. There is considerable doubt, therefore, that such an abandonment method could satisfy the hydrologic criteria considered necessary to prevent the impacts outlined previously.

It is estimated that decommissioning costs for both shafts would be on the order of $436,000. This high cost derives from the relatively large volume of material required to fill the shafts (cost of procuring, transporting and preparing) together with the cost of placing the fill. Although it is not compacted, such fill should be dumped at the working level. It is not considered feasible to place such material by dumping from the surface in shafts containing all their original fixtures. The

decommissioning cost is sensitive to the availability of acceptable clay fill. If suitable material is not available within 20 miles of the mine, costs could be substantially higher. In addition, no allowance has been made for improving the quality of the clay by bentonite addition, for example. Depending on the type of clay available, this could be an important consideration for material to be placed in uncompacted condition.

It is not recommended that this decommissioning procedure be applied to the Wills-Weaver mine shafts as a result of its high cost and suspect technical performance.

Procedure H - Shaft Cap

The placement of shaft caps on the Wills-Weaver mine shafts, together with concreting of the shafts' pump stations, would be a very inexpensive method of shaft abandonment. Although such a decommissioning procedure offers no structural or hydrologic control, it is considered to be comparable to Procedure F and possibly not significantly inferior to Procedure G.

At Wills-Weaver, there would also be benefits in protecting the shaft mine station areas from dissolution by placing clay blankets to conduct ingressing water away from the shaft pillar region. Such action would prevent dissolution in this region with subsequent subsidence of the overlying strata and probable exposure of the Salado salts behind the liners. The estimated cost of capping the shafts, concreting the pump stations, and protecting the mine stations from dissolution is $160,000.

Much of the estimated cost is related to mine station protection. Although such action could be effective at the Wills-Weaver mine, there are a number of other shafts in the potash fields of southeast New Mexico which are unlined through the salt. If these latter shafts are not effectively sealed by

use of hydraulic plugs, it would appear ineffective to apply abandonment procedures which were more elaborate than simple capping. It could therefore be argued that the additional expenditure (related to mine station protection), applied to only some of the mine shafts (e.g., Wills-Weaver) of the potash mining area, cannot be justified.

1.0 INTRODUCTION

1.1 BACKGROUND

This report describes a study of the Wills-Weaver potash mine and mine shafts undertaken by Golder Associates for the Department of the Interior, U.S. Geological Survey. The shafts and workings of the potash mines of southeast New Mexico are located on public land and are held under Federal Potassium Leases. At present, there are several potash mines in southeast New Mexico in which mining operations have ceased. The U.S. Geological Survey is currently evaluating whether all potash reserves have been recovered from the Wills-Weaver workings. Over the next few decades it is expected that approximately 30 shafts will be abandoned within the Potassium Leases. Since no procedures for proper abandonment of the mines and mine shafts exist, there is a requirement for the development of abandonment regulations and recommendations. Proper abandonment has been defined as that which results in minimum environmental impacts, minimizes the potential for future public liability, allows permanent release of bond to the leasee, and fulfills all other state and federal government regulations.

In January 1979, Golder Associates was awarded a firm fixed-price contract to develop abandonment procedures for the Wills-Weaver mine of Duval Corporation. It is anticipated that the developed procedures and recommendations will assist in formulating guidelines for mine and shaft abandonment at other mines located within the southeast New Mexico Potassium Leases.

1.2 HISTORY OF MINE DEVELOPMENT

The Wills-Weaver potash mine of Duval Corporation is located approximately 30 miles northeast of Carlsbad, New Mexico (Figure 1-1), in the SW 1/4, sections 14, 15, 22, and 23; T. 18S, R. 30E; New Mexico Principal Meridian. The shafts and workings are located on public land and are held under Federal Potassium Leases NM-016330-A and NM-016330-B. The mine has not been worked since 1971.

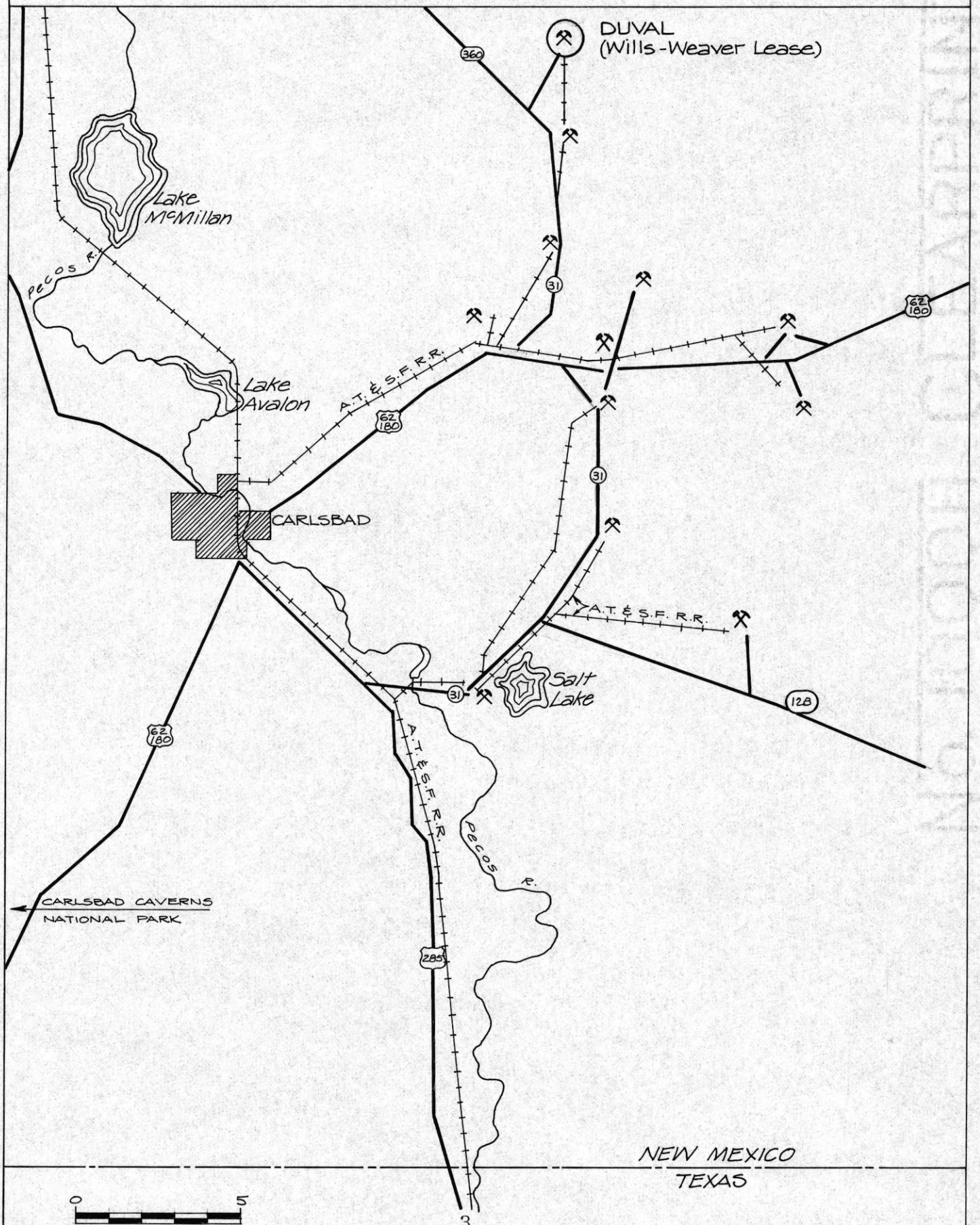
There are two mine shafts and approximately 720 acres of workings (Figure 1-2). Room and pillar mining methods were used on first mining, resulting in an average extraction ratio of about 60 percent. The production rooms were mined at just over 4 feet in height by drilling and blasting, following overcutting and undercutting for blasting efficiency and dilution control. Subsequently, pillar mining over considerable areas of the workings raised extraction ratios to in excess of 70 to 80 percent. Large areas of the mine have therefore either closed completely or are inaccessible for safety reasons. Within the vicinity of the shafts, located in the extreme southwest of the workings, a region of low extraction has been left in order to protect the shafts from subsidence damage. This area extends some 550 feet from the No. 3 shaft and 850 feet from the No. 4 shaft. The depth of the mining horizon is approximately 1,000 feet.

The mine is accessed by two shafts which were sunk in 1960 and designated No. 3 and No. 4. The shafts are concrete lined over their entire lengths with minimum lining thickness of about one foot. In specific regions (e.g., aquifer zones), the concrete linings are substantially thicker.

The No. 3 shaft has an internal diameter of 14 feet and a total depth of 1,082 feet. This shaft is presently making small

LOCATION OF WILLS-WEAVER SITE

Figure 1 – 1



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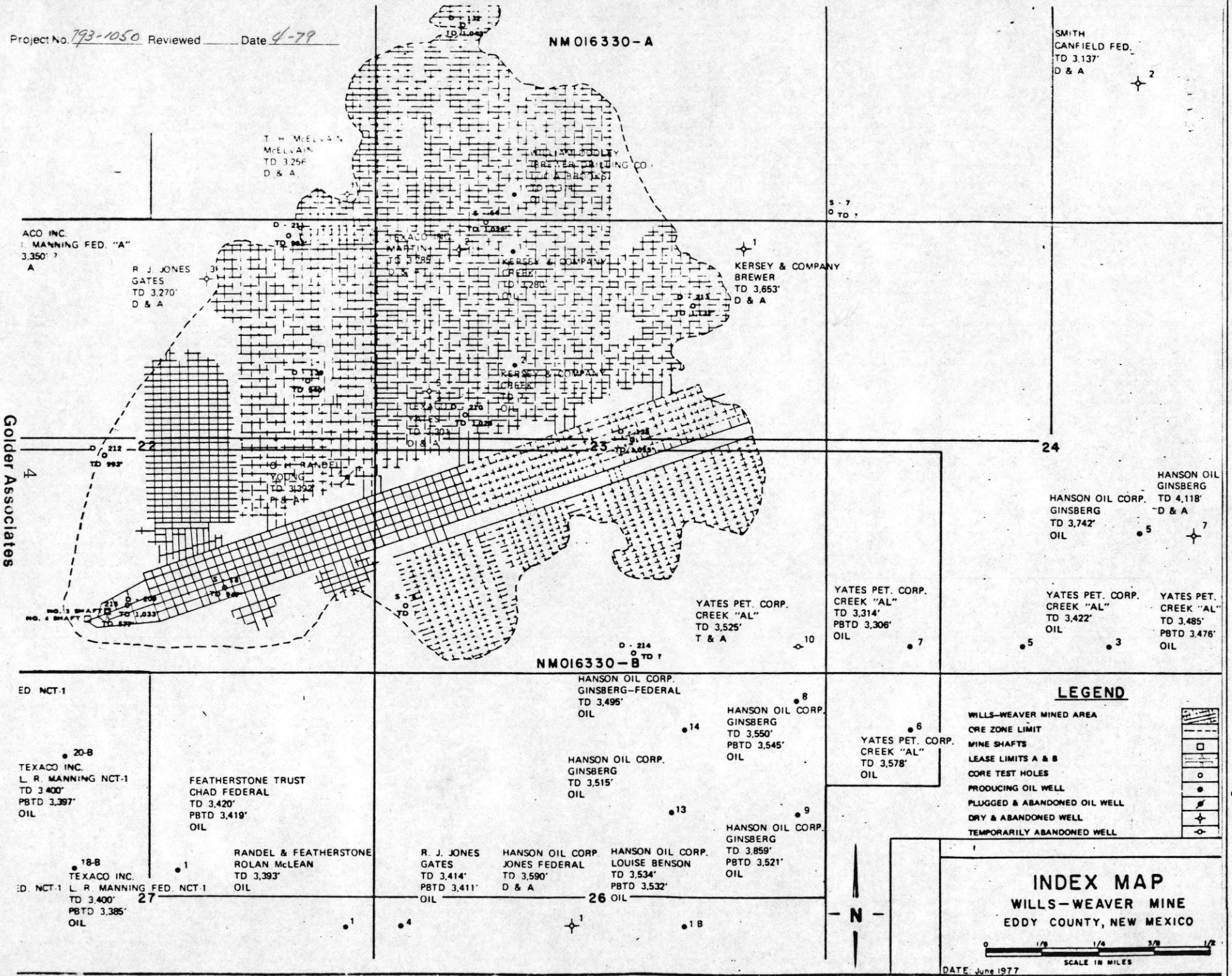

LEGEND

- WILLS-WEAVER MINED AREA
- CRE ZONE LIMIT
- MINE SHAFTS
- LEASE LIMITS A & B
- CORE TEST HOLES
- PRODUCING OIL WELL
- PLUGGED & ABANDONED OIL WELL
- DRY & ABANDONED WELL
- TEMPORARILY ABANDONED WELL

INDEX MAP

WILLS-WEAVER MINE

EDDY COUNTY, NEW MEXICO

0 1/8 1/4 3/8 1/2

SCALE IN MILES

DATE: June 1977

WILLS-WEAVER MINE PLAN

Figure 1-2

amounts of water (10 to 35 gph), most of which is piped directly into a worked-out area of the mine. There is about 30 feet of water in the tailshaft. During the operating life of the mine, the No. 3 shaft water was piped to storage tanks located in the shaft pump station (approximately 660 feet below the shaft collar) and periodically pumped to the surface.

The No. 4 shaft is about 300 feet southwest of the No. 3 shaft and has an internal diameter of 9 feet and a total depth of about 1,011 feet. Ingressing water is completely evaporated, and no water is presently entering the workings through this shaft. There is about 4 feet of water in the No. 4 tailshaft.

1.3 SCOPE OF WORK

The total scope of work is outlined in the proposal and incorporated into the contract. The technical discussion of the proposal is included in Appendix A. This discussion outlines the major work items necessary to fulfill the conditions of the contract.

Phase 1 of the program involved principally data collection and evaluation. The end product of the first phase of the work was an estimation of the lives of the mine workings and mine shafts. This evaluation was based on the following information:

- Review of mine and shaft geology
- Review of mine and shaft hydrology
- Evaluation of relevant rock mechanics data
- Evaluation of present condition of mine workings
- Evaluation of surface subsidence effects
- Evaluation of present condition of mine shafts.

Phase 2 of the program evaluated abandonment options appropriate to the Wills-Weaver mine. The relevance of each abandonment alternative was determined, in part, by the conditions of the shafts and workings.

1.4 OUTLINE OF PRESENTATION

The format of this report is briefly described:

- Section 2 provides a general review of the regional and site geology.
- Section 3 discusses the regional and local surface and groundwater hydrology, together with the mine and shafts' hydrology.
- Section 4 reviews subsidence phenomena within the southeast New Mexico potash fields and considers data specific to the Wills-Weaver property.
- Section 5 evaluates the present condition of the mine workings. A review of pertinent rock mechanics data is also undertaken in order to explain present conditions and to provide the technology for predicting future behavior.
- Section 6 reports on the detailed conditions of the mine shafts.
- Section 7 evaluates the future performance of the mine workings and mine shafts.
- Section 8 considers the effects of permitting brine migration down the shafts' linings and into the workings.

- Section 9 evaluates shaft abandonment options for both short- and long-term abandonment and identifies the optimum solution for each case.

- Section 10 reviews possible alternative uses for the Wills-Weaver mine workings and mine shafts.

- Section 11 discusses procedures which might be applied to the waste stockpile located adjacent to the mine shafts.

A detailed discussion is presented within the text of each section. In order to improve comprehension and allow the casual reader the opportunity to review the work without getting involved in the details, each section (where appropriate) contains a summary review which itemizes the most significant points discussed within that section.

An inventory of documents researched during the work program is provided in Appendix B.

1.5 ACKNOWLEDGEMENTS

A significant item of the Phase 1 program was a review of data contained within USGS and Duval files. Full cooperation was offered by Mr. John Burleson, USGS, Carlsbad, New Mexico, and Mr. Jack Hunt and his colleagues of the Duval Corporation in providing the above data. During site visits, Duval staff provided every assistance, and their efforts are gratefully acknowledged.

Throughout the program USGS staff have reviewed this document, and their comments and suggestions have been most helpful. In particular, we would like to acknowledge the assistance and support of Mr. Don Libbey and Mr. Ed Sandell, Jr.

7 (p. 9 follows)

2.0 GEOLOGY

2.1 GENERAL GEOLOGY OF THE DELAWARE BASIN

The Delaware Basin in southeast New Mexico and western Texas is a crudely circular embayment roughly 90 miles in diameter bounded on three sides by the Capitan Reef Front and open to the southeast. Surface topography across the basin is characterized by low hills and widespread pediments and alluvial plains. These features are modified by drainage channels and local collapse structures. The Pecos River winds its way across the basin, northwest to southeast. The basin is underlain by Precambrian granitic, metasedimentary, metavolcanic, and volcanic rocks.

The potash mines lie generally east of Carlsbad, New Mexico, in the northern reaches of the basin. At this location, the basement rocks are overlain by approximately 16,000 feet of Paleozoic sediments. Some 5,000 feet of these are pre-Permian marine deposits; the remainder are Permian sediments grading upwards from limestone (Bone Spring Formation) through sandstone and shales (Brushy Canyon Formation), sandstones and limestones (Cherry Canyon Formation), the Capitan Reef limestone, dolomites of the Yates and Tansill Formations, evaporites of the Castile, Salado, and Rustler Formations (anhydrite-salt-anhydrite), and finally the terrestrial redbeds of the Dewey Lake Formation.

Overlying the Permian section, depending on locale, are sandstone beds representing the Santa Rosa and Chinle formations (Triassic), the Tertiary Ogallala Formation, Quaternary deposits of the Gatuna Formation, and younger alluvium. Beds younger than Permian may be missing completely, or preserved in sections ranging up to 1,000 feet in thickness.

2.2 GENERAL GEOLOGY ACROSS THE WILLS-WEAVER MINE AREA

The Wills-Weaver Mine area is shown on Figure 2-1 and is enclosed within sections 14, 15, 22, and 23; T18S, R30E; Eddy County, New Mexico. Ground elevation in the area, Figure 2-2, varies between 3,400 and just over 3,500 feet above sea level.

Terrestrial siltstones and sandstones, principally of the Dewey Lake Formation, range in thickness generally between 300 and 500 feet above the anhydrite of the Rustler Formation. The upper surface of the Rustler Formation is shown on Figure 2-3.

The Rustler Formation is primarily massive anhydrite with interbeds of siltstone, shale, and dolomite/limestone. The two principal dolomitic beds have been named: the Magenta Dolomite member (upper) and the Culebra Dolomite member (lower). These beds are impure dolomite, containing anhydrite and/or limestone.

Detailed stratigraphic information in the Rustler and overlying sediments is not available in the Wills-Weaver mine area, as all of the potash exploration drillholes were churn-drilled to the top of the Salado Formation. Summary logs of the cuttings are available. It is not always possible, from these, to identify even the dolomite members.

Between the Rustler and the massive halite of the Salado Formation is a zone of solution breccia, containing a mixture of salt fragments in a matrix of insoluble clays and sands. This is the "brine aquifer" that is most significant in the solutioning of the halite in the upper part of the Salado Formation.

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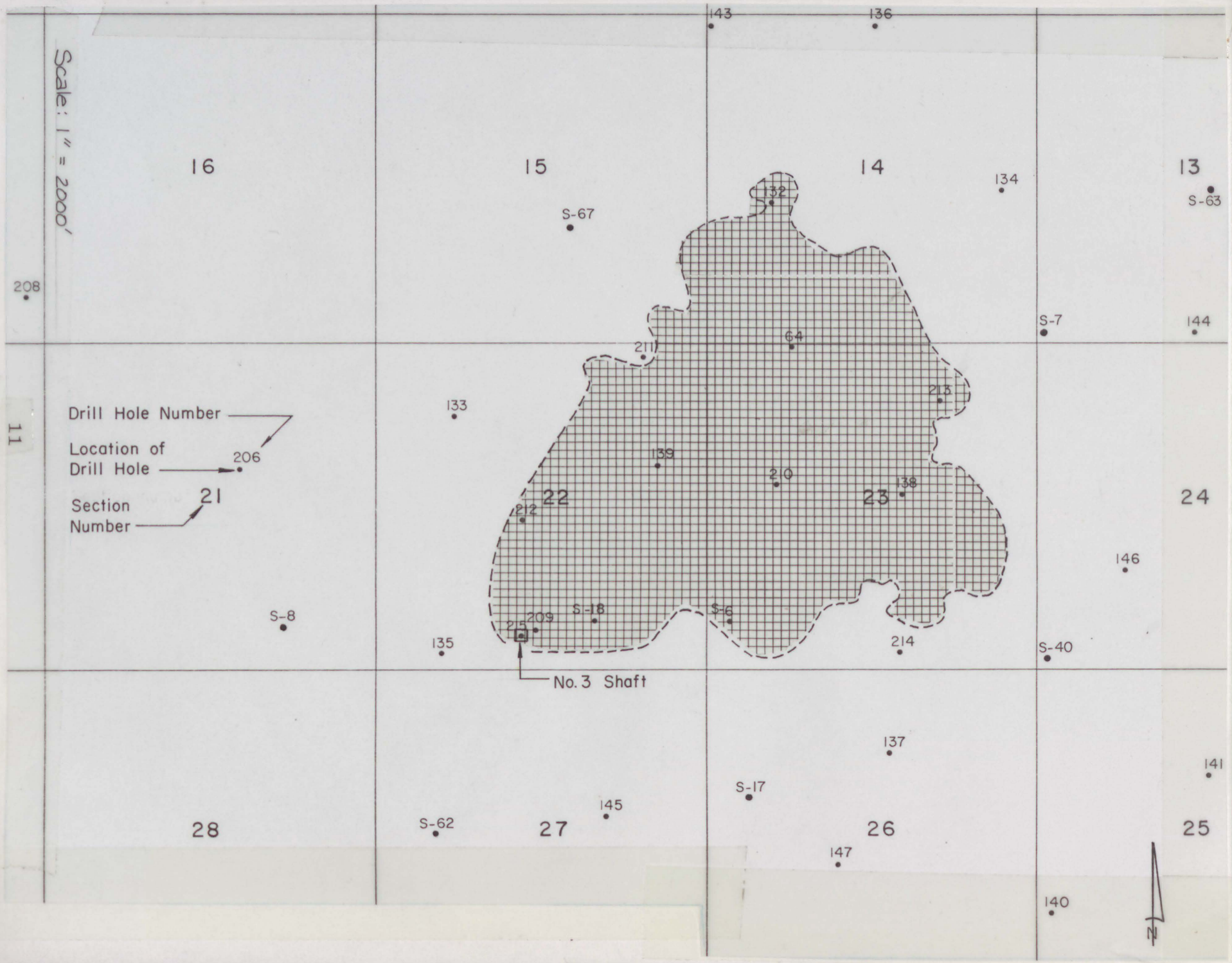

WILLS - WEAVER MINE AREA
T18S; R30E Eddy Co., N.M.

Figure 2-1

Golder Associates

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GROUND SURFACE TOPOGRAPHY (generalized)

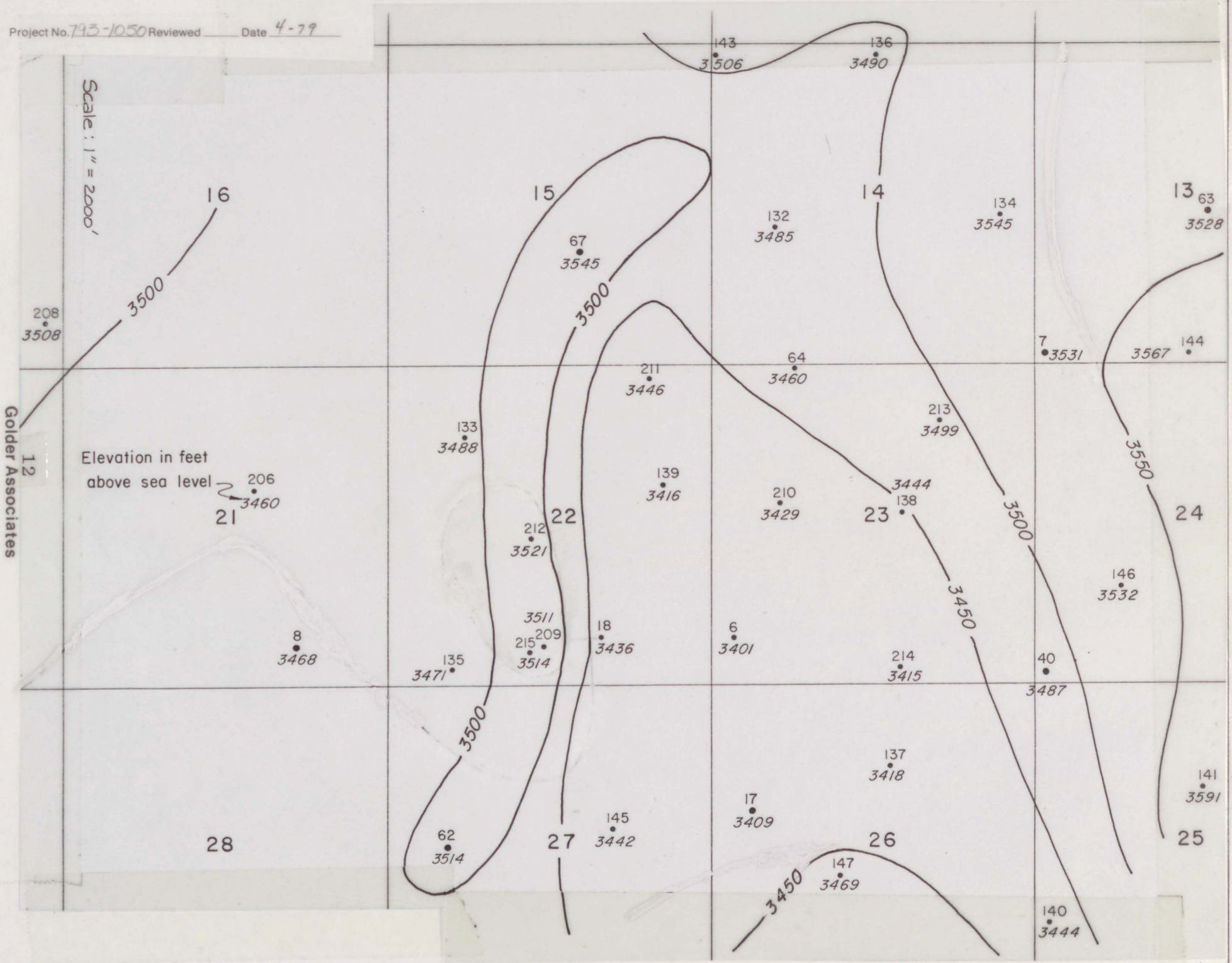
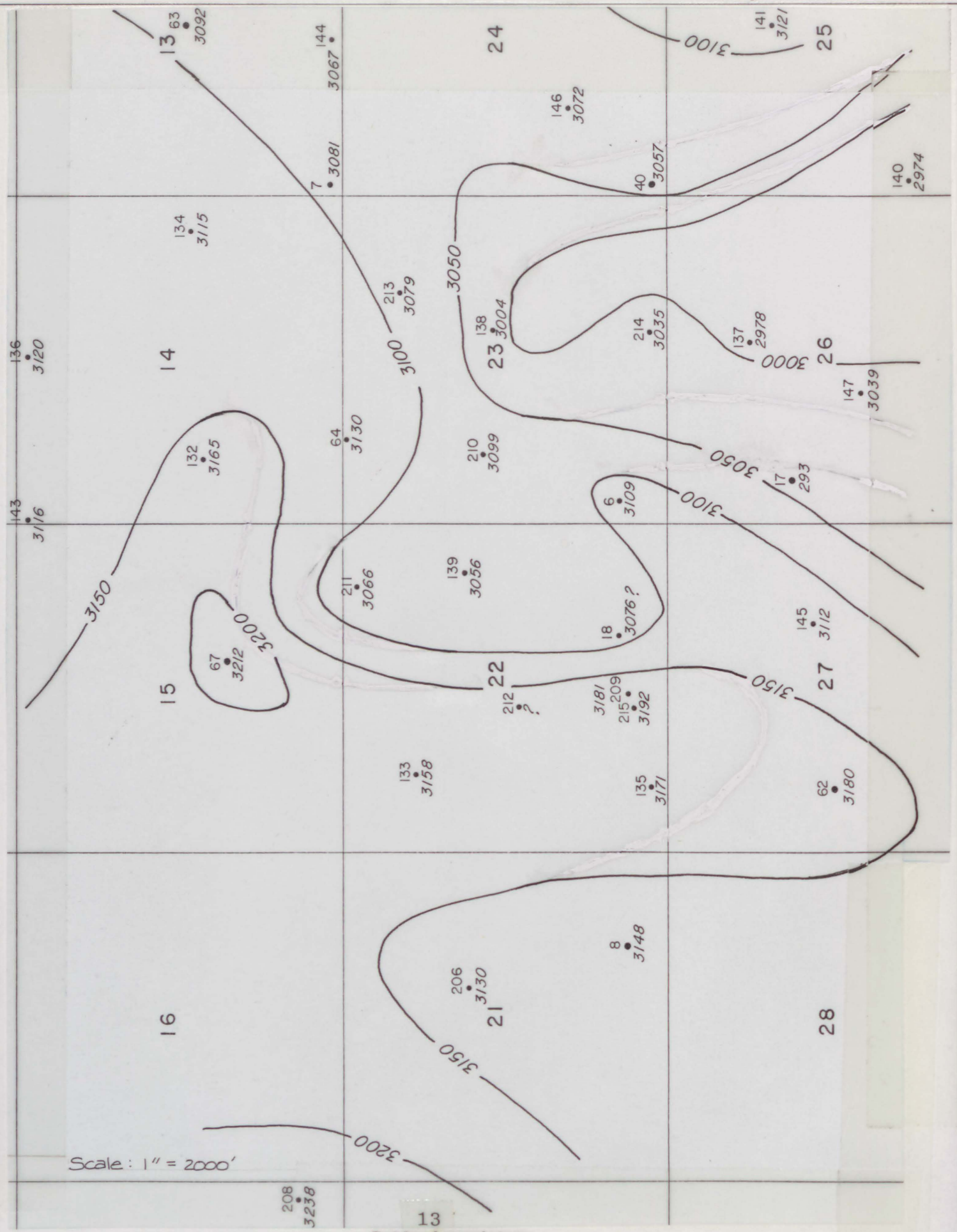


Figure 2-2

STRUCTURAL CONTOURS ON TOP OF RUSTLER Fm.

Figure 2-3



Project No. 793-1050 Reviewed ____ Date 4-79

Figure 2-4 shows the top of the massive salt of the Salado Formation. A section varying between 250 and 400 feet in thickness overlies the Wills-Weaver mine horizon shown on Figure 2-5. This section is primarily halite with varying minor percentages of clay and sand and numerous thin anhydrite beds. Stratigraphic details are shown on Figure 6-2, the No. 3 shaft section.

2.3 GEOLOGY OF THE WILLS-WEAVER MINE HORIZON

The First Ore Zone at Wills-Weaver dips uniformly to the east at about one degree. The elevation of the horizon at No. 3 shaft is about 2,520, dropping to 2,400 in the eastern extent of the mine. The ore zone averages 40 inches in thickness: 6 inches above and below were mined, resulting in a 52-inch room height.

According to Ed Sandell (USGS, Grand Junction--personal communication), the Wills-Weaver orebody is a lens of sylvite enclosed within an envelope of carnallite/halite, which thickens towards the periphery of the body. This in turn is enclosed within more massive halite of the Salado Formation. Mr. Sandell interprets this orebody as being the result of secondary enrichment processes which concentrated the potash salts from evaporite beds originally lower in potassium grade. The carnallite/ halite envelope is characteristic of sylvite lenses in the northern portion of the Delaware Basin, as it is also encountered in the northern reaches of the AMAX and National Potash Company's properties.

As shown on the assay profiles (Figures 2-6 and 2-7), the heart of the sylvite lens grades about 40 percent K_2O (equivalent weight percent). In the periphery, the grade drops to about 30 percent. (Pure sylvite runs 63.17 equivalent weight percent K_2O.)

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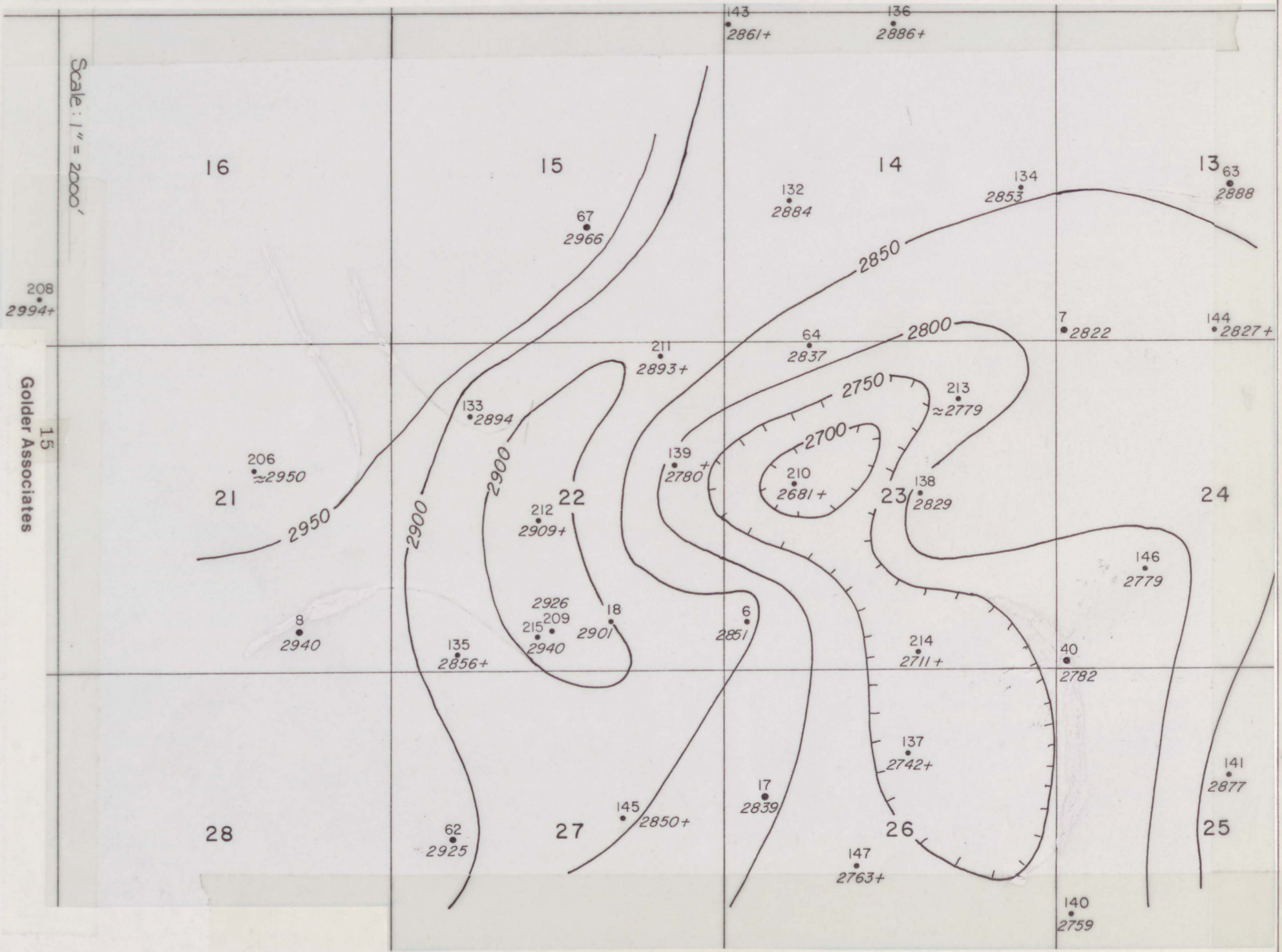

Golder Associates

TOP OF SALADO Fm.

Figure 2-4

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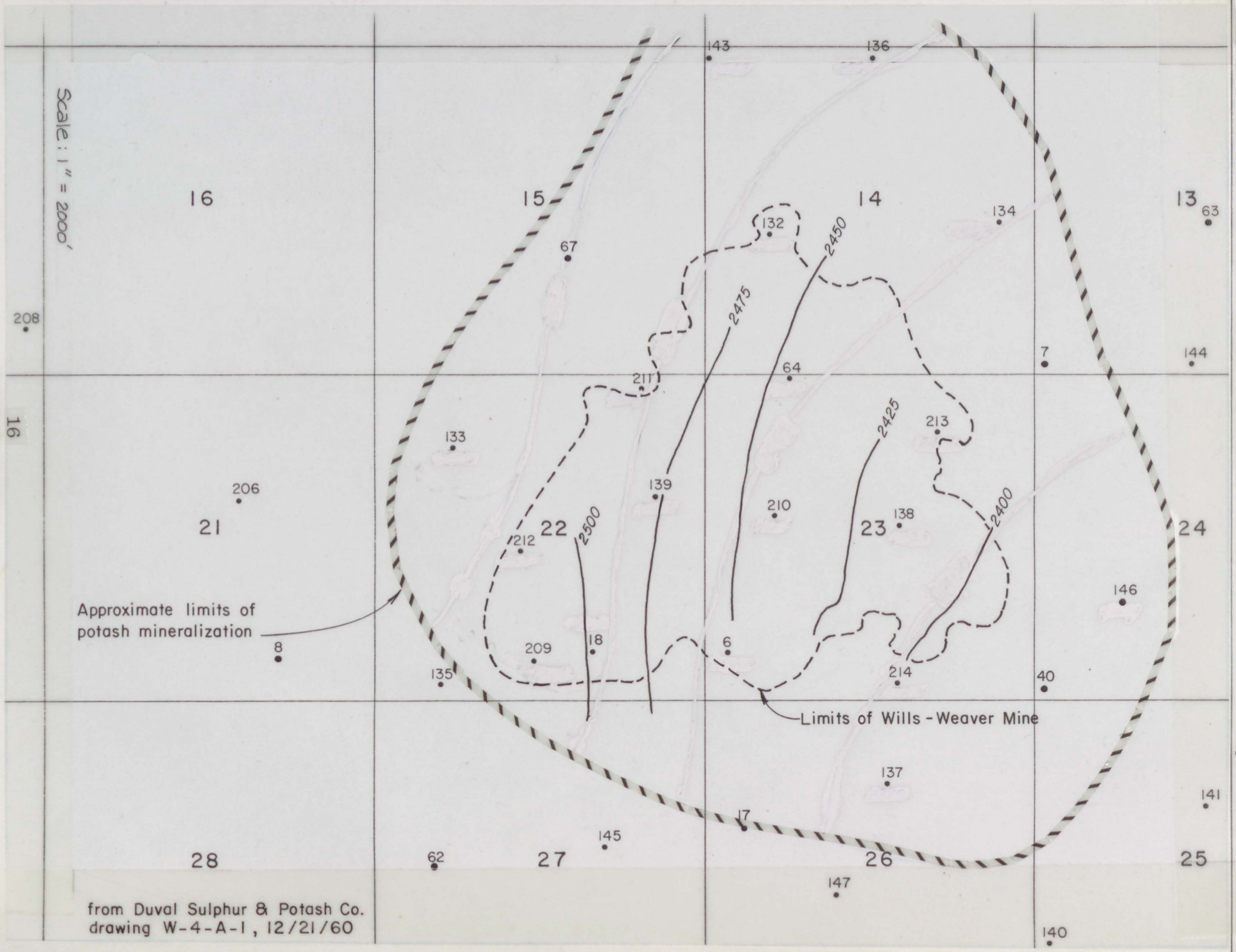


CONTOURS ON FLOOR OF WILLS - WEAVER MINE HORIZON

Figure 2-5

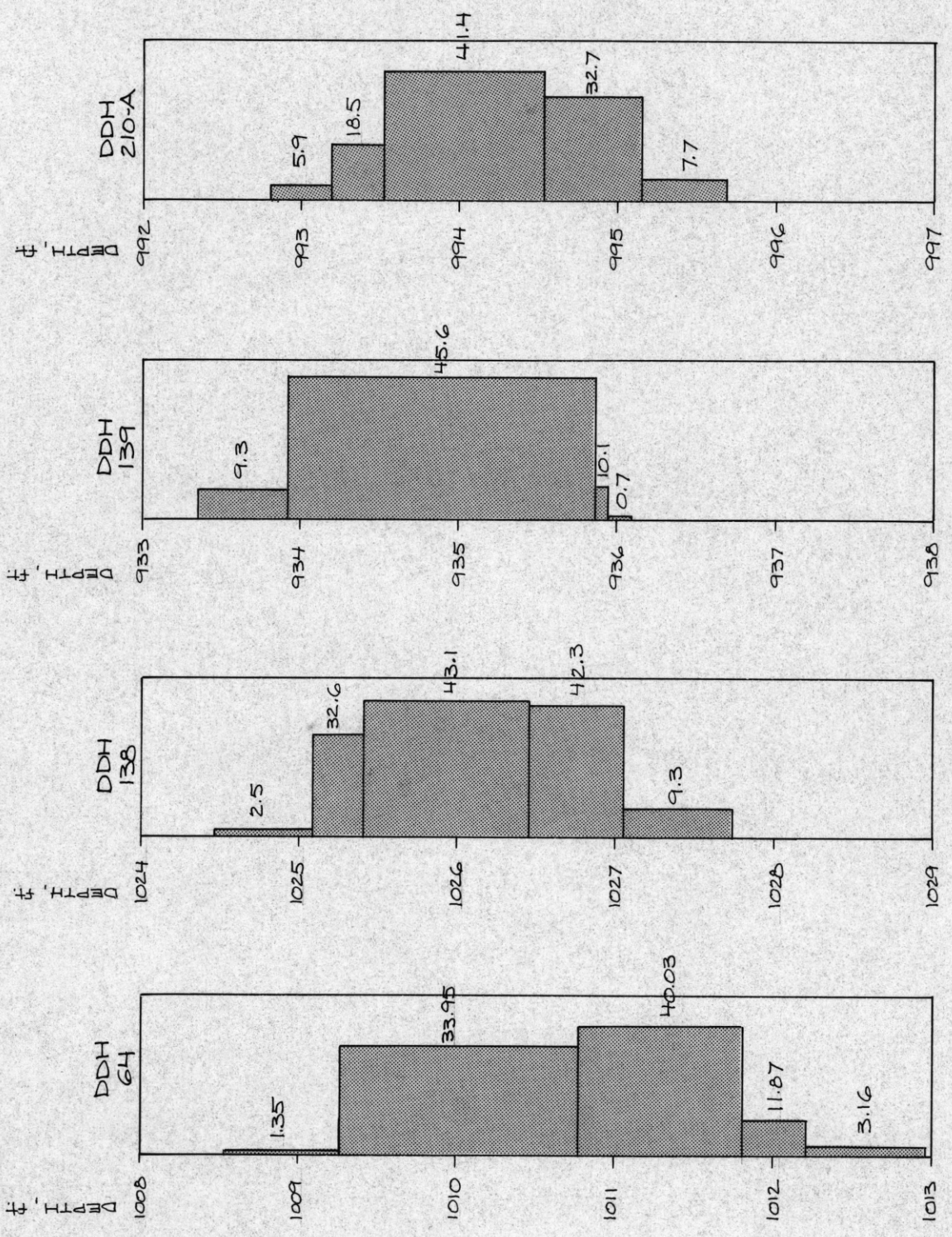
Golder Associates

K_2O ASSAY PROFILES IN CENTER OF WILLS-WEAVER MINE AREA

Figure 2-6

(First Ore Zone)

Vertical Scale: 1" = 1'
Horizontal Scale: 1" = 50% K_2O



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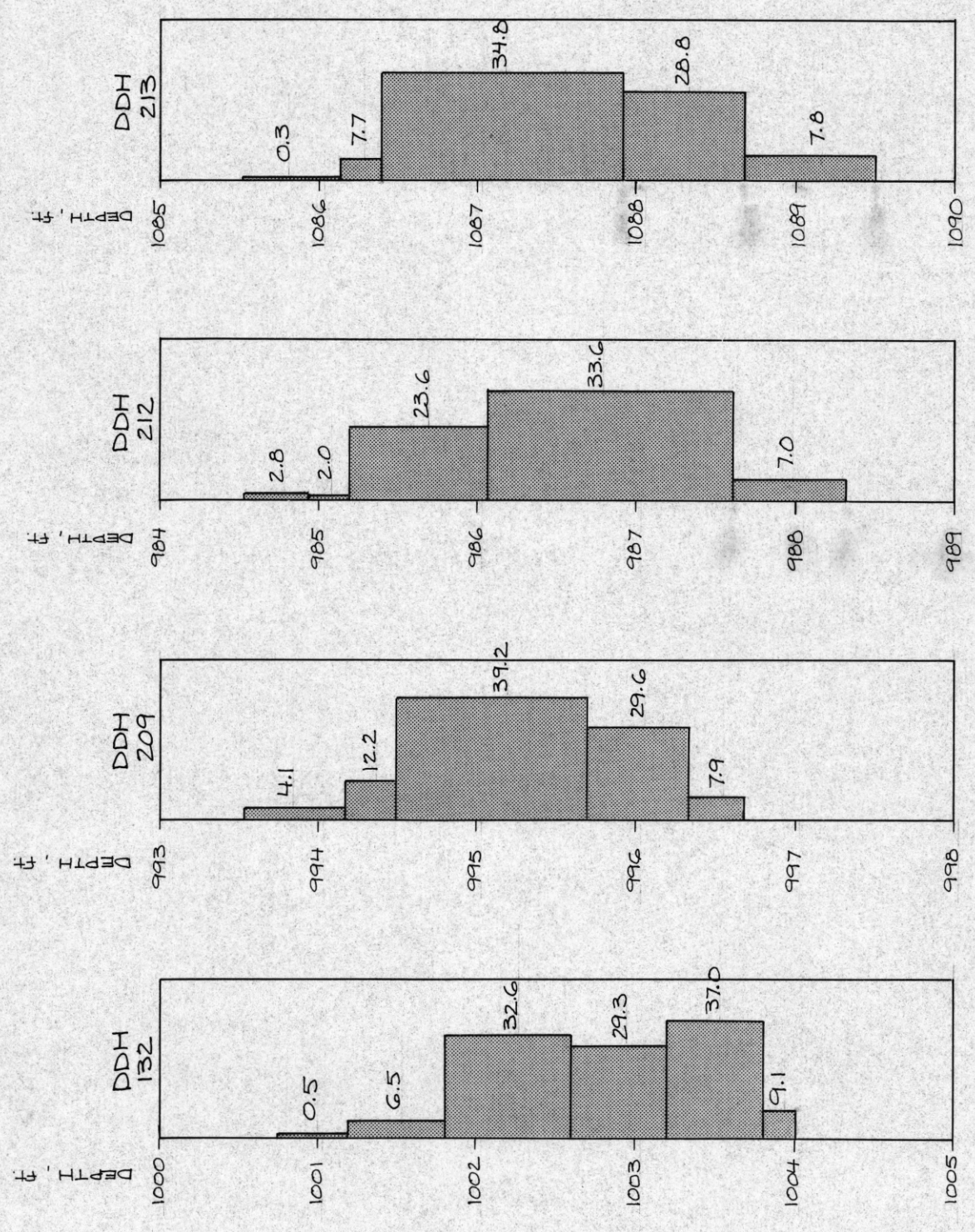
K_2O ASSAY PROFILES IN PERIPHERY OF WILLS-WEAVER MINE AREA

Figure 2-7

(First Ore Zone)

Vertcal Scale : 1" = 1'
Horizontal Scale : 1" = 50% K_2O



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Above the heart is a zone about 8 inches in thickness that assays at roughtly 1 to 5 percent. This zone is principally halite with the potash being represented as polyhalite and lesser amounts of carnallite. Underlying the high grade is a zone, also about 8 inches in thickness (varying between 6 and 18 inches), that runs near 7 percent. This lower zone is the carnallite/halite envelope as indicated by the high magnesium content. (Pure carnallite is 16.95 equivalent weight percent K_2O.)

The clay/polyhalite beds which occur throughout the salt sequence represent periods of invasion by unsaturated water and are therefore residue from solution phases in the evaporite deposition processes.

There is a 6 inch clay/polyhalite bed between 5 and 6 feet above the top of the First Ore Zone in the Wills-Weaver area and a clay parting directly beneath the underlying carnallite/halite zone.

In mining at Wills-Weaver, the 6-inch undercut was made in the lower carnallite/halite zone, so that much of this material, which was wasted, has been removed from the mine floor. It can be seen forming the base of the remaining pillars where it is distinguished by its sugary appearance.

3.0 HYDROLOGY

3.1 CLIMATE

The climate at the Wills-Weaver mine site and the surrounding Carlsbad area is semiarid. Average precipitation is 13 to 14 inches annually, with more than two-thirds occurring from June to October primarily as thunderstorms (Hendrickson and Jones, 1952). Large variations in yearly precipitation amounts are common.

The average temperature is about 44° F in January, rising to about 80° F in July, as measured at Carlsbad, New Mexico.

3.2 SURFACE WATER

The only through-flowing perennial stream in the Carlsbad area is the Pecos River. It is located about 20 miles west of the Wills-Weaver mine and flows towards the south. Near the town of Carlsbad it bends southeastward until it is located about 30 miles directly south of the mine. The Pecos River receives most of the surface runoff and a large portion of the subsurface discharge in the Carlsbad area.

Surface runoff at the Wills-Weaver mine flows into local depressions forming playa lakes. There it evaporates or infiltrates into the ground. A series of local depressions continues southward from the mine leading into a large-scale depression called the Clayton Basin.

3.3 REGIONAL GROUNDWATER CONDITIONS

3.3.1 Principal Aquifers

The principal aquifers in the area of the Wills-Weaver mine are the Magenta and Culebra dolomite members of the Rustler Formation (Figure 6-2). Other aquifers found in the surrounding region of Eddy County east of the Pecos River include the basal solution breccia zone of the Rustler Formation, the Santa Rosa sandstone, and alluvium. Groundwater is found in other units overlying the Salado Formation, but generally they yield only minor amounts of water to wells. Aquifers below the Salado are sealed off from the strata above the Salado by the thick intervening layer of salt and thus do not affect the hydrologic system in the upper units.

The Culebra dolomite member is the most important aquifer in the region. It supplies water for many stock wells and a few domestic wells. Transmissivity of the Culebra measured at the Project Gnome site approximately 30 miles south of Wills-Weaver ranged from 17.2 to 4,000 gpd/ft (Brokaw et al. 1972).

The Magenta dolomite member yields small amounts of water to stock wells in the region, but is generally not an important aquifer.

The basal solution breccia zone made up primarily of porous gypsum immediately overlies the Salado. It is a very productive saline aquifer from Nash Draw southwest to Malaga Bend (Hendrickson and Jones, 1952). The transmissivity of the breccia zone was measured to be 60,000 gpd/ft at the Project Gnome site (Brokaw et al. 1972). The zone overlying the Salado in the area of the Wills-Weaver Mine contains anhydrite, gypsum, and much

brown clay, as described in the drill log of Hole DSP 209. It does not appear to be a water producing zone.

The Salado shows an irregular surface at its contact with the Rustler indicating solutioning and removal of salt by groundwater. Depressions in the surface of the Salado often correspond to depressions in the land surface such as at Nash Draw and parts of the Clayton Basin (Hendrickson and Jones, 1952).

3.3.2 Direction of Flow

Strata overlying the Salado are apparently hydrologically connected and act as a single multi-layered aquifer system. Average vertical permeability is probably much lower than the average horizontal permeability of the aquifer system owing to numerous horizontal layers of anhydrite, gypsum, and claystone which have low permeability.

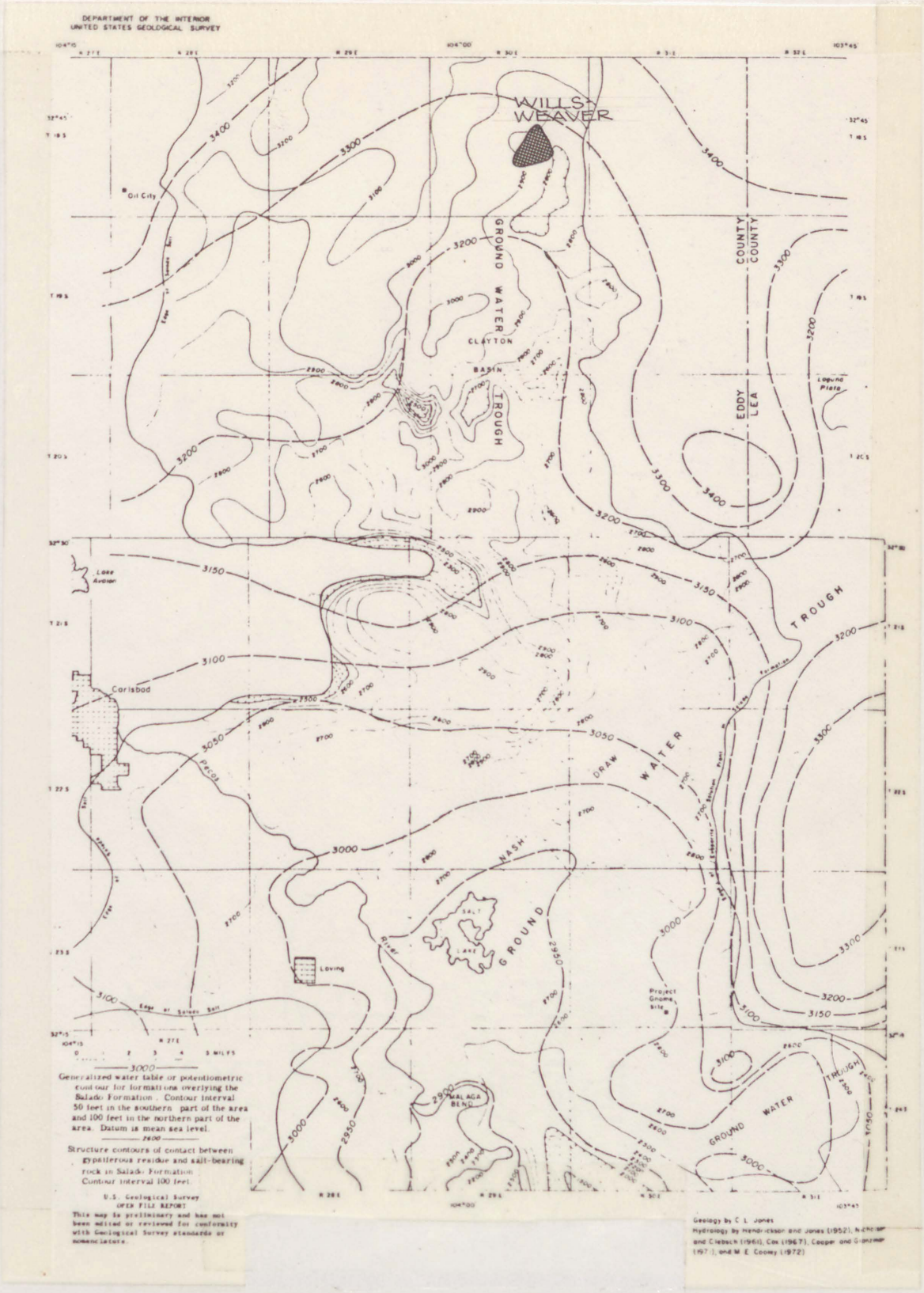
Regional piezometric levels as shown on Figure 3-1 indicate that regional flow within the aquifer system overlying the Salado is to the south and southwest towards the Pecos River at Malaga Bend. Piezometric levels define several groundwater troughs, including the Clayton Basin and Nash Draw. These troughs probably signify zones of increased permeability due to solutioning of salt and the formation of a breccia zone.

3.3.3 Water Quality

Water quality is generally low within the units overlying the Salado in the entire region surrounding the Wills-Weaver mine, as shown on Figure 3-2. The water is used for stock and a few domestic supplies where it is potable. Water is generally impotable where the Salado is close to the surface.

REGIONAL PIEZOMETRIC LEVELS

Figure 3-1



Reference: Brokaw et al, 1972

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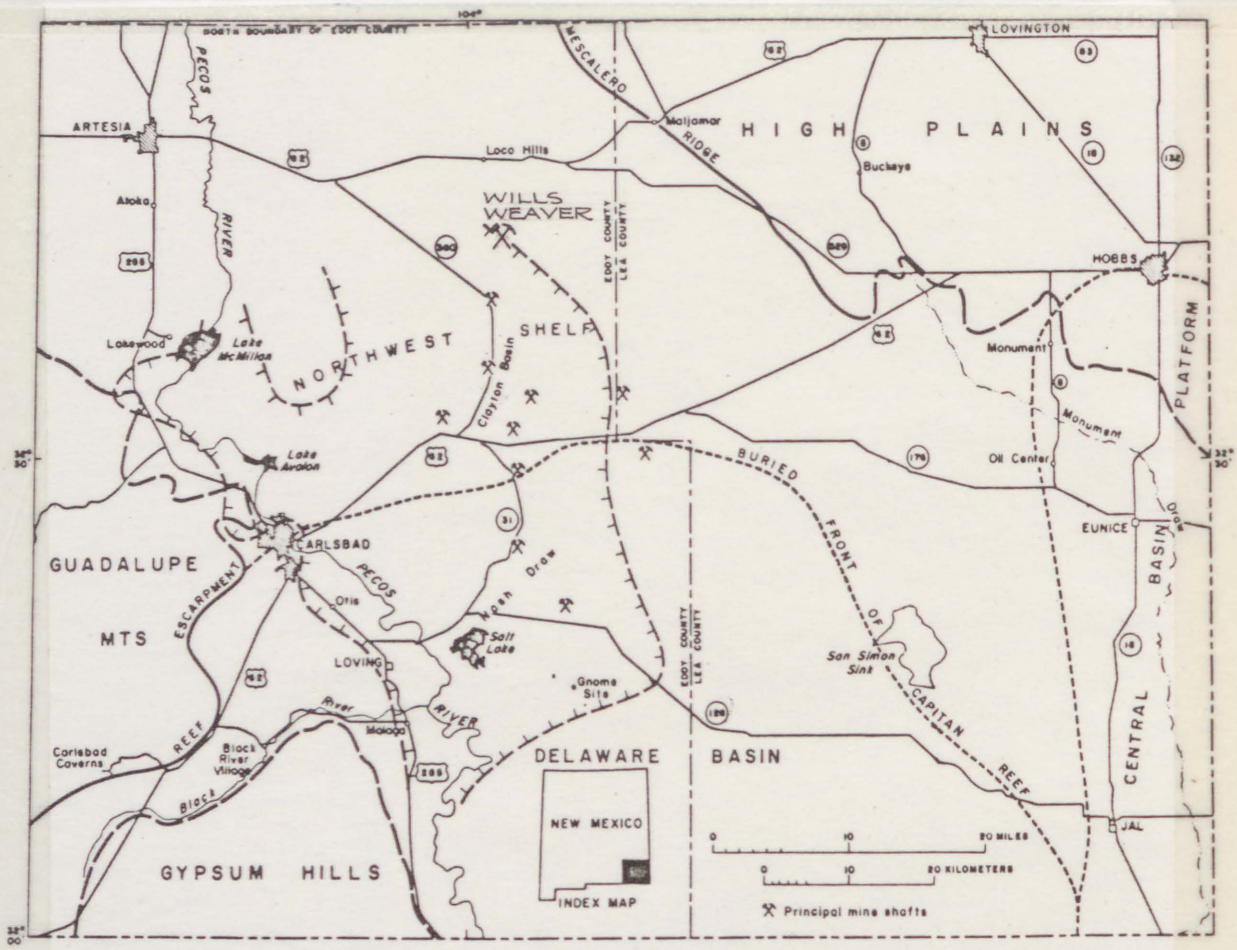


Generalized distribution of total dissolved solids in well water in the rocks overlying the Salado Formation, Carlsbad mining area and adjacent region of southeastern New Mexico. Hachures indicate area where ground water usually contains more than 2,500 ppm of dissolved solids. Data compiled from Bjorklund and Motts (1959), Cooper and Glanzman (1971), Cox (1967), Hendrickson and Jones (1952), and Nicholson and Clebsch (1961).

U.S. Geological Survey
OPEN FILE REPORT
This map is preliminary and has not been edited or reviewed for conformity with Geological Survey standards or nomenclature.

Reference: Brokaw et al, 1972

Project No. _____ Reviewed _____ Date _____

The solution breccia zone, where it exists, contains saline water which is impotable due to high dissolved solids content.

Several wells in the Clayton Basin south of the Wills-Weaver mine obtain fair quality water from alluvium. This water is recharged from local precipitation and has not been extensively mixed with low quality water from deeper strata.

3.4 LOCAL GROUNDWATER CONDITIONS

3.4.1 Aquifers

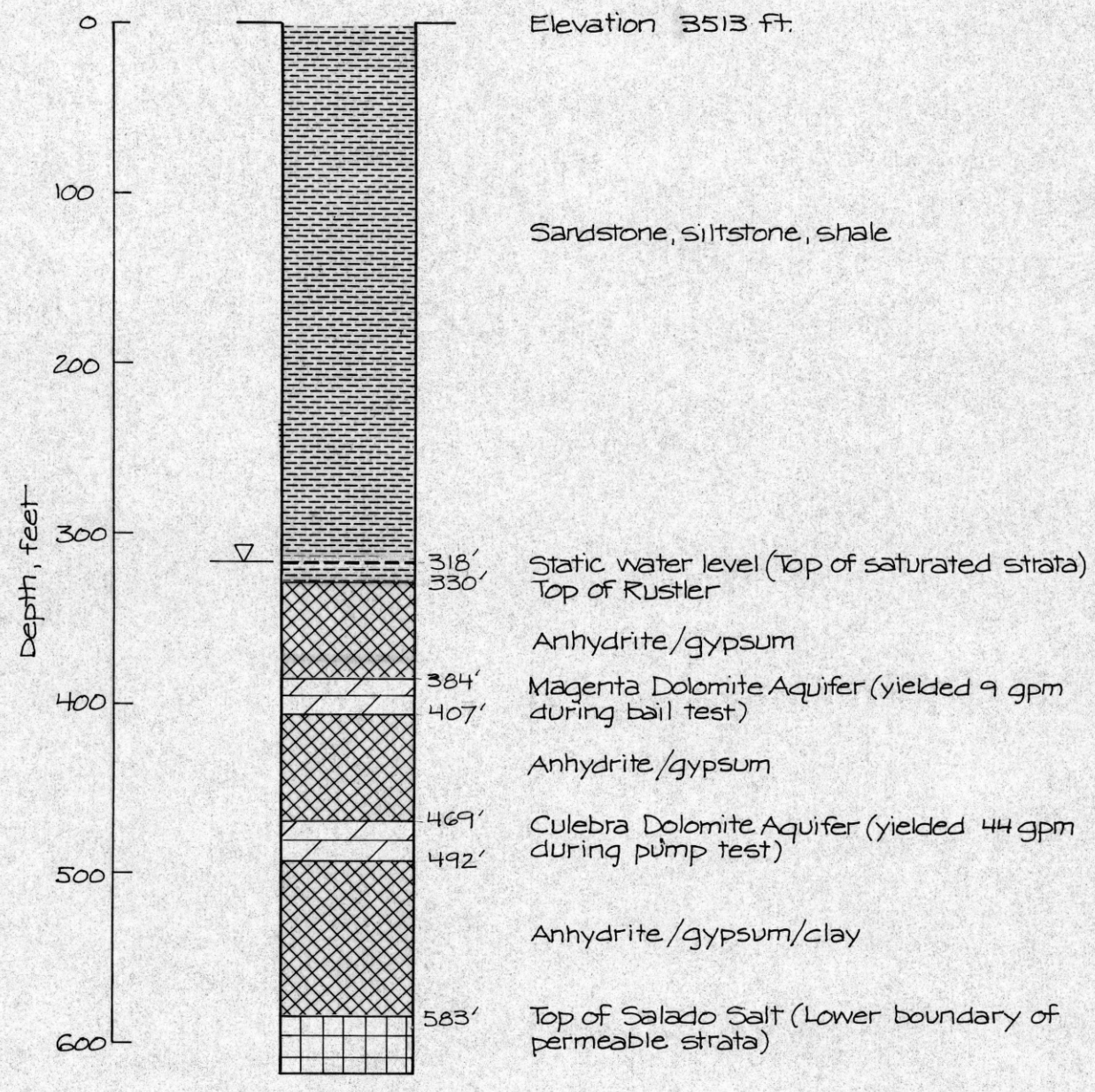
Two aquifers are important locally at the Wills-Weaver mine site: they are the Magenta dolomite, which yields minor amounts of water, and the Culebra dolomite, which is the major water producer. Data from two drillholes provide information about the local water conditions.

Hole DSP 209 is located near the Number 3 Shaft and was drilled to a depth of 620 feet by cable tool rig and completed in October 1958 (the hole was subsequently cored to a total depth of 1,033 feet). It was drilled specifically to test for and, if necessary, produce water for the mining operations. Figure 3-3 shows the location of the aquifers and the static water level in hole DSP 209. The Magenta dolomite was encountered at a depth of 384 feet, but did not produce water until a depth of 393 feet. The Magenta produced about 9 gpm during a bail test. The Culebra dolomite was encountered at a depth of 469 feet. It produced 44 gpm during a pump test. A permeable solution breccia zone immediately overlying the Salado was not encountered in hole DSP 209.

Hole DSP 215 was cored down to a depth of 577 feet for the purpose of obtaining geologic information for the shaft sinking

GROUNDWATER CONDITIONS IN DSP 209

Figure 3-3



Hole drilled with cable tool rig
Completed October 1958

Project No. 783-1050 Reviewed ____ Date 4-79

operation. It was drilled at the site of the No. 3 shaft. Core logs describe the Magenta dolomite as a minor water producing zone and the Culebra dolomite as an active aquifer. A permeable solution breccia zone was not encountered in DSP 215.

3.4.2 Direction of Flow

Piezometric levels measured in DSP 209 were essentially the same in both the Magenta and Culebra dolomites, indicating hydraulic connection between them. The connection is probably along fractures developed by salt solutioning and the resulting collapse of overlying strata.

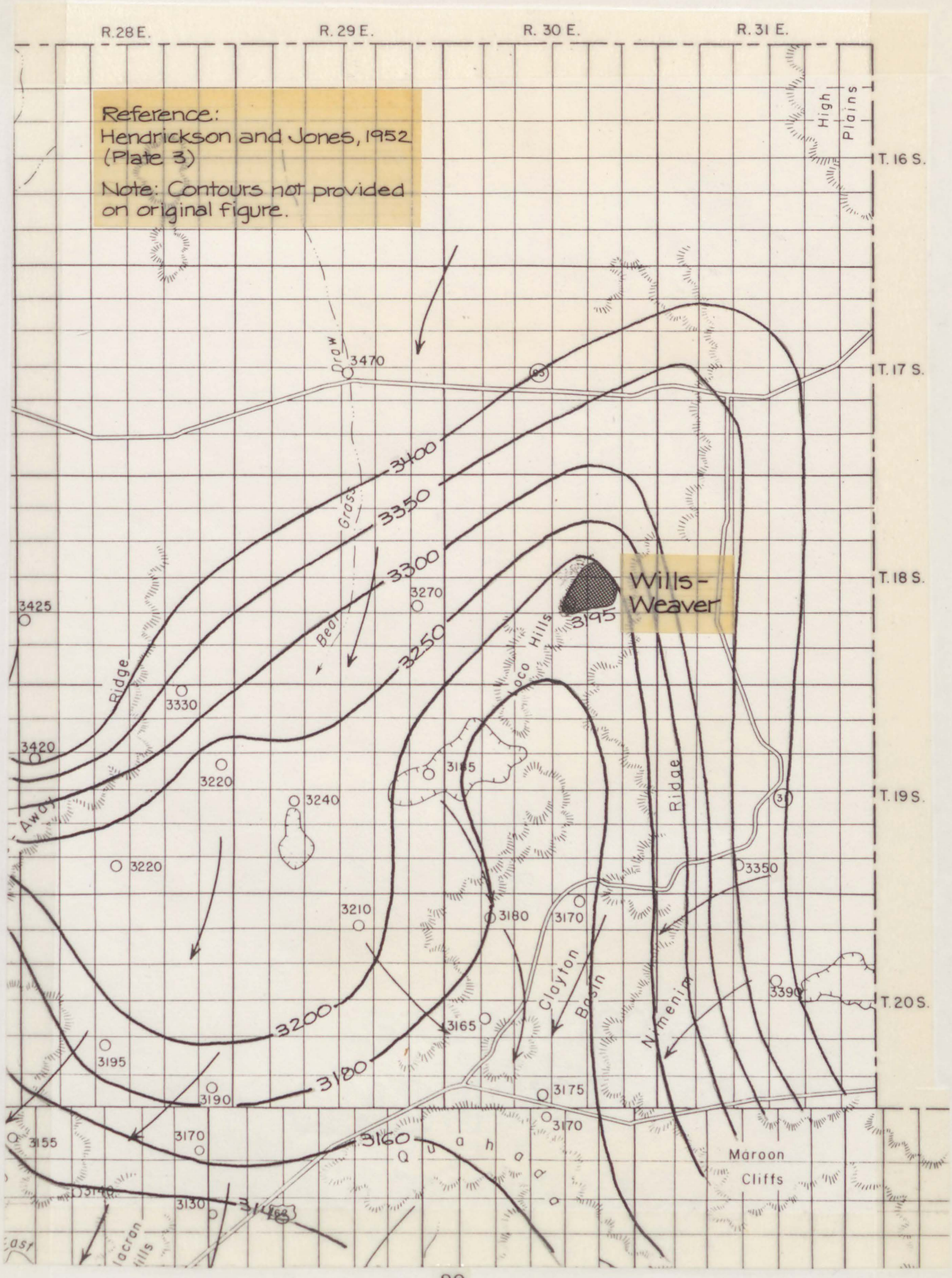
Flow in the immediate area around the Wills-Weaver mine is generally towards the groundwater trough in the Clayton Basin south of the mine. Figure 3-4 shows piezometric elevations in various wells surrounding the mine as compiled by Hendrickson and Jones (1952). The authors have provided arrows indicating the general direction of flow. The piezometric elevations have been roughly contoured by hand and the resulting contours clearly delineate the groundwater trough in the Clayton Basin.

The static piezometric level in hole DSP 209 rises to within 318 feet below ground level. This is 66 feet above the top of the Magenta dolomite and 151 feet above the top of the Culebra dolomite. Although the two dolomite members are the only units which yielded water to DSP 209, the remainder of the strata below 318 feet down to the Salado are probably saturated and at nearly the same piezometric pressure as the two dolomite units. They do not produce measurable amounts of water due to low permeabilities.

The grouting log during sinking of the No. 3 shaft indicates that water pressures of 68 psi were measured during grouting of

LOCAL PIEZOMETRIC LEVELS

Figure 3-4



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the Culebra dolomite from the 454-foot level. This would indicate a piezometric head level within the No. 3 shaft at about 300 feet below ground level.

No values of permeability or storage are given in the information obtained from DSP 209 or any surrounding holes in the Wills-Weaver vicinity.

3.4.3 Water Quality

Water samples were taken at three depths during the drilling of DSP 209. Sample No. 1 was taken from the Magenta dolomite at a depth of 391 to 393 feet. Sample No. 2 was taken just below the Magenta at 411.5 feet, and samples No. 3 and 4 were taken from the Culebra dolomite at a depth of 503 feet. The analyses of the four water samples are shown in Table 3-1. Two water samples, one from the Magenta and one from the Culebra, were collected from the No. 3 shaft in December 1972. These analyses are shown in Table 3-2. All water samples from the aquifers at the Wills-Weaver mine were of low quality, with concentrations of total dissolved solids near or over 10,000 mg/l.

The ion concentrations determined by water analysis are plotted on Figure 3-5 in terms of milliequivalents per liter. Milliequivalents per liter are calculated by dividing the measured concentration of the ion in milligrams per liter by the combining weight of the ion. Anion and cation concentrations in milliequivalents per liter should balance if the analysis is accurate and all ions have been considered. In general, the results appear good.

The quality of water in the Magenta is somewhat higher than that in the Culebra; however, all water contains concentrations

of dissolved solids too high for domestic and stock use without treatment.

TABLE 3-1

WATER QUALITY ANALYSES FROM DSP 209
Water Samples Taken September and October 1958

	Sample Number and Depth			
	No. 1 391-393 ft	No. 2 411.5 ft	No. 3 503 ft	No. 4* 503 ft
pH	7.7	7.7	7.3	7.3
Total Alkalinity (ppm)	120.0	108.0	122.0	120.0
Total Hardness (ppm)	3817.8	4027.8	4906.2	4818.9
Ca (ppm)	959.9	1008.4	1269.5	1243.6
Mg (ppm)	340.3	361.6	415.8	410.3
Na (ppm)	1610.0	1775.0	2790.0	2767.0
Cl (ppm)	3415.2	3883.8	5900.1	5824.8
SO_4 (ppm)	2195.0	2227.7	2203.2	2167.3

*Sample No. 4 was taken 1 hour after Sample No. 3.

TABLE 3-2
WATER QUALITY ANALYSES* FROM SHAFT NO. 3
Water Samples Taken December, 1971

1. Sample from Magenta Dolomite
Est. Depth 500 Feet

Conductivity		15900	Iron, Diss (as Fe)	µG/L	20
Manganese, Dissolved	µ G/L	10	Ph		7.5
Silica	MG/L	29	Phos, Ortho (as P)	MG/L	0.03
Phosphate, Dissortho	MG/L	0.09	Alk, Tot (as CaCO3)	MG/L	106
Total Hardness	MG/L	4200	Noncarb Hardness	MG/L	4100
SAR		15	Percent Sodium		53
Calc Dissolved Solid	MG/L	10900	Dis Solids (Ton/A-Ft)		14.8

Major Ions

Cations (MG/L)		Anions (MG/L)	
Calcium, Diss.	1000	Bicarbonate	129
Magnesium, Diss.	410	Carbonate	0
Potassium	78	Chloride	4800
Sodium	2200	Flouride Diss.	2.2
		Sulfate	2300
		NO2 + NO3 (as N)	3.9

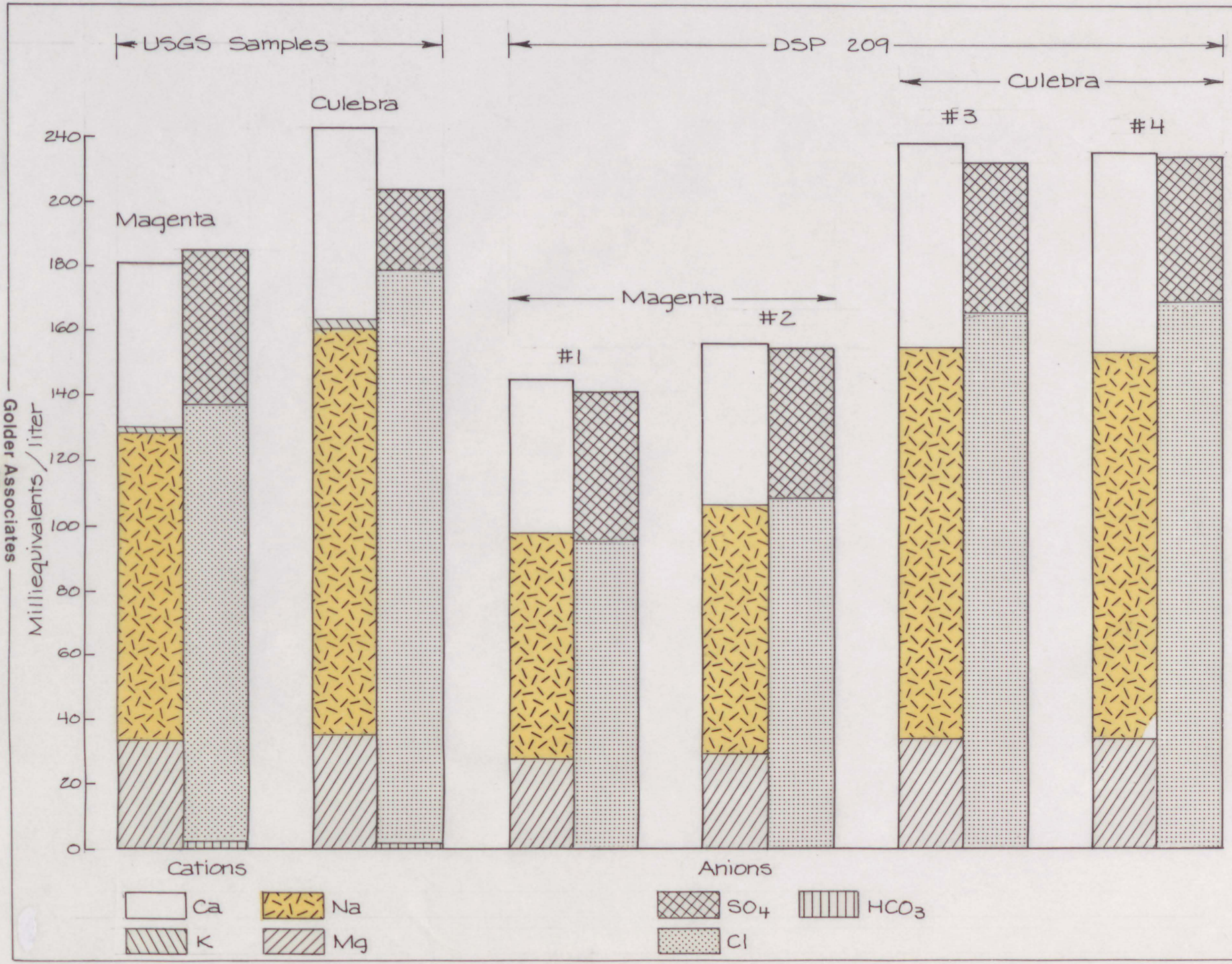
2. Sample from Culebra Dolomite
Est. Depth 550 Feet

Conductivity		19700	Iron, Diss (as Fe)	µ G/L	10
Manganese, Dissolved	µ G/L	10	Ph		7.5
Silica	MG/L	29	Phos, Ortho (as P)	MG/L	0.03
Phosphate, Dissortho	MG/L	0.09	Alk, Tot (as CaCO3)	MG/L	88
Total Hardness	MG/L	4800	Noncarb Hardness	MG/L	4700
SAR		18	Percent Sodium		56
Calc dissolved solid	MG/L	13400	Dis Solids (Ton/Aft)		18.2

Major Ions

Cations (MG/L)		Anions (MG/L)	
Calcium, Diss.	1200	Bicarbonate	107
Magnesium, Diss.	430	Carbonate	0
Potassium	110	Chloride	6300
Sodium	2900	Flouride, Diss.	1.5
		Sulfate	2400
		NO2 + NO3 (as N)	

*Sample analyzed by U.S. Geological Survey, Water Resources Division Central Water Quality Lab, Salt Lake City, Utah.



RESULTS OF WATER QUALITY ANALYSIS ON WATER FROM THE MAGENTA AND CULEBRA DOLOMITE MEMBERS OF THE RUSTLER FORMATION

Figure 3-5

3.5 SHAFTS' HYDROLOGY

A detailed inspection of the No. 3 shaft indicated that the shaft is completely dry to a depth of about 300 feet. At this location, there is an apparent crack around the entire shaft circumference (not associated with a construction joint) through which water is ingressing. The very clear demarcation between the dry and wet portion of the shaft is shown on Plate 1. The elevation corresponding to the ingress of groundwater approximately coincides with the piezometric surface determined within the No. 3 shaft during grouting operations for the Culebra dolomite aquifer. Static head determinations within pilot hole DSP 209, located approximately 250 feet to the east of No. 3 shaft, suggested a piezometric surface approximately 20 feet lower than that observed within the No. 3 shaft.

In general, beneath this elevation the shaft is partially damp. At elevation 390 feet below shaft collar, a water collection drain has been formed in the concrete lining. Similar water collection drains have been installed at elevations 470 feet and 490 feet beneath the shaft collar. The water collected from these drains is subsequently piped into the mine workings. Water exiting from one shaft drain is shown on Plate 2.

In the lower regions of the shaft, beneath a water ring which has apparently never functioned (located at 527 feet below the shaft collar), the shaft wall was generally dry, although damp spots still existed. The No. 3 shaft was downcasting and the relatively dry air was probably helping to maintain the fairly dry appearance of the lining. From the quantity of deposited salts on the shaft wall and a knowledge of the concentrations of the ingressing brines, it is estimated that a maximum of about 2 to 3 gph of water has been removed by evaporation.



Plate 1

Water line in no. 3 shaft, 300 feet beneath shaft collar



Plate 2

Water collection drain in no. 3 shaft

The moisture taken into the mine workings by the downcast air resulted in a damp condition in the mine floor, in the immediate vicinity of the No. 3 shaft mine station. This is due to the presence of the hygroscopic mineral carnallite located in the floor.

There is approximately 30 feet of water in the No. 3 tail-shaft. The water piped into the mine workings from the No. 3 shaft had a measured flow rate of about 12.5 gph, although in-flows up to 36 gph have been measured at other times.

The No. 4 shaft was generally dry, and no water was entering the mine workings through the shaft. Salt concentrations on the shaft wall indicated, however, that the shaft was not totally dry but that the inflow rates were sufficiently low that all water was being evaporated by the upcasting air. First evidence of water ingress was indicated by a small salt deposit originating from a construction joint located approximately 435 feet beneath the shaft collar. Approximately 10 feet beneath this location, a more general buildup of surface salt was present (maximum average thickness of 0.25 inches over the complete shaft lining). Heaviest salt concentrations occurred at approximately 470 feet to 480 feet beneath the shaft collar, where an average thickness of about 1 inch existed. Locally, up to 6-inch thick deposits were present, with heaviest concentrations occurring on the east wall of the shaft. A small water drip also existed at this location, which corresponds to the Culebra dolomite aquifer and which represents the location of the majority of the No. 4 shaft brine ingress.

Salt deposits in lower concentrations existed below this elevation and down to the pump station located at an approximate depth of 652 feet. Below the pump station there was no signifi-

cant build-up of salt. At approximate depth 840 feet, there existed a small brine seep (as evidenced by salt deposits) from the Salado salt. This seep was ingressing through a construction joint.

Just above the shaft water ring, which was salt encrusted, salt flowers were observed exiting from grout holes in the lining. The most recent areas of brine ingress could be identified by the relatively fresh and clean appearance of the deposited salts.

Observations of the quantity of salts in the No. 4 shaft indicated that approximately 1 to 2 gph of water is being evaporated. Since all ingressing water in the No. 4 shaft is evaporated, the above value also represents an average rate of brine inflow. There is approximately 4 feet of water in the No. 4 tailshaft.

3.6 MINE HYDROLOGY

3.6.1 Introduction

Because of the nature and thickness of the salt beds overlying and underlying the mine horizon, the only possible sources of significant water supply to the mine workings are man-made openings (e.g., shafts, wells, core test holes, etc.). It was not possible to inspect any core test holes or wells from the workings, and their capacity for transmitting water to the mine has been studied by reviewing the corehole abandonment procedures employed.

The No. 4 shaft is evidently supplying virtually no water to the mine, but the No. 3 shaft is supplying moisture through two

mechanisms--water directly piped into the mine and moisture carried in by the air circulation. Evidence for the latter is provided in the damp floor conditions noted in the areas of the mine, adjacent the No. 3 shaft mine station, which were directly exposed to the downcast air. The quantity of salt on the No. 3 shaft walls indicates that, on average, moisture carried into the workings by circulating air is limited to 2 to 3 gph.

3.6.2 Core Test Hole and Well Plugging Procedures

The Wills-Weaver mine area has been penetrated by core test holes and oil wells. A list of core test holes and wells intersecting the Wills-Weaver mine is presented in Table 3-3. Their locations are shown on Figure 3-6. Available information concerning plugging procedures in each hole and well within the mine area is presented on Figure 3-6. The indicated seal thicknesses within the core test holes are based on an interpretation of the data presented on the USGS "Sundry Notices and Reports on Wells" together with field notes by the Duval Corporation, and are considered "representative" of in situ conditions. Golder Associates' interpretation of the data could differ from actual conditions, but probably not significantly. The following assumptions were made in the calculations of individual seal thicknesses: (1) a 4-inch casing "fits" a 6.5-inch diameter hole, (2) core size within the Salado Formation is 4 inches in diameter, (3) one sack of cement will yield 1.1 cubic feet of seal, and (4) the presence of soluble carnallite will not greatly influence the volume of the completed hole. This last assumption is examined in detail further on.

The information on the oil wells did not allow a calculation check of seal thickness to be made, as hole and casing diameters were not available. The indicated thickness is plotted directly from the USGS information.

Additionally, within the mine area there are presently three producing oil wells. These wells are encapsulated by pillars within the mine. They should present no problem with regard to leakage of groundwater from overlying aquifers into the mine.

TABLE 3-3

CORE TEST HOLES AND OIL WELLS

WITHIN MINED AREA

Identification	Location Within T-18-S R-30-E	Condition: Solid Cement Plug From Total Depth To Top of Salt or Above Except as Noted
D-215	587S, 2349W SW-22	Hole ended at top of salt
D-210	2291N, 1130W NW-23	
D-211	218N, 998E NE-22	
D-212	2420S, 2342W SW-22	
D-213	910N, 1540E NE-23	
D-132	2250S, 1064W SW-14	
D-138	2419N, 3121W NW-23	
D-139	1956N, 780E NE-22	
S-18	800S, 1762E SE-22	44'void, 76' cement, 89' void, 308'cement
S-64	32N, 1338W NW-23	251' cement, 116' void, 41' cement
S-6	740S, 388W SW-23	18'void, 51' cement, 292' void
Martin No. 2	330N, 990W NW-23	293' cement
Martin No. 5	1980N, 660W NW-23	346' cement
Randel Duval No. 1	2310S, 330E SE-22	

In general, all holes and wells were advanced through the formations overlying the salt by using churn drilling or rotary drilling techniques. The core test holes were then diamond cored in the Salado Formation to the mining horizon. The oil wells were likewise advanced through the mine area, but to "producing zones" far below the mining horizon. Common drilling practices were employed, using casing, drilling mud, and/or brine water to advance the holes. Each hole was drilled under regulations provided by the USGS, allowing for documentation of the plugging operations.

The "balance method" of plugging all holes was followed. This is common "oil field" practice, in which no packers are used, but the cement (a per sack mix) is simply pumped down the drill rods in quantities calculated to provide the desired seal thickness. Zonal sealing of all encountered aquifers and total sealing of the Salado Formation was attempted in all plugging operations. As indicated on Figure 3-6, the two aquifers above the Salado Formation are the Magenta dolomite and Culebra dolomite. The plugging operation sealed these zones by placing a seal through the aquifer or by placing a seal above and below the aquifer.

The type of cement used in the plugging operation is not documented. Within the Salado Formation the cement was mixed with brine and 3 percent calcium chloride. Above the Salado Formation it was mixed with fresh water only. The calcium chloride additive quickens the "set time" of the grout.

In only two core test holes, wells Nos. S-6 and S-18, were wooden bridge plugs used. They provide positive positioning of seal locations, if properly placed. No mention is made within the documentation if obstacles or difficulties were encountered in their placement.

In general, all casing used in the drilling operation to maintain hole stability was removed prior to seal placement. However, in some holes the casing could not be pulled. Some sections remained in place after overlying sections were cut and removed, as noted on Figure 3-6. In addition to cement seals and casing, "mud and cuttings," "cuttings," and "soil" were placed in nonsealed areas of some holes. It is expected that mud-laden drilling fluid also occupies nonsealed zones as a remnant of the drilling operation.

In order to evaluate the influence of highly soluble carnallite on the effectiveness of the borehole seals, three holes through the Wills-Weaver mine horizon were selected for further study. These holes, S-6, S-18, and S-64, were selected because in each, portions of the Salado Formation were not sealed, whereas the entire salt section was cemented in the remaining holes that penetrate the mine horizon, as shown on Figure 3-6. Figure 3-7 depicts the occurrence of carnallite, as shown on the geologic log of each hole. In more than 75 percent of the Salado intercept, carnallite is noted as occurring in amounts varying between trace and 1 percent, typically as isolated grains enclosed within salts of lower solubility. Occasionally, thin stringers, carnallitic clay seams, and discrete bands of essentially pure carnallite were noted. The total thickness of core logged in each hole that was estimated to contain 1 percent or more carnallite is less than 20 feet, out of 643 feet in S-6, 257 feet in S-18, and 379 feet in S-64. The high grade (greater than 5 percent) stringers and bands amount to less than 3 feet in each hole.

The mode of occurrence (i.e., isolated grains or continuous seams) of the carnallite is considered important in contemplating the condition of the borehole walls and, accordingly, the

condition of the seals. The isolated grains, when dissolved by the drilling fluid, will result in a pitted surface and a hole volume only slightly larger than the theoretical smooth wall would produce. However, leaching within the stringers and seams of higher carnallite grade could be more extensive, limited only by the exposure time to circulating drilling brines and the ability of fluids to migrate within the solution cavity, which would reflect the thickness of the seam. Accordingly, a 1-inch seam occurring near the top of the Salado would be expected to show a larger cavity than an identical seam closer to the mine horizon (longer period of exposure to drilling fluids). Equally, a 3-inch seam might result in a cavity of greater lateral extent than a 1-inch seam of similar character in the same vicinity.

There are only a few locations within these holes where the carnallite content is high enough and the seams thick enough to result in a cavity of noticeable dimensions. These are listed in Table 3-4. Accordingly, it is not considered that the estimated seal locations will be in significant error as a result of increased corehole volume associated with carnallite dissolution.

The results of the plugging procedures review are presented below. As minimal information on actual plugging procedures used within each hole is available, the conclusions reached are based partly on the stated facts and partly on Golder Associates' interpretation of the data. The conditions under which the seals were probably placed can be summarized as follows:

- Stable hole conditions
- Small inflow rates (less than 50 gpm) from aquifers
- Minimal hole erosion as a result of the drilling.

TABLE 3-4
CARNALLITE-RICH ZONES

Hole No.	Depth in Hole (feet)	Thickness of Interval (inches)
S-6	705.8	2
	737.8	2
	747.9	3
	791.8	1
	963.8	24 (bottom of ore)
S-18	704.4	8
	729.3	1
	775.6	2
	863.6	2
	931.0	23 (bottom of ore)
S-64	663.8	1
	794.5	3
	805.2	5
	991.0	12
	1,012.2	10 (bottom of ore)

These conditions would allow for the relatively easy and accurate placement of the seals. Contamination of the cement and poor bond to the host rock would be unlikely. However, no mention is made in the documentation of cement type used in the plugging operation. Because of the brackish nature of the groundwater in the area, cement seals, if not properly designed, could deteriorate with time. The core test hole seals have been emplaced for 20 to 30 years (Corehole S-6 since 1948), while the oil well seals have been in place since 1964/1965.

Of all the holes and wells reviewed, only two core test holes would appear to present any future seal problems. These holes are core holes Nos. S-6 and S-18. They are not totally

sealed within the Salado Formation, but have zonal seals. Depending on future plans for the mine, hole No. S-6 should be opened up and completely resealed. Hole No. S-18 is marginally sealed within the Salado Formation with two 50-foot-plus seals and could require resealing. It should be noted that hole S-6 does not pass through the present mine workings because the 2nd South panel was not fully developed. Hole S-18 is isolated from the mine horizon by a 50-foot radius salt pillar.

In conclusion, the plugging procedures followed were those commonly used in the "oil field." Although the technical procedures are not documented, the seals are most likely functioning satisfactorily, owing to the stable environment in which they were originally placed. The core test hole seals have been emplaced for quite a long time, and consideration should be given to checking their condition, depending on plans for the decommissioning of the mine. It is not considered likely that any of the abandoned core test holes or oil wells are responsible for water entry into the mine workings.

3.6.3 Mine Water

In general, the Wills-Weaver mine is very dry. Water piped from the No. 3 shaft is drained into the location indicated on Figure 3-8. At this location, the water has caused substantial local dissolution. A flowrate of 12.5 gph was measured during the site visit, although flowrates to 36 gph have been measured in the past. These variations are evidently related to the surface precipitation cycle. It was noted that the No. 3 mine shaft walls were somewhat drier in February than they were during a July inspection.

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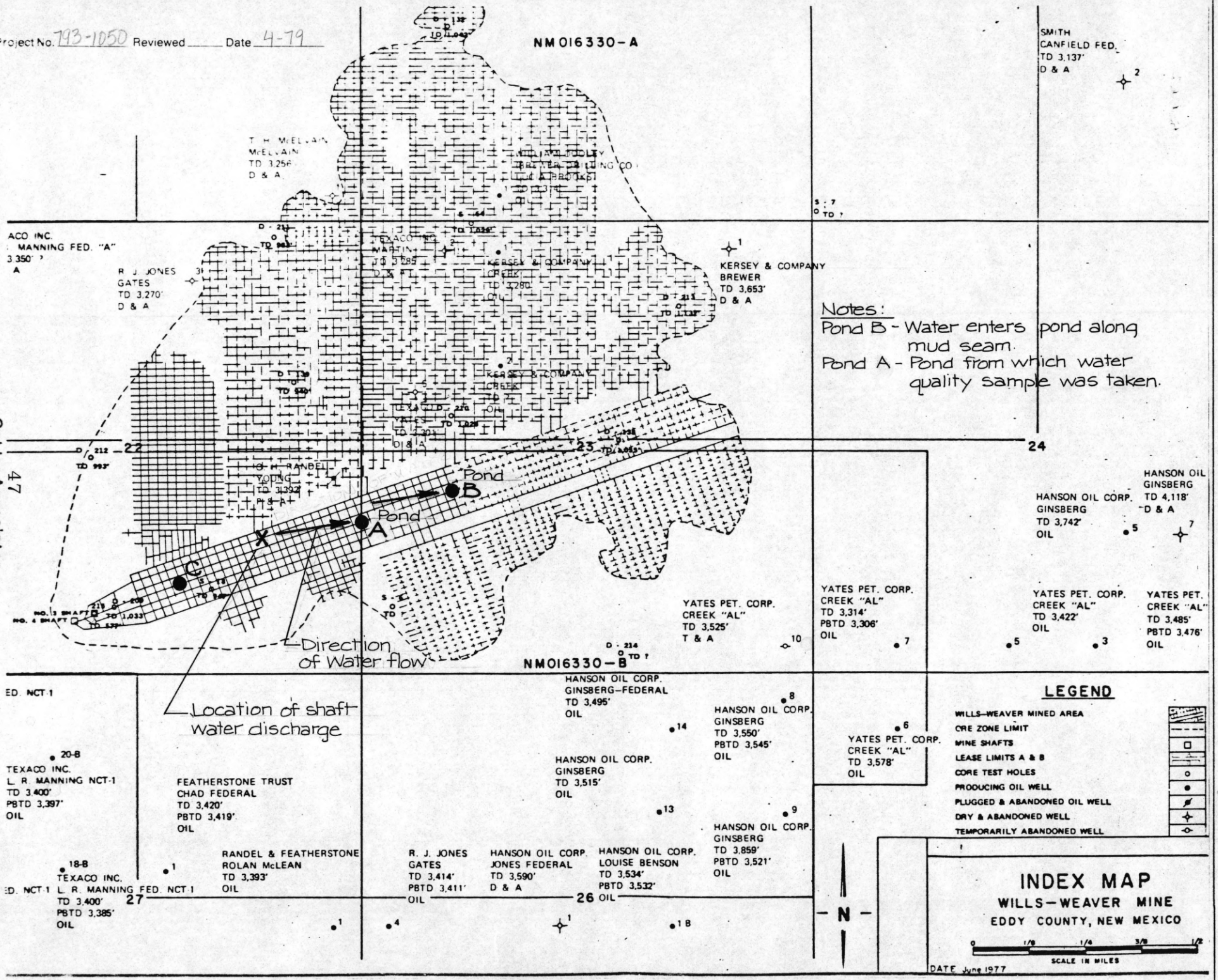


LOCATION OF MINE WATER

Figure 3-8

Dissolution of the discharge area by the brackish shaft water has resulted in the formation of an approximate 25-foot by 45-foot floor trough, which is about 5 feet deep at its center. The collapsed floor is shown on Plate 3 and undercutting of the corner of an adjacent support pillar is shown on Plate 4.

From the discharge area, the water apparently migrates towards the east, following the natural gradient of the mine workings (Figure 3-8). Water ponds were observed at the locations indicated on Figure 3-8. These ponds consisted of topographic lows in the mine floor and typically occupied the majority of the intersection between the drive and cross-cut. The water appears to migrate along mud seams just below the present floor level of the mine. In these areas, the mine floor is experiencing some limited floor heave and this process would also tend to provide a flow path for the migrating water. At location B (Figure 3-8), water was observed entering the pond through a mud seam located 1 to 2 feet below the mine floor. It is presumed that the mine shaft water discharge migrates to the extreme east of the workings.

Water samples were taken at two locations in the mine during the site visit and later analyzed for chemical constituents. The results of the chemical analysis are presented in Table 3-5 and on Figure 3-9.

Sample No. 1 was taken from water leaking into the No. 3 shaft, as described in Section 3.5. Its water chemistry is typical of water existing in the aquifers overlying the mine. This is apparent from a comparison with Tables 3-1 and 3-2 and Figure 3-5. Ion concentrations in sample No. 1 are given in terms of parts per million (ppm), which are approximately equal to milligrams per liter (mg/l) since the density of the water is



Plate 3

Dissolution damage at no. 3 shaft water disccharge

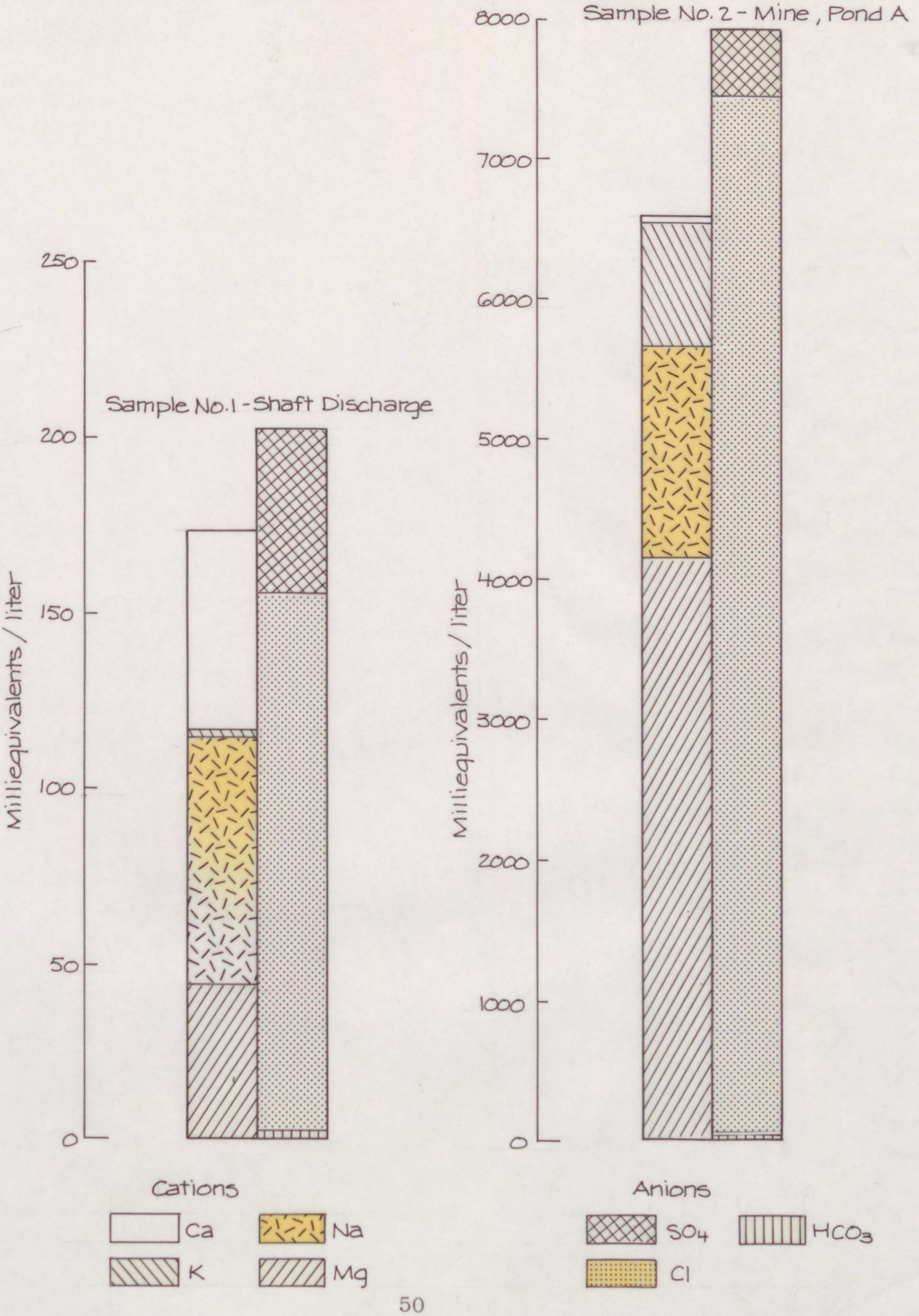


Plate 4

Undercutting of mine pillar at no. 3 shaft water discharge location

RESULTS OF WATER QUALITY ANALYSIS ON WATER FROM THE SHAFT DISCHARGE AND MINE
February, 1979

Figure 3-9



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nearly equal to that of fresh water. The total dissolved solids (TDS) concentration is 13,500 mg/l, indicating no significant contact with evaporites.

Sample No. 2 was taken from Pond A (Figure 3-8), located in the mine workings. It contains extremely high ion concentrations resulting from salt dissolution as water flows from the shaft drain outlet (Figure 3-8) towards the east. Ion concentrations given in ppm must be multiplied by the specific gravity of the solution to convert to mg/l, because the ion concentrations significantly affect the density of the solution [specific gravity equals 1.229 (from water quality analysis) compared to 1.0 for pure water]. Total dissolved solids concentrations are in excess of the total ion concentrations, and this is due to hydrated water molecules associated with the magnesium chloride.

The water quality analysis indicates that the principal salts in the saturated solution are the chlorides of sodium, potassium, and magnesium. These salts have their source in the halite, sylvite, and carnallite minerals present in the mining horizon. The concentrations of KCl, $MgCl_2$, and NaCl present in the mine water are, respectively, 53,000, 162,000, and 71,000 ppm. For comparison purposes, laboratory analysis of a prepared solution saturated in KCl, $MgCl_2$, and NaCl indicated respective concentrations of 69,000, 136,000, and 90,000 ppm. These results are shown in Table 3-6.

The laboratory results indicate that if all the chloride minerals are present in sufficient amounts (i.e., carnallite, halite, and sylvite), 1 gallon of fresh water will dissolve approximately 0.06 cubic feet of carnallite and about 0.01 cubic feet of halite. At the same time, a small amount of sylvite will

be deposited as a result of the excess KCl provided by the carnallite

TABLE 3-5

WATER QUALITY ANALYSES FROM SHAFT AND MINE*

Water Samples Taken During Site Visit, February 1979

Constituent	Sample No. 1 Shaft Discharge	Sample No. 2 Pond A in Mine
Alkalinity (ppm)	196	465.5
Chloride (ppm)	5445	208,000
Specific Conductivity (mhos/cm)	6.73×10^{-3}	1.38×10^{-2}
Nitrate (ppm)	3.95	7.9
Nitrite (ppm)	<0.250	92.5
pH	7.6	6.15
Total Dissolved Solids (ppm)	13,500	512,000
Sulfate (ppm)	2260	18,405
Calcium (ppm)	1132	487
Magnesium (ppm)	536	41,150
Potassium (ppm)	76.3	28,250
Sodium (ppm)	1620	28,250
Total Hardness (ppm)	5000	175,000
Carbonate (ppm)	0	0
Bicarbonate (ppm)	196	465.5
Fluoride (ppm)	1.2	0.39

*Samples analyzed by Food, Chemical and Research Labs, Inc., Seattle, Washington.

Table 3-7 gives estimated volume dissolution rates for two flow rates into the shaft (see Section 3.5). Based upon a flow rate of 12.5 gph, which is that measured during the site visit, the volume dissolution rate will be 260 cubic yards per year. Based upon the previously observed maximum flow rate of 36 gph, the volume dissolution rate would be 760 cubic yards per year.

TABLE 3-6

WATER QUALITY ANALYSIS FOR PREPARED SOLUTION
SATURATED IN CHLORIDES OF POTASSIUM, MAGNESIUM, AND SODIUM*

Constituent	Concentration (g/l)
Potassium	45
Magnesium	43
Sodium	44
Chloride	220
Total Dissolved Solids	357
	Specific Gravity = 1.25

*Test performed by Am Test, Inc., Seattle, Washington

Assuming an average inflow rate of the order of 20 gph, approximately 420 cubic yards of solids would be dissolved each year. Since 1971, over 3,500 cubic yards of material would have been removed by dissolution. The approximate volume of the dissolution void at the shaft water discharge point (Figure 3-8) is 150 cubic yards, and this void therefore probably represents less than 5 percent of the material that has been dissolved since cessation of mining in 1971.

TABLE 3-7
ESTIMATED VOLUME DISSOLUTION RATES
(Cubic Yard Per Year)

Volume Flow Rate From Shaft Drain	
12.5 gal/hr	36 gal/hr
260	760

3.7 REVIEW OF HYDROLOGY

The following points summarize the factors of principal interest in relation to the Wills-Weaver mine and shaft hydrology.

- The Magenta and Culebra dolomite members of the Rustler Formation are the two aquifers which are important locally at the Wills-Weaver mine site. The former yields minor amounts of water, and the Culebra dolomite is a major water producer. The aquifers overlying the Salado are apparently hydraulically connected, and the piezometric level in the vicinity of the mine shafts is approximately 300 feet below ground surface.

- Water from the aquifers is of low quality with concentrations of dissolved solids of the order of 10,000 mg/l.

- The No. 3 shaft is making a small amount of water. The majority of this is collected in shaft drains and piped into the workings. The flow rate of the water entering the mine from the No. 3 shaft was measured at 12.5 gph, although flow rates to 36 gph have been measured in the past. In addition, some moisture is carried into the

workings by the downcast air (2 to 3 gph), and this results in a damp floor in the vicinity of the No. 3 shaft mine station. There is about 30 feet of water in the No. 3 tailshaft.

- There is no water entering the mine workings from the No. 4 shaft. Salt concentrations on the No. 4 shaft wall suggest an inflow rate of 1 to 2 gph. The majority of the inflow is entering the shaft at the elevation of the Culebra aquifer and is totally evaporated by the upcasting air.

- Based on a review of the sealing procedures for the 12 core test holes and 3 abandoned oil wells which intersect the Wills-Weaver mine, it is not considered likely that any of these features are permitting the ingress of water into the mine workings. In general, the Salado salt above the workings is totally sealed with a cement plug. Two core test holes, S-6 and S-18, have zonal seals within the Salado, and these holes might require re-sealing depending on planned future utilization of the mine. Both holes, however, are at present protected from the mine horizon by salt barriers.

- The water piped into the workings from the No. 3 shaft constitutes the most significant supply of water to the mine. This water has considerable dissolution capacity and is probably dissolving material (principally carnallite) at a rate of approximately 420 cubic yards per year. Since 1971, about 3,500 cubic yards of material from within the mine floor would have been dissolved.

4.0 SUBSIDENCE DEVELOPMENT

4.1 INTRODUCTION

An understanding of the nature of the deformations of the strata overlying a caving mine horizon is important for a number of reasons. First, deformations of the strata can damage engineered structures located within the zone of influence of the movements. At the Wills-Weaver mine the principal structures of concern are the two mine shafts. There are also a number of operating oil wells located on the Wills-Weaver property and these have been protected from subsidence damage by salt pillars left within the mining horizon. In certain circumstances, surface subsidence related to underground mining activity may be unacceptable because of damage to surface structures, but this is not a constraint within the Potash leases of southeast New Mexico. The reasons for this are:

- The area is sparsely populated
- Surface structures are few
- Existing surface structures are protected by mine pillars. Linear structures such as power and telephone lines are more difficult to protect and practices such as incorporating extra wire into the length between pole supports, have been adopted.

Second, subsiding strata may cause damage to overlying aquifers. There has also been a number of mining disasters in salt mines as a result of flooding caused by aquifer disturbance.

Finally, the observation of rates of surface subsidence at the Wills-Weaver mine provides valuable rock mechanics data in

the form of rough estimates of the rates of closure of the mining panels which have suffered high extraction ratio mining. These types of data could not be obtained from observations within the mine workings themselves.

In order to understand the general nature of subsidence above a mine panel which has suffered high extraction, a brief review of published data has been undertaken. Finally, the subsidence monitoring data obtained at the Wills-Weaver mine have been examined.

4.2 SUBSIDENCE - A GENERAL REVIEW

Subsidence of strata overlying high extraction ratio mining horizons has received considerable attention over the years. An area in which considerable subsidence data are available is the longwall mining of flat lying coal beds within the United Kingdom. Field measurement and theoretical evaluation have confirmed that the amount of surface subsidence will be influenced by:

- The depth below ground level of the mine horizon
- The lateral extent of the mine workings
- The thickness of the mining seam
- The character of the strata overlying the mined horizon.

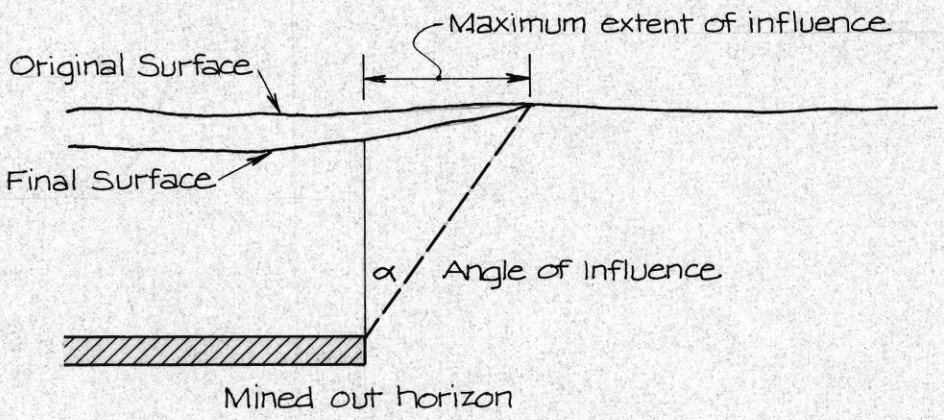
For the coal fields of the United Kingdom, the maximum vertical surface settlement is typically 90 percent of the thickness of the extracted seam, providing the lateral extent of the workings is to at least 0.7 times the depth of the mine in all directions from the point in question (Panek, 1969). If the mine workings extend to only 0.35 times the mining depth from the point under consideration, the maximum surface settlement experienced will be only 45 percent of the thickness of the extracted

horizon. This phenomenon may be explained in terms of an angle of influence concept (Figure 4-1). In general the zone of surface subsidence does not stop at the limit of the mine panel, but extends beyond the mine boundary and is characterized by an angle of influence 'α' (Figure 4-1). It can therefore be seen (Figure 4-1(b)) that a mine panel must have a certain minimum lateral extent before any point on the surface experiences maximum subsidence. The region marked "zone of maximum subsidence" on Figure 4-1(b) will not be influenced by further extensions of the mine workings. Simple trigonometry also suggests that the lateral development of the workings must equal '2D tanα'(D = depth of mine, α = angle of influence) before any point on the surface experiences maximum movement. The areal extent of the mine workings required to produce maximum surface movements at any point must also vary as the square of the mining depth 'D' i.e., doubling the mining depth requires the extraction of four times as much material in order to produce the same maximum surface effect. A corollary of this is that surface effects are less when mining at greater depths, unless the lateral extent of the workings is substantially larger.

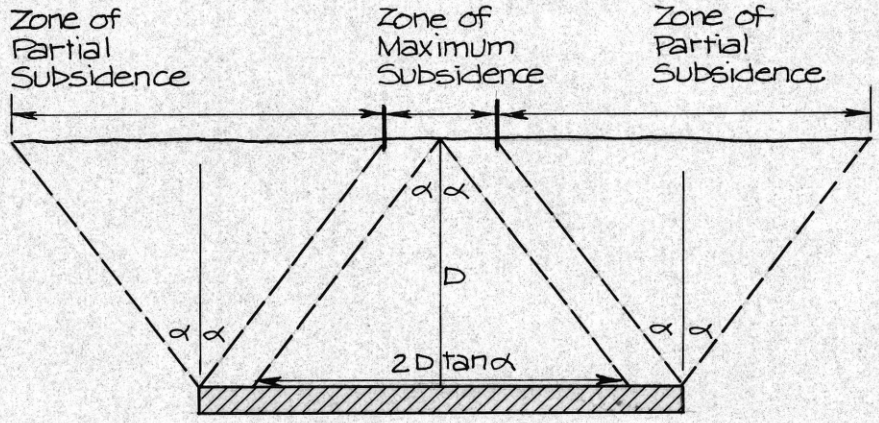
The level of surface subsidence must also obviously depend on the thickness of the extracted seam. If the overlying strata were completely undisturbed by undermining and simply settled into the developed void, surface subsidence would be equal in magnitude to the thickness of the mined horizon (assuming 100 percent extraction). If a certain amount of break-up and loosening of the strata occurs, particularly in the immediate roof of the mine, then bulking of the material results and the surface movements will be correspondingly less. The structural characteristics of the undermined strata will therefore also contribute to the level of surface subsidence developed.

SURFACE SUBSIDENCE EFFECTS

Figure 4-1



(a) DEFINITION OF ZONE OF INFLUENCE OF SURFACE SUBSIDENCE



(b) DEFINITION OF ZONE OF MAXIMUM SUBSIDENCE

NOT FOR PUBLICATION

The nature of the surface movements associated with ground subsidence due to mining are quite complex. In addition to the vertical movements which result in a surface depression of the type indicated on Figure 4-2 (after Deere, 1961), horizontal displacements also occur. Surface horizontal strains as a function of location with respect to the mining face, are shown on Figure 4-2. The data given on Figure 4-2 are reproduced from a report by Deere on surface subsidence effects at the U.S. Borax and Chemical Corporation Potash Mine, Carlsbad, New Mexico. As the mining front advances, a zone of horizontal tension develops above the face position. Behind this (i.e., above the worked-out area) another zone of surface compression exists. Behind this compression zone, very small residual tensions or compressions may remain.

With advance of the mine face, the material above the face is progressively subjected to a tension phase, a compression phase, and finally a long-term relatively undisturbed phase. The rate of advance of this "surface wave" is determined entirely by the rate of mine face advance. The tension phase may be indicated by surface cracks up to one inch wide, and these close as the face advances. At the extremities of the mine panel, where the tension zone persists indefinitely, the surface cracks may develop up to 6 inches to 2 feet in width, become quite deep, and ultimately fail by sloughing of the crack walls.

There are various ways of characterizing the extent of the surface disturbance as a result of high recovery mining methods, and the most usual method is by an "angle of influence" or "angle of draw." Figure 4-2 shows four such angles. These are as follows.

- 'α_1' - the angle from the mine face to the point of maximum horizontal tensile strain.

- 'α_2' - the angle from the mine face to the point of zero horizontal tensile strain. Beyond this point (To, Figure 4-2), minor tensions or compressions may occur. Inside this point, structures are liable to experience substantial disturbance.

- 'α_3' - the angle from the mine face to the point beyond which vertical subsidence is limited to a rather arbitrary value of 0.02 feet.

- 'α_4' - the angle from the mine face to the point beyond which essentially no vertical subsidence occurs.

These various limits were defined by Deere (1961) in an attempt to characterize zones of relative surface disturbance above the U.S. Borax and Chemical Corporation Potash Mine. Because of the location of this mine, its depth below the ground surface, etc., the data are liable to have particular relevance to the Wills-Weaver property. It should also be noted (Figure 4-2) that different influence angles were obtained at different ends of the mining panel. One of these ends was adjacent to an unmined region, whereas the other was located adjacent to an area which had been first mined (typical extraction of 60 percent). Although surface settlements over the first mined region were negligible, its presence obviously influenced the relative locations of maximum surface movements and strains. The data do indicate, however, that the region of the surface liable to experience movement as a result of high recovery mining will correspond roughly to an angle of influence of 45 to 55 degrees. More severe movements and strains will occur within a zone characteristized by an approximate 30 to 35 degree influence angle.

Since the region of subsidence influence extends beyond the extent of the underground workings, it is obvious that as mining

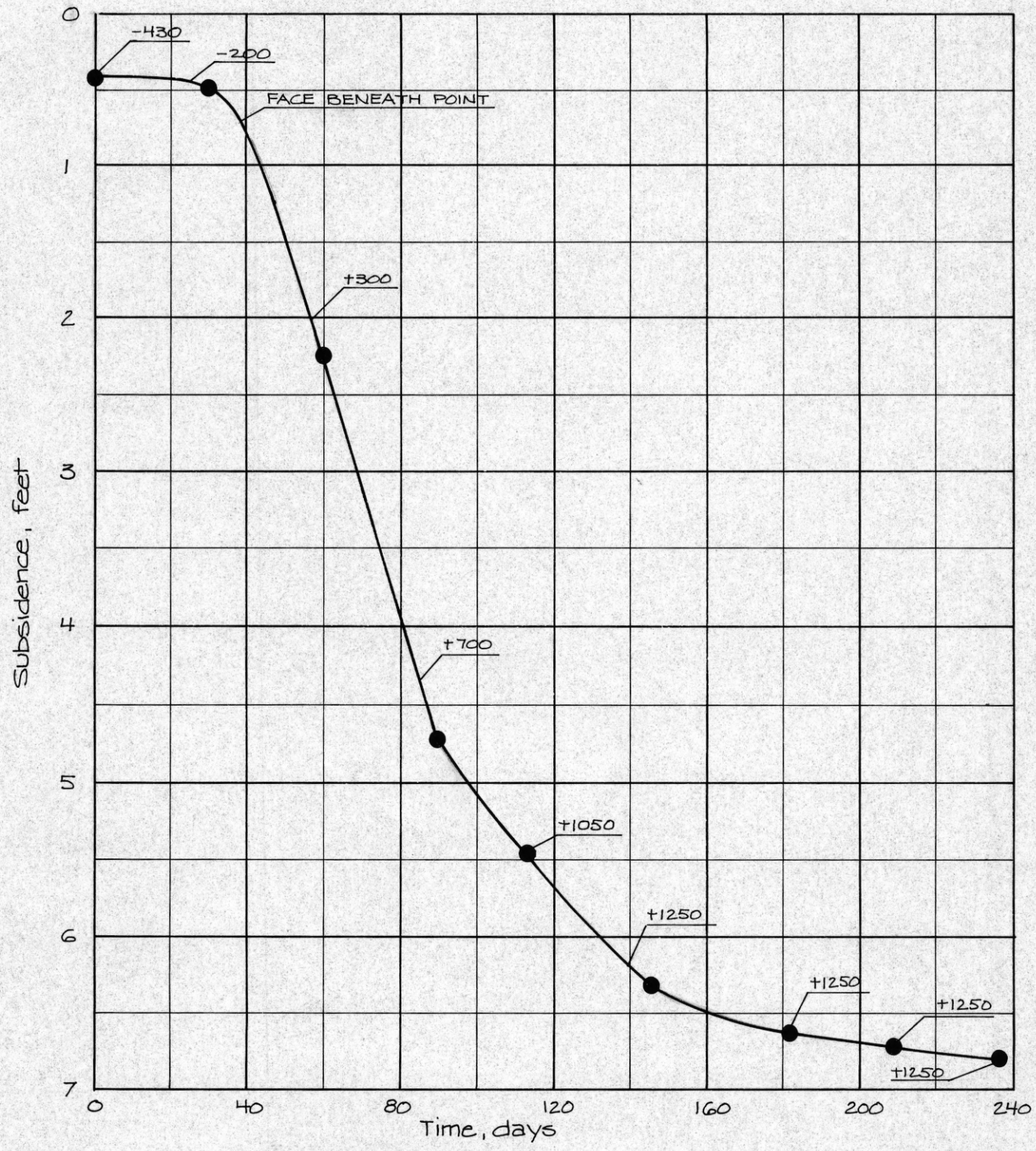
proceeds some limited subsidence will precede the mining face. Up to the stage when the mining face is located directly beneath the surface location, comparatively limited movements occur. As the mining face advances beyond this point, however, the surface settlement rate increases greatly. This behavior is indicated on Figure 4-3, and pertains to observations taken at the U.S. Borax and Chemical Corporation Potash Mine. As the face passes out of the area of influence of the particular surface location, the rate of subsidence gradually decreases. In general, surface movements continue to occur with time even after the mining face has advanced well beyond the area of influence. This secondary surface subsidence effect is probably related to gradual closing of the mine void, which still exists as a result of the residual pillars; at the U.S. Borax Mine, average extraction was approximately 80 percent.

In the potash fields of southeast New Mexico, the relation between the extent of vertical surface subsidence and the thickness of the mining horizon varies with the degree of extraction. For 100 percent extraction, it is probable that the maximum surface subsidence would be very close to the thickness of the mined orebody. There appears to be very little break-up and bulking of the overlying strata, which would tend to limit the level of surface subsidence. Evidence for this type of behavior comes from mining activity carried out in the caved hangingwalls, about 50 to 100 feet above earlier mining horizons. In these applications, the ore beds had suffered no noticeable structural deformation other than the elevation changes caused by subsidence; the hangingwall appeared to retain its cohesive nature even quite close to the mine horizon (Brokaw et al., 1972). Frequently, maximum surface subsidence is somewhat less than the mine horizon thickness (60 to 70 percent) as a result of average extraction ratios of something less the 100 percent.

TIME vs SUBSIDENCE

Figure 4-3

(-) Face advancing toward Point
(+) Face receding from Point



(after Deere, 1961)

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4.3 SUBSIDENCE - WILLS-WEAVER PROPERTY

Surface subsidence profiles were taken along the center lines of two of the earlier production panels of the Wills-Weaver mine (i.e., 1st North and 2nd North panels) in which pillar mining was carried out, resulting in average extraction ratios of the order of 80 to 85 percent. The traverses along which subsidence profiles were taken are indicated on Figure 4-4.

Subsidence development with time along the center line of the 1st North panel (traverse A) is shown on Figure 4-5. First mining within the panel was completed in July 1963. The subsidence values given on Figure 4-5 are referred to initial values taken in June 1963. Measurements taken at other areas above the workings indicated that surface movements accompanying first mining were negligible. Pillar mining was subsequently initiated, retreating southwards along the length of the panel, and this was completed in February 1964.

Inspection of Figure 4-5 shows the subsidence profile to be quite irregular, and this is attributed to local extraction ratio variations within the panel. Subsidence commenced at the northern end of the panel and followed pillar mining towards the south. The data also show that subsidence was continuing at an approximately uniform rate until monitoring ceased. Vertical movements for two stations (Nos. 10 and 18) are shown plotted as a function of time on Figure 4-5. Since pillar mining within this panel was completed in February 1964, it is evident that subsidence continued to develop well beyond the cessation of pillar mining. The data suggest a mean rate of subsidence of the order of just less than 1 inch per month.

At the completion of monitoring, a maximum surface movement of approximately 1.8 feet had occurred. Since the mining height

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LOCATION OF SUBSIDENCE TRAVERSE LINES AT WILLS - WEAVER

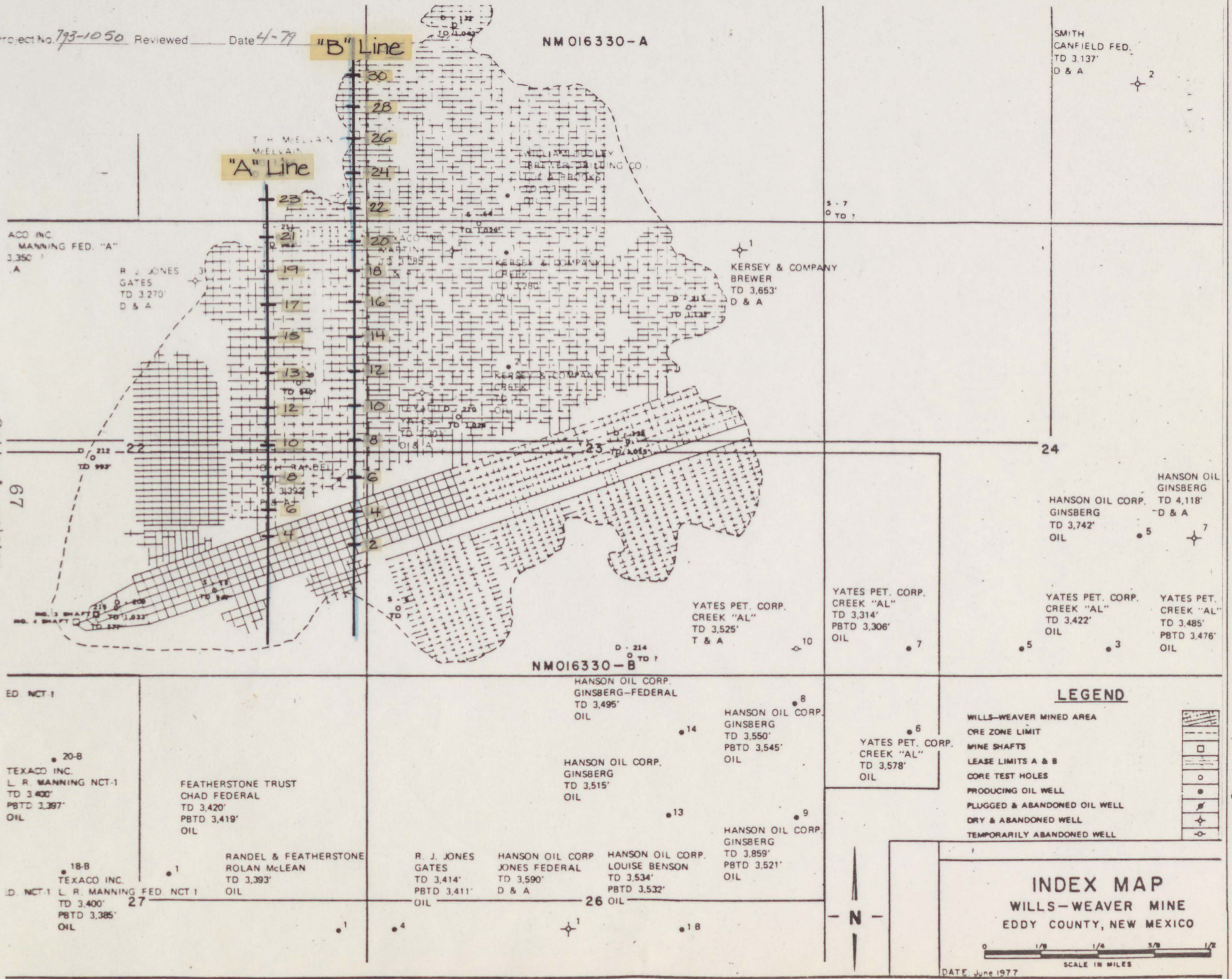

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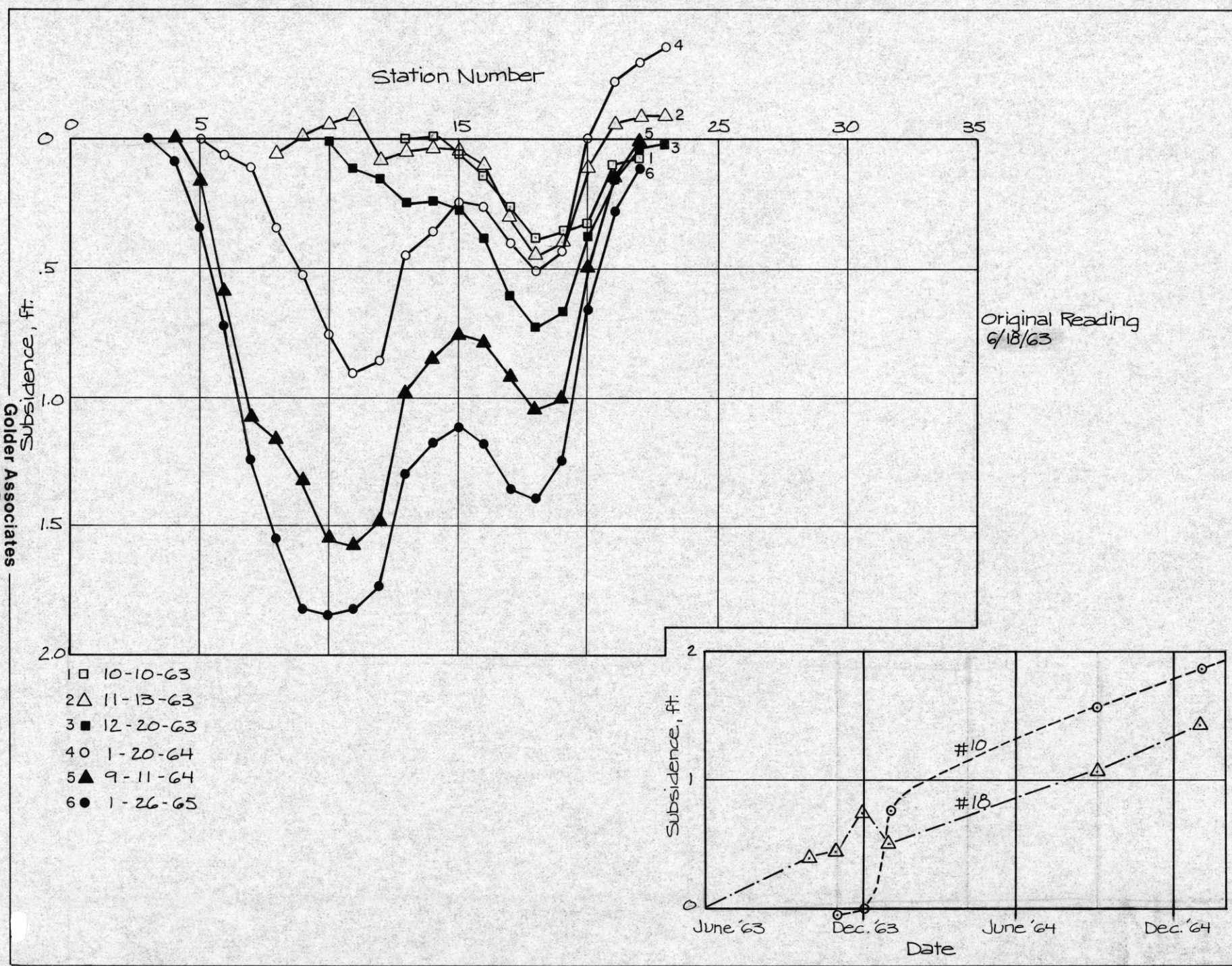
Figure 4-4



SURFACE SUBSIDENCE PROFILES ALONG 'A' LINE OVER PANEL 1 NORTH

Figure 4-5

was approximately 4.3 feet, this represents a surface movement of about 40 percent of the thickness of the mined ore zone. For an average extraction ratio of 80 to 85 percent, this figure would appear to be low by comparison with experience elsewhere in the potash leases of southeast New Mexico. The following points should be noted, however.

- Movements were evidently continuing and the observed movements to the end of the monitoring period do not represent total movements.

- The proximity of the edges of the mine panel to the subsidence traverse ensured that maximum subsidence would not develop.

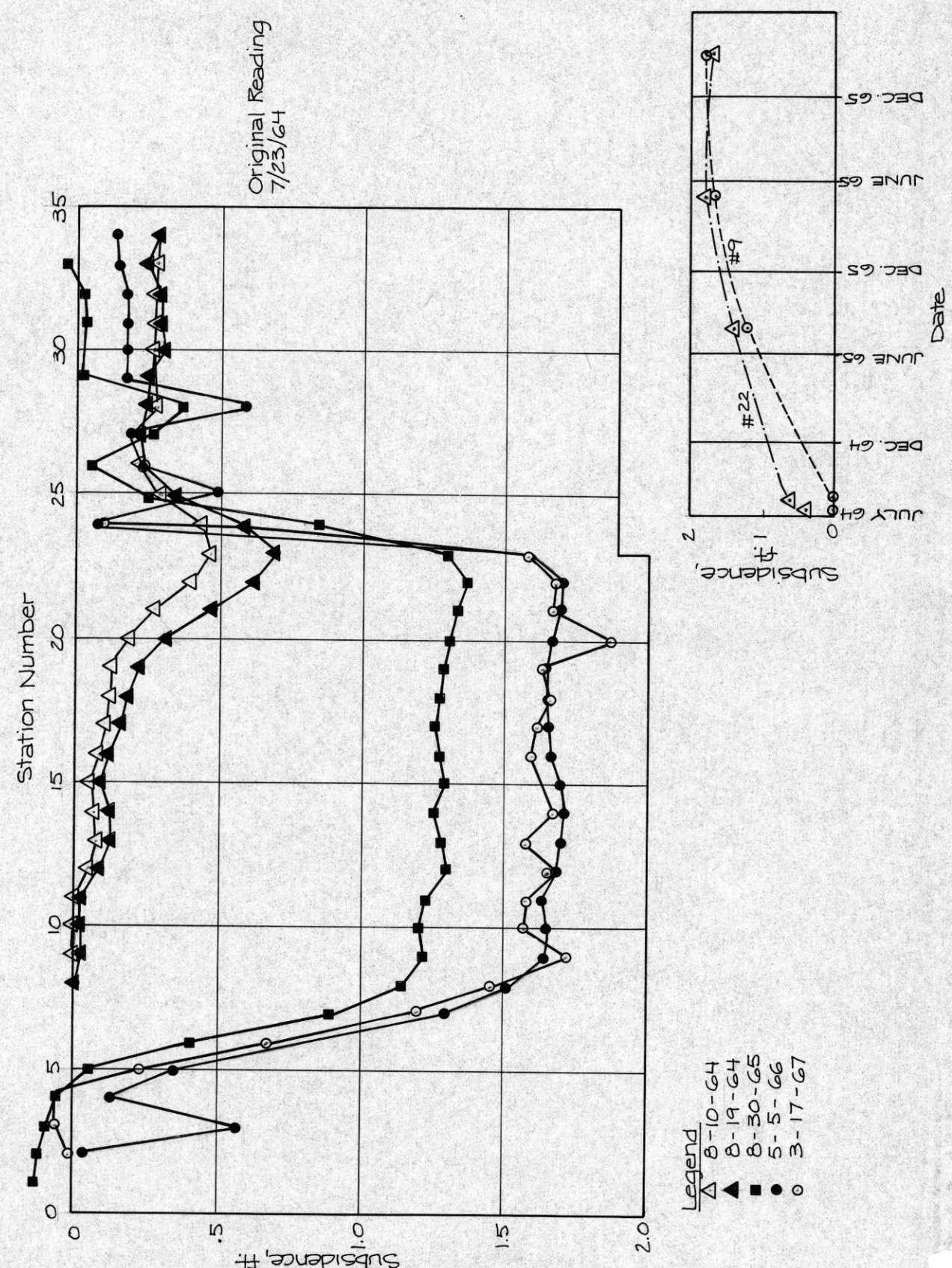
- The relatively low mining height provided for high aspect ratio pillars (aspect ratio equals pillar width/ pillar height) remaining within the mine horizon and hence substantial pillar support capacity.

Similar data are shown for the traverse (Traverse B - Figure 4-4) taken along the center line of the 2nd North production panel (Figure 4-6). The initial readings for these data were taken in July 1964, and the movements indicated on Figure 4-6 are referred to this date. First mining within the 2nd North panel was completed in July 1964, and surface movements prior to this time were therefore probably negligible. Pillar mining was finished in the 2nd North panel in January 1965.

The subsidence profiles shown on Figure 4-6 demonstrate the fairly regular nature of the movements over the majority of the traverse, presumably as a result of a more uniform extraction ratio than that employed in the 1st North panel. Once again, the

SURFACE SUBSIDENCE PROFILES ALONG 'B' LINE OVER PANEL 2 NORTH

Figure 4-6



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data show movements continuing well beyond the completion of pillar mining. Unfortunately, monitoring was carried out very infrequently in relation to the mining phase, and a significant amount of detail was therefore lost, e.g., a traverse was run in August 1964, and the next measurements were taken in August 1965, even though the entire panel was pillar mined between July 1964 and January 1965. Nevertheless, the data do show that surface settlement continued well beyond the completion of mining. Between August 1965 and May 1966, an average vertical settlement rate of about 0.6 inch per month occurred.

Once again, maximum surface settlements of the order of 1.8 feet were experienced, although the nature of the movements appeared to be somewhat different from those experienced above the 1st North panel. At the completion of approximately 19 months of monitoring above the 1st North panel, the surface subsidence showed every sign of continuing to develop. Above the 2nd North panel, however, surface movements were indicated as having virtually ceased after approximately 20 months. Because there are no detailed data available regarding the nature of the residual pillars left within these panels, it is not possible to comment in more detail on these observed variations in behavior.

4.4 REVIEW OF SUBSIDENCE AT WILLS-WEAVER

Comparatively limited subsidence monitoring has been undertaken at the Wills-Weaver mine, but more detailed studies have been carried out at other mines (U.S. Borax and Chemical Corporation) on the potash leases of southeast New Mexico. This has resulted in a fairly detailed understanding of subsidence phenomena within this area. The following points are significant.

- In areas of high extraction, the mining horizon will close and surface subsidence follows.

- The overlying subsiding strata suffer comparatively limited distortion and disturbance, and this suggests that for 100 percent extraction the surface movements would be similar in magnitude to the thickness of the mined ore zone, providing the lateral extent of the workings was large enough.

- In general, residual pillars are left within the workings, and at Wills-Weaver the average extraction ratio in the monitored areas where pillar mining has been carried out is of the order of 80 to 85 percent. These pillars provide a level of support and subsidence develops at a controlled rate. This rate will be determined by the depth of the mining horizon and the extraction ratio (which determine the average pillar vertical stress), together with the horizontal dimensions of the residual pillars and the height of the mined horizon (which determine the pillar aspect ratio). Maximum surface subsidence rates at Wills-Weaver following the conclusion of pillar mining were on the order of 0.5 to 1.0 inch per month. It can therefore be assumed that the residual pillars were closing at a minimum rate on the order of 1.0 inch per month. Although surface movements did not appear to be fully developed at the completion of subsidence monitoring, the rates of observed subsidence would suggest that within these specific areas, very little surface movements are occurring at this time since the mining horizon would be largely closed.

- Apart from the residual pillars, the proximity of the surface to the edge of the mine workings (or to an area of the mine which is supported) will also inhibit the

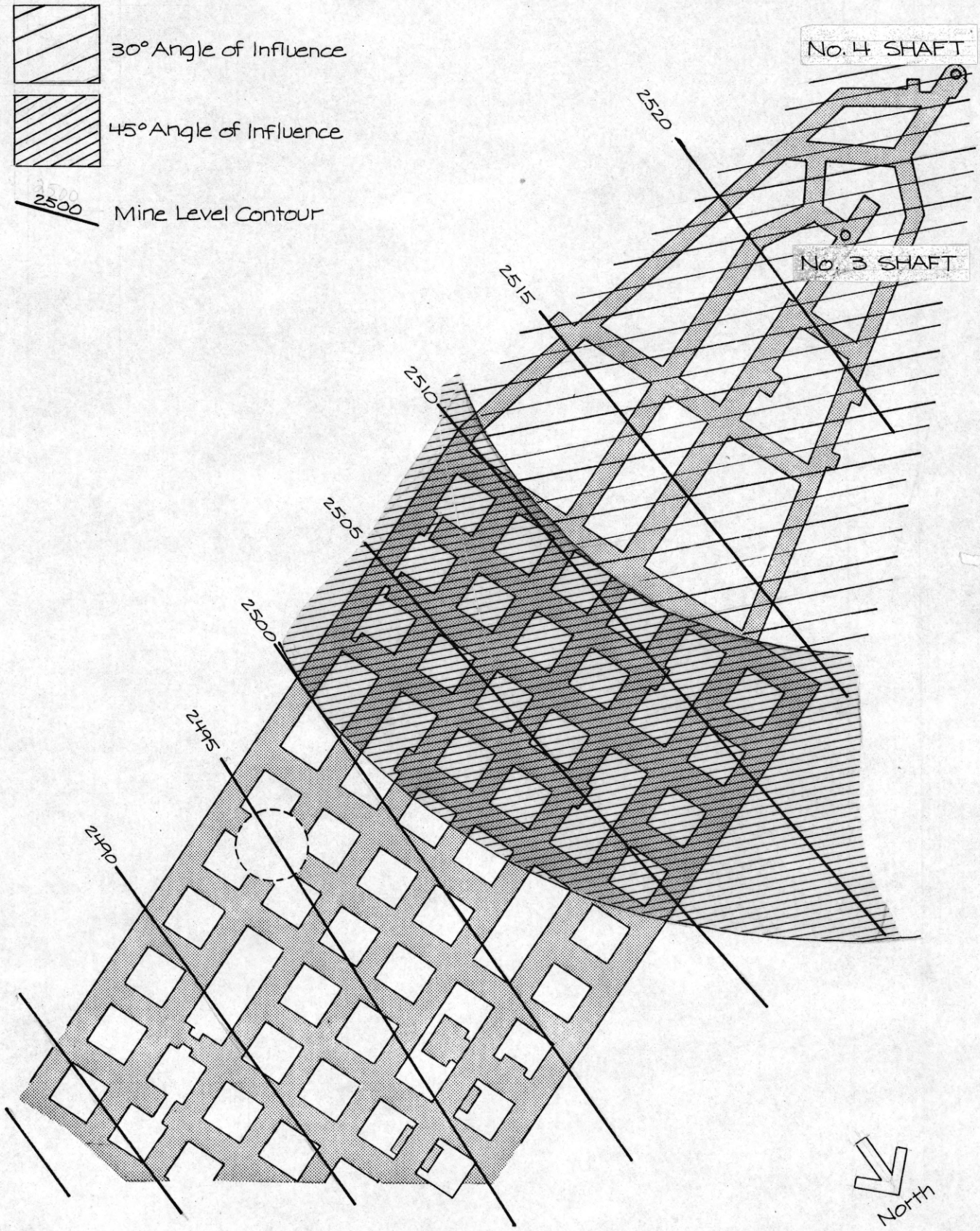
rate and total amount of surface subsidence which occurs.

- The zone of disturbance of strata above the mine workings extends beyond the limit of the mine workings and data from the southeast New Mexico potash fields suggest that a reasonable limit for defining this zone of disturbance would be an angle of 45 degrees from the vertical. More significant disturbance would occur within an influence angle of about 30 degrees. These areas of influence, with respect to possible damage to the No. 3 shaft, are indicated on Figure 4-7. The 30 degree influence cone is seen to include only the shaft pillar protection region (average extraction of 40 percent), and no significant closure of the mine horizon, with consequent overlying strata disturbance, is anticipated from this region. The 45 degree influence cone encompasses a substantial region with an extraction ratio of about 60 percent, and any significant pillar closure within this region could have an effect on the strata within the vicinity of the No. 3 shaft. The rate at which closure of the mining hozizon is expected to develop within this area is discussed in Section 5.

- The three producing oil wells currently located on the Wills-Weaver mine site are protected by salt pillars with radii of approximately 150 feet. Surrounding these pillars, the average ratio of extraction is about 70 to 75 percent, and some closure of the mining horizon would probably have occurred within these areas. These closures at the mine horizon level would have impacted the oil wells because of the limited pillar sizes, but the level of disturbance has evidently not been sufficient to cause problems.

MINE HORIZON REGIONS WHICH COULD IMPACT SHAFTS IF SUBSIDENCE OCCURS

Figure 4-7



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- No data are available on the effect that strata subsidence has on movement of water within the aquifers overlying the Salado Formation. It is evident, however, that the hangingwall suffers a minimum of disturbance and largely retains its cohesive nature.

5.0 PRESENT CONDITION OF MINE WORKINGS

5.1 INTRODUCTION

The Wills-Weaver mine covers approximately 720 acres of workings, mined by the room and pillar method of mining, with large areas of secondary pillar recovery. Mining commenced in 1960 and was completed in 1971. The mine floor is approximately 990 feet below ground level in the vicinity of the No. 4 shaft which is located in the extreme southwest of the workings. The No. 3 shaft, which was the main supply shaft for the mine, is located approximately 300 feet to the northeast of the No. 4 shaft. The mining horizon dips gradually towards the east with an average dip of about one degree.

In general, mine development proceeded towards the northeast. The Main East development, providing access to the remainder of the mine, was developed as an elongated panel with 60-foot square pillars and an extraction ratio of about 60 percent. The majority of the mine workings lie to the north of the Main East entry, with more limited workings to the south. In general, initial mining of each of the production panels employed extraction ratios of about 60 percent, with subsequent pillar mining raising extraction ratios in some areas to in excess of 80 percent. Pillar mining in the workings to the south of the Main East entry and within the Main East entry (east of the 3rd North panel), has resulted in extraction ratios in these areas of the order of 70 to 75 percent. Pillar mining within the 1st and 2nd North panels has achieved extraction ratios of the order of 85 percent. In the remainder of the north panel workings, extraction ratios in the vicinity of each of the main entries to the panels have remained at about 55 to 60 percent, with pillar mining achieving approximately 70 to 75 percent extraction over

the remainder of each of the panels. The Main East entry (west of the the 3rd North panel) has suffered no pillar mining and has an extraction ratio of approximately 60 percent. An overall summary of ore extraction within the Wills-Weaver mine is shown on Figure 5-1.

Within the immediate vicinity of the shafts, ore extraction has been limited. The Main East entry begins its development approximately 550 feet to the northeast of the No. 3 shaft and 850 feet to the northeast of the No. 4 shaft.

In general, the mining height within the Wills-Weaver mine was 52 inches. The room and pillar panels were developed from main entries driven along the long center lines of the panels. The main entries are typically 30 to 40 feet wide and 5 to 7 feet high. The panel rooms were developed by undercutting and overcutting for dilution control and blasting efficiency, drilling, blasting, and mucking. On initial mining, the panel rooms were typically 30 feet wide and approximately 4 feet high. The first mining phase achieved extraction ratios approaching 60 percent.

Pillar mining was carried out by retreating from the outer edges of the developed panel towards the main haulageways. Rooms were typically widened to 50 feet and extraction ratios increased to 70 to 85 percent. Although roof conditions during first mining were generally good, it is understood that some roof problems were experienced during pillar mining. Problems of this type can generally be alleviated by limiting room widths and extraction ratios during first mining, minimizing delays between first mining and pillar mining, and providing roof support during first mining. Only sporadic roof bolting had been carried out in the area of the Wills-Weaver mine which could be inspected, although it is understood that heavier bolting patterns were employed during the later stages of mine development as a result of problems during the development of the earlier mining panels.

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EXTRACTION RATIOS AT WILLS-WEAVER

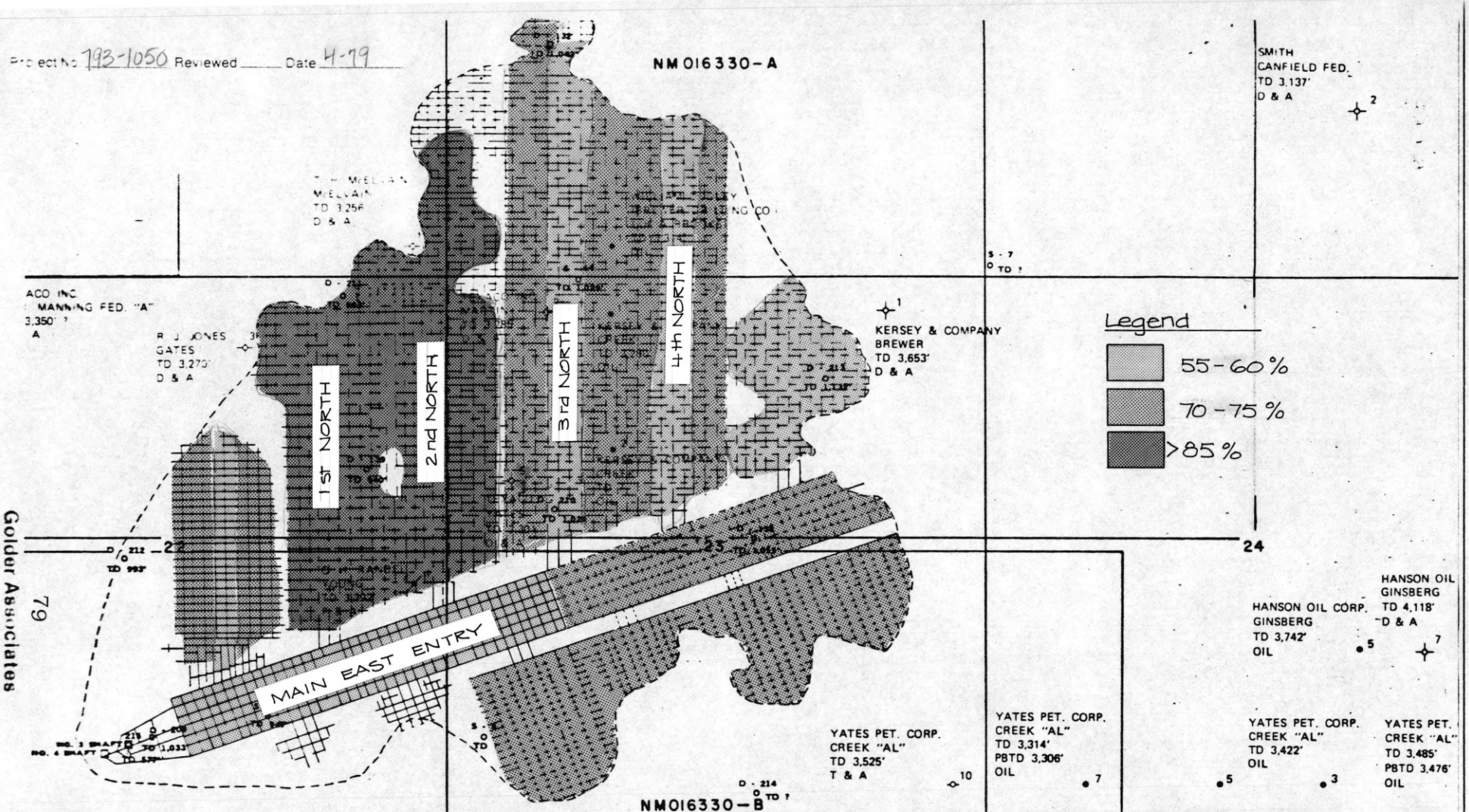

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Figure 5-1

The present condition of the mine workings has been evaluated by the following procedures.

- Geotechnical inspection of the workings which are presently accessible. A detailed inspection was undertaken within approximately 1,000 feet of the shafts, and a much more general evaluation of the remaining accessible workings was also carried out.

- Consideration of site specific and general rock mechanics data which assisted in an evaluation of the observed current level of mine stability. These data included laboratory and in situ information.

There are two basic reasons for establishing the present stability conditions within the mine workings. First, temporary abandonment procedures for the mine shafts implies that the mine workings may be utilized at a future date. The nature of any potential future utilization includes resumption of mining operations if ore reserve determinations indicate economic deposits, energy storage or waste storage reservoirs, etc. These uses require that the future condition of the mine be predicted and the present condition of the mine therefore consitutes a fundamental starting point for this type of evaluation.

Second, the stability of the mine workings is closely related to the question of subsidence of the geological strata overlying the workings. Subsidence of the overlying strata can adversely impact the performance of engineering structures located on or within the the affected region, in particular the mine shafts. The potential for future strata subsidence is

therefore determined by the present and future stability of the mine workings.

5.2 DETAILED CONDITIONS WITHIN VICINITY OF SHAFTS

A detailed geotechnical inspection of the mine workings was carried out to an approximate distance of 1,000 feet from the shafts. This zone was identified on the basis of the region of the mine workings which could potentially cause subsidence damage to the mine shafts. (Details of the zone of influence of subsidence are discussed in Section 4.) The conditions of the roof, floor and pillars within this region were examined.

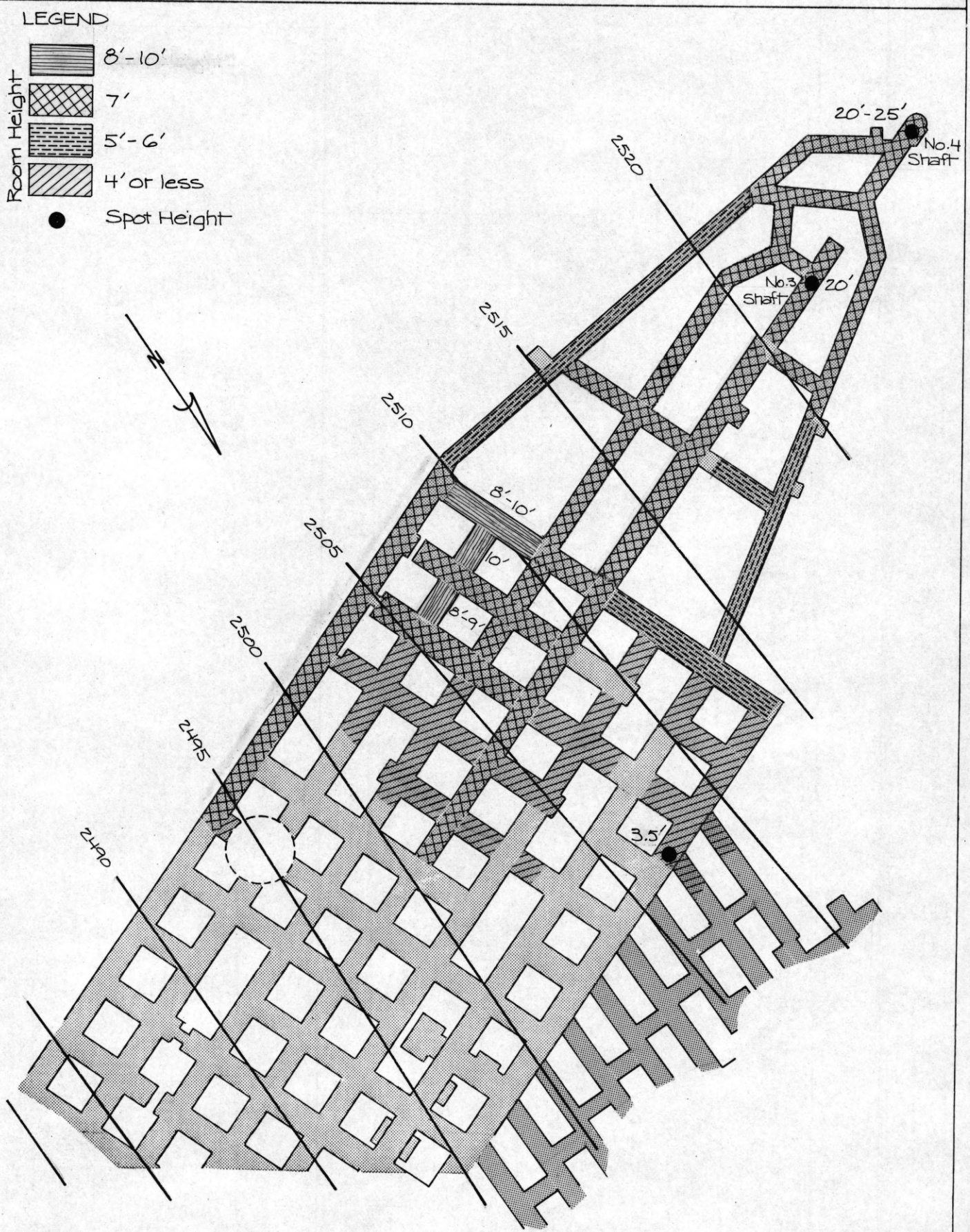
Development of the mine workings proceeded in a direction northeast of the mine shafts. The Main East access panel commences approximately 550 feet northeast of the No. 3 shaft and 850 feet northeast of the No. 4 shaft. The shafts are connected to the Main East panel by four access drifts, with room spans of 25 to 35 feet and room heights of 5 to 7 feet. The access drifts are separated by a minimum horizontal distance of 70 feet. Ore extraction within this pillar protection zone was therefore limited.

The region subjected to detailed examination is indicated on Figure 5-2. Room heights within the Main East panel are generally of the order of 4 feet, although higher rooms are developed at the start of the panel. These rooms are typically 7 feet high, with limited areas 8 to 10 feet high. The distribution of ceiling heights within the vicinity of the shafts is indicated on Figure 5-2. The mine level stations of both the No. 3 and No. 4 shafts have ceiling heights of about 20 to 25 feet.

Within the Main East panel, the mine is developed on a regular room and pillar arrangement. Pillars are 60 to 65 feet

AREA OF DETAILED GEOTECHNICAL INSPECTION

Figure 5-2



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square with typically 35-foot span rooms, giving an overall extraction ratio of about 60 percent.

5.2.1 Pillar/Sidewall Conditions

In general, pillar and sidewall conditions within this region of the mine are excellent. Although the original overburden stress at the mine level is approximately 1,000 psi, and an extraction ratio of 60 percent implies average pillar stress levels of 2,500 psi, the very high pillar aspect ratios (aspect ratio equals pillar width/pillar height) impart a high degree of stability to the pillars. This is discussed in more detail in the following sections.

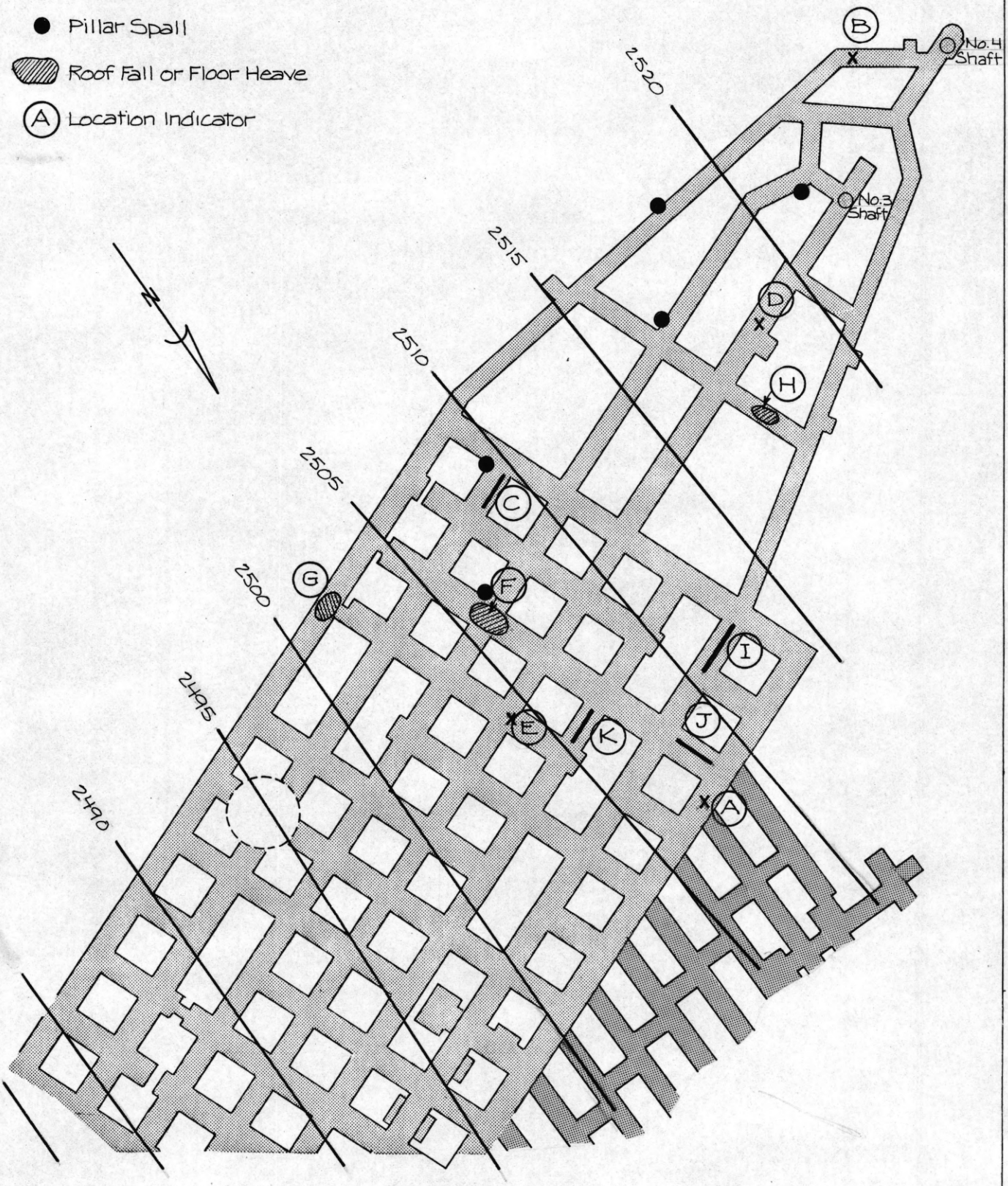
Very localized sidewall spalling was observed at a few locations, indicated on Figure 5-3. Two of the areas correspond to pillar corners within the Main East panel. These pillars were 7 to 10 feet high, and the corner slabbing typically dipped steeply from the roof-line towards the floor, reaching a maximum depth of 1 to 3 feet into the pillar sidewall.

Minor sidewall spalling was also observed within the shaft pillar protection area, at the locations indicated on Figure 5-3. Maximum depth of sidewall spalling was approximately one foot. (Plate 5).

Sidewall spalling within the Main East panel (60 percent extraction) appeared to be no more severe than the spalling within the shaft pillar zone, and this is evidence of the very stable condition of the pillars within this area. The drill and blast production cycle produced sidewalls which were somewhat irregular, but there is no evidence of pillar distress of any sort within the area defined on Figure 5-2.

LOCATIONS OF AREAS
OF GEOTECHNICAL INTEREST

Figure 5-3



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5.2.2 Roof Conditions

The condition of the roof within a 1,000 foot radius of the No. 3 shaft was generally good, despite very limited roof bolting. Within the 4 foot high production rooms, where the roof had been formed by overcutting for dilution control, the roof had a very clean and smooth appearance. There were regular steps in the roof-line as mining followed the slightly eastward dipping potash bed, but there was no evidence of any tendency of the roof beds to separate at these 4 to 6 inch steps. There was no roof bolting at all within the four foot rooms, and no evidence of roof falls or roof distress of any sort. Room closure had apparently occurred, however. At location A (Figure 5-3) the room height was approximately 42 inches, which is about 10 inches less than the presumed mining height. It is not believed that this closure is due to pillar compression, but to a combination of roof sag and floor heave. It is not possible to identify whether roof sag or floor heave was the major contributor to this room closure, but evidence from other parts of the workings would suggest that the latter was probably more significant. The roof condition in this region appeared to be very sound.

In the remaining rooms of greater height (typically 5 to 7 feet high), the roof had a rougher appearance because it had been formed directly by drilling and blasting, without the benefit of overcutting. Once again, however, the roof condition was generally good with only occassional minor spalls. Apart from a regular pattern of bolting (6 to 8 foot centers) within the Main East heading, roof support was sporadic and confined to limited regions. At location B (Figure 5-3), where an abrupt change in roof elevation showed some evidence of separation of a one foot thick bed along a shale parting, a more intense bolting pattern was employed. Limited room closure within this area was indicated by diagonal cracks within a block bulkhead.

At various locations (C and D, Figure 5-3), slots had been cut into the roof, and this provided an opportunity to inspect the roof strata directly. Roof bolting had been installed around these slots (typically 4 to 8 feet deep), and the roof strata showed no signs of separation or distress of any sort. A further opportunity to observe the roof strata existed at locations where the 4-foot high production rooms intersected the Main East heading (typical location E, Figure 5-3). At these intersections, there is an abrupt step in the roof-line of 2 to 3 feet (i.e., the elevation of the roof of the production room is 2 to 3 feet lower than that of the Main East heading). Although no roof bolts were installed, the 35-foot span roofs appeared very stable with no evidence of bed separation within the immediate roof strata. The typical appearance of these roof strata is indicated on Plate 6.

One major roof fall was observed at the intersection of the 1st Right adit and the Bt6 crosscut (location F, Figure 5-3). The scope of the fall encompassed the entire intersection and included 20 to 30 feet of the Bt6 crosscut roof to the north of the intersection. A one-foot thick bed of material fell from the roof en masse, separated from the roof along a thin shale parting. Surrounding the perimeter of the fall, the roof bed has been torn away, leaving an approximate 6-inch separation (Plate 7). The general appearance of the collapsed roof bed is shown on Plate 8.

It is understood that this roof fall occurred about 2.5 years ago, approximately 5 years after the completion of mining activity at Wills-Weaver. This area had not been previously bolted, and had shown no signs of distress. The original roof within this region had been formed by drilling and blasting. There appears to be no rational way of predicting the occurrence

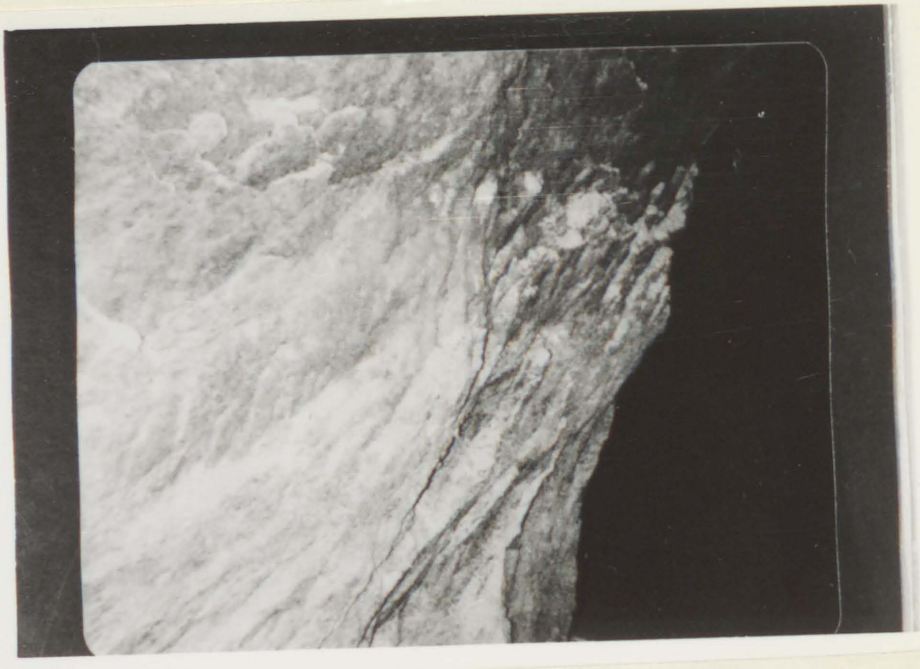

Plate 5

Sidewall spalling



Plate 6

Production room roof strata

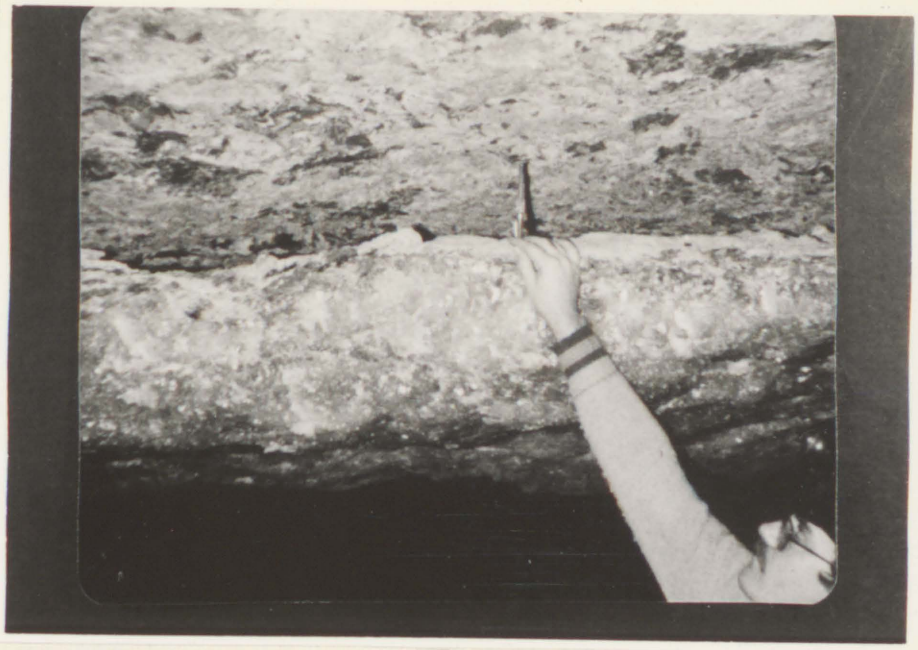

Plate 7

Separated roof strata at perimeter of a roof fall



Plate 8

Collapsed roof strata

of such roof falls, although the generally good conditon of the roof in virtually all visited areas of the mine suggests that this fall was more or less an isolated occurrence. The possibility of further such falls in the future cannot be discounted, although there is no reason to suggest that roof problems within the presently accessible areas of the mine workings will develop and become widespread. In general, roof conditions in the accessible areas of the Wills-Weaver mine must be regarded as excellent.

The roof bolting installed consisted of torqued and mechanically anchored bolts varying between 2 and 4 feet in length. There is no evidence of distress in any of the bolts, as indicated by the undistorted nature of the faceplates.

5.2.3 Floor Conditions

The mine floor within a distance of 1,000 feet of the shafts appeared to be experiencing virtually no problems. At two locations (G and H, Figure 5-3) there was evidence of floor heave, as indicated by a drummy sound to the floor. At location G, some floor buckling had occurred, showing that the thickness of the buckled floor slab was approximately 6 to 12 inches. Although floor damage within this general area was virtually negligible, severe floor heave was observed in the areas of the mine which had suffered high extraction ratio mining. In general, the mine was obviously much more prone to problems of floor heave than it was to roof sag difficulties, and this has been attributed to the presence of carnallite located within the mine floor. Carnallite is known to creep at a faster rate than either halite or sylvite (Baar, 1971), and differential straining could result in the observed floor heave problems.

5.2.4 Other Factors

Within the area subjected to a detailed geotechnical inspection, the mine workings were generally in a clean condition and free of major obstruction. In the vicinity of location G (Figure 5-3), there were various items of abandoned mining equipment (e.g., cutter bars, etc.). Some of the 4 foot high production rooms (locations I, J, K - Figure 5-3) had also been backfilled with waste salt. Within these backfilled rooms the roofs were generally standing free of the stockpiles and appeared to be in good condition.

5.3 GENERAL CONDITIONS WITHIN ACCESSIBLE MINE AREA

Inspection of the mine workings generally was limited to the Main East Entry panel, as far east as the main entry to the 3rd North panel. Further to the east, pillar robbing within the Main East panel has resulted in the mine workings being inaccessible. Similarly, to the north of the Main East Entry Panel, high extraction ratio mining prevented entry to the workings. The region of the mine workings which was considered safe to enter is indicated on Figure 5-4.

Generally speaking, roof, floor, and pillar sidewall conditions within the Main East panel were similar to those observed during the detailed inspection of the western end of the panel. This is to be expected since room and pillar dimensions and extraction ratios were the same throughout this area. Roofs were stable, pillars showed limited sidewall spall, and there was occasional evidence of floor heave.

The more unstable areas of the mine were inspected at three locations. These are indicated on Figure 5-4. The three areas were:

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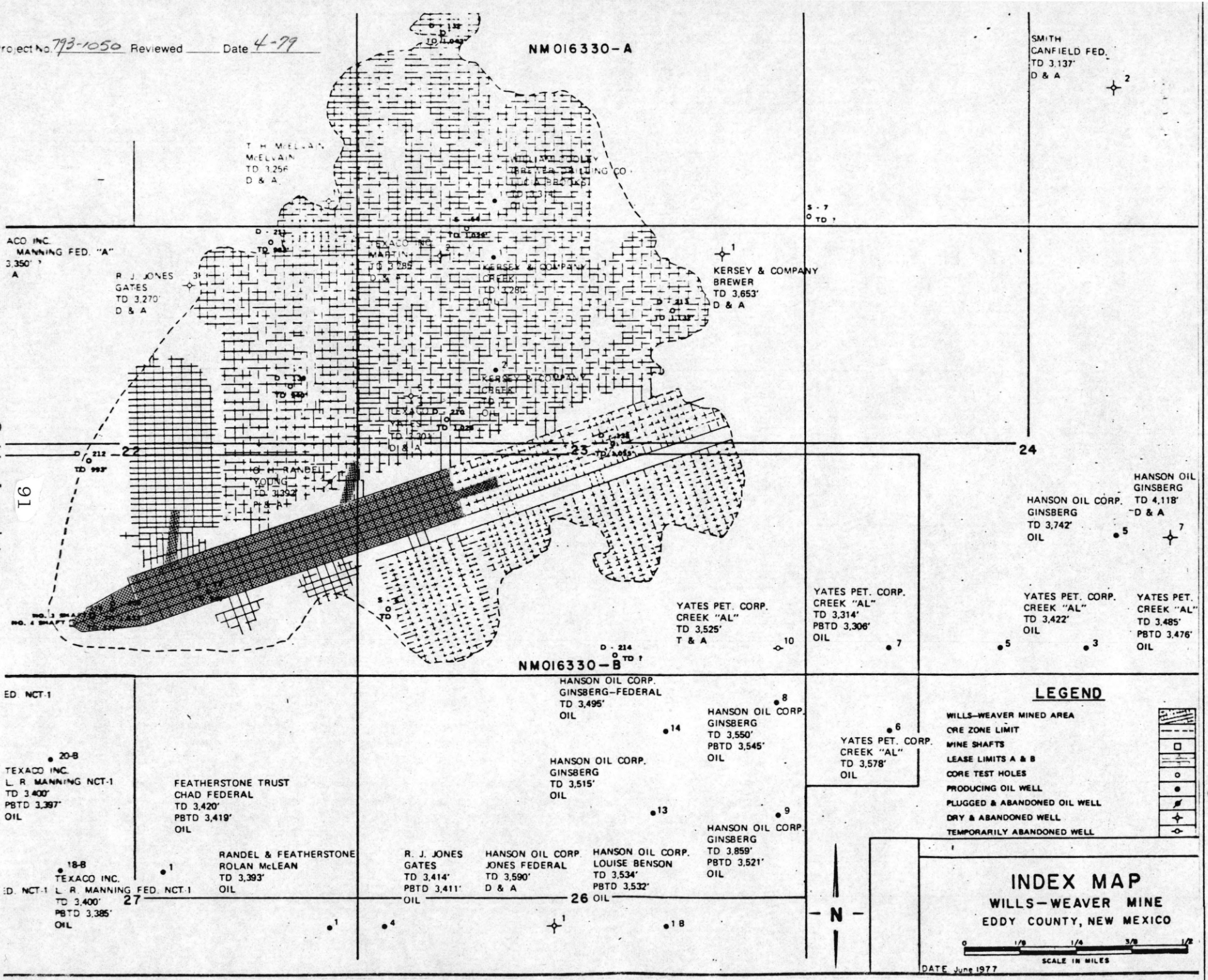

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AREA OF MINE INSPECTED

Figure 5-4

- Entry to the last mined North panel located closest to the shafts
- Entry to 2nd North panel
- Within the high extraction ratio region of the Main East panel.

Within these areas, the extraction ratios are sufficiently high that substantial pillar closures have been experienced. Areas of almost complete room closure were not observed for obvious reasons, but these areas have generally been associated with substantial surface subsidence. Within the areas visited (which were located at the edges of the areas of high ore extraction), roof conditions again appeared to be very stable, but floor conditions deteriorated badly. In many areas, floor slabs had buckled sufficiently that the floor beds in the center of the rooms were in contact with the ceilings. There was also evidence, according to Duval personnel who had visited these areas periodically, that floor heave was continuing to develop within these regions.

5.4 ROCK MECHANICS DATA

5.4.1 Introduction

The engineering performance of evaporite minerals has been widely studied, both within the laboratory and within the field. From a mining point of view, the factor of principal interest is the condition of the mine openings (walls, roof, and floor). Another phenomenon which is influenced and determined by deformations within the mining horizon is surface subsidence.

When a mine panel is developed, the pillars are subjected to increased vertical stress. The magnitude of this stress increase is determined by the extraction ratio, or percentage of the panel which is mined. For a panel which is of sufficient horizontal extent such that the full weight of the overburden must be supported by the pillars, the average pillar vertical stress is given by:

$$\sigma_{AV} = \frac{\sigma I}{(1-e)}$$

σI = initial overburden stress
= γD (D = depth of mine horizon below ground surface; γ = material specific weight)
e = extraction ratio
= mined area/initial area

In general, evaporite minerals will continue to deform with time when they are subjected to differential or shear stresses. The mine pillars will therefore shorten and the room height will diminish. This type of pillar behavior leads not only to room closure but also to surface subsidence. It is therefore of fundamental importance to understand the performance of the mine pillars.

Within the mining horizon, other factors will contribute to room closure. The immediate roof and floor will close as a result of plastic flow of the salt. This is the same process that is responsible for the deformation of the pillars. If the mine is of large horizontal lateral extent, the pillars cannot transfer their loads to mine abutments, etc. By contrast, high loads within the immediate roof and floor can generally be transferred deeper into the roof or floor. Closure of the mine opening by

plastic flow is therefore generally more dependent on the processes occurring within the mine pillars than on the processes occurring within the roof and floor.

Within bedded salt deposits, however, roof and floor strata may tend to separate along bedding structures, e.g., clay seams. When this occurs, these roof and floor slabs may tend to buckle into the opening and severe roof and floor closures can occur. A principal driving force in this process is the force provided by lateral expansion of the pillars. As the pillar compresses vertically under the overburden weight, it also expands horizontally. This lateral expansion provides a driving force to the ends of the horizontal slabs, and buckling occurs. These processes are shown diagrammatically on Figure 5-5.

These roof and floor buckling phenomena (together with normal plastic flow processes within the roof and floor) do not generally contribute directly to surface subsidence, but may be very important factors in determining room closure and stability. The performance of the pillars is the factor of principal interest with regard to surface subsidence.

The above discussion indicates that pillar behavior is of considerable importance in determining the stability of the mine openings and overlying strata. The reasons for the critical nature of the pillars are:

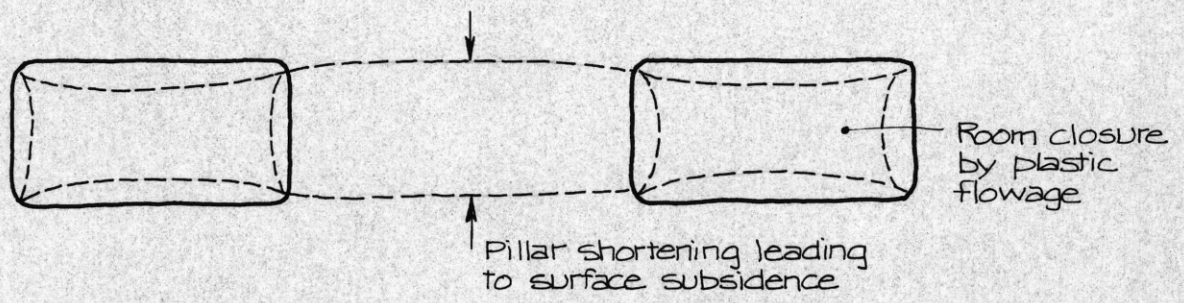
- Pillars support the overlying strata and pillar closure will eventually result in surface subsidence

- Pillar closure will cause closure of the rooms

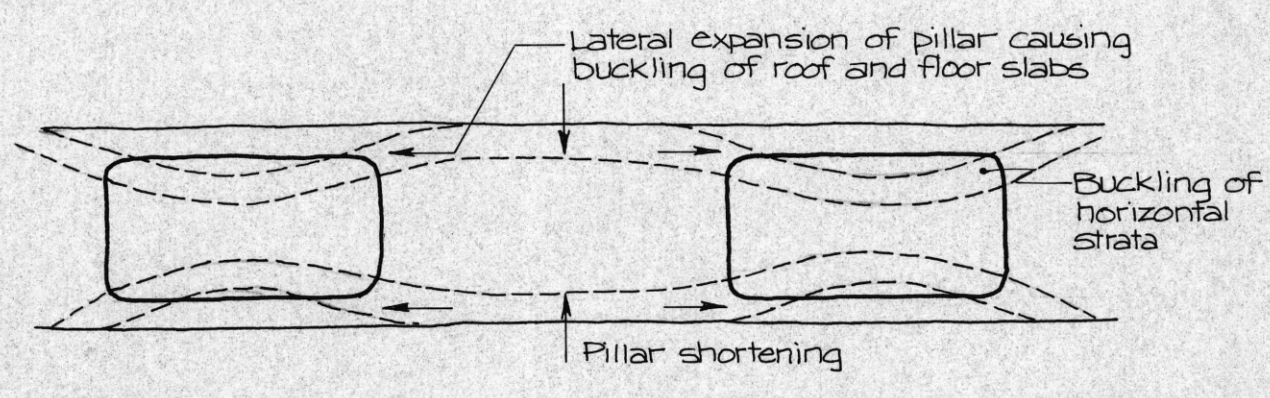
- Vertical compression and accompanying lateral expansion of the pillars will provide driving forces to separated

ROOM AND PILLAR DEFORMATION MODES

Figure 5-5



(a) ROOM AND PILLAR CLOSURE WITH NO BED SEPARATION (Plastic Flowage)



(b) ROOM CLOSURE WITH BED SEPARATION AND BUCKLING

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roof and floor slabs, causing them to buckle and lessen the life of the opening. Quite spectacular buckling of floor slabs can occur. With roof slabs, the bed separation generally results in a roof fall.

In order to evaluate future stability conditions of the mine openings and to explain existing conditions, a review of available information on the performance of pillars within potash deposits has been undertaken.

5.4.2 Pillar Performance

Pillar performance in evaporite materials has been studied by:

- Observation of in situ mine pillars
- Laboratory tests on small model pillars.

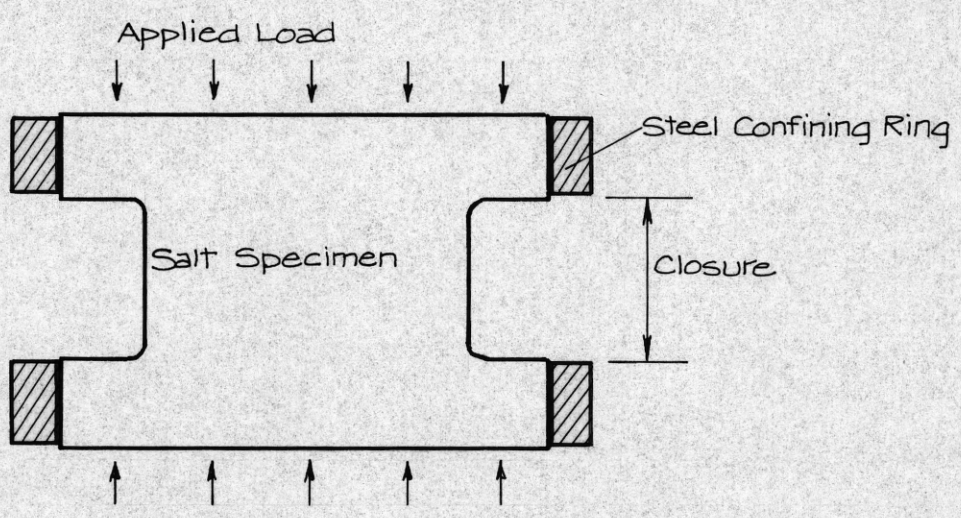
The nature of the laboratory tests generally consists of simulated mine pillars of the type shown on Figure 5-6, to which constant loads are applied and average vertical deformations are monitored as a function of time. The shape of the model pillars (with lateral restraint provided by steel rings) is intended to simulate restraint to lateral expansion, which is provided by the roof and floor of the mine opening.

In situ investigations generally comprise deformation monitoring (lateral expansion across the pillar, wall shortening, room closure, etc.) of pillars which are presumed to be under average stress conditions, determined by the depth of the mining horizon and the level of extraction.

In order to demonstrate the complex nature of deformations within a pillar, reference is made to Figure 5-7. This figure

MODEL PILLAR

Figure 5 – 6



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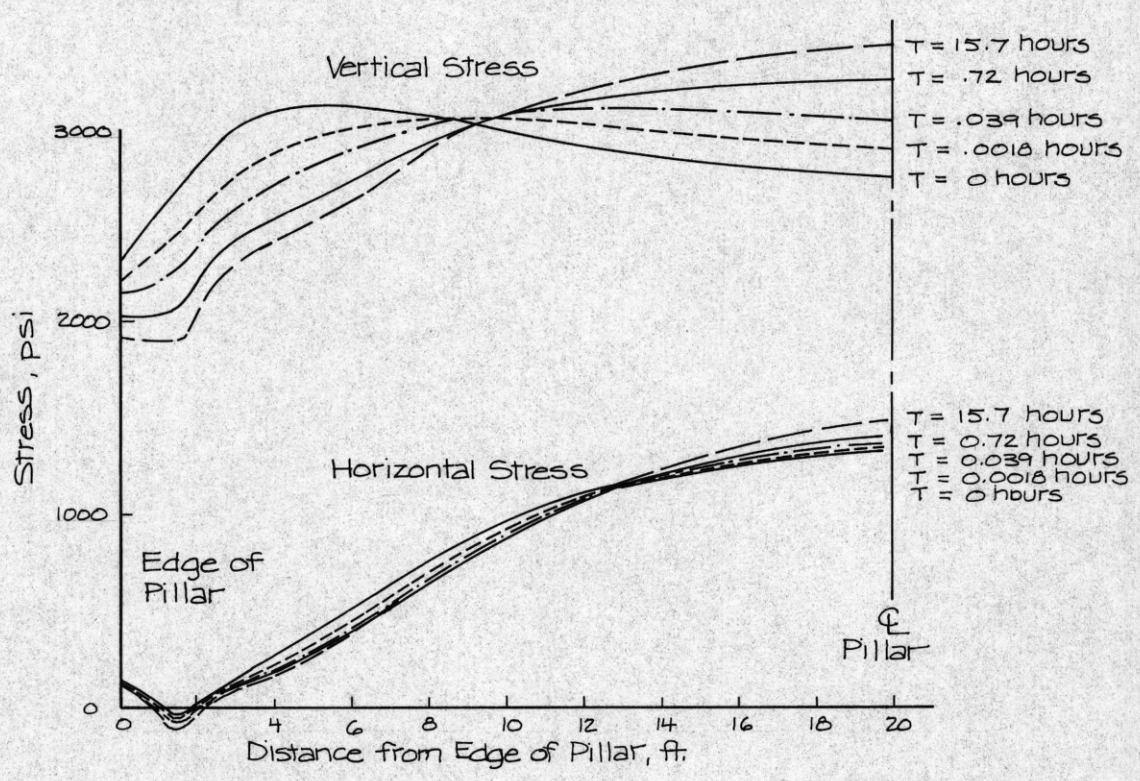
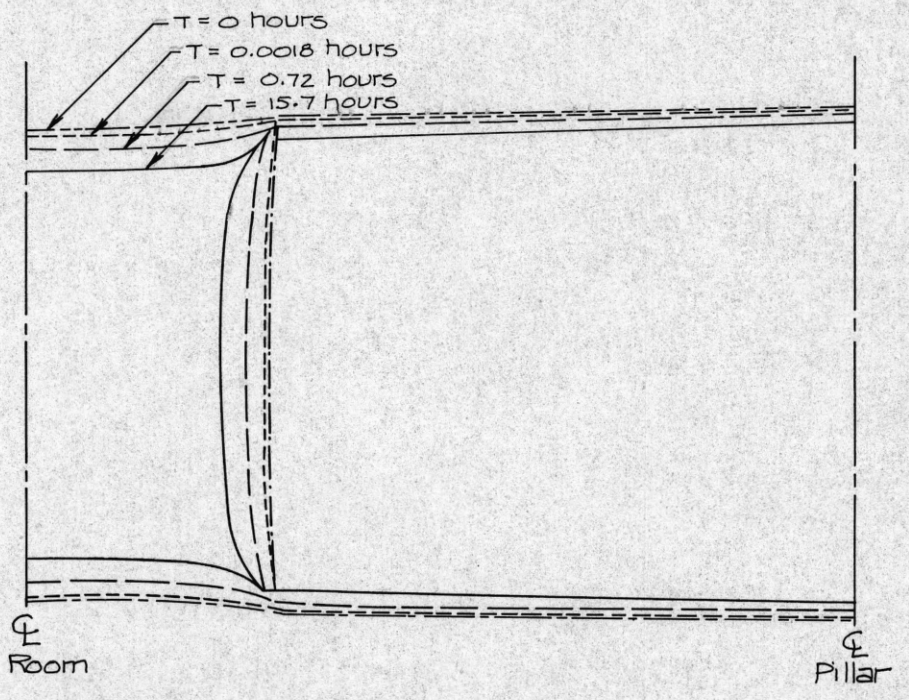
STRESS AND DISPLACEMENT CONDITIONS
WITHIN EVAPORITES UNDERGOING CREEP

Figure 5-7



(a) STRESSES AT PILLAR MID-HEIGHT



(b) DISPLACEMENT PROFILES OF ROOM AND PILLAR

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shows stress conditions across the mid-height of a pillar (assumed to be part of a mine of large horizontal extent), together with the room and pillar deformations, as a function of time. The data given on Figure 5-7 were developed numerically.

The initial elastic stress distribution indicates a vertical stress concentration towards the edge of the pillar, but stress is gradually transferred away from the edge and towards the center of the pillar. The reason for this type of behavior is that the rate of deformation of the material is a function of the level of deviator stress (i.e., the difference between the vertical stress and the horizontal confining stress). The initial elastic solution shows that the deviator stress is highest towards the edge of the pillar (high vertical stress and low horizontal confining stress) and lowest towards the center of the pillar. The pillar walls therefore tend to deform more rapidly than the pillar core, and load is subsequently transferred to the core region. This stress redistribution continues to occur until the deviator stress within the core of the pillar more nearly resembles the stress level near the pillar edges. Since confinement is obviously greater towards the pillar core, vertical stress levels are greater in this region in order to develop the same deviator stress. This type of behavior is shown on Figure 5-7. When the deviator stress levels are more nearly uniform over the pillar cross-section, the pillar will tend to compress vertically at a more uniform rate.

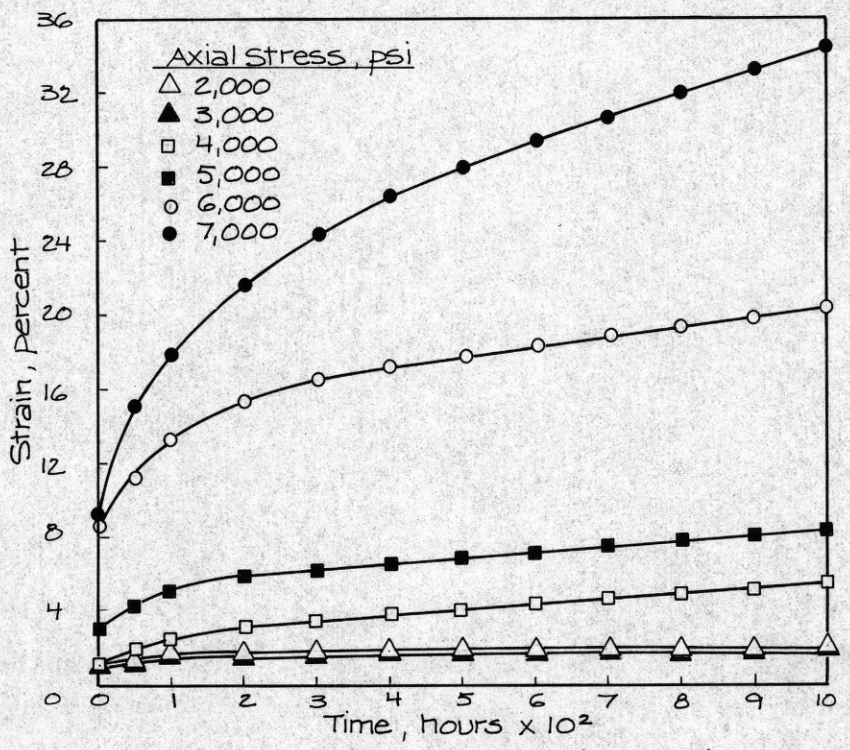
The room and pillar deformation profiles on Figure 5-7 illustrate the above behavior. In general, the pillar does not deform uniformly across its full width but shortens more towards the wall of the room. This type of differential straining across the width of the pillar has been noted both experimentally (Bradshaw and McClain, 1971) and in situ (McClain, 1964).

The above discussion helps to illustrate the complex nature of the deformations which occur within a mine pillar. It is often convenient to refer to specific components of the pillar deformation (e.g., pillar wall deformation in situ, average specimen deformation of a laboratory model pillar), and the nature of the overall movements which occur within the pillar should be kept in mind when comparisons are being made.

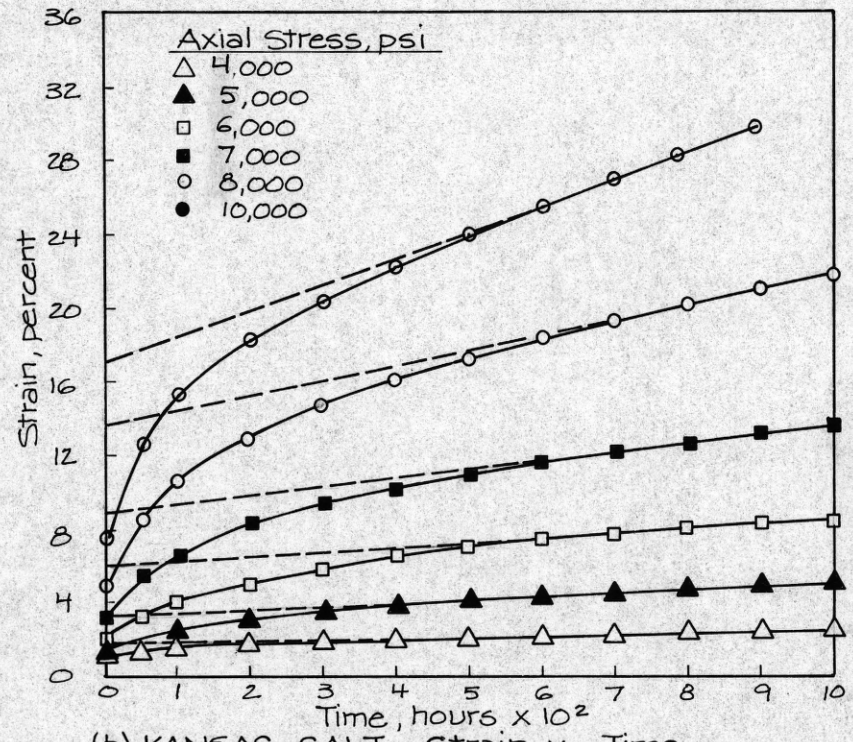
When a rock salt/potash mine pillar or laboratory specimen is initially loaded, it deforms relatively rapidly, and subsequently strain hardens so that the deformation rate is reduced. The long-term deformation behavior of such materials is a subject of considerable debate. Certain authors (e.g., Baar, 1977) maintain that ultimately, a steady rate of deformation is reached and this continues indefinitely. Others (Bradshaw and McClain, 1971) maintain that hardening continues and that the deformation rate continues to decrease. Laboratory testing has done little to resolve this question, primarily as a result of the limited duration of the majority of laboratory test programs. Obert (1965) carried out creep tests on model pillars of salt, trona, and potash ore (pillar diameter of 4 inches, pillar height of 1 inch), with test durations of 1,000 hours. The test results for Kansas salt showed creep rates continually decreasing with time, whereas the New Mexico potash ore appeared to indicate constant deformation rates following an initial hardening phase (Figure 5-8). Similarly, although mine closure measurements have been carried out over long durations, they have not definitely confirmed that material hardening ceases to occur. In some respects, the question of decreasing or constant strain rates is somewhat academic. Available data do indicate, however, that deformations continue for very long periods of time and probably, for all practical purposes, they continue indefinitely.

LABORATORY CREEP BEHAVIOR

Figure 5-8



(a) NEW MEXICO POTASH - Strain vs Time
(after Obert, 1965)



(b) KANSAS SALT - Strain vs Time
(after Obert, 1965)

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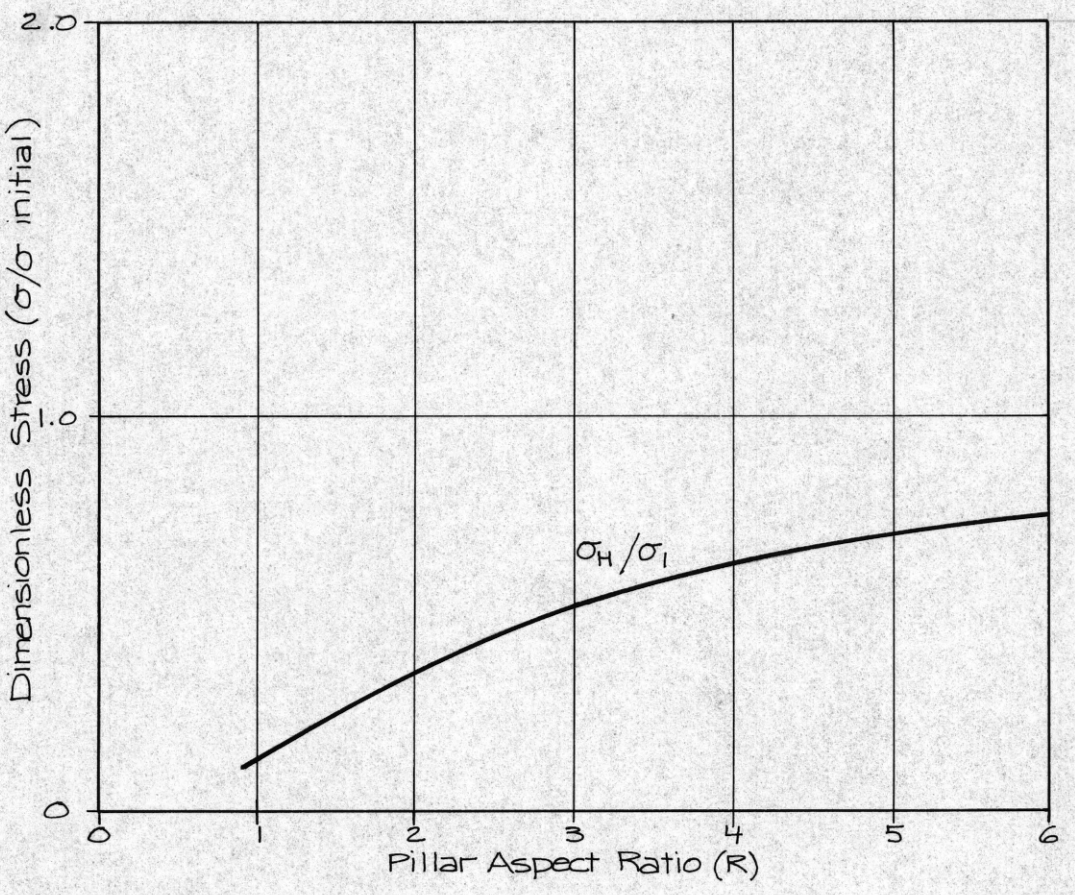
The rate of deformation of a potash material specimen under constant load conditions (at any time following loading) is a function of the level of applied deviator stress (or shear stress) and the temperature. For most mining applications, temperatures are relatively constant, and this parameter need not therefore be considered. In specific applications (e.g., disposal of heat generating waste materials) temperatures may be of significance, and deformation rates can increase greatly as the temperature rises. For the present purposes however, it is necessary to consider the influence of the level of stress only on the rate of material deformation.

There are therefore two principal factors which will influence the rate of creep deformation suffered by a mine pillar. Both of these factors affect the level of applied deviator stress, which is responsible for causing material flow. First, the level of applied average pillar vertical stress is an important factor and this is a function of the depth of the mining horizon below ground level, the specific weights of the sediments comprising the overlying strata, and the extraction ratio. Second, the level of effective pillar confinement also determines deformation rates, and this factor is principally determined by the aspect ratio (relation of pillar width to pillar height) or relative proportions of the pillar. Short and squat pillars will have a substantially higher level of lateral confinement than relatively tall and slender pillars. The influence of pillar aspect ratio on the average level of horizontal confinement at the mid-height of the pillar is demonstrated on Figure 5-9. This relation was developed by integrating elastic solutions for horizontal stress across the width of the pillar, assuming an hydrostatic initial in situ stress. As shown on Figure 5-7, the initial elastic solution for the horizontal stress is a good approximation to the actual stress levels following creep. Also,

PILLAR MEAN HORIZONTAL STRESS vs
PILLAR ASPECT RATIO

Figure 5-9



$R = \frac{W}{H}$

W = Pillar Width
H = Pillar Height

σ_H = Horizontal Stress

σ_I = Initial Hydrostatic Stress

Note: This relation assumes square rooms (room height equals room width) and is not general.

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average vertical and horizontal pillar stresses have a real meaning in terms of establishing the deformation characteristics of the pillar, since the vertical stresses in particular appear to readjust so that the deviator stress is ultimately approximately uniform across the full pillar width. This final deviator stress is the difference between the average vertical and average horizontal pillar stress levels.

The data given on Figure 5-9 were developed assuming square rooms (i.e., room height equals room width), and hence implies a specific relation between the extraction ratio and the pillar aspect ratio. It is not, therefore, a general relation, as the average horizontal stress within the pillar is not only a function of the pillar aspect ratio but also a function of the pillar average vertical stress.

A number of investigators have carried out laboratory studies employing model pillars of the type shown on Figure 5-6. Although the level of experimentation of this type has been quite extensive for rock salt (halite), a more limited amount of data are available for potash ores. Obert (1965) tested pillars (aspect ratio of 4) made of New Mexico potash ore under applied axial stresses varying from 2,000 to 7,000 psi. The data are summarized on Figure 5-11. It was found that the steady state strain rate (achieved after 200 to 400 hours of loading) could be represented by:

$$\dot{\varepsilon} = K\,\sigma_o^{3.3}$$

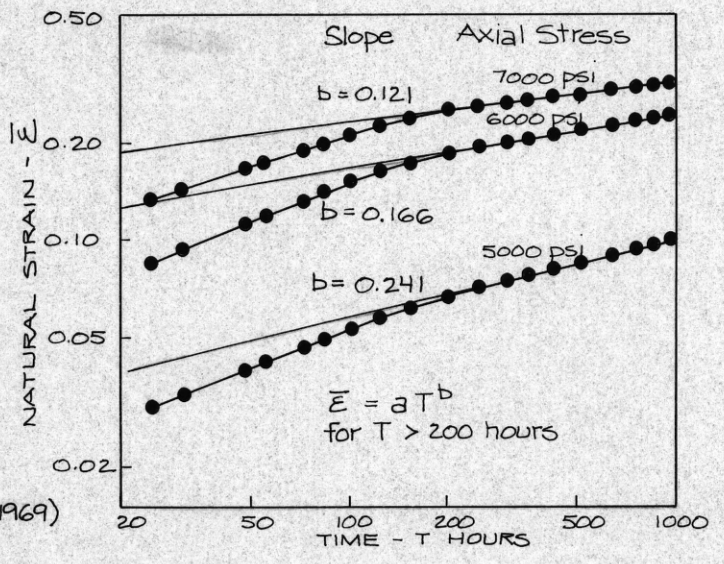
where $\dot{\varepsilon}$ = mean axial rate of strain
σ_o = applied vertical stress
K = constant

The level of horizontal confinement within these model pillars is unknown, and σ_o does not represent the applied deviator loading. Assuming that the level of lateral confinement is

approximately proportional to the applied vertical stress, an equation of the above type may also be used where σ_o represents the average deviator loading. (The value of K would increase, if σ_o were taken to represent the deviator stress rather then the applied axial stress.) It therefore appears as though there is a very non-linear relation between the long-term rate of strain and the applied deviator stress, with the former varying approximately as the third power of the latter.

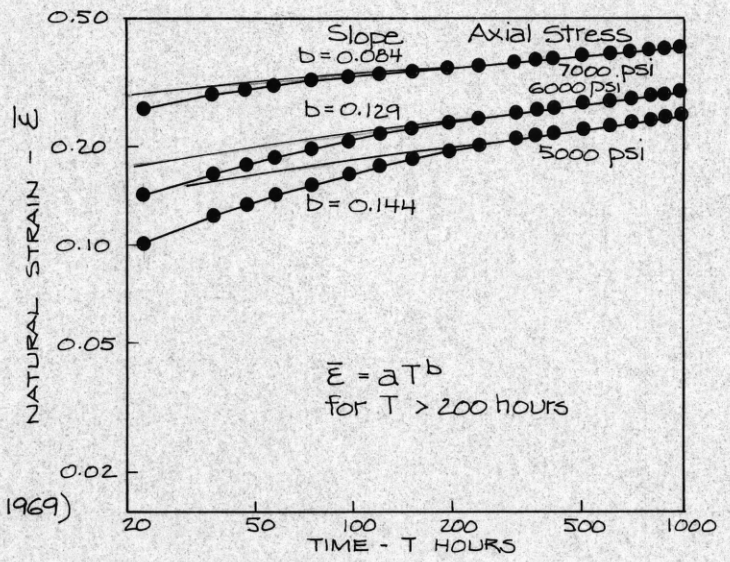
The above type of laboratory testing may provide very useful data of a relative type, but more limited data of an absolute nature. There are a number of reasons for this. One of the most important is that it is not entirely clear what is the exact relation between total stress levels developed within a mine pillar (formed by developing openings within an hydrostatic stress field) and those developed within a model pillar (formed by axial loading of the specimen with lateral restraint provided by steel rings at the ends of the sample). It is considered that the lateral confinement within the model pillar will be less than that within an in situ pillar (even for identical aspect ratios), and this has led Obert (1965) to suggest that creep rates predicted from the models may be somewhat higher than those developed in situ.

King and Acar (1969) carried out similar laboratory tests on model pillars (aspect ratio of 4) of Saskatchewan potash; at two temperatures of 80°F and 110°F; and at applied axial stress levels of 5,000, 6,000, and 7,000 psi. The data are shown on Figure 5-10. Over the period 200 to 1,000 hours following initial loading (test duration was 1,000 hours), the rate of vertical straining of the model pillars was expressed as a power function of time, i.e., the rate of deformation continued to decrease. In addition, there appeared to be some inconsistencies, e.g., for the specimen tested at 80°F, the variation in



(after King & Acar, 1969)

(a) SASKATCHEWAN POTASH
Strain vs Time - 80° F



(after King & Acar, 1969)

(b) SASKATCHEWAN POTASH
Strain vs Time - 110° F

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developed creep strains between applied axial loadings of 6,000 and 7,000 psi was somewhat less than expected. This was attributed to variations in the level of confinement provided by the restraining rings at the ends of the sample. These problems made it difficult to identify a regular pattern of behavior.

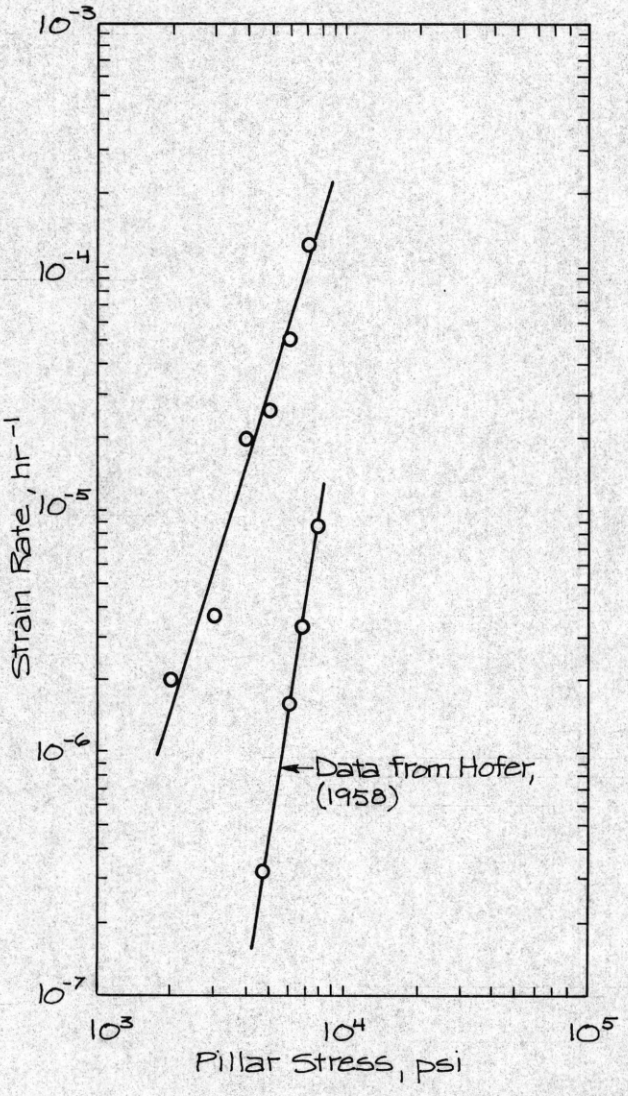
Although room and pillar convergence data within potash mines are available (Baar, 1977), the data are often sufficiently incomplete that it is not possible to establish a general pattern of behavior of the mine pillars. Variations in pillar aspect ratios are often not discussed in detail, and this makes comparisons difficult. The field data do indicate, however, that creep deformations continue for very long periods of time, probably indefinitely.

Hofer (1958) measured the in situ lateral deformation in a number of potash pillars over periods of up to 500 days. Constant creep rates were indicated and the lateral strain rates were plotted as a function of pillar axial stress (calculated from depth of the mining horizon, together with the extraction ratio). These data are shown on Figure 5-11. The strain rate may once again be expressed as a power function of the level of stress, but in this case the exponent has a value of 6.3 compared with the value of 3.3 derived for New Mexico potash using model pillars. Part of this difference may be due to the fact that the in situ pillar aspect ratios varied between 1 and 2, and hence the level of horizontal confinement varied between pillars. Since creep rates are dependent on the level of deviator stress, the same level of applied pillar vertical stress will induce different rates of deformation within pillars with different aspect ratios. In addition, the mineralogy of the German potash referred to by Hofer is not equivalent to that of the New Mexico potash, and variations in behavior might therefore be expected.

NEW MEXICO POTASH
STRAIN RATE vs AXIAL STRESS

Figure 5-11

(after Obert, 1965)



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The discussion presented in Section 5.4.1 indicated that an understanding of the mine pillar performance was a necessary prerequisite to understanding the present stability conditions within the mine, and to evaluating future likely stability conditions. The most appropriate available rock mechanics data which will assist in an evaluation of stability conditions at Wills-Weaver mine, consist of the following:

- Model test pillar experiments carried out by Obert (1965) on New Mexico potash
- Observations of deformations and stability conditions at the mine itself.

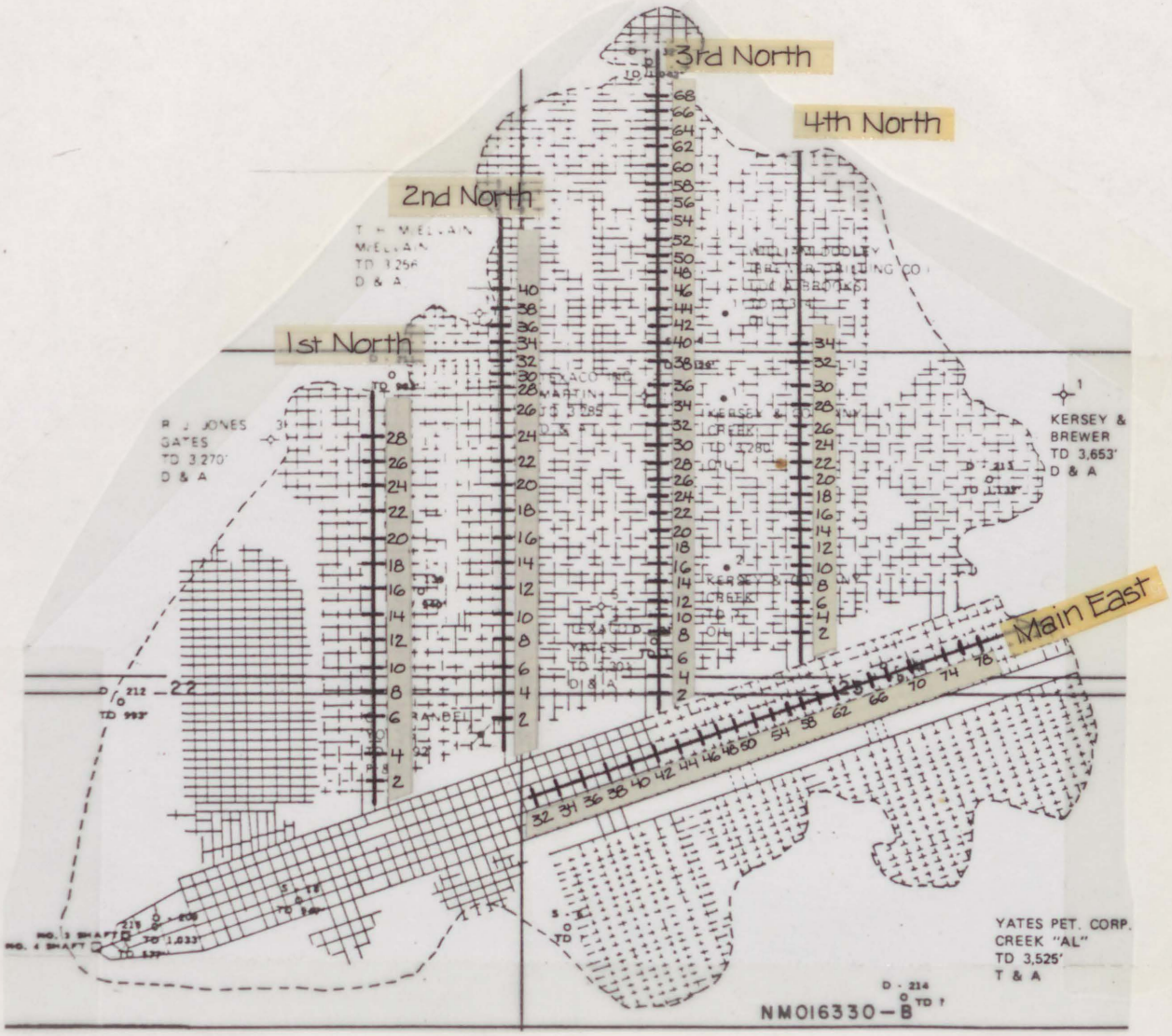
5.4.3 Mine Site Data

Because of the inherent stability of the Wills-Weaver mine workings and the subsequent limited operational problems experienced during mining, comparatively limited site specific rock mechanics data are available. The existing data consist of surface subsidence monitoring and a limited amount of room closure measurements. The details of the subsidence data have been considered in Section 4 and will be only briefly referred to in the present discussion.

Room closure measurements were taken over varying periods of time at a number of locations. The purpose of the room convergence monitoring was primarily for safety control during pillar mining. Some of these locations are indicated on Figure 5-12. Reference heads were located in the roof and floor (suitably recessed for protection purposes), and closures were monitored by taping the distance between the reference pins. The measuring stations were generally located at the centers of room intersections and the closures included components from pillar

LOCATION OF ROOM CLOSURE STATIONS

Figure 5-12



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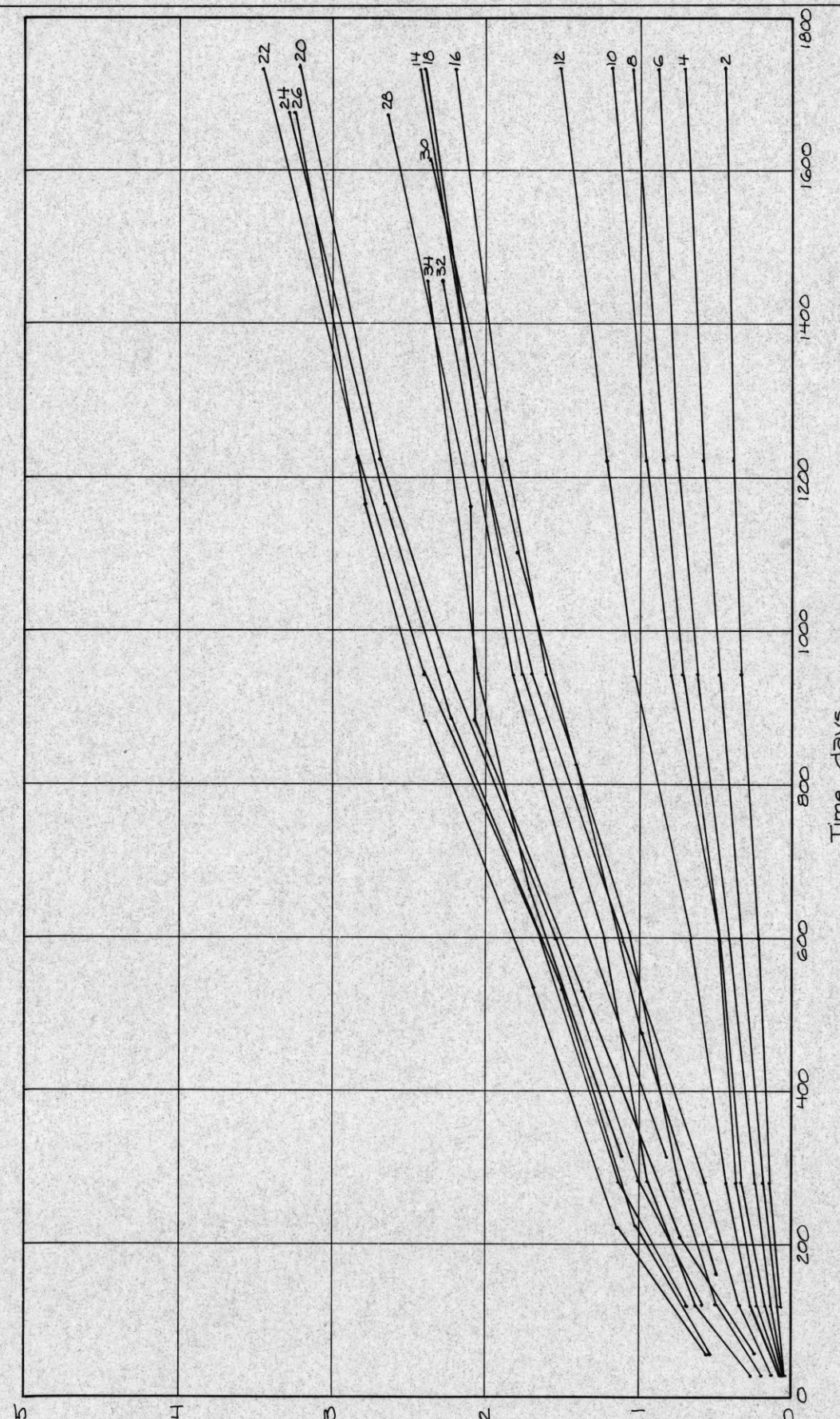
compression, roof sag, and floor heave. There is no way of establishing the relative magnitudes of these individual deformation components.

Two sets of closure stations which were monitored over a period of approximately 5 years were located along the main entries to the 3rd and 4th North panels. The data for the 4th North panel main entry are shown on Figure 5-13. There appears to be a fairly regular pattern of behavior in that the stations (2 through 12) located nearest to the panel entry show progressively greater deformations as the distance from the entry increases. Thereafter, the room closures appear relatively uniform. Total maximum room closures of 3 to 4 inches were noted over the 5 year period. At the end of the 5-year monitoring period (1970), the room closure rate was something less than 0.5 inches per year. It is probable that a relatively small proportion of the above movement was directly attributable to pillar compression. Although extraction ratios in the majority of the 4th North panel were of the order of 75 percent, the main entry was protected by a lower local extraction ratio ($\simeq$ 60 percent), and this undoubtedly contributed to the relatively controlled rate of deformation indicated by the closure monitoring within this area.

Corresponding data for the main entry to the 3rd North panel are shown on Figure 5-14. Once again, the stations nearest the entry to the panel (2 through 4) showed more limited closures. The remaining stations indicated room closures of 2.5 to 6.5 inches over the 5 year period. At the end of the 5 years of monitoring, the rates of room closure were typically less than 0.5 inches per year. The measuring stations were again located within the main entry to the panel. Pillar mining resulted in extraction ratios of 75 percent within a large portion of the

ROOM CONVERGENCE - 4th North Panel

Figure 5-13



First data: 9/1/65
Last data: 5/8/70

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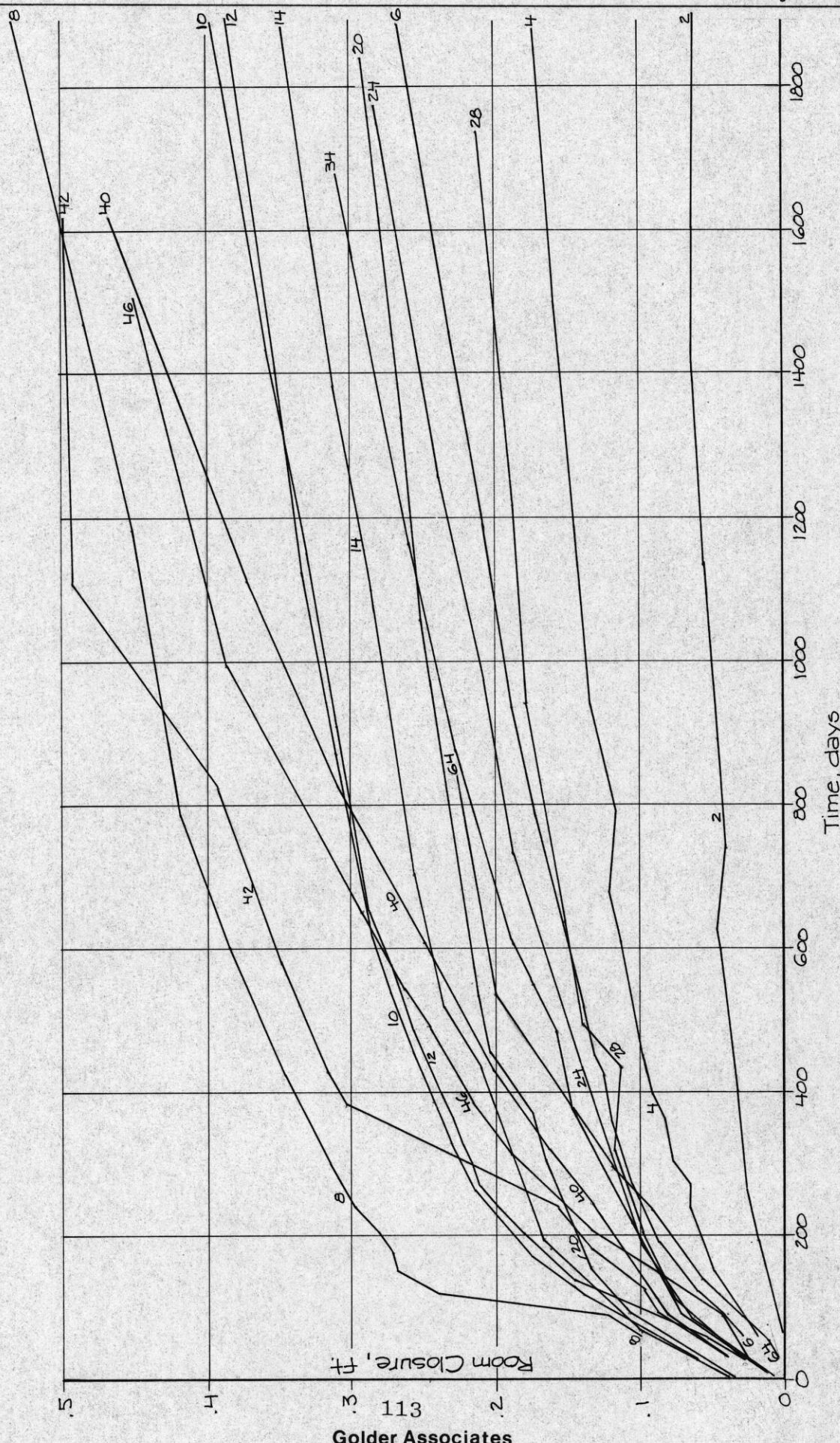
ROOM CONVERGENCE – 3rd North Panel

Figure 5-14



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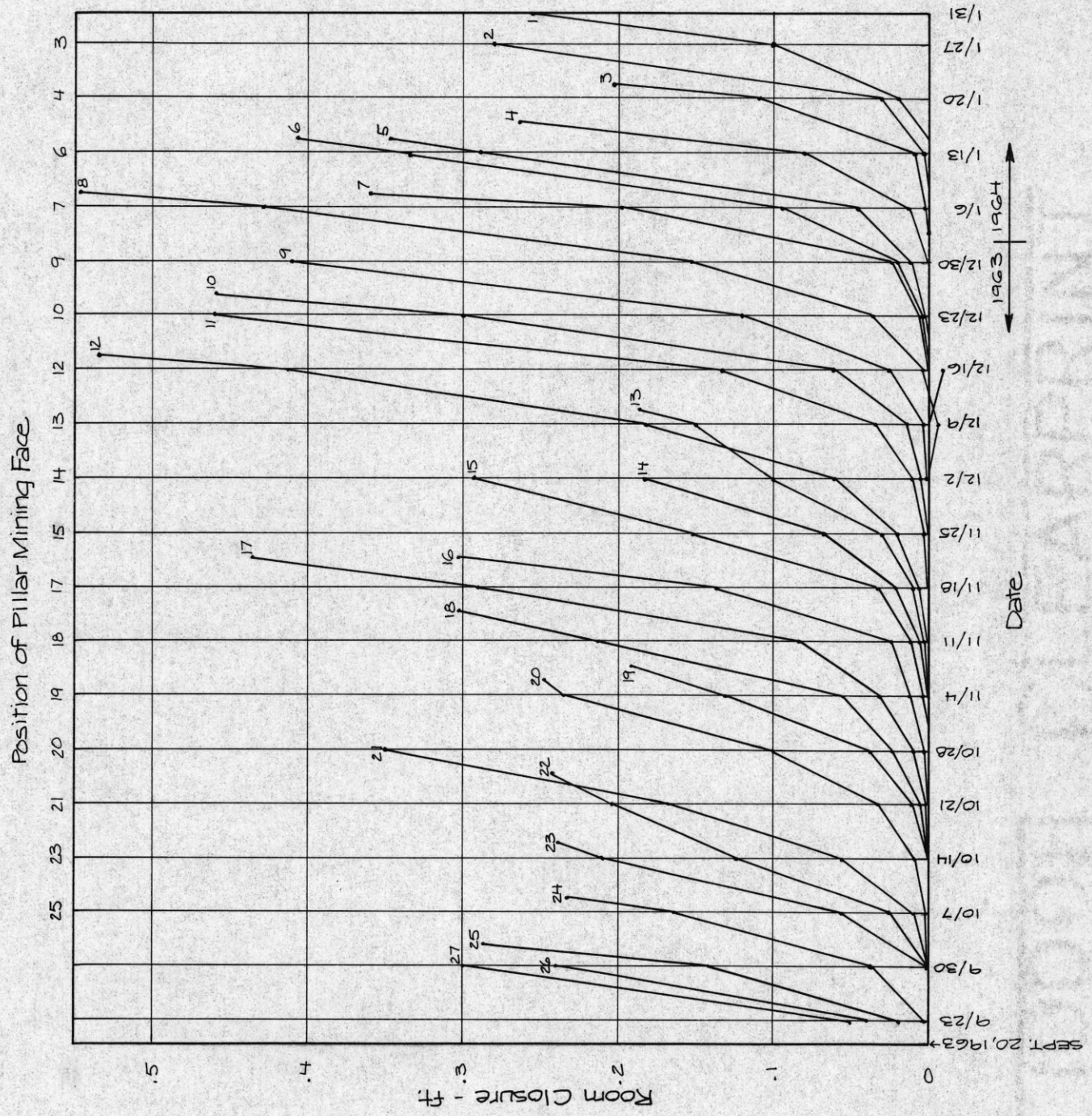
panel, but the main entry was with an area of localized lower extraction (≃ 60 percent), since the pillars on either side of the entry were not mined.

Room convergence data for the 1st and 2nd North panels are shown on Figures 5-15 and 5-16. The convergence stations within the 1st North panel were located along the main entrance to the panel and within the 2nd North panel, the stations were located along the 1st Left entrance. The surface subidence profiles discussed in Section 4 were located directly above these two convergence monitoring profiles. Each figure indicates the time of installation of each station and the period for which each station was monitored prior to abandonment as pillar mining retreated southward along the panels.

Fairly regular behavior is demonstrated by all stations. The initial convergence rates are limited in magnitude, but increase rapidly as the pillar mining front approaches the station. Before abandonment, room closures of up to 6 or 7 inches were monitored and the closure rates were typically 12 to 18 inches per month. These rates are very large in comparison with those observed in the main entries for the 3rd and 4th North panels (residual closure rates of about 0.5 inches per year). However, the extraction ratios within these latter panels were lower than those in the 1st and 2nd North panels, and in addition the main entry pillars of the 3rd and 4th North panels were not mined. The closure rates of 12 to 18 inches per month are very large compared with the rate of surface subsidence monitored directly above the mine level stations (typically 1 inch per month). The large rates of closure evidently included significant components of roof sag and, in particular, floor heave. The contribution of pillar shortening to the total closure rates observed at the centers of the rooms was probably only a small fraction of the total observed rate of convergence.

ROOM CONVERGENCE – 1st North Panel

Figure 5 – 15



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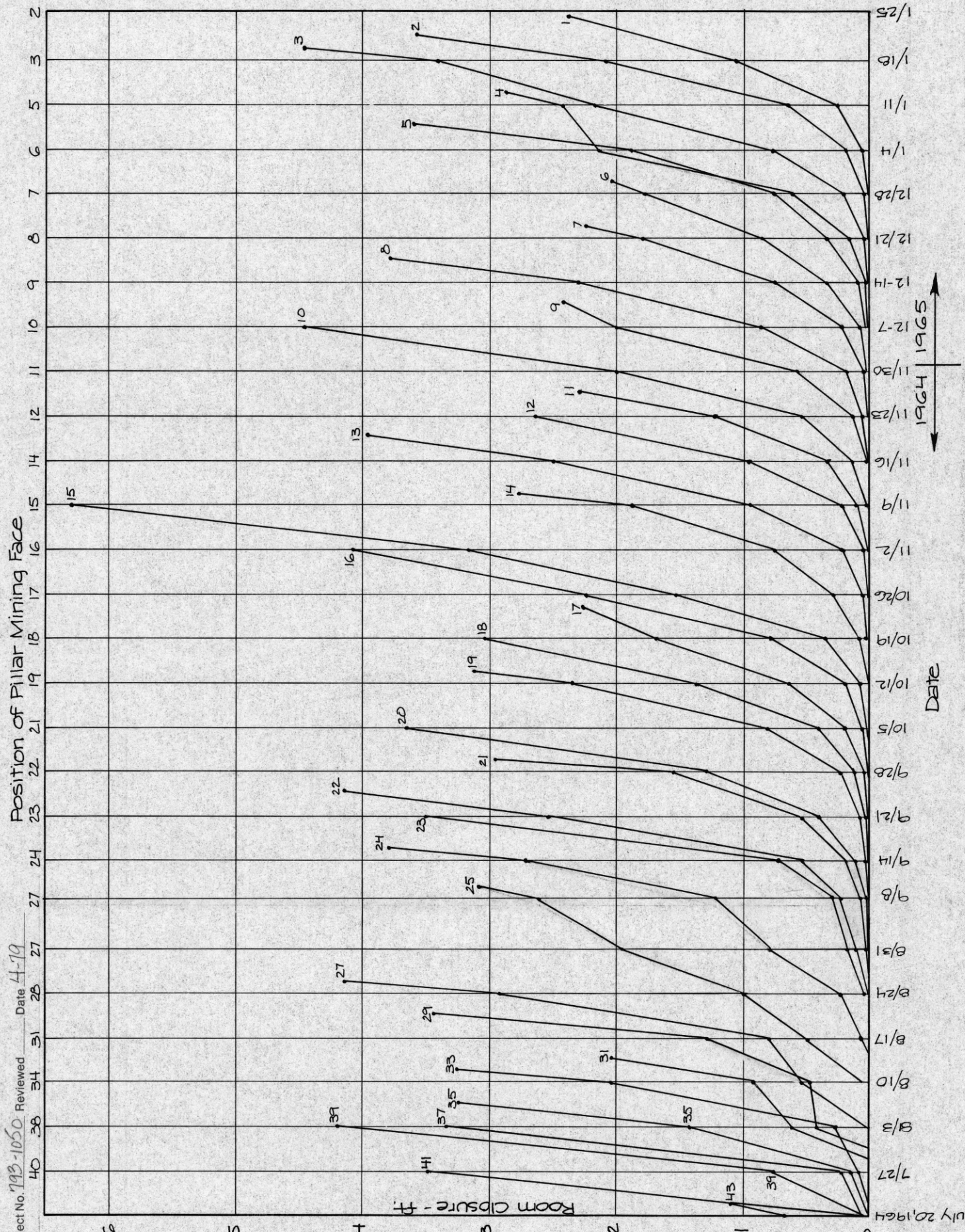
ROOM CONVERGENCE - 2nd North Panel

Figure 5-16



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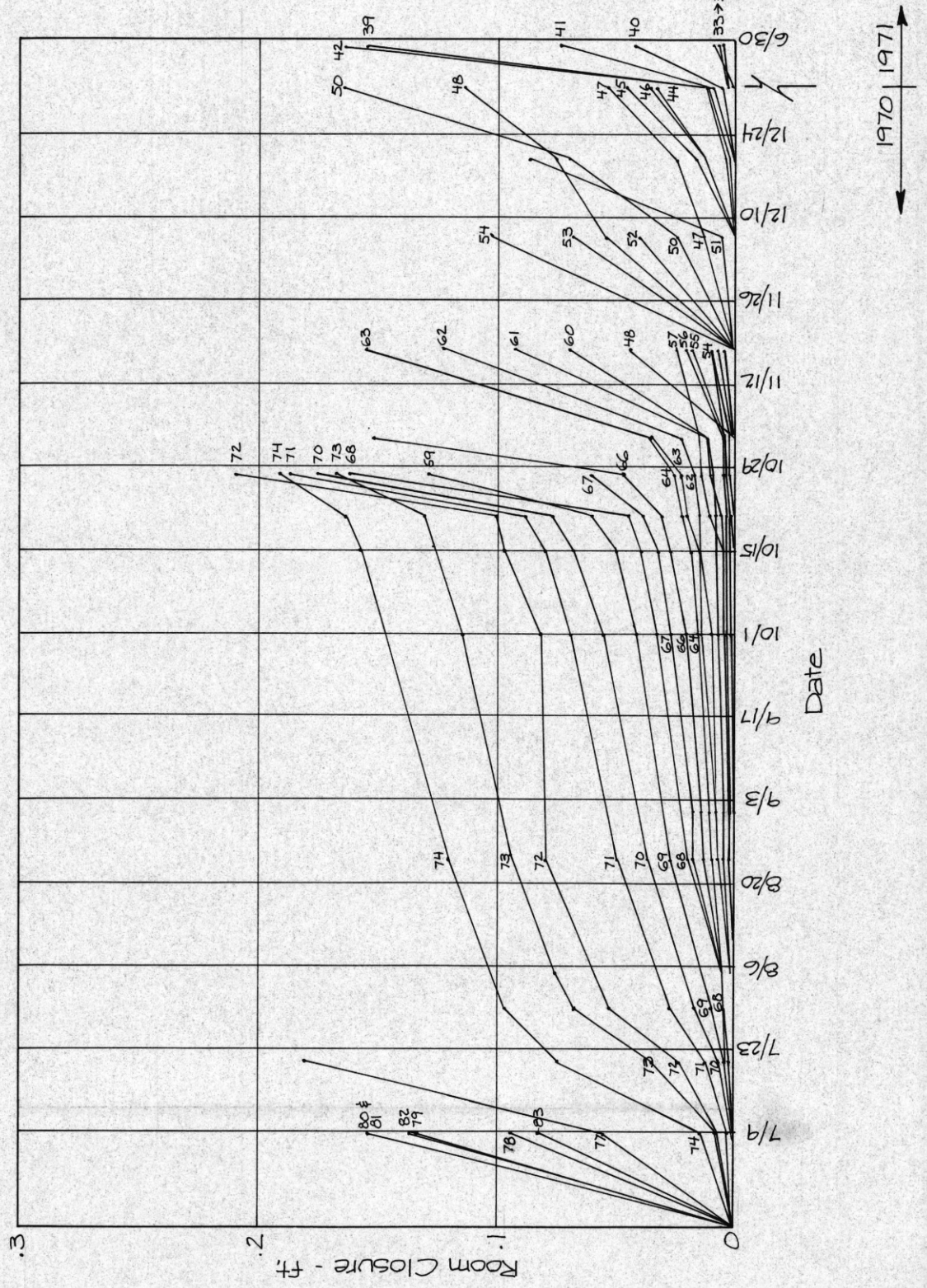
Convergence data for the Main East panel, monitored during pillar mining of the eastern end of the panel, are shown on Figure 5-17. Within the area of the Main East panel, which was subjected to pillar mining, comparatively large convergence rates were again measured. Visual inspection of some of these areas confirmed that floor buckling was the major contributor to closure of the rooms. In the area of the Main East panel where no pillar mining was carried out, the rate of room closure was indicated as being from 0.1 to 0.2 inches per year. As with all the mine convergence data, these closure rates include contributions from pillar shortening, floor heave, and roof sag.

It is of interest to evaluate some of the closure monitoring data, in order to determine our ability to make reliable predictions about the future performance of the mine. The area of particular interest consists of the region of the Main East panel, which has not been pillar mined. In particular, the western extremity of the Main East panel is of concern, since closure of the mining horizon in this area could have impacts on the stability of the No. 3 shaft, as discussed in Section 4. In this regard, the present rate of closure of the pillars within this area is of considerable interest.

The majority of the closure monitoring data cannot readily be employed to determine pillar deformation rates, since the excessive room closures are primarily the result of floor buckling and heave. As discussed in Section 4, the surface subsidence data may be used to obtain a rough estimate of pillar closures, since it is primarily the pillar deformations which are responsible for inducing surface movements. On the basis of this argument, pillar closure rates of the general order of 1 inch per month (Section 4) may be assumed for the 1st and 2nd North panels where high extraction ratios ($\simeq$ 85 percent) were employed.

ROOM CONVERGENCE - Main East Panel

Figure 5-17



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Detailed data on the dimensions of the residual pillars left within the 1st and 2nd North panels following pillar mining are not available. Assuming an average extraction ratio of 85 percent and pillars with approximate horizontal dimensions of 20 feet (i.e., aspect ratio of 5), the average vertical and horizontal pillar stresses within these areas are shown in Table 5-1. Similar information is shown for the Main East panel areas, with an extraction ratio of 60 percent and 60-foot square pillars. These data suggest that deviator stresses within the 1st and 2nd North panel pillars are about 3.5 times those within the region of the Main East panel which has not been pillar mined. The experimental data of Obert (1965) on model New Mexico potash pillars suggest that the rate of deformation is proportional to the 3.3 power of the deviator stress, and this would indicate that the rate of pillar closure within the 1st and 2nd North panel area would be about 60 times that within the Main East pillar region. The differences may be even more significant than this, particularly if the extraction ratios exceed 85 percent within the North panels.

These data would indicate, therefore, that the rate of pillar closure within the Main East panel is of the order of 0.2 inches per year (i.e., approximately 1.5 percent of the rate of 1 inch per month deduced for the 1st and 2nd North panels).

Convergence monitoring with the region of the Main East panel which was not pillar mined, showed closure rates of about

TABLE 5-1
AVERAGE PILLAR STRESS LEVELS WITHIN HIGH AND LOW EXTRACTION RATIO AREAS

Location	Av. Vertical Stress (psi)	Av. Horizontal Stress (psi)
1st and 2nd North Panels	6,700	3,200
Main East Panel	2,500	1,500

0.1 to 0.2 inches per year, and these values are very similar to the above derived value. Since the monitored convergence values also included contributions from roof sag and floor heave, it is likely that the pillar closure rates are somewhat less than these values, and a value of 0.1 inch per year to represent the present rate of pillar closure within the Main East panel is probably a realistic upper bound.

A rate of closure of 0.1 inch per year would imply pillar shortenings of the order of 1 to 2 inches, since the development of the mine in the early 1960's. Room closures within the Main East panel area of up to 10 inches were observed during the inspection of the mine workings (location A, Figure 5-3), but movements of this magnitude are primarily the result of floor heave and roof sag.

5.5 REVIEW OF PRESENT MINE STABILITY

A review of the present stability conditions within the Wills-Weaver mine workings, together with an understanding of the physical processes at present operative within the mine through an evaluation of the appropriate rock mechanics data, provides the information necessary to establish future likely stability conditions within the mine. The above information has been developed by a geotechnical inspection of the currently accessible workings, together with an evaluation of pertinent rock mechanics information. The performance of the mine workings is of interest not only in relation to the stability of the mine openings themselves, but also in relation to subsidence of the overlying strata and the possible damage which such behavior might inflict on the mine shafts.

The following points highlight the major items of interest with regard to the present condition of the Wills-Weaver mine openings.

- Within the areas of the mine where no pillar mining has been carried out (principally the western half of the Main East panel), stability conditions are very good and comparatively little deterioration has occurred since the cessation of mining activity.

- The pillars within these areas show virtually no signs of distress, with only very occasional surface slabbing occurring. The inherent stability of the pillars derives from the modest extraction ratio ($\simeq$ 60 percent) in relation to the mine depth, together with the limited ceiling height (on average about 4 feet).

- Consideration of available rock mechanics data (including convergence monitoring and surface subsidence monitoring at the Wills-Weaver property) suggests that the pillars are shortening at a rate of less than 0.1 inches per year, within the supported area.

- Roof conditions in the accessible area of the mine are generally very good, despite the almost complete absence of roof bolting. One isolated major roof fall (encompassing an entire intersection and consisting of a one foot thick slab) has occurred, but elsewhere the roof shows no signs of distress.

- Floor conditions within the 60 percent extraction ratio area were good, although isolated examples of floor heave were evident. In general, the mine workings are

susceptible to problems of floor heave, and the performance of the mine pillars is critical in determining the severity of any floor heave problems. Pillars which are heavily loaded and compressing at a relatively fast rate will also expand laterally at a comparatively fast rate. This lateral expansion provides a horizontal driving force to the ends of the floor beds, and floor buckling results. The limited vertical closure of pillars within this area has ensured no serious floor problems to date. Notwithstanding this, room closures of up to 10 inches were noted in some of the production rooms within the Main East panel (representing an average rate of closure of approximately 0.5 inch per year). A major contributor towards this closure is considered to be the floor heave component.

- Within the areas which have suffered pillar mining and very high extractions (≃ 85 percent), room closure rates have been very substantial. Deduced pillar closure rates would suggest that the majority of these areas have largely closed by this time (typical minimum rates of closure of the pillars of one inch per month are considered reasonable). Closure of the rooms themselves proceeded at a very much faster rate of course, principally as a result of excessive floor heave which occurred in conjunction with the relatively large rates of pillar compression. Areas located on the extremities of the pillar mined regions, which could be observed, demonstrated floor beds which had buckled sufficiently to make contact with the roof.

- Observations of room closure and surface subsidence within the 1st and 2nd North panels, which were pillar

mined to an extraction ratio of in excess of 85 percent, indicate that as a result of roof sag and principally floor heave, the centers of the rooms closed approximately one order of magnitude faster than the pillars compressed (e.g., deduced pillar closure rate of 1 inch per month versus observed room closure rates in excess of 1 foot per month). Within the supported areas which have not been pillar mined, observed room closure rates of up to 0.5 inch per year have occurred (e.g., convergence monitoring of the supported main entries to the 3rd and 4th North panels, observations of room closures within the Main East panel). By analogy, a pillar closure rate of 0.05 inch per year might be deduced if the same relations hold within the pillar supported region. This value compares well with the figure of less than 0.1 inch per year, derived from a consideration of available rock mechanics data.

6.0 PRESENT CONDITIONS OF MINE SHAFTS

6.1 INTRODUCTION

There are two mine shafts at the Wills-Weaver mine, located in the extreme southwest area of the property and protected from the workings by a region of limited extraction. This region extends approximately 550 feet from the No. 3 shaft and 850 feet from the No. 4 shaft. Since closure of the mine horizon to within about 1,000 feet of the shafts could inflict damage on the shaft linings, it is necessary to understand the engineering performance of the mine pillars within the area of the workings that could potentially influence the shafts. To date, no damage to the shafts as a result of subsidence of the strata overlying the mine has occurred. The nature of the movements at present occurring within the mine and the overlying strata has been considered in Sections 4 and 5, and estimates of future behavior are given in Section 7. Potential shaft deterioration which can be directly attributed to strata disturbance is therefore not considered further in this section.

Shaft deterioration may also occur as a result of factors which can be regarded as purely local to the shaft, e.g., shaft buckling caused by loss of bond to the rock as a result of dissolution, shear compressive failure of the concrete lining caused by excessive earth pressure, deterioration of the concrete lining and shaft fittings by aging and corrosion, etc. An evaluation of these factors is given in this section. The present conditions of the Wills-Weaver shafts have been evaluated by three specific approaches.

- A general visual inspection of the shafts to evaluate the appearance of the lining and the condition of the

guides, timbers, pump stations, mine stations, gathering rings, etc. In addition, a review of shaft sinking records, grouting records, etc., was undertaken in order to evaluate the construction, reinforcing, and sealing details.

- A quantitative evaluation of the conditions of the concrete linings was undertaken by running Schmidt hammer profiles. This test provides a rapid, nondestructive in situ procedure for estimating the concrete compressive strength.

- Radar profiles were taken along the length of the No. 3 shaft in order to determine the condition of the lining-rock interface, i.e., whether any voids are present behind the lining. Problems with mine shafts (located in evaporites) as a result of dissolution behind the lining have led to the loss of many shafts.

The present conditions of the shafts at the Wills-Weaver mine provides important data required to evaluate the expected lives of these structures. This will determine, in part, the optimum shaft abandonment procedure.

6.2 GENERAL DESCRIPTION OF SHAFTS

6.2.1 Number 3 Shaft

The Wills-Weaver No. 3 shaft is a 14-foot diameter, concrete lined shaft which extends from surface to the sylvite level at a depth of 990 feet. Stations have been cut at the 661.5-foot and 990-foot levels. The concrete lining is continuous from the collar to a depth of 969 feet, having been poured in 20 and 30 foot

lifts. The actual shaft bottom is at a depth of 1,082 feet to allow for skip loading and spill pocket facilities.

The shaft is divided down the middle by a single 10WF 39 bunton which is bolted to the concrete by clip angles. The clip angles are bolted to the bunton as shown on Plate 9. The buntons are on 5-foot centers down the shaft. The ladder road location is shown on Figure 6-1, and ladderway landings are on 20-foot centers. Utility brackets consist of "Z" section steel bolted to the shaft walls on 15-foot vertical centers. The layout of pipes and the main power cable is detailed on Figure 6-1.

The conveyances in the No. 3 shaft consist of two single deck cages over two bottom dump skips.

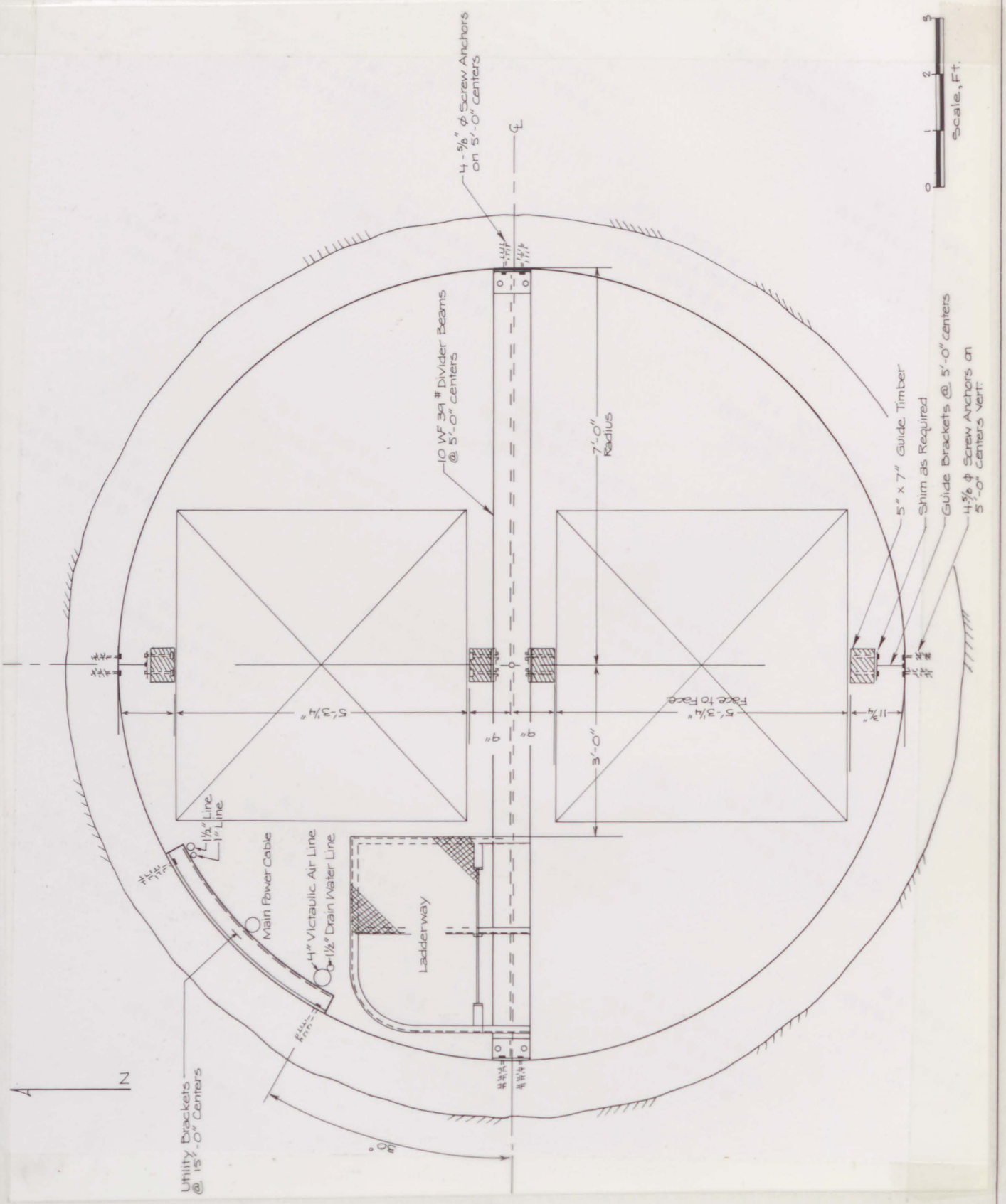
Concrete in the shaft varies from a thickness of 9 inches to a design wall thickness of 24 inches. As can be seen on Figure 6-2, the actual average wall thickness is somewhat greater due to the need to meet minimum thickness standards. The mean shaft wall thickness for each concrete pour was calculated from the concrete quantities given in the shaft sinking log. Various areas in the shaft are reinforced with steel rebar. These reinforced areas are in the collar area and in the aquifer zones. The location and specification of the reinforcing are detailed on Figure 6-2.

One water ring was installed at a depth of 527 feet. Water stops were installed at points designated on Figure 6-2.

The grouting program during shaft sinking consisted of a cement grout curtain into the Magenta dolomite from a shaft depth of 365 feet and from the 454-foot depth into the Culebra dolomite.

SHAFT No. 3
GENERAL ARRANGEMENT

Figure 6-1



Project No. 78-10-36 Reviewed RJB Date 11-79

Grouting in the Magenta dolomite was a two-stage procedure. In the first stage, holes were drilled at an angle of 63 degrees from horizontal to a depth of 22 feet, then drilled to 30 feet. This resulted in a ring pattern having a diameter of 41.4 feet at a depth of 392 feet. Grout hole spacing at hole bottom averaged 5 to 6 feet. A total of 716 feet of grout holes was drilled, and total grout take was 3,270 gallons containing 222 sacks of cement. Grout injection pressures varied from 510 to 760 psi. The only holes that made water at depth 385 feet were holes No. 7 and No. 8 (Figure 6-3). Grout taken in hole No. 8 was 80 sacks of cement at 660 psi.

The main water inflow occurred at depth 387.5 feet, which is 2.5 feet above the bottom of the Magenta dolomite. Maximum water inflow into a 2.5 inch grout hole was 7.5 gpm.

The second stage of grouting consisted of 30-foot holes drilled at an angle of -70 degrees, which resulted in a grout ring having a diameter of 34.5 feet at a depth of 393 feet. This resulted in a total grout take of only 250 gallons containing 5 sacks of cement at a pressure of 760 psi. Subsequent test holes drilled within the shaft were dry.

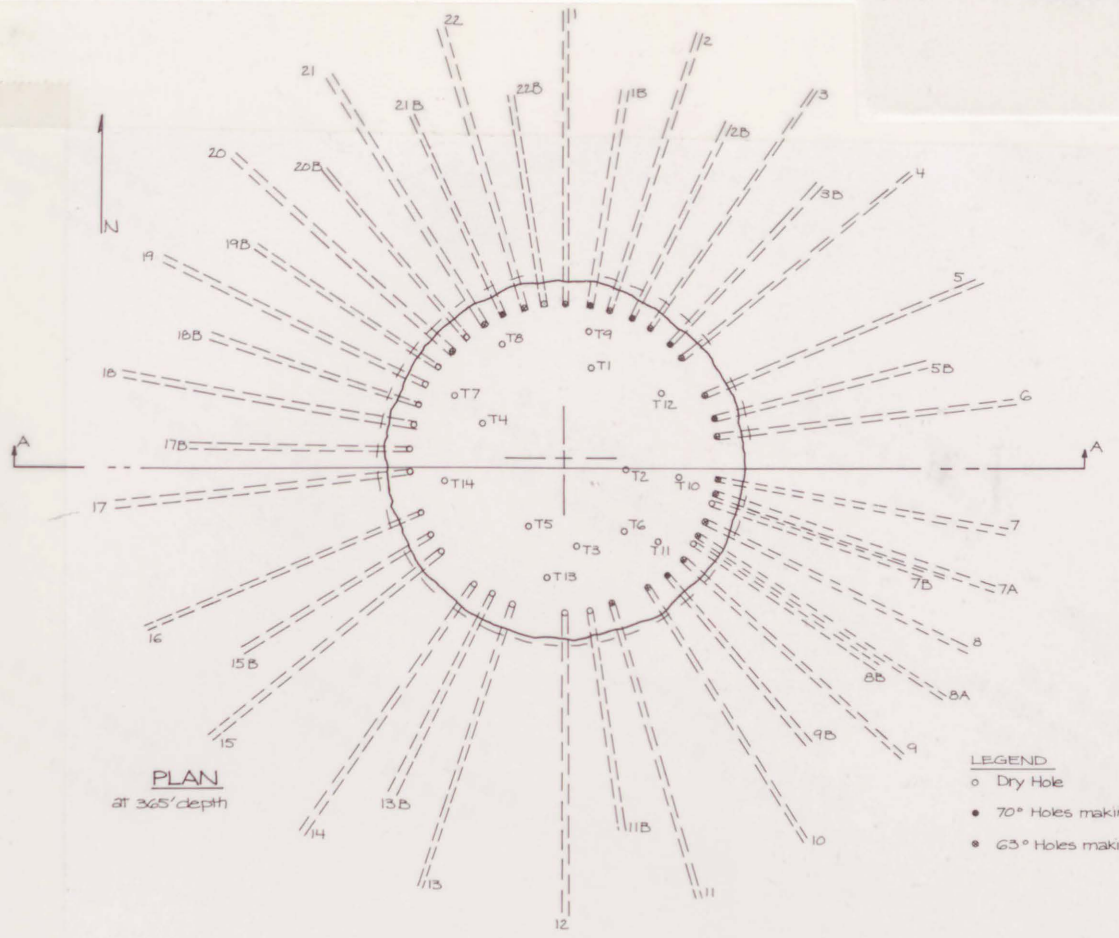
Some interesting features of the grouting program in the Magenta dolomite were as follows:

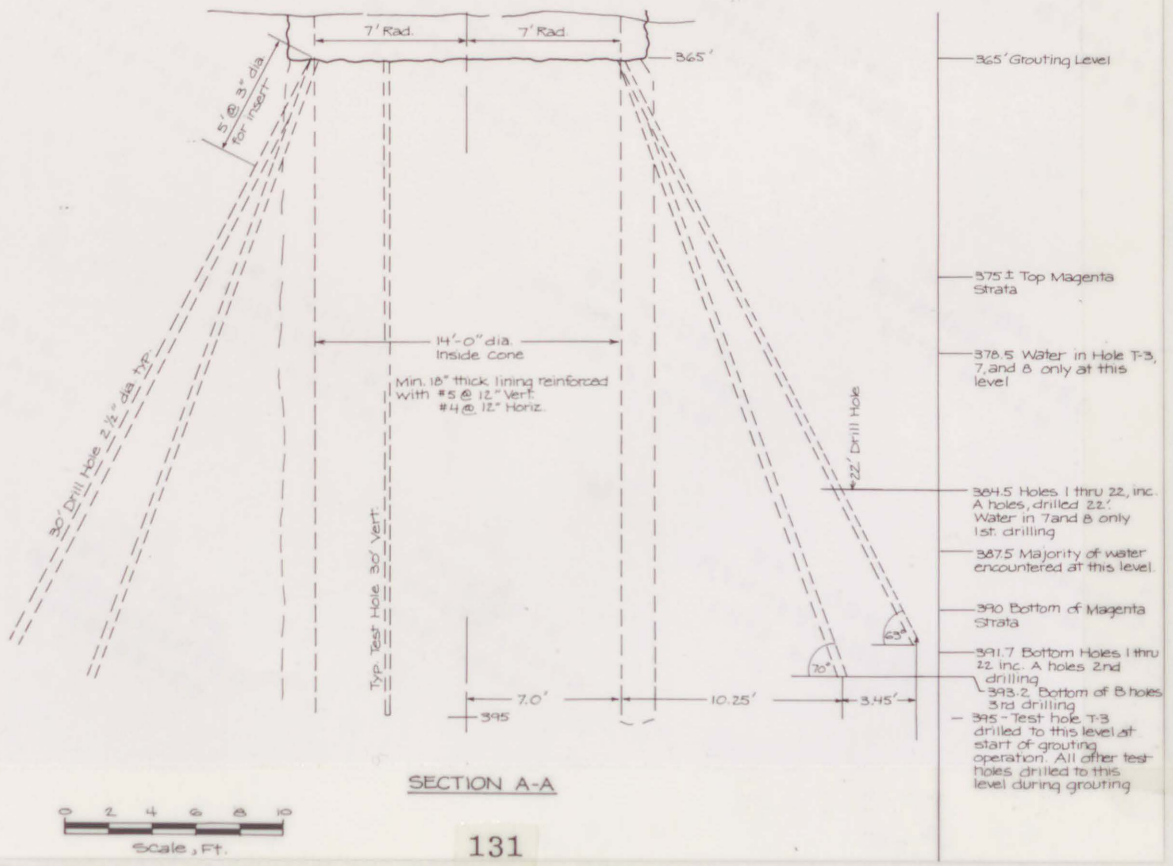
- Holes making water were only in the northwest, northeast, and southeast quadrants of the shaft from azimuth 312 degrees to azimuth 162 degrees.

- Grout injected into the formation at depth 392 feet appeared in the wall at depth 358 feet.

SHAFT No. 3
GROUTING AT 365 LEVEL

Figure 6-3





SECTION A-A

Scale, Ft. 0 2 4 6 8 10

Project No. 783-1050 Reviewed RB Date 11-77

Grouting in the Culebra dolomite was done in several phases. In the first phase, holes were drilled to a depth of 10 to 14 feet at an angle of -70 degrees, forming a grout ring at depth 465 feet with a diameter of 25 feet. Water pressure encountered was 68 psi, or about 157 feet of head.

Out of 25 holes drilled, only 15 holes made water, although one hole (Hole No. 22, Figure 6-4) was making water at the rate of 132 gpm. Four holes were injected with grout at this depth, with the wet hole taking 10,255 gallons of grout with 599 sacks of cement. Ninety-five cubic feet of sand were added to the grout in this hole. Grouting of this one hole resulted in the blocking of nearly all of the remaining holes in the 465 Grout Ring.

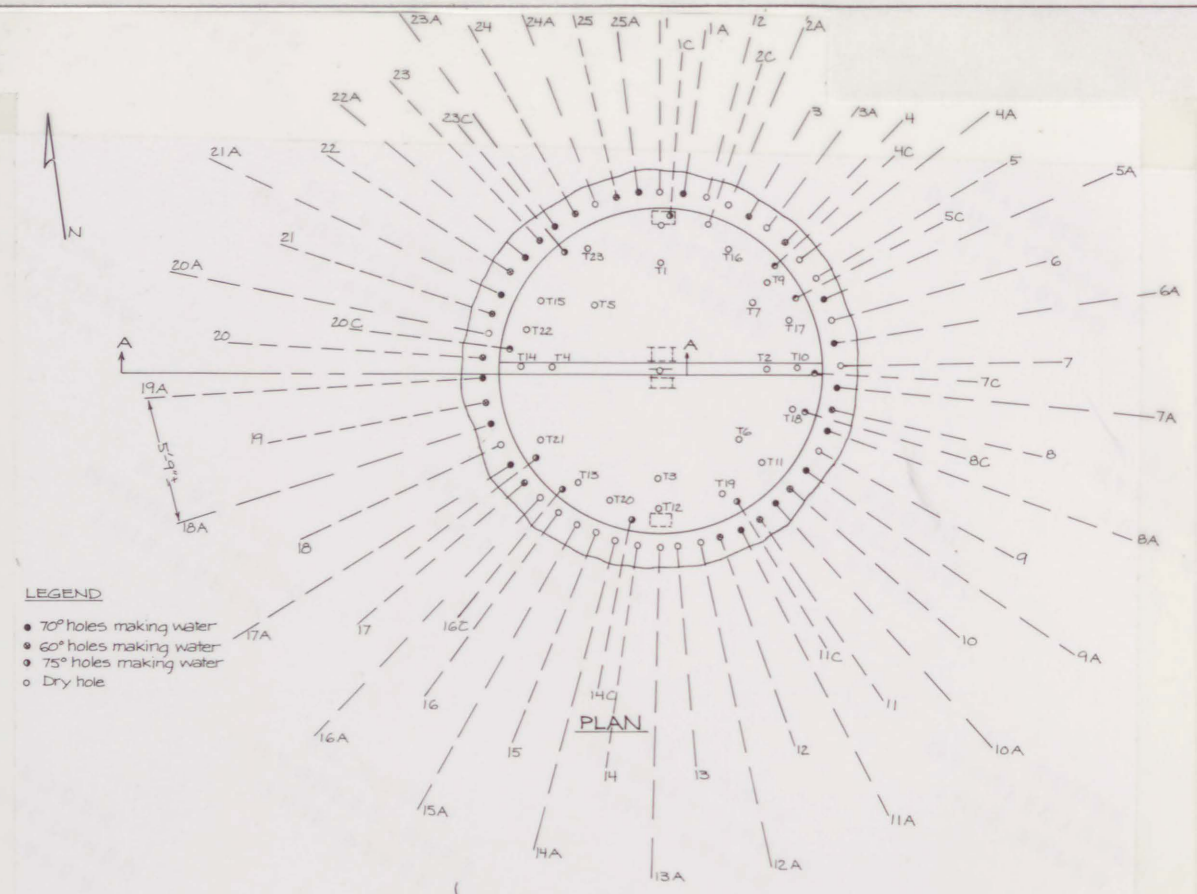
Holes 1 through 25 (Figure 6-4) were then deepened to 26-30 feet, and an additional 1,460 gallons of grout containing 33 sacks of cement were injected at 500 to 600 psi.

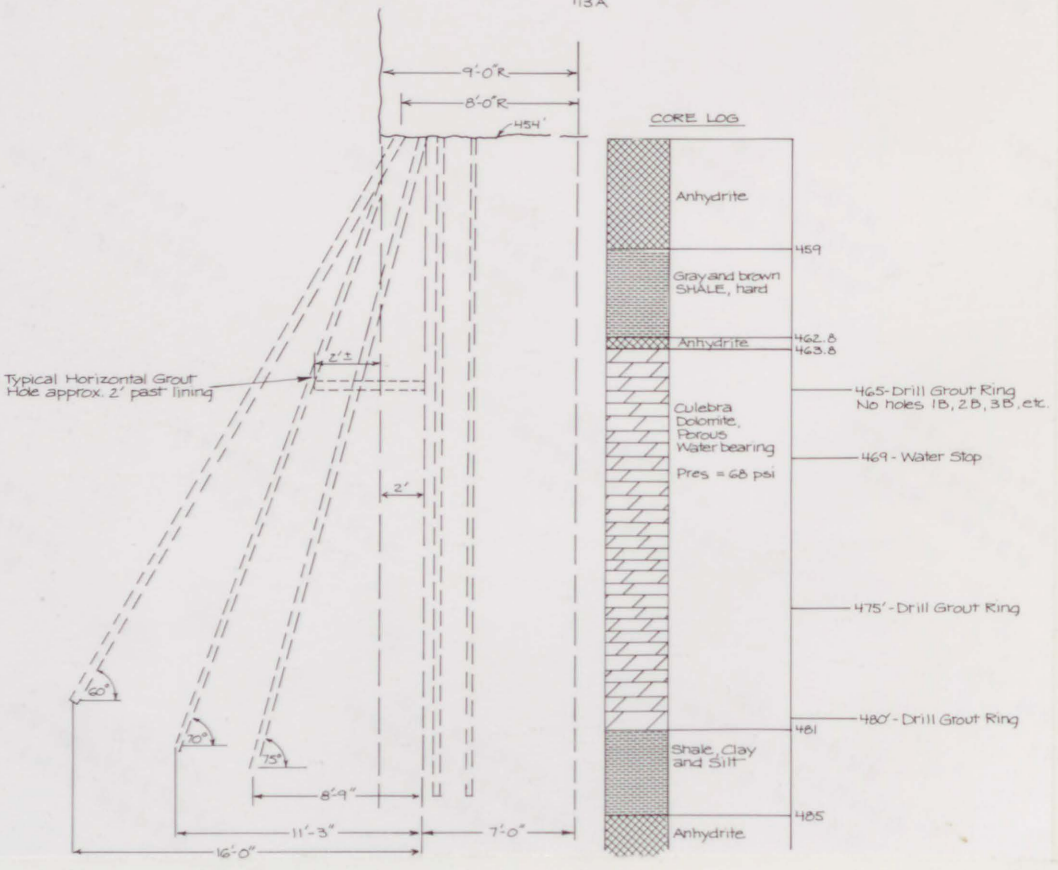
Holes 1A through 25A were then drilled to 30 feet at -60 degrees in order to inject the 475 Grout Ring (6 feet above the bottom of the Culebra dolomite). This ring had a diameter of 46 feet. The grout take was relatively minor compared with the first injection.

Holes 1, 3, 9, 16, 17, 19, 20, and 23 were then redrilled to a depth of 30 feet and chemical grout was tried. The chemical grout consisted of 725 gallons of sodium silicate and 465 gallons of calcium chloride. The chemical grout was apparently ineffective, as the water flow from the test holes was 9.1 gpm before injection and 9.5 gpm after injection. Chemical grout was injected at 600 psi.

SHAFT No. 3
GROUTING AT 454 LEVEL

Figure 6-4





Project No. 783-1636 Reviewed RJB Date 11-78

Cement grout was again injected in the same holes in the 1 to 25 series and the 1A to 25A series. The shaft log carries a notation that all holes were sealed off. Subsequent test hole readings show an inflow of 10.9 gpm.

Some of the above holes were again redrilled and another minor amount of grout was injected (2,000 gallons containing 26.5 sacks of cement). Pressures were from 450 to 700 psi.

A final ring containing 16 No. 30 foot holes at -75 degrees was drilled to the 480 Grout Ring. This ring had a diameter of 29 feet and penetrated nearly 2 feet into the shale, clay, and silt bed underlying the Culebra dolomite. This "C" ring took only 925 gallons of grout containing 18.5 sacks of cement. The injection pressure was 600 to 700 psi. At this point, a total of 801 sacks of cement had been injected into the Culebra dolomite. Of this, 599 sacks had been injected into one 10-foot hole.

Shaft sinking was resumed using paving breakers. After the shaft lining reached a depth of 60 feet below the Culebra dolomite, short holes were drilled horizontally about 2 feet into the rock and 70 sacks of cement were pumped behind the liner to seal water coming through relief holes and concrete joints.

Subsequently, Duval Corporation had a shaft crew put in 8 or 10 shifts in this shaft in an attempt to grout through the liner.

A storage pocket has been excavated below the sylvite level floor elevation for storage of crushed potash ore and salt. The general dimensions of this pocket are 25x25x31 feet deep. The wall of the storage pocket facing the shaft is of steel plate; the other three walls are natural salt. The floor of the pocket is concrete, 3 feet 6 inches thick.

The bin is divided into two compartments: one is a 125T salt pocket and the other is a 300 ton ore pocket.

The measuring pockets are of steel plate construction and have arc gates. The measuring pocket layout is a typical volumetric system. Figure 6-5 shows details of the skip loading system.

The tailshaft cleaning system consists of a screw conveyor in the bottom of the spill pockets feeding a bucket elevator which lifts the spill material to the elevation of the measuring pockets. The measuring pocket floor is accessed by a man lift from the sylvite level as well as by a ladder road.

Shaft guides are 5x7 inch straight-grained, clear Douglas fir in 20-foot sections. The inside guides are bolted to the buntons, and the exterior guides are bolted to a bracket which consists of a 5 inch long piece of 6 inch WF. This bracket is in turn bolted to the shaft wall through the flanges of the bracket. Shimming is done on the exterior guide brackets.

6.2.2 Number 4 Shaft

Wills-Weaver No. 4 is a 9 foot diameter shaft which has a total depth of 1,011 feet. A pump station has been cut at the 652.5-foot level and the sylvite sill is at 990 feet. The shaft is concreted from the collar to the 964-foot level with a water ring at 517 feet. Concrete lining is continued from the 990 sill to the bottom of the tailshaft.

Design wall thickness is 9 inches for most of the shaft depth, but design thicknesses of 12 and 24 inches are used in the aquifer areas. Actual average wall thicknesses are detailed on

Figure 6-6. Rebar was used in the lining near the collar and in the aquifer area, as shown on Figure 6-6.

The shaft has one small cage, offset from center (shown on Figure 6-7), and utility lines. During production the shaft was only occasionally used for hoisting men and supplies, due to the antiquated hoist and small cage that were used to equip the shaft.

Wooden guides were bolted directly to the shaft wall. Guides are 5x7 inches and are 7 feet face-to-face.

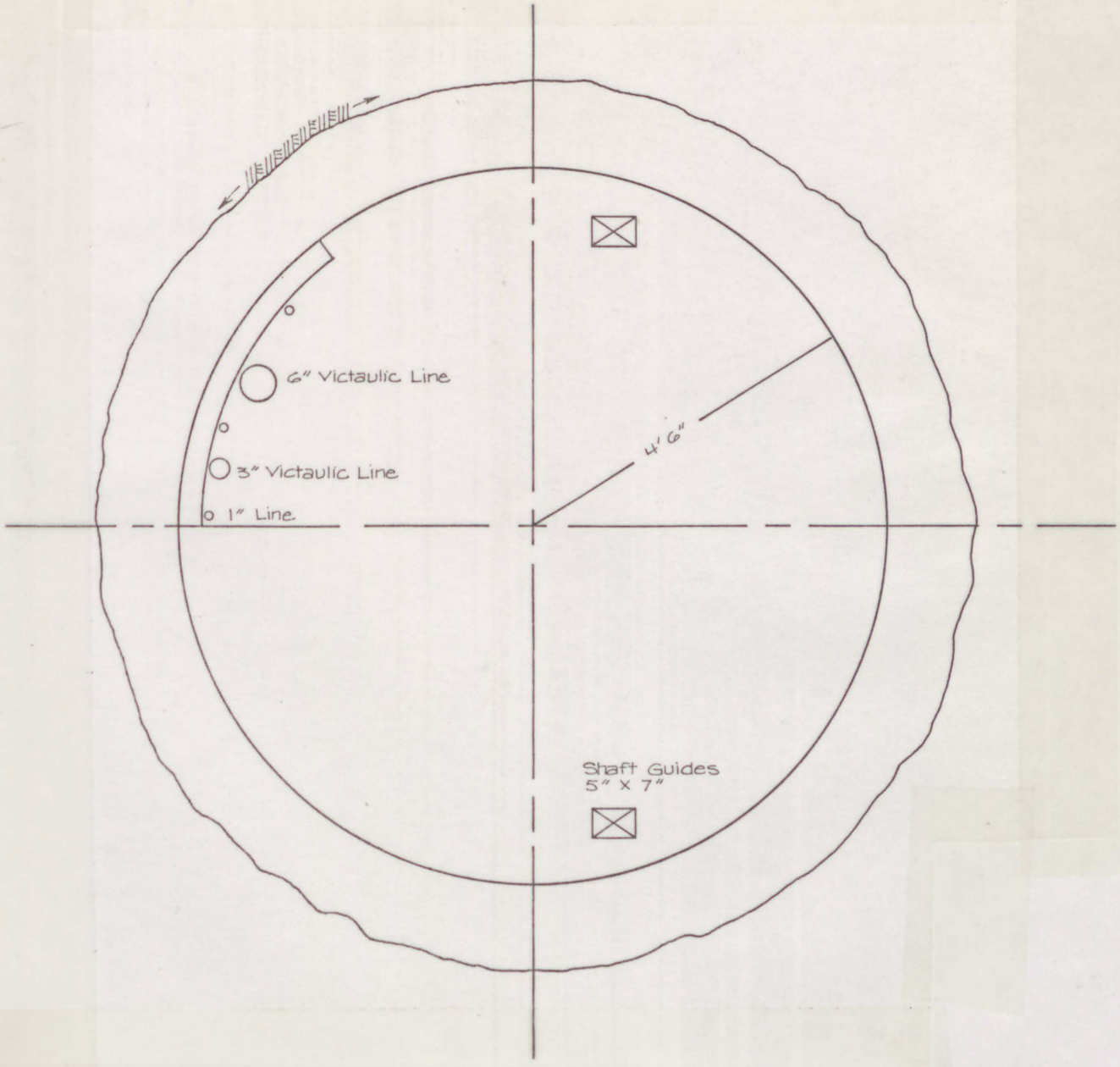
Four test holes varying from 32 to approximately 49 foot depth were drilled from the shaft bottom at the 440-foot level. These four holes flowed a total of 50.5 gpm from the Culebra dolomite. Grout was injected into the Culebra from this point, and the grout take was 565 sacks. The grouting record indicates that grout ran from the lining 50 feet above shaft bottom. Pressure was 700 psi and water pressure was 68 psi.

The shaft was then deepened to 445 feet, and a ring of 16 holes was drilled to depths of 26 and 30 feet at -60 degrees. This gave a grout ring 39 feet in diameter which bottomed about 2 feet above the bottom of the Culebra dolomite. The grout take was 331 sacks of cement, with one hole accounting for 276 sacks. Aggregate water inflow from the grout holes was 118.4 gpm.

Fifteen holes were then drilled at an angle of -75 degrees to depths of 23 to 30 feet. This provided a grout ring 24.5 feet in diameter, and the 30 foot holes bottomed 1 foot below the bottom of the Culebra dolomite. Grout take for this phase was only 28.5 sacks and aggregate water inflow was 59 gpm.

SHAFT No. 4
GENERAL ARRANGEMENT

Figure 6-7



Project No. 783-1052 Reviewed RJB Date 11-78

A "B" ring of grout holes was then drilled at -75 degrees and grouted in two stages. These holes were all drilled or re-drilled to 26 feet, which would be about 3 feet above the bottom of the Culebra dolomite. Total grout take was 58 sacks for these two phases, and all holes were dry at completion of the final phase.

Shaft sinking resumed using light blasting with modeling being done at the shaft walls. After the lining was 60 feet below the Culebra dolomite, 75 sacks of cement were injected behind the liner. The result of this work was a dry shaft.

6.3 CONDITION OF SHAFTS

6.3.1 Number 3 Shaft

During a shaft inspection made by Golder Associates on February 5, 1979, the following conditions were observed in the Wills-Weaver No. 3 Shaft.

Steel. The shaft buntons were observed to have a light corrosion on all surfaces, but particularly on horizontal surfaces facing upward. These surfaces have collected a coating of potash ore and salt which have combined with the moisture in the shaft to cause some corrosion. At present, the buntons are quite solid and appear to have an adequate thickness of solid steel to provide nearly the strength of new sets.

The ladder road steel is sound at present but, due to the fact that it is of lighter gauge material, corrosion could weaken this metal in the near future.

Probable weak points in the structural steel system are in the bolts which fasten the steel to the concrete. These bolts

are subject to corrosion from saline water running down the shaft walls (Plate 9) and would be subject to high shearing and tensile forces during ore hoisting. As these bolts could not be examined, it is not possible to estimate their strength.

Utilities. The Zee brackets are of heavy gauge steel and therefore still have much of their original strength. Pipes in the wet areas of the shaft are badly corroded and will prove to be troublesome in the immediate future. The pipe carrying the shaft seepage water has corroded through at several points and has been replaced by rubber hose in some sections. The bolts which fasten the Zee brackets to the shaft wall are of unknown quality, but are not subject to the dynamic loadings that the bunton bolts are subject to. These fastenings should not be ignored, because of the possible disastrous effects that would result from the failure of two or three of these fastenings. In particular, the integrity of the fastenings should be examined before any additional loading is placed on the utility bracket system (i.e., startup of pumping, use of any of the line for grout or concrete slickline).

Guides. Wooden shaft guides are in excellent condition, having been treated with oil and paraffin. There is no evidence of unusual wear or misalignment in the shaft.

Concrete. From the inside surface of the shaft the concrete appears to be generally in good condition. The concrete is sound, not spongy or spalling. For the most part, construction joints between pours are tight and honeycombing is rare. There was no point in the concrete that was observed to be leaking as a

result of a poor construction or honeycombing. One unusual feature of the concrete lining was the lack of latence on the concrete surface. Coarse aggregate is exposed at the surface of the concrete.

There were some circumferential cracks observed in the area of the aquifers. Most of these cracks were weeping small amounts of water. In addition, other cracks had been scarfed out and water collection systems installed. The method of treating the cracks is detailed on Figure 6-8. Water is manifolded into the water collection system and piped into a worked-out area in the mine.

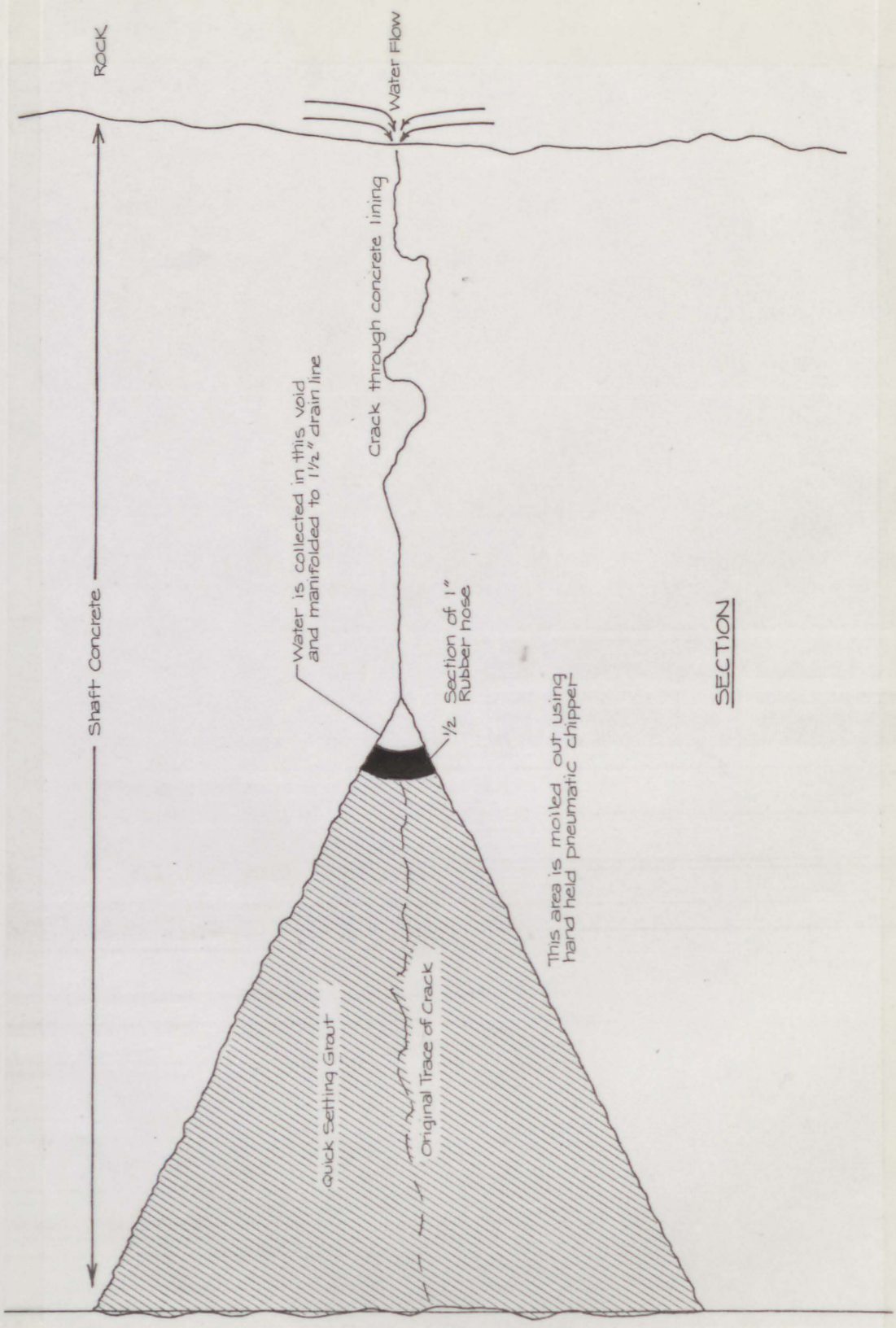
The water ring at the 527-foot level is in disrepair and appears to have never functioned as a water ring. No water is leaking from the bottom of the concrete liner.

Skip Loading Station. Due to the presence of large amounts of fine potash ore and salt on the surfaces in this area, corrosion is more pervasive although most of the chutes, gates, and bins are now in usable condition. Most of the plate in this area is of fairly light gauge, so it can be expected that some of the plate will have to be replaced before startup could be effected. It appeared that bin hinge bearings were frozen by corrosion and would have to be replaced.

Tailshaft. The bottom of the shaft contains 30 feet of water. The bucket elevator is partially in the water and would require extensive repair or complete replacement. The condition of the spill pocket is unknown due to flooding of the shaft bottom. The manlift is in very poor condition and would require replacement or major repair before startup.

SHAFT No. 3
METHOD USED TO COLLECT WATER FROM CRACKS IN SHAFT LINER

Figure 6-8



Project No. 783-1058 Reviewed RJB Date 11-77

6.3.2 Number 4 Shaft

The No. 4 shaft was inspected by Golder Associates on July 2, 1979, and the following conditions were observed:

Utilities and Guides. Shaft utilities and guides are generally in good condition, considering that no maintenance has been carried out since the shutdown of the mine. There is some corrosion of the brackets and pipes in the vicinity of 475 to 510 feet beneath the shaft collar. This region corresponds to an area of significant salt deposits on the shaft wall as a result of small but continuing brine inflows from the Culebra dolomite aquifer.

In addition to a 1-inch diameter pipe and a 2.5-inch diameter pipe running down the length of the shaft, a 2-inch drain line runs from the water ring (520 feet beneath the shaft collar) to the pump station (652 feet beneath the shaft collar). There is no flow from the 2-inch drain line into a 55-gallon drum located in the pump station. This drum is two-thirds filled with salt and brine.

Some utility brackets are missing, e.g., at approximate depth 400 feet. At an approximate depth of 435 feet, a utility bracket has pulled loose from the wall at a consstruction joint.

In general, the shaft guides appear to be quite solid.

Concrete. The surface of the concrete liner in the No. 4 shaft is somewhat different to that in the No. 3 shaft. A large portion of the surface of the No. 3 shaft liner is characterized by the lack of latence and the exposure of aggregate. This is not a feature of the No. 4 shaft liner.

From the surface to a depth of 445 feet, the No. 4 shaft is dry and the concrete appears to be in sound condition with some minor cracking noted. Cracks in the liner exist at depths 29 feet, 140 feet, 312 feet, 340 feet, 352 feet, 375 feet, and 428 feet. These cracks are typically 1/16-inch in width and in general appear to be more well developed in the east wall of the shaft. At depth 433 feet, some minor salt buildup exists as a result of brine ingress through a poor construction joint.

Below depth 445 feet, a general buildup of salt occurs on the wall, and this persists with varying intensitites to the pump station (652-foot depth). Below the pump station there are no significant salt deposits on the shaft wall, although small salt crustations occur very occasionally.

Heaviest salt deposts occur at depth 475 to 480 feet, and in this vicinity there is an average salt thickness of about 1 inch over the entire surface area. Heaviest deposits occur on the east wall, and locally the thickness is up to 6 inches.

Since the No. 4 hoist had not been utilized for approximately 8 years prior to the inspection, these substantial thicknesses of deposited salt interferred with the passage of the cage at locations where minimum clearances between the cage and the wall existed. Passage of the cage removed the offending salt from the wall and, in addition, approximately 1/4 to 1/2-inch of concrete was removed along with the salt. Close inspection of these freshly exposed areas of concrete in the region of heavy surface salt (approximate depths of 475 to 505 feet) indicated poor quality material that is apparently degrading under the action of the shaft water (Plate 10). The concrete appears to be losing its cohesion and degrading to a sand-aggregate assemblage. It was possible to remove pieces of aggregate from the exposed



Plate 9

Bunton supports - no. 3 shaft

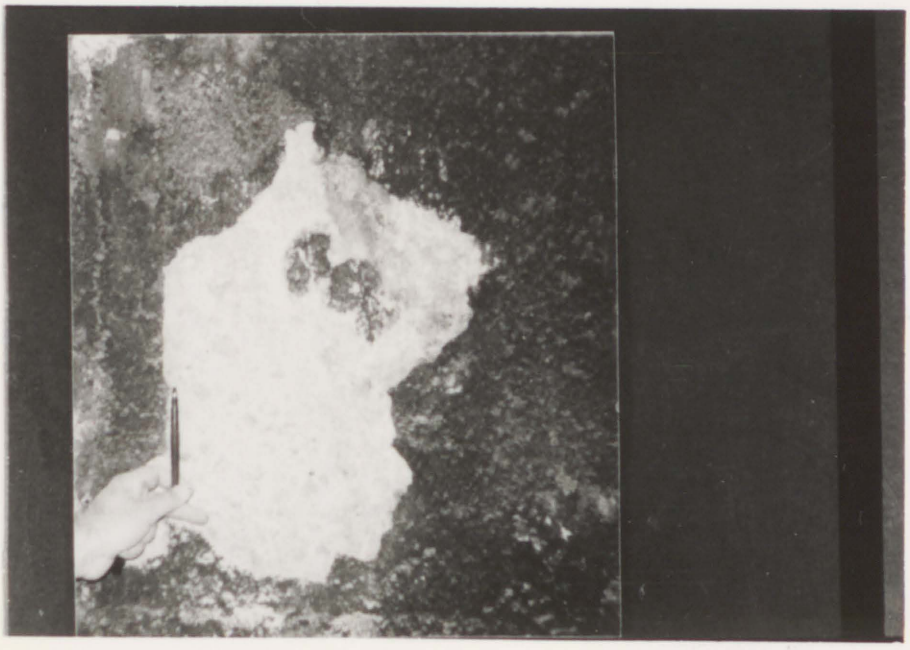


Plate 10

Degrading concrete in vicinity of Culebra Dolomite no. 4 shaft

areas with the fingers. Although there exist substantial salt deposits on the shaft wall between depths 440 and 650 feet, very little shaft water was noted. A few small damp spots were observed, but the ingressing water is totally evaporated by the upcasting air and no water is entering the mine through the shaft. The water ring, located at approximate depth 520 feet, is crusted over with salt and is not performing any useful function. Approximately 5 feet above the water ring there are a number of drillholes used for grouting behind the lining following construction. Salt deposits exist at the collars of these holes indicating limited brine ingress.

From the vicinity of the pump station to the mine station, the shaft wall is generally free of salt and the concrete appears to be in good condition. There are some circumferential cracks in the lining in the vicinity of 745 to 755 foot depth.

Evidence of concrete deterioriation in the No. 4 shaft (between depths 475 and 505 feet) which was exposed when salt deposits were removed from the wall, resulted in a further inspection of the No. 3 shaft liner. Concrete deterioration in the No. 3 shaft is not nearly so obvious. Some thin deposits of surface sand were noted, probably caused by the same processes responsible for the loss of cohesion in limited areas of the No. 4 shaft concrete. It must be presumed, however, that the regions of the No. 3 shaft concrete that have been exposed to the high sulfate water are in no better condition than the corresponding regions of the No. 4 shaft.

6.4 SCHMIDT HAMMER TESTING

The Schmidt hammer test provides a quick, low-cost, non-destructive method of establishing the compressive strength of

concrete. This provides a quantitative evaluation of the condition of the concrete linings of the shafts. The principle of the Schmidt hammer test is based on an established correlation between the rebound of a steel hammer and the compressive strength of the concrete.

The test hammer is held perpendicular to the concrete surface and the plunger is depressed until it impacts. The rebound is determined by a scale reading which measures the level of rebound and, indirectly, the compressive strength.

Since the rebound can be significantly affected by the condition of the concrete surface, points chosen for the test must be carefully selected. Ideally, the surface should be smooth and uniform and preferably cast against a form. A poor surface condition will give underestimates of the concrete strength. There are a number of factors which will affect the accuracy of the strength values indicated by the hammer. These are:

- Surface texture
- Surface/internal moisture ratio
- Size, shape and rigidity of the specimen
- Aggregate size and concentration near the area being tested
- Type of aggregate
- Age of specimen.

Because of the number of variables which can influence the accuracy of the results, a large number of readings are taken at each location (at least 15), and about 10 of these readings (deviating by less than ± 3 on the impact scale) are used to define an average value. The rebound readings may then be converted directly to compressive strength by establishing a calibration curve or by using a standard relation such as that shown

on Figure 6-9 (Journal of the American Concrete Institute, 1954). For the present purposes, the latter procedure was employed, and the estimates of compressive strength should therefore be considered as approximate.

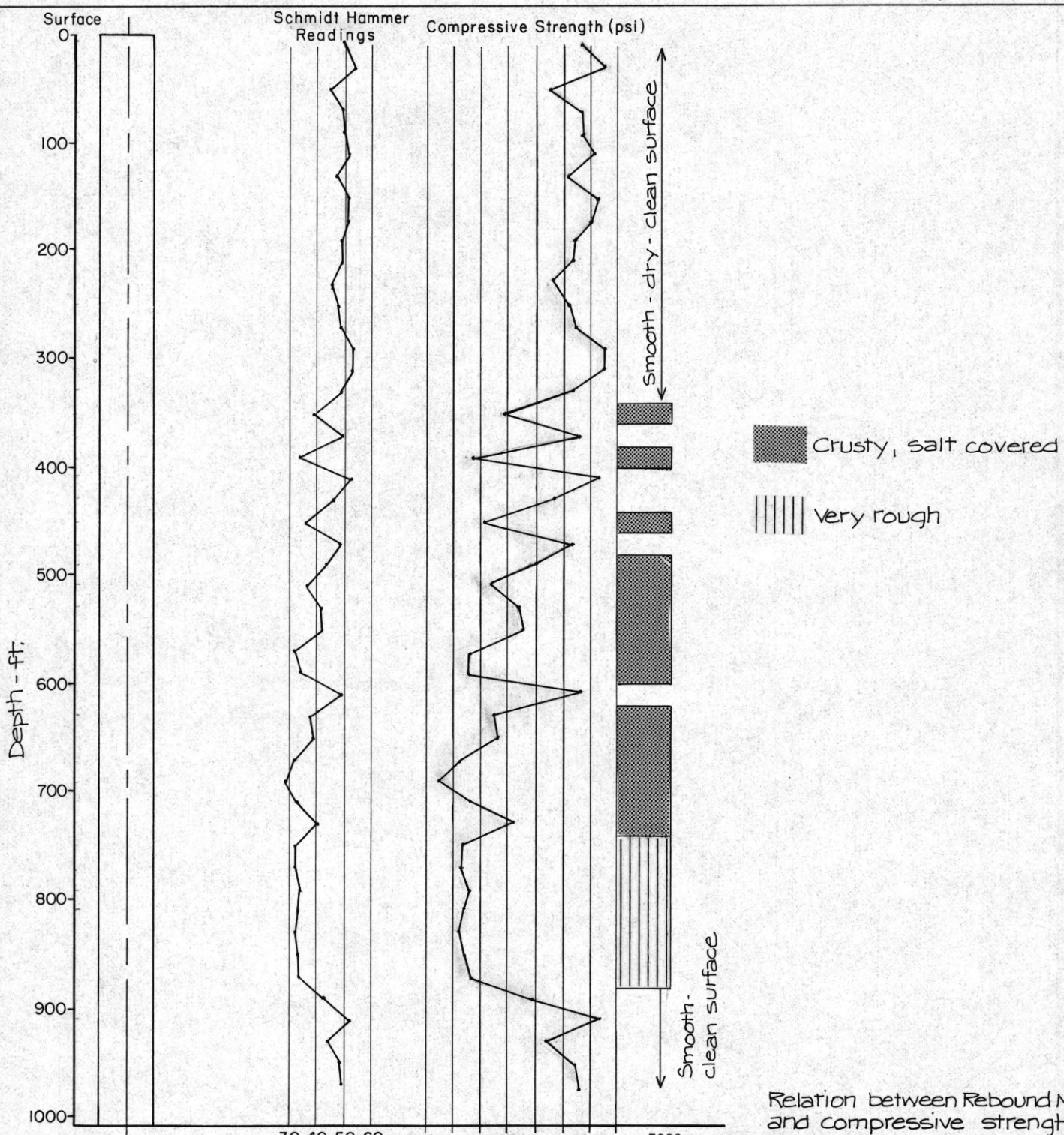
A Schmidt hammer profile was run down the No. 3 shaft. The traverse was taken along the ladderway and readings were taken at 20 foot increments along the full length of the shaft, i.e., at each ladder station. Approximately 15 determinations were made at each location, and the average Schmidt hammer readings obtained over the length of the shaft are shown on Figure 6-9. Also shown are interpreted compressive strength values, determined from the correlation indicated on Figure 6-9.

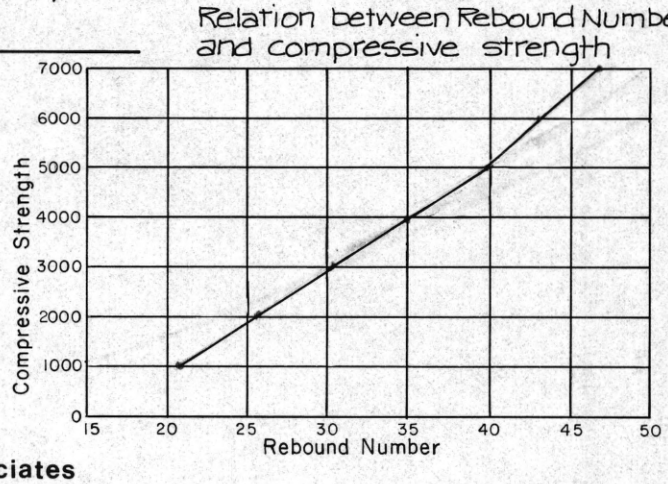
For approximately the first 300 feet the shaft wall was dry and clean and the surface was relatively smooth. The readings indicated a competent concrete with a compressive strength of 7,000 to 8,000 psi. Similarly, in the bottom 100 feet of the No. 3 shaft where the concrete surface was relatively smooth and clean, a compressive strength of the order of 7,000 psi was indicated. In the central 500 to 600 feet of the shaft, the readings obtained were substantially lower and suggested compressive strengths between 3,000 and 6,000 psi. At specific locations within this apparent low strength zone (depths 370, 410, 430, 470, and 610 feet), areas of higher strength (7,000 to 8,000 psi) were obtained. These higher strength regions were again associated with relatively good surface conditions in the concrete.

The low strength regions were invariably associated with a poor concrete surface condition. The concrete surface was either heavily salt-encrusted or very rough (these areas are indicated on Figure 6-9). Attempts were made to clean the salty surfaces with a wire brush, but this was only partially successful. In

SCHMIDT HAMMER TESTING (No. 3 SHAFT)

Figure 6-9





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many locations, the poor nature of the surface resulted in some damage to the concrete by the impacting hammer (i.e., punch marks). The range of surfaces encountered within the No. 3 shaft is shown on Plates 11 through 15.

It is interesting to note that although a large number of low strength readings were obtained over the length of the shaft lining, concrete strengths of 6,000 to 7,000 psi were indicated in the vicinity of the Culebra dolomite member (470 to 490 foot depth). This region would be anticipated to be adversely impacted if sulfate damage to the concrete was occurring. In some respects, the No. 3 shaft Schmidt hammer results were inconclusive. The low strength readings were invariably associated with a poor surface condition of the concrete, i.e., surface salt or very rough and uneven surface, and high strength readings were obtained within the top 330 feet, the bottom 100 feet, and at a number of intermediate points, including the region of the Culebra aquifer, where the surface was free of salt deposits. There was therefore no consistent trend in behavior and some evidence to suggest that the low strength readings were simply a reflection of the surface condition of the concrete rather than indicative of the entire thickness of the liner. The large number of indicated low strength values indicated raises some doubt, however. The sensitivity of all nondestructive techniques for concrete strength testing (e.g., Schmidt hammer, Windsor probe) to the surface condition of the material is a weakness which can often be resolved only by destructive testing of cored samples.

A Schmidt hammer profile was also run down the No. 4 shaft. Readings were taken less frequently as there was no ladder access and all work was done from the cage. Conditions also appeared to be rather more uniform in the No. 4 shaft as a result of the



Plate 11

Smooth, fresh liner surface - no. 3 shaft



Plate 12

Rough, fresh liner surface - no. 3 shaft



Plate 13

Salt crusted liner surface - no shaft



Plate 14

Heavily salt crusted liner surface - no. 3 shaft

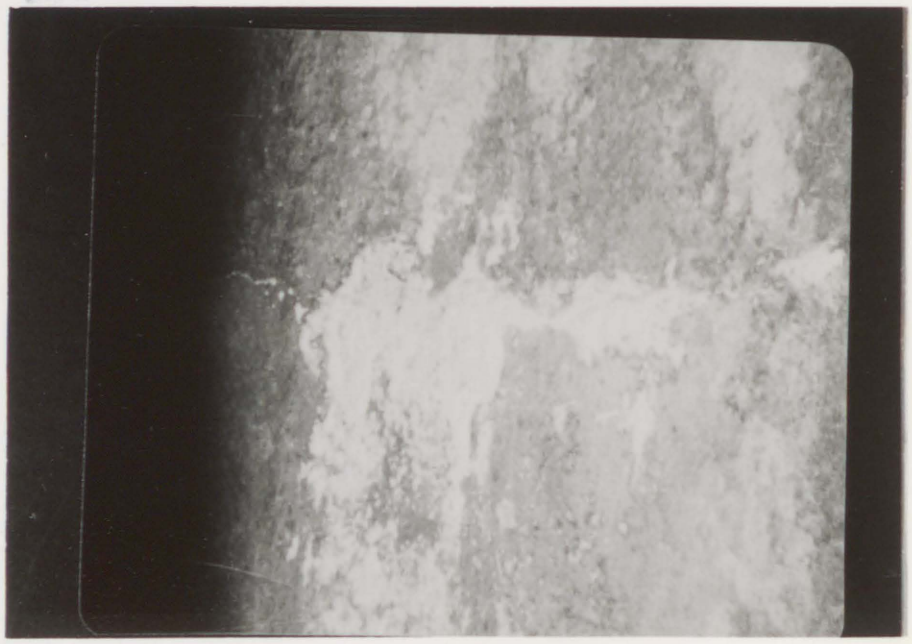


Plate 15

Light salt crustation on lower region of no. 3 shaft liner

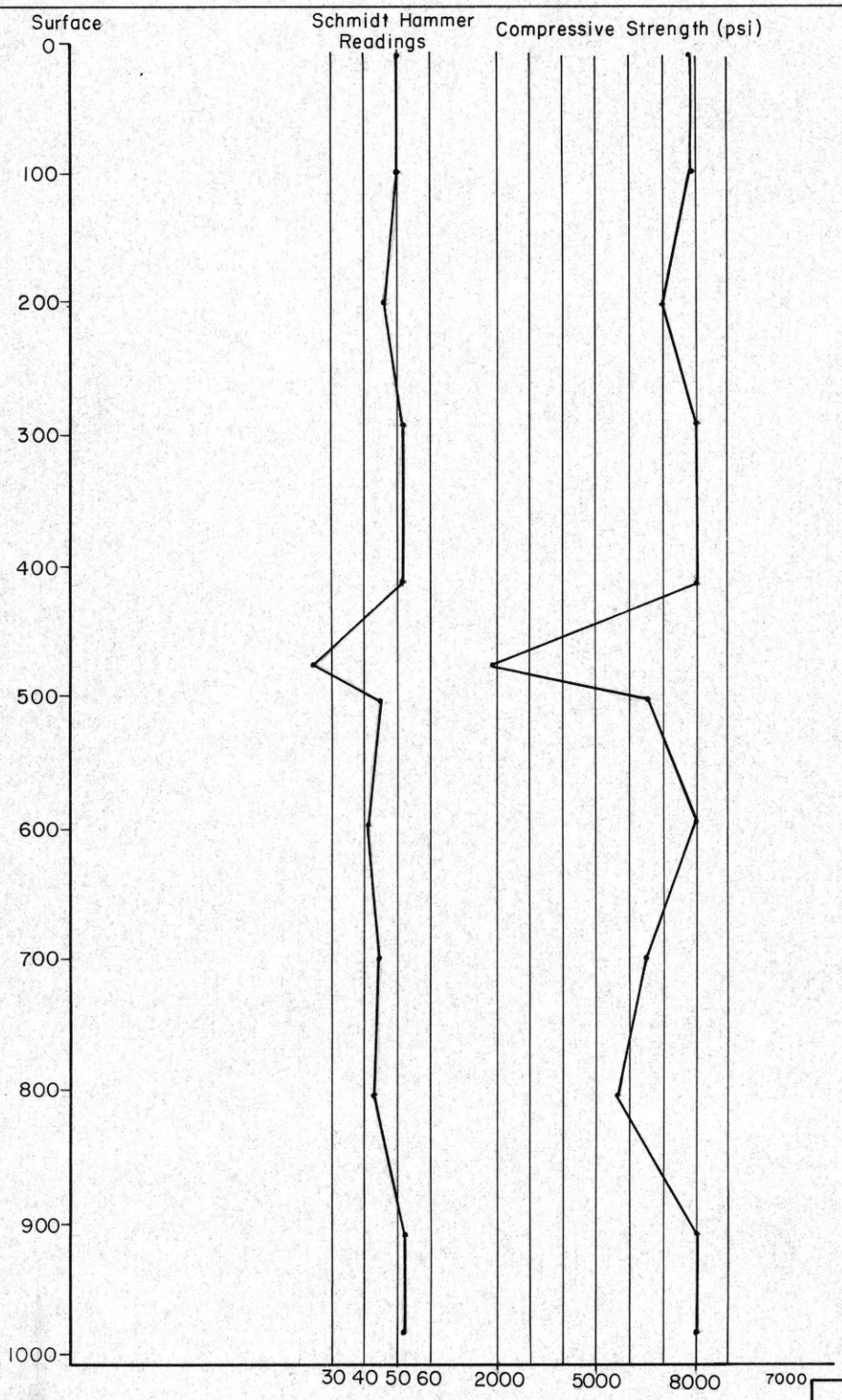
smaller quantities of water present. The results are presented on Figure 6-10. In general, consistently high compressive strengths (6,000 to 8,000 psi) are indicated down the length of the shaft. One conspicuously low set of readings was obtained at depth 475 feet, suggesting a compressive strength of less than 2,000 psi. As mentioned previously, this region of the shaft is coated with a heavy salt crust and corresponds to the elevation of the Culebra aquifer. The reading was taken on a region of the surface subject to lesser salt buildup, as elsewhere the deposits were too thick to be cleaned with a wire brush.

Evidence to indicate that the low strength reading is not purely a surface phenomenon was provided by some small sections of concrete (1 to 2 square feet in area) which were exposed when thick salt deposits were removed by the cage. Approximately 1/4 to 1/2 -inch of concrete has been torn from the wall and at this depth the freshly exposed material has a very sandy/gravelly appearance; there is evidence that the cement bond is being destroyed. The area of the shaft wall which appears to be affected is an approximate 30 foot section ($\simeq$ 470 to 500 foot depth), primarily on the east side of the shaft where salt deposits are heaviest.

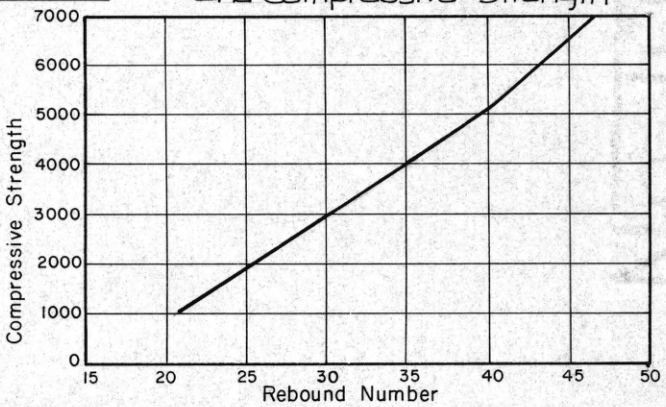
It is not possible to determine to what depth degradation of the shaft lining might be occurring, using non-destructive techniques. Evidence within the No. 4 shaft (Schmidt hammer and visual observations) suggests that there is a good possibility that a limited length of the shaft liner is suffering concrete deterioration as a result of the high sulfate content of the groundwater. This region of the shaft corresponds to the area of most significant water ingress, as evidenced by the heaviest salt deposits present, and is also coincident with the major aquifer

SCHMIDT HAMMER TESTING (No.4 SHAFT)

Figure 6-10



Relation between Rebound Number and compressive strength



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of the area - the Culebra dolomite. The majority of the No. 4 shaft lining appears to be in sound condition, with the concrete exhibiting substantial strength. It is believed that sulfate resistant cement was used in the concrete of the shafts' liners although there is no information on either Duval files or the shaft sinking contractor's files to confirm this.

Evidence for concrete deterioration in the No. 3 shaft is less obvious, but it must be recognized that similar concrete deterioration to that suggested in the No. 4 shaft may also exist within shaft No. 3.

6.5 RADAR PROFILING

An item of great concern to shaft stability for shafts constructed in evaporite materials is the possibility of dissolution of the salts behind the lining, leading to loss of bond and eventual shaft collapse. The condition of the concrete lining itself has been established by Schmidt hammer testing and by visual inspection. The state of the shaft fixtures has also been evaluated by visual inspection. In order to examine the condition of the lining-salt/rock contact, a technique of electromagnetic subsurface profiling (ESP) was utilized. Golder Associates employed a subcontractor, Geophysical Survey Systems, Inc., of Hudson, New Hampshire, to perform this work.

The ESP system consists of a portable antenna unit which transmits and receives radar pulses. The pulses are partially refracted and partially reflected at physical interfaces, with the degree of reflectance governed by the differentials in the electrical characteristics of the materials--the dielectric constant and the conductivity. The reflected radar signals are picked up by the transceiver and recorded on magnetic tape. This enables the traverses to be conducted at a much faster rate than would be possible if a hard copy printout were generated during the operation. Within the Wills-Weaver No. 3 mine shaft, the

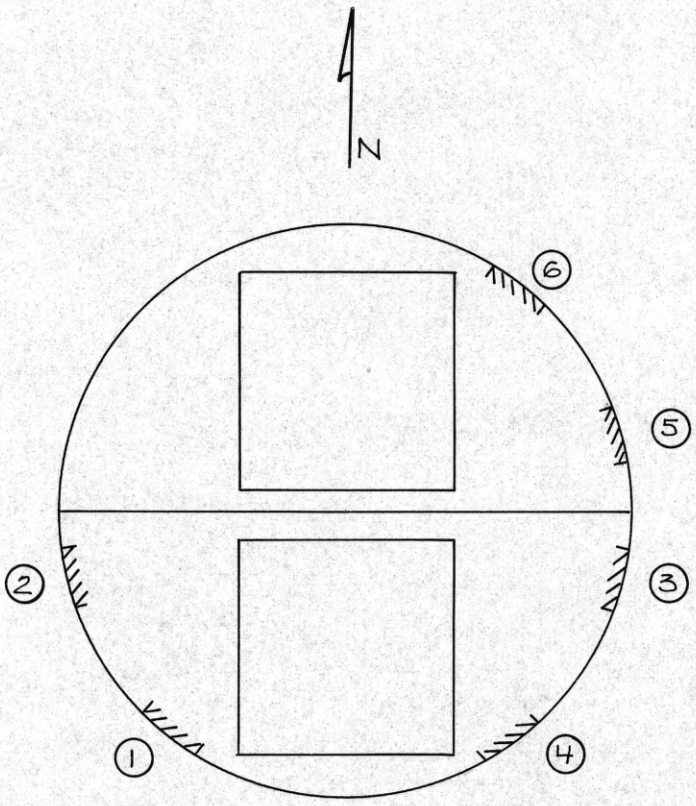
cage was lowered at a uniform rate of about 100 feet per minute, and each traverse required approximately 10 minutes of travel time. A further advantage of recording the signal on magnetic tape is that computer processing can subsequently be carried out in order to filter out the input signal, which is recorded automatically along with the reflected echoes. Hard copy printouts from the magnetic tape recordings were generated at a later time.

The depth of penetration of the radar signal is governed by the conductivity of the material. Highly conductive materials (e.g., reinforcing mesh, brine solution, etc.) will inhibit penetration, although the presence of the interface itself will be indicated by a strong signal.

On February 26, 1979, Geophysical Survey Systems, Inc., in conjunction with Golder Associates, surveyed the No. 3 shaft of the Wills-Weaver mine using a 101B transducer. Because the signals were weaker than expected, the survey was repeated on February 27 using a 3105 transducer (higher power unit). This latter instrument was borrowed from Sandia Laboratories, who were testing equipment in Duval's Nash Draw mine. In both cases, the transducer was held against the wall of the shaft from the cage as it was lowered at a constant rate to produce a vertical traverse down the shaft. Bench marks were recorded onto the magnetic tape every 20 feet, using the ladderway platforms to determine the distance. Figure 6-11 shows the approximate locations of the six traverses made with each transducer. A single calibration run was also undertaken with each of the two instruments and, in all, about 14,000 feet of traverses were recorded. As shown on Figure 6-11, two traverses (with each transducer) were run in each quadrant of the No. 3 shaft, with the exception of the quadrant containing the ladderway and the service pipes.

LOCATION OF RADAR VERTICAL PROFILES
IN SHAFT No. 3

Figure 6-11



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The data from the 3105 transducer, after being computer processed to remove the background signals and reverberations, proved to be the most useful, and most of the conclusions listed below were based on the results from this instrument. In most cases the conclusions were verified by the 101B data. The data analysis was primarily concerned with locating possible voids behind the lining, but additional information such as the location of the lining rebars was also obtained.

The apparently weak signals which were obtained with the 101B transducer resulted in calibration tests being carried out at the mine on February 26. The first of these tests consisted of monostatic probing through a 2-foot thick footing in the hoist house. The reflection from the back side of the footing (concrete-air interface) was of only moderate strength but was easily identified. A metal shovel moved past the back of the footing also gave a clear signal.

A second test was carried out on the concrete lintel of the entrance to the No. 3 shaft pump station. The transmitter was placed inside the pump station and the receiver was placed on the inside surface of the shaft. A clear signal was received through the 3.5 feet of concrete. With the transducer in the shaft operating in the monostatic mode, the motion of a metal sheet on the pump station side of the lintel was not detected. Unfortunately, the geometry of the test was not satisfactory in that it was not possible to hold the metal plate flush with the concrete and perpendicular to the line of sight to the transducer. It is probable that the energy reflected from the metal plate was not directed toward the receiving antenna.

The above tests indicated that the less powerful 101B transducer was able to penetrate the shaft lining when it was 2 feet or less thick.

The reflection coefficients of the interface between the concrete shaft liner and the surrounding materials have been estimated as shown in Table 6-1.

Table 6-1
REFLECTION COEFFICIENTS AT VARIOUS INTERFACES

Surrounding Material	Reflection Coefficient
Sandstone	0.2
Limestone	0.1
Halite	0.1
Air	0.5
Water	-0.5

Although the reflection coefficients are not large, they are not so small that detection of a reflection would be expected to be difficult, particularly if the liner was less than a foot thick. It is also evident that the reflection coefficients for air and water are larger, making the detection of voids much more likely than detecting the interface between the concrete and the surrounding medium.

The roughness of the interface behind the concrete can have a strong influence on the reflected energy. Facets which are longer than the radar wavelength will act like a mirror and reflect the transmitted energy to one side, unless they are perpendicular to the rays from the antenna. When the wave length is larger than the size of the facets, the surface "looks" smooth

and the reflection direction is determined by the average surface. In either case there is scattering of energy from all sharp edges, and this spreading of the reflected energy may result in a weak signal at the receiver.

When a rough interface exists the following behavior would be expected:

- Weak echoes
- Occasional stronger reflections from perpendicular facets and scattering edges
- Stronger reflection when longer radar wavelengths are used.

All of these characteristics are present in the Wills-Weaver data. It should be noted that the 3105 transducer provides a wavelength in concrete of the order of 1 foot and the 101B transducer gives a corresponding wavelength of 0.4 feet.

In general, all of the traverses showed similar behavior and records of the six traverses obtained with the 3105 transducer (with the input signal filtered out) are shown on figures 6-12 to 6-17. The locations of the rebars are immediately obvious in each of the records. The rebars are located over the intervals 0 to 100 feet, 360 to 420 feet, and 445 to 580 feet. This is in contrast to the shaft sinking log, which indicates continuous reinforcement over the intervals 0 to 100 feet and 360 to 550 feet. In general, the presence of the rebars confused the reflections to an extent that prevented evaluation of the deeper reflections from within the reinforced sections of the lining. The rock types behind the region of the lining in which rebar confused the signals consisted primarily of anhydrites and shales of the Rustler Formation, in addition to the dolomites/limestones of the

aquifers. Dissolution voids would not be anticipated within these strata.

A change in the echo pattern is noted on some of the traverses (e.g., 1 and 2) at approximate elevation of 300 feet below the shaft collar, and this corresponds to the elevation of the piezometric surface in the region of the No. 3 shaft.

A number of other signals are evident which are due to surface obstructions within the shaft, and these are clearly delineated. For example, on traverse Nos. 1 and 2, there was an obstruction approximately 40 feet beneath the shaft collar which required that the transducer be lifted from the wall. Similarly, the locations of the pump room (traverse Nos. 1 and 2) and the water ring (all traverses) are clearly evident.

The general pattern of behavior recorded did not suggest that dissolution damage was occurring behind the shaft lining. Dissolution of the Salado salts behind the shaft liner would progress from the Rustler/Salado contact downwards. There would therefore exist a pattern of solution voids which would be evident from the top of the Salado to the depth of dissolution into the Salado. No such pattern of voids was recognized from the radar profiling.

Further evidence for this position includes:

- There is no evidence of seepage behind the lining at either the mine station or the pump station

- The shaft wall within the Salado is comparatively dry i.e., there is no significant water circulation associated with the Salado salts in the vicinity of the shaft

- Total water flow into the shaft is small, the majority of this coming from the Culebra dolomite aquifer and being piped directly into the workings.

One region of the shaft which showed consistent reflections on all traverses was the section located at approximately 760 to 790 feet beneath the shaft collar. This may correspond to a region of higher moisture content. It is not considered that it is possible that a solution cavity exists at this location, since there is no consistent evidence that such behavior is occurring. Unsaturated water capable of inflicting dissolution damage would originate from the dolomite aquifers of the Rustler Formation. In order for this water to cause dissolution damage, the following observations would be made:

- Dissolution would commence at the top of the Salado and progress downwards

- A region of water discharge from behind the lining would occur, since only flowing unsaturated water will cause damage.

Since the egress of water from within the Salado was not observed and indications of cavities above the region 760 to 790 feet were not obtained, it is unlikely that the reflections obtained in this area are indicative of a solution cavity. Nevertheless, it might be prudent to investigate this region of the shaft (by coring through the lining) if the shaft were to be extensively utilized at some time in the future. Significant reflections were also obtained on most of the traverses within the interval 900 to 950 feet beneath the shaft collar.

Other consistent minor reflections can be noted on all traverses over the full length of the shaft, and many of these correspond to the construction joints. Some of these reflections are indicated on the figures.

6.6 REVIEW OF CONDITIONS OF WILLS-WEAVER MINE SHAFTS

The conditions of the mine shafts at Wills-Weaver mine were established by the following methods:

- Visual inspection of the shafts and shafts' fittings. Review of shafts' sinking and grouting logs
- Schmidt hammer testing to assess the quality of the concrete linings
- Radar profiling to determine the nature of the lining-rock/salt interface

The following points were noted.

- Grouting of the principal aquifers of the Rustler Formation (i.e., Magenta and Culebra dolomites) in multi-stage operations, achieved a virtually dry No. 4 shaft and a No. 3 shaft making very little water (≃20 to 40 gph).
- The water rings installed at 527-foot depth in the No. 3 shaft and at the 520-foot depth in the No. 4 shaft are in disrepair. The majority of the No. 3 shaft water is presently collected and piped into the mine workings. All of the No. 4 shaft minor inflow (1 to 2 gph) is evaported.

- There is no water entering either the No. 3 or No. 4 shafts' pump stations or mine stations from behind the lining, and the concrete linings beneath the top of the Salado are basically dry, although limited size damp areas are present in the No. 3 shaft.

- The No. 3 shaft steel is lightly corroded. The buntons are in good condition, but lighter gauge metal (such as the ladder way) could be weakened in the near future. Probable weak points in the steel are the bolts fastening the steel to the concrete. These bolts are susceptible to corrosion from saline water and would be subject to high stresses during operation.

- Utility pipes are badly corroded in places and would require attention before they could be utilized. The supporting Zee brackets are in good condition, but the fastenings should be examined in detail prior to future utilization.

- Corrosion is quite pervasive throughout the No. 3 shaft skip loading station and some of the plate would require replacing before start-up could be effected. Most of the chutes, gates, and lines are, however, in a usable condition at present.

- There is 30 feet of water in the No. 3 tailshaft. The bucket elevator and the manlift would require extensive repair or replacement, and the condition of the spill pocket is unknown as it is flooded. The No. 4 tailshaft contains only about 4 feet of water.

- Wooden shaft guides show no evidence of unusual wear or misalignment. The guides in the No. 3 shaft have been treated with oil and paraffin and are in excellent condition. The No. 4 shaft guides are also in good condition, considering that they have not been maintained since closure of the mine.

- Schmidt hammer readings within the No. 4 shaft indicated a generally sound lining over the full length of the shaft. An area of low indicated strength was obtained in the vicinity of the Culebra dolomite aquifer (480 to 500 foot depth), where substantial salt deposits on the wall indicated some minor water inflows into the shaft. Freshly exposed concrete in this area also showed signs of deterioriation, suggesting that sulfate attack is a distinct possibility. This could be further investigated by coring and testing.

- Although significant areas of the No. 3 shaft liner gave low indicated compressive strengths when tested with the Schmidt hammer, the overall distribution of the strengths down the shaft wall suggested that many of the low readings were the result of purely surface effects. This is considered particularly true for the section of the liner within the Salado salts, which exhibited a quite rough surface condition. Although direct evidence of concrete deterioration within the No. 3 shaft was less obvious than in the No. 4 shaft, it must be recognized that those areas of the No. 3 shaft liner exposed to significant groundwater would be in no better condition than that suggested for the No. 4 shaft. Such deterioration is likely to be more widespread in the No. 3 shaft as a result of the more significant amounts of water present.

- Radar probing in the No. 3 shaft indicated that no significant air or water filled voids exist behind the lining. There was no consistent evidence of a void pattern indicative of solutioning, and this was in accord with the visual inspection. The majority of the recorded echoes were explained in terms of known physical features (e.g., construction joints, pump room, water ring, reinforcing bar, etc.). Consistent reflections were obtained on all traverses at approximately 760 to 790 feet beneath the shaft collar. These cannot be explained in terms of known physical features. Similarly, reflections were also noted in the interval 900 to 950 feet.

- The locations of the No. 3 shaft lining rebar were readily identified from the radar probing. The presence of the rebars (locations 0 to 100 feet, 360 to 420 feet, and 445 to 580 feet) generally confused the signals sufficiently to prevent an evaluation of deeper reflections in these areas. It is not considered likely that dissolution voids would exist behind the section of reinforced lining.

- One somewhat disappointing aspect of the radar profiling was its inability to identify the concrete lining/rock contact.

7.0 PREDICTION OF MINE AND SHAFTS' LIVES

7.1 INTRODUCTION

The life of the mine opening is determined by geomechanical and hydrological factors. The rate of mine closure will depend first on the state of stress within the pillars and on the dimensions and structural characteristics of the strata in the immediate vicinity of the rooms. Second, the ingress of relatively fresh water into the mine workings will cause dissolution damage and occupy the existing mine voids.

There are also a number of factors to consider in estimating the probable life of the mine shafts. The integrity of the concrete linings might be compromised by excessive earth pressures, by general strata subsidence associated with closure of the mining horizon, by dissolution damage caused by water flowing behind the shaft lining, or by deterioration of the concrete under the action of the brackish shaft water. The shaft fittings' lives will also be influenced by the corrosive action of shaft water.

The data required to estimate the Wills-Weaver mine and shafts' lives have been reviewed and summarized in the previous sections. These data are subsequently interpreted to provide summary estimates of the future performance of the mine voids and mine shafts. The estimates will be qualified, since the expected lives will to some extent be determined by action taken during abandonment of the property.

7.2 PREDICTION OF FUTURE PERFORMANCE OF MINE

Considering the mechanical performance of the major structural elements of the mine horizon, the following points can be made:

- Large areas of the Wills-Weaver mine workings have undergone pillar mining with subsequent extraction ratios on the order of 75 to 85 percent. These regions have suffered substantial pillar closures and associated surface subsidence, together with excessive floor heave. Such regions are no longer accessible.

- The only area of the present mine workings which is considered accessible is the western region of the Main East Entry panel (west of the entry to the 3rd North panel), where no pillar mining has been carried out. In this area, an extraction ratio of approximately 60 percent has been employed, and the relatively short and squat pillars (60 feet square by 4 feet high) have imparted a high degree of stability to the workings. These pillars show virtually no signs of distress and are probably closing at a rate of less than 0.1 inch per year.

- Room closure rates are somewhat greater than pillar compression rates, as a result of roof sag and floor heave. Although room closures of up to 0.5 inch per year have been noted in some areas of the supported region of the Main East Entry panel, the rates of closure are in general less than this. Roof conditions are generally very good despite the limited roof bolting in this area. Isolated roof falls of considerably proporations may

develop from time to time, but these are not expected to become widespread. Floor conditions within the western half of the Main East Entry panel are also acceptable, although floor heave will develop with time at a rate which is closely linked to the rate of pillar compression.

- It is not expected, therefore, that stability conditions within the supported area of the Main East Entry will deteriorate significantly within the next several decades. Over this period, isolated roof falls may occur, and floor heave will develop progressively.

- Extraction within the immediate pillar protection zone (up to 550 feet from the No. 3 shaft) is limited and this area of the workings is not expected to deteriorate significantly with time.

The second factor which can influence mine stability is the dissolution capacity of water entering the workings. The following items should be noted.

- Virtually all of the water at present entering the mine workings is originating from the No. 3 shaft. In general, the core test holes and abandoned oil wells which penetrate the mining horizon have been adequately sealed, and it is not considered that water will enter the workings from these sources. Two core test holes have suspect seals (Holes S-6 and S-18), but both holes are at present protected from the mine horizon by salt barriers. There is no water entering the mine through the No. 4 shaft.

- The No. 3 shaft water is at present piped into the mine workings. The flow rate is variable (between 12 and 36 gph have been measured at various times), but 20 gph is probably a good estimated average rate of discharge. Although the water is brackish (total dissolved solids concentration in excess of 10,000 mg/l), it has a very substantial dissolution capacity. It is estimated that approximately 420 cubic yards of material (principally carnallite) are dissolved from within the mining horizon each year. The above value represents approximately the volume of a mine pillar within the Main East Entry panel.

- The present discharge point for the No. 3 shaft water is at the entry of the 1st North panel. From this point, the water flows eastward across the Main East Entry panel. It is evident that a significant proportion of the salt dissolution is occurring within the supported area of the Main East Entry. It would appear appropriate that any water which is piped into the mine should have its discharge location positioned toward the entry to the 3rd North production panel. This would prevent dissolution damage to a presently stable area of the workings.

- Shaft water should not be permitted to simply drain into the tailshaft as part of any abandonment option which calls for future utilization of the workings. This could result in substantial dissolution damage to the area in the vicinity of the shaft mine station. The impact of allowing water inflow to the mine workings is discussed more fully in Section 8.0.

7.3 PREDICTION OF FUTURE PERFORMANCE OF SHAFTS

With respect to the estimated performance of the Wills-Weaver mine shafts and shafts' fittings, the following factors are of importance:

- The shafts have suffered no damage as a result of subsidence of the strata overlying the mining horizon. A substantial area of the Main East Entry panel (extraction ratio of 60 percent) could cause subsidence damage to the No. 3 shaft if the mine pillars within this region were to close significantly. As outlined in Section 7.2, this area of the workings is very stable, and rates of closure are such that subsidence damage to the No. 3 shaft is unlikely to occur for many decades. There will be no subsidence damage to the shafts as a result of movements within the shafts' protection pillar, unless dissolution damage to this area occurs.

- In general, the condition of the concrete of the shafts' liners is sound, although specific regions of the concrete appear to be degrading as a result of sulfate attack. Within the No. 4 shaft there is evidence that the concrete in the vicinity of the Culebra dolomite aquifer has suffered over the last two decades and deterioration may be somewhat more widespread in the No. 3 shaft where substantially more water is present. Under presently existing conditions, the majority of the concrete of the shafts' liners can be expected to remain structurally sound for a considerable period (e.g., many decades). Specific regions of the concrete which have been exposed to the sulfate laden groundwater show evidence of deterioration and are probably degrading much more rapidly.

- Visual inspection and subsurface radar profiling yielded no evidence to suggest that dissolution was occurring behind the No. 3 shaft lining. Consistent signals were obtained in the lower regions of the No. 3 shaft (760 to 790 feet, 900 to 950 feet below shaft collar elevation), and these could not be explained in terms of known physical features. Depending on future utilization of the shafts, it might be prudent to investigate these areas by coring through the liner. Visual inpsection of the No. 4 shaft showed no evidence of dissolution damage occurring behind the lining.

- The shafts' fittings have suffered to varying degrees, depending upon their location and character. There is approximately 30 feet of water in the No. 3 tailshaft, and the bucket elevator and manlift will require extensive repair or replacement before future utilization. Corrosion is pervasive throughout the skip loading station, but most of the chutes, gates, etc., are in a usable condition at present. Some of the plate would require replacement before startup. The remainder of the shaft steel has a life which is primarily dependent on the gauge of the metal. Lighter gauge metal (e.g., ladderway) can be expected to be weakened in the near future, and the existing utility pipes are badly corroded in places and would require attention before they could be used. The heavier gauge metal (e.g., shaft buntons) is lightly corroded and retains most of its original strength. Supporting bolts for the buntons, utility Zee brackets, etc., might be significantly weakened by corrosion and would require examination and possible replacement prior to startup.

- The wooden shaft guides in the No. 3 shaft are at present in excellent condition, and have been well maintained since the cessation of mining activity. The No. 4 shaft guides are also in good condition, considering they have not been maintained since cessation of mining. Should the guides be left in place over substantial periods as part of any abandonment alternative, they can be expected to suffer dry rot and deteriorate.

- At present, therefore, there are a number of areas in the shafts' fittings that would require attention before start-up in the near future (e.g., bucket elevator, tailshaft manlift, some plate within the skip loading station, utility pipes, supporting bolts). Start-up in the more distant future would probably require replacement of or extensive repair to most of the shafts' fittings.

- Although the water entering the No. 3 shaft is limited, it has substantial dissolution capacity and could cause significant damage to the mine station over a period of time if simply allowed to drain into the tailshaft. With regular maintenance, the water has been controlled since the completion of mining. Shaft abandonment (with loss of maintenance) will require an alternative solution to the control of shaft water, if it is required to prevent dissolution damage of the mine station.

8.0 POTENTIAL IMPACTS OF SHAFT FLUID INGRESS

8.1 INTRODUCTION

Although the Wills-Weaver mine is comparatively dry, the relatively small flow rates entering the workings through the No. 3 shaft have substantial dissolution capacity, as indicated in Section 3.6.3. To date, the mine water has been largely controlled and dissolution damage has not significantly impacted the mine or mine shafts' stabilities. Abandonment procedures may lead to loss of control over the No. 3 shaft water and this section presents an evaluation of short and long-term effects that may result if water continues to enter the mine. Initially, a general overview of a potential long-term scenario is discussed and details of potential dissolution damage etc., are considered subsequently.

8.2 SHAFT WATER INGRESS - A GENERAL DISCUSSION

It is estimated that there is approximately 60 million cubic feet of void space at present available in the Wills-Weaver mine. This estimate is based on total areas and extraction ratios for the workings, together with estimated room heights for each of the areas. This mine void is continuing to close with time as the pillars compress. Different regions of the mine workings are closing at different rates, depending on the extraction ratios employed. Only order of magnitude estimates can be made of the rates of closure and the following limits have been estimated:

Region	Lower Limit	Upper Limit
60% Extraction	0.02 in/yr	0.1 in/yr
70% Extraction	0.08 in/yr	0.4 in/yr

The area of high extraction (≈85%) is presumed to have largely closed. The above values represent estimated pillar closure rates and these are utilized to estimate the rate of mine void closure. The void closure occurs because of two phenomena:

- The roof and floor move closer together (with associated surface subsidence) and the void is reduced.

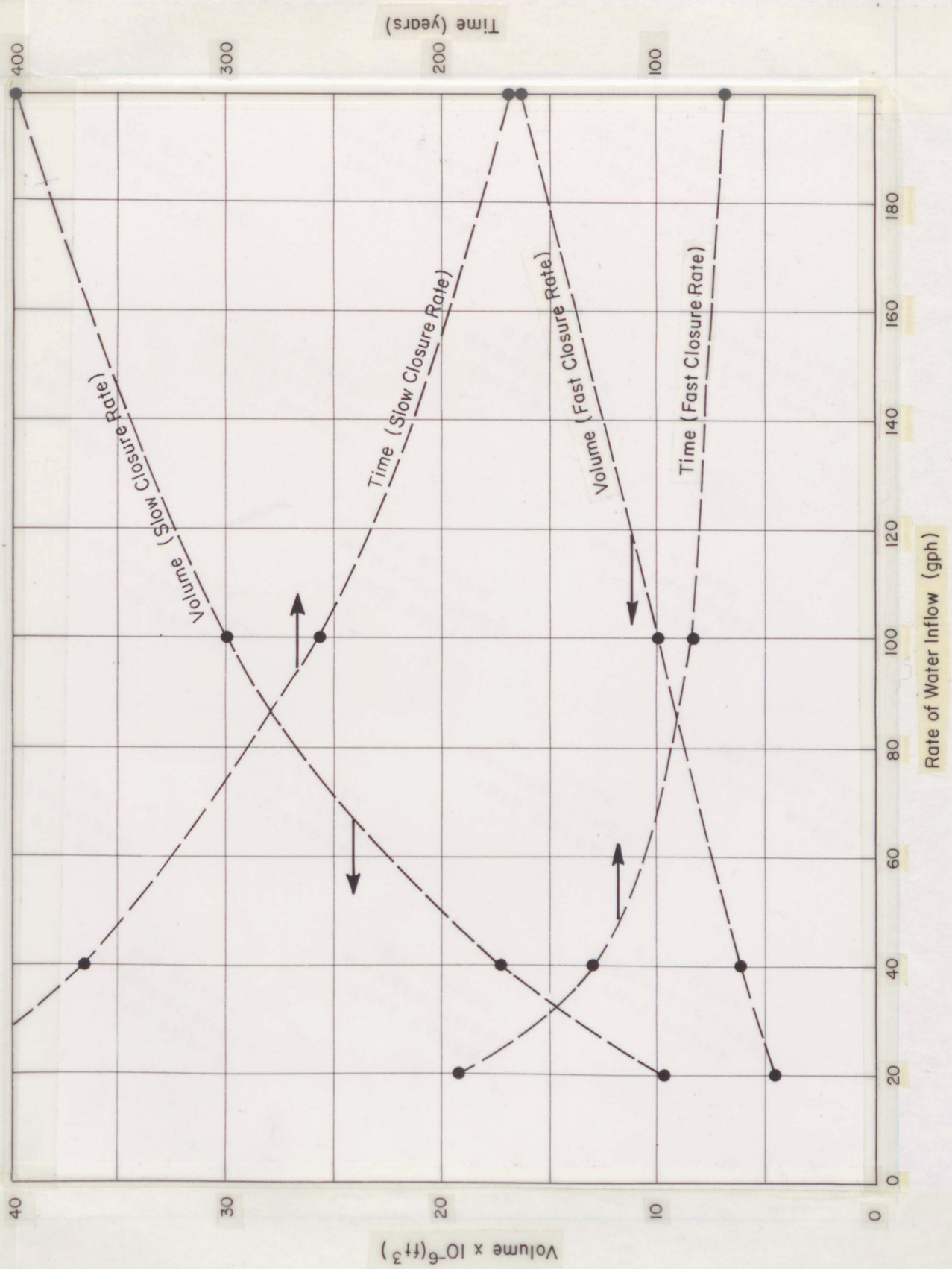
- The void is further reduced as a result of lateral expansion (plastic flow/pillar spalling) of the in situ pillars as they are compressed.

Both of the above phenomena have been considered in an approximate manner in estimating the rate of void closure.

If comparatively fresh water entered the mine void, it would occupy the existing void space until the mine was filled with water. In addition to occupying the original void space, the 'fresh' water would create new voids by dissolution. As the relatively fresh water became saturated however, it would occupy a greater volume. It can be assumed that the additional volume occupied by the fluid as a result of brine saturation would approximately balance the additional dissolution void generated. The total volume then available to the 'fresh' water would approximately equal the initial (i.e., present) void volume less the volume lost as a result of mine closure prior to filling the mine. The total volume of 'fresh' water which would enter the workings, together with the time taken to flood the mine, are shown as a function of the rate of water inflow on Figure 8.1 (for the upper and lower estimated limits of mine void closure rate). Figure 8.1 indicates therefore, that for an inflow rate of 20 gph, it would take between 190 and 420 years to flood the mine and that between 5 and 10 million cubic feet of water would have entered the workings at that stage. For an inflow rate of 100 gph, the mine would flood within 80 to 260 years and would

TIME AND VOLUME OF 'FRESH' WATER TO FILL MINE VOID

Figure 8-1



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have been inundated by 10 to 30 million cubic feet of relatively fresh water.

At the completion of flooding of the mine, the total volume of saturated brine existing within the workings would be a function of the amount of 'fresh' water which entered the workings, together with the nature of the salts available for dissolution. For an average inflow rate of 20 gph, it is estimated that the total volume of brine saturated in chlorides of sodium, potassium and magnesium which could exist in the workings at the time of complete flooding would be in the range 7.3 to 14.5 million cubic feet. For an inflow rate of 100 gph, the total volume of brine saturated in chlorides of sodium, potassium and magnesium which could exist within the mine at flooding would be within the range 14.5 to 43.6 million cubic feet.

The above estimates presume that the chlorides of sodium, potassium and magnesium would be present in sufficient quantities to provide a solution which was saturated in the presence of each of the solid chlorides. If this were the case, the material which would suffer the majority of net dissolution would be the carnallite. For an inflow rate of 20 gph, the final volume of water in the mine voids would have a carnallite dissolution capacity of between 2.2 and 4.5 million cubic feet. For an average inflow rate of 100 gph, the final volume of water in the mine voids would have a carnallite dissolution capacity of between 4.5 and 13.5 million cubic feet, depending on the rate of which the mine voids were closing.

The volume and distribution of carnallite present in and around the Wills-Weaver mine workings is therefore of considerable interest in establishing possible lateral growth of the mine voids as a result of dissolution. This question is considered in detail later.

The preceeding scenario indicates that if water were permitted to enter the mine workings, the mine void would eventually be filled with a large volume of saturated brine. The exact volume of brine is a function of the rate of closure of the mine voids, the rate of water ingress and the type of salts dissolved. A minimum volume of brine is estimated to be approximately 7 million cubic feet, and substantially larger volumes may ultimately exist (of the order of 50 million cubic feet or greater). It is also estimated that the time required to completely flood the mine would be from approximately one-hundred to several hundred years.

There are a number of factors which complicate the above picture. Dissolution of mine pillars would tend to hasten closure of the workings but there is no way of quantifying this. Damage to the shaft lining in the vicinity of the aquifers (e.g., subsidence damage) might result in high rates of inflow and this would tend to increase the ultimate volume of brine existing within the mine void. Large volumes of brine within the mine following flooding would be associated with rapid rates of water inflow and small rates of mine closure and smaller volumes of brine would occur for small rates of water inflow and a rapid rate of mine closure.

When the mine had completely flooded, the rate of closure could be expected to diminish. As substantial hydrostatic heads (pressures) were developed within the mine water, these pressures would provide support to the back (reducing the average vertical stress levels with the pillars) and lateral confinement to the supporting pillars. Thus, pillar deviator stress levels would be significantly reduced and it is estimated that this would lead to an approximate order of magnitude drop in the rate of mine closure. Since mine volume closure rates prior to flooding have been estimated at 120,000 to 600,000 cubic feet per year, it is considered that following flooding of the mine, the rate of void

closure would be of the order of 10,000 to 50,000 cubic feet per year. This volume of saturated brine would be returned either to the surface or to the groundwater aquifers overlying the Salado (depending on hydraulic connectivity) and would represent a continuing source of pollution for many years. It is estimated that approximately 10,000 acre feet (450 million cubic feet) of brine effluent is discharged to tailings ponds each year by the refineries of the southeast New Mexico potash industry (BLM, 1975). In terms of annual rates of brine effluent discharge to the environment overlying the Salado salts, the discharge from the Wills-Weaver mine following flooding would be considered as negligible.

The above analysis has assumed that the Wills-Weaver mine void is 'water tight' and that there exists no continuing source of leakage from the void as it is filled by shaft water ingress. Observations of transient water ingress to mines in bedded salt along clay interbeds and local movements of water within the mine horizon have led some to consider that there is substantial capacity for fluid transmission within undisturbed bedded salts. Such observations are not, however, necessarily indicative of a permeable bedded salt formation in the overall sense. Laboratory testing of salt samples demonstrates generally very low permeabilities and tight clay interbeds consolidated by overburden would also be expected to have very low permeabilities. In order to examine the suggestion, however, that the nature of these clay seams is such that they impart an overall hydraulic conductivity to the formation along the bedding, some in situ evidence should be considered.

- Although in situ mesurements of hydraulic conductivity of salt are rare, Sandia laboratories performed a number of Drill Stem tests to measure the hydraulic conductivity of

bedded salt at the Wipp site and found that the salt yielded no fluid to the measurement test well. Hydraulic conductivities on the order of 5×10^{-9} cm/sec were estimated for vertical intervals ranging from 12 to 77 meters in thickness, and this included the influence of the clay interbeds (Griswold, 1977).

- Indirect field data also suggest that salt may have essentially no permeability. Nitrogen - brine artesian pockets are found in bedded salt deposits throughout the world (Baar, 1977; Griswold, 1977) and it can be inferred that the pressures in excess of those of aquifers above or below the salt should long ago have dissipated if salt had any significant permeability.

- Inflows of brines along clay interbeds are often observed in salt and potash mines and such occurrences are sometimes suggested as evidence of the permeability of bedded salt formations. These flows are, however, transient in nature and it is suggested that the transient inflows are the result of consolidation of the clay layers by stress concentrations associated with mining activities. Further evidence for this explanation (Geotechnical Engineers, Inc., 1978) is provided by observations in the potash mines of southeastern New Mexico; rates of water inflow appear to be proportional to the thickness of the clay seams and the magnitude of the stress concentrations (as determined by the extraction ratio).

- The formation of an underground opening creates a large hydraulic sink and groundwater will therefore flow towards the cavity until it is filled and hydraulic gradients are eliminated. If the saturated formation is in fact permeable, significant water inflows should be observed. No such significant inflows have been noted in the potash

mines of southeastern New Mexico. Geophysical Engineers, Inc., (1978) as part of a study to define parametric values for nuclear waste repositories, attempted to back calculate the hydraulic conductivity of bedded salt by estimating flow rates into a number of the potash mines of southeast New Mexico. Flow rates were estimated by evaluating the quantity of precipitated salt on the walls of or roof of the mines. Very low effective permeabilities or hydraulic conductivities for the bedded salt were estimated.

- Numerous salt and potash mines have flooded over the years as a result of shaft fluid ingress suggesting relatively water-tight evaporite formations which are unable to absorb significant quantities of water.

In summary, there is substantial indirect evidence to suggest that bedded salt formations at depth are highly impermeable and that leakage of ingressing shaft water from the mine workings (except by dissolution and this aspect is limited as discussed later) is unlikely. In fact, because the mine constitutes an hydraulic sink until it is filled with water it would be expected that any fluid flow would be towards the voids rather than away from them (assuming the formation itself is saturated).

It has been suggested, however, that evaporate salts are somewhat more porous than indicated by the above evidence and that shaft water entering the tailshafts will flow downwards into the underlying porous bedded salt formation. Should this be the case, then the mine void might never fill with brine as described above (depending on inflow rates and the rate at which water can flow into the formation). If the mine void were never filled then the following scenario (Section 8.3) describing lateral

growth of the cavity by preferential carnallite dissolution, would not take place. Dissolution damage would occur at the point of water ingress to the workings (e.g., tailshaft) but the effect that this action might have on shaft stability depends upon how the migration of fluid into the underlying formations is envisaged. If the shaft inflow rate is sufficiently high that water flows into the mine station areas generally (i.e., is not contained within the tailshaft), then dissolution of the station could result in the impacts described in Section 8.4. If the shaft inflow migrated vertically downwards such that the tailshaft did not overflow into the mine station, then dissolution would still occur but it might not necessarily be of a form which would compromise the stability of the shaft and expose the Salado salts behind the lining to dissolution (as discussed in Section 8.4).

It is not considered that the permeabilities of the bedded salts underlying the Wills-Weaver mine are of a nature similar to that described above and the following scenarios, describing potential impacts which would accompany shaft fluid inflow, are considered realistic. The capacity of the formations underlying the Wills-Weaver mine for transmitting fluid away from the workings could initially be investigated by filling the tailshafts with water and measuring the rate of fluid loss. The No. 3 tailshaft currently contains about 30 feet of water which is apparently showing no signs of flowing into the underlying salts.

8.3 LATERAL GROWTH OF MINE VOIDS

The chlorides of sodium, potassium and magnesium are the principal salts available for dissolution within the mine horizon. These materials occur in the minerals halite, sylvite and carnallite. If all salts were present in sufficient

quantity, carnallite would be preferentially dissolved as discussed in Section 3.0. If the carnallite were not available, then an alternative equilibrium solution of halite and sylvite would develop. An halite/sylvite saturated brine which contacted carnallite would, however, preferentially dissolve the magnesium salt and deposit chlorides of sodium and potassium in the solid phase.

It is estimated that the volume of carnallite available for dissolution within the existing mine void (located principally within the bases of the mine pillars) does not exceed 2 million cubic feet. It was previously determined (Section 8.2) that the volume of water which might ultimately enter the mine void could have a carnallite dissolution capacity in excess of 13 million cubic feet. It is evident that if the mine void water were to be ultimately saturated in magnesium chloride, the great majority of this material would have to originate from carnallite located beyond the limits of the present mine.

Mineral percentages were determined in a number of coreholes surrounding the Wills-Weaver mine, at the level of the mine horizon. The intervals examined and the equivalent thicknesses of pure carnallite within those intervals, are indicated on Figure 8.2. If it is presumed that the nature of the carnallite distribution within this horizon is such that it is virtually all available for dissolution, then an approximate average thickness of 3 inches of pure carnallite is available. In order to satisfy a dissolution requirement of 13 million cubic feet, an area of approximately 52 million square feet, external to the present mine void, would be affected.

Assuming that only those areas with in excess of 10 percent disseminated carnallite will yield material for dissolution

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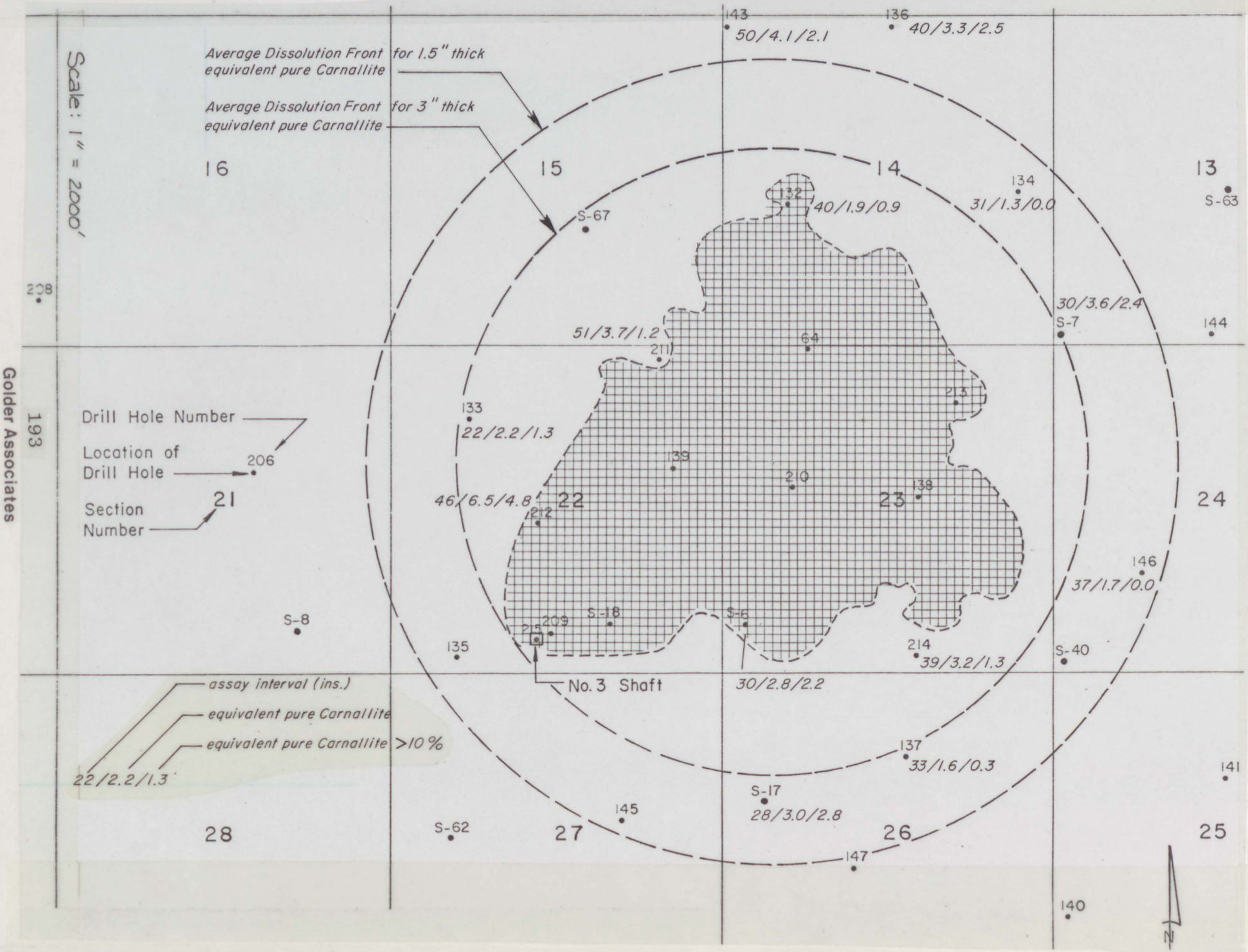
CARNALLITE DISTRIBUTION AND POTENTIAL RANGES OF DISSOLUTION



Golder Associates

Figure 8.2

(i.e., it is assumed that at low carnallite concentrations, the carnallite is enveloped in other less soluble salts such as halite and sylvite), then the equivalent average thickness of pure carnallite which is available is closer to 1.5 inches. For this situation, an area of 104 million square feet of carnallite, external to the existing mine voids, would be required to provide 13 million cubic feet of carnallite for dissolution.

The average areas which could be potentially influenced by carnallite dissolution are shown on Figure 8.2. It is indicated that on average, the dissolution front could extend beyond 2000 to 3500 feet from the existing limits of the mine workings. In specific directions the extent of the dissolution may reach much farther depending on factors such as the distribution of the carnallite, the rate of water ingress and rates of dissolution, the relative densities of carnallite, halite and sylvite saturated brines, etc.

It is instructive to examine in some detail the processes which would occur at the boundaries of the present mine void if that void were filled with a volume of brine having a substantial carnallite dissolution capacity. There are two processes which might be envisaged as inhibiting lateral dissolution of carnallite by carnallite unsaturated brine - brine stratification whereby exposed carnallite is protected from the solvent by a layer of carnallite saturated fluid, and blinding of the exposed carnallite by salt deposition which might accompany carnallite dissolution, depending on the concentration and nature of the salts contained in the solvent prior to its contacting the carnallite.

Protection by brine stratification is totally dependent on the geometric distribution of the dissolving salt together with the relative densities of the various salt saturated brines. For

example, if carnallite were exposed only in the eastern extremity of the mine workings it would be anticipated that brine stratification would quickly protect the carnallite from significant dissolution. Ingressing shaft water would flow down dip towards the east, contact the exposed carnallite and protect the material from further dissolution because of the relatively high specific gravity of the carnallite saturated brine. At Wills-Weaver however, the distribution of the carnallite is such that brine stratification cannot provide a mechanism for halting carnallite dissolution although it can have a very significant effect on the predominant directions in which the dissolution front tends to migrate, i.e., the shape of the final dissolution void.

Blinding of the exposed carnallite by salt deposition is a second process which should be considered in terms of inhibiting carnallite dissolution. If fresh water contacts carnallite, halite and sylvite, it will dissolve a relatively large volume of carnallite (approximately 45 percent of the volume of water) a smaller volume of halite (5 to 10 percent of the volume of water) and deposit a very small amount of sylvite (a few percent of the volume of the water). The net salt dissolution under these circumstances would be of the order of 50 percent of the initial volume of water.

Fluid contacting the carnallite at the extremeties of the mine void would not be "fresh" but would more likely be heavily loaded with salts available for dissolution within the mine floor. Under these circumstances, large volumes of carnallite would still be dissolved but salts would also be precipitated from solution. An halite saturated brine would deposit mainly sodium chloride during carnallite dissolution (with small amounts of potassium chloride also precipitated) and a sylvite saturated

brine would precipitate relatively large amounts of potassium chloride during carnallite dissolution. The net volume of material dissolved at the mine boundary would therefore be reduced compared with the action of fresh water, and if the precipitated volume of salts were of the same order of magnitude as the potentially soluble volume, blinding of the carnallite horizon might be anticipated. It is estimated that for both halite and sylvite saturated brines, the volume of salts which would be precipitated when these brines contacted carnallite would be on the order of 35 to 45 percent of the volume of salts which would be dissolved. Hence, the net dissolution at the mine boundary would be reduced from about 50 percent of the "fresh" water volume as described above to approximately 30 percent of the original volume of fresh water. It is not evident therefore that blinding of the exposed carnallite by solute precipitation can prevent lateral dissolution of carnallite external to the present mine boundaries.

Although it is not possible to predict in detail what the shape of the dissolution void might be, the above analysis suggests that at the Wills-Weaver mine, the dissolution front might eventually extend considerable distances (of the order of 1 to several miles) from the mine limits if water were permitted to migrate into the workings. Because of the distribution and availability of carnallite external to the mine limits, dissolution of the carnallite will not be prevented by brine stratification or by "blinding" as a result of solute precipitation.

8.4 DISSOLUTION DAMAGE IN SHAFTS' VICINITY

The preceeding section has indicated that if shaft water is simply permitted to migrate down the shaft liner, a very substantial volume of comparatively 'fresh' water (from the point

of view of dissolution capacity) would enter the workings and eventually fill what remains of the mine void. The quantity of water would depend on the rate of water ingress and the rate of mine void closure. Although no precise calculations are possible because of the complicated sequence of events which might occur (eg. subsidence damage to shaft lining resulting in increased rates of inflow, dissolution damage to support pillars causing increased rates of mine void closure) an order of magnitude estimate of the total inflow would be from 5 to in excess of 30 million cubic feet of water.

It is possible that these volumes might enter the workings through the No. 3 shaft. At present, this shaft provides the great majority of the water entering mine and is also susceptible to shaft lining cracking from any mine horizon subsidence that occurs within the western end of the Main East Entry panel. Since the ingressing water has very substantial dissolution capacity, it will dissolve very considerable volumes of mine material. At present, it is not considered that any significant dissolution of salts behind the shaft lining is occurring, but this situation might not persist into the future.

At present, the Salado salts in the vicinity of the No. 3 shaft are protected from dissolution by the presence of the lining. For this situation to continue, the lining in the lower portion of the shaft must remain intact. There are two possible methods by which the concrete lining within the Salado might be compromised. First, the lining might suffer shear failure as a result of increasing confining loads as the salt creeps. This problem might also be made worse by progressive loss of the concrete material strength. Second, although the region of the No. 3 shaft within the Salado is currently protected from subsidence damage by the limited mine horizon development in the

immediate vicinity of the shafts, dissolution damage of the mine station could alter this situation.

The probability of ultimate shear failure and cracking of the lower portion of the existing concrete liners as a result of excessive earth pressures, can be briefly considered. The No. 3 shaft has an internal diameter of 14 feet and an average shaft wall thickness within the Salado of approximately 16 inches. The average compressive circumferential stress within the lining is therefore given by:

$\sigma c = 6.5\ p$

σc = circumferential compressive stress
p = radial pressure external to shaft

The maximum external pressure 'p' would equal the overburden pressure and this would imply an ultimate circumferential loading of about 6,000 psi within the lower regions of the lining. In practice the applied external earth pressure 'p' will be a function of the overburden pressure, the time of lining placement after shaft excavation, the material properties of the salt and the time since construction of the lining. This relatively complicated situation is related to the time dependent behavior of salt rocks. When the salt is initially excavated, the differential stresses created within the shaft result in creep of the material and an inward deformation of the shaft wall. With time, the rate of inward deformation decreases as the material hardens. Placement of a concrete lining provides a resistance to this inward movement and develops loads within the lining. The rate of load development will depend primarily on the rate of inward deformation of the salt and may therefore be reduced by delaying lining contruction for some time after excavation. As the lining is loaded, the rate of inward deformation of the salt

is further reduced because the support provided by the concrete liner reduces the deviator stress within the salt.

It is conventionally assumed that ultimately an equilibrium situation is developed where the applied load on the liner is substantially lower than the overburden pressure, i.e., the salt ceases to deform even though it is stressed by differential loads. The phenomenon of long-term creep of salt-like rocks is poorly understood and there are limited suitable data to demonstrate conclusively that such materials cease deforming while under deviatoric loading. If the salt continues to creep until hydrostatic conditions develop, the lining would ultimately be loaded by a circumferential pressure equal to overburden. It is not possible to estimate how long this process might take or even if full hydrostatic pressures ever develop, but it is considered that hydrostatic loading represents a realistic upper bound for the long-term loading of a shaft constructed in salt. In this case, the lower regions of the Wills-Weaver No. 3 shaft might ultimately be subjected to circumferential stresses which are unreasonably high in relation to acceptable design loadings. In addition, although the quality of much of the concrete in the shafts' liners is considered adequate at present, it must be expected to degrade somewhat with time. It is not, therefore, considered reasonable to presume that the long-term integrity of the present shafts' linings within the Salado salts will be guaranteed. There is a substantial probability that the lower regions of the linings in particular might suffer significant deterioration as a result of high confining pressures and a progressive loss of concrete strength.

The above scenario describes a process whereby high stresses/decreasing strength might cause lining failure and expose the Salado salts behind the lining to the dissolution

action of water migrating down the shaft wall. It is also possible for the shaft water itself to induce lining disturbance by causing subsidence within the protective shaft pillar.

If shaft water is permitted to migrate down the shaft in an uncontrolled manner, its initial point of entry to the workings will be the mine shaft station. The very considerable dissolution capacity of such water has already been discussed, and it is evident that a large dissolution void could develop in the vicinity of the mine station. Removal of pillar support within this area would induce subsidence of the overlying strata in which the mine shafts are located. The ground movements would seriously impact the condition of the concrete lining, causing cracking and failure, and once again exposing the salts behind the lining to the dissolution action of the ingressing shaft water.

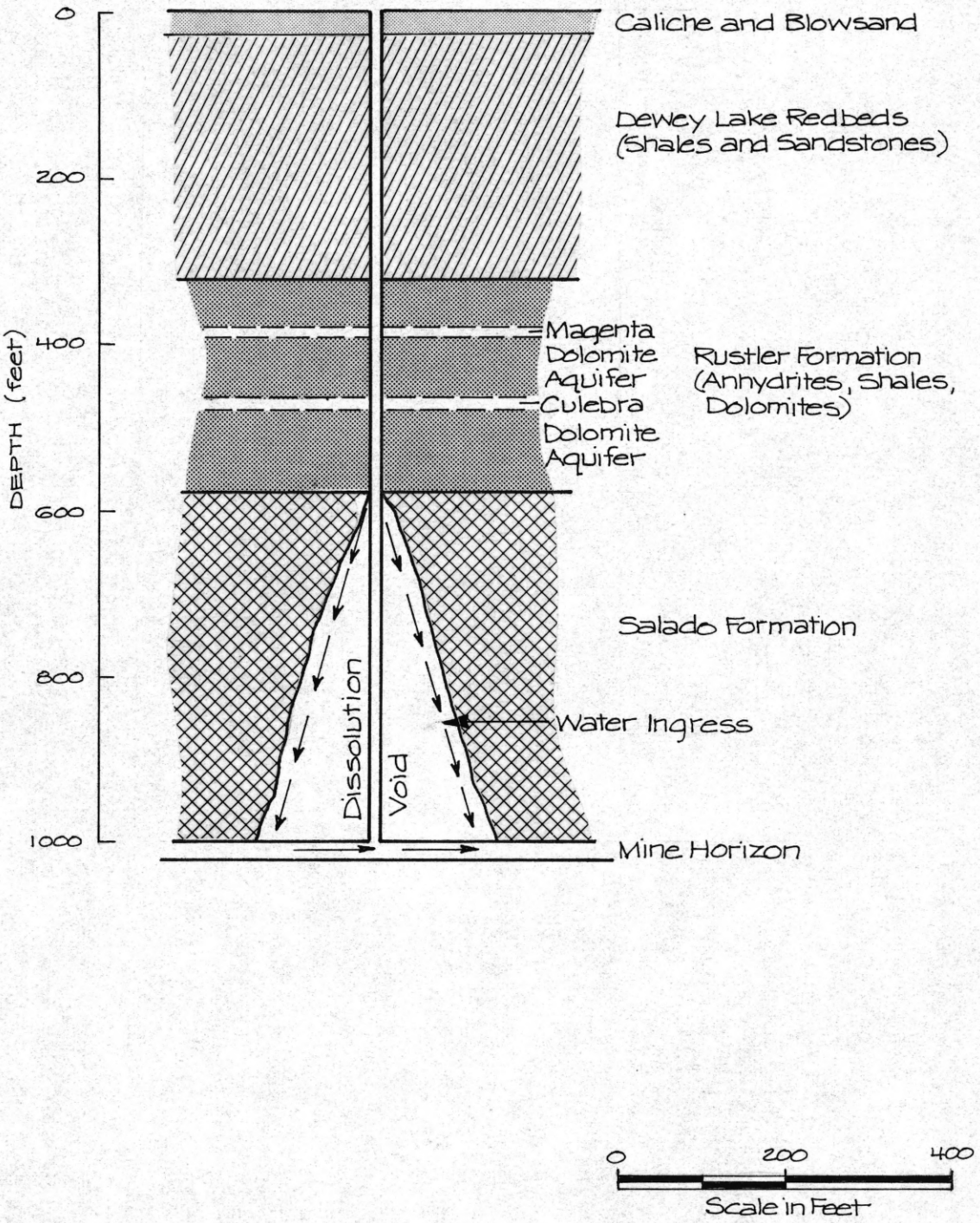
If water is permitted to migrate down the mine shaft(s), the eventual exposure of the salts behind the lining to the shaft water must be anticipated. It has been previously estimated that from 5 to in excess of 30 million cubic feet of water could ultimately enter the mine workings. This volume of water would have an halite dissolution capacity of approximately 1 to 6 million cubic feet.

The upper limit of potential salt dissolution from around the No. 3 shaft could occur under inflow rates of 100 gph and a slowly closing mine void. In order to visualize the volume of salt that could potentially be removed from within the Salado formation surrounding the shaft, a conical void of volume 6 million cubic feet is shown diagramatically on Figure 8.3.

Dissolution of the Salado salts surrounding the lower half of the mine shaft would obviously result in a loss of this

POTENTIAL DISSOLUTION DAMAGE
BEHIND SHAFT LINING

Figure 8-3



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portion of the shaft. In addition, the resulting strata disturbance would provide the maximum opportunity for development of a direct hydraulic connection through to the overlying aquifers (the Culebra member is only 100 feet above the Rustler/Salado contact). This could result in substantial rates of inflow into the mine workings, well in excess of flowrates discussed above. The impact of this would be twofold:

- The total volume of fluid entering the workings prior to flooding would be significantly increased. The total amount of dissolution in the mine voids would be correspondingly increased.

- The capacity for salt dissolution at the point of entry of the water to the mine workings (i.e., around the shaft - Figure 8.3) would increase and an even larger dissolution void would develop.

The formation of such a potentially large dissolution void (of the type indicated on Figure 8.3) in an uncontrolled manner could lead to collapse of the overlying strata surrounding the mine shaft and the propagation of a localized subsidence feature through to the ground surface.

8.5 DISCUSSION OF IMPACTS OF SHAFT FLUID INGRESS

A brief summary of conditions which might develop under various assumptions concerning the water-tightness of the Wills-Weaver mine void, is now presented. Also considered in each case are the shaft abandonment requirements to inhibit the development of the various impacts.

8.5.1 Water-Tight Mine Void

As discussed previously, the scenario involving a watertight mine void which is continuing to close and which will ultimately fill with brine is considered to be the most likely. The potential impacts of a brine filled mine reservoir include probable lateral migration of the void by preferential dissolution of carnallite together with the development of a large dissolution void (possibly accompanied by local subsidence in the vicinity of the shaft) at the point of entry of the water into the workings. In order to minimize the probability of either of these impacts occurring, the shaft abandonment procedure should limit inflow rates for a sufficient length of time so as to minimize the ultimate volume of brine contained within the void and should also provide suitable protection to the mine station area so that dissolution damage does not occur in this region. If these two objectives are met, then significant lateral dissolution migration will not occur and the probability of dissolution of salts at the point of entry (and resulting loss of the shaft) will be minimized.

8.5.2 Leaky Mine Void with Mine Station Dissolution

It has been suggested that clay seams impart an overall mass permeability to the bedded salt deposit and that the mine void will never fill with brine, i.e., ingressing fluid passes out of the mine void. In this event, it would not be anticipated that lateral preferential dissolution of carnallite would occur. It is very likely however, that shaft water migrating into the tailshaft and mine station would cause considerable dissolution within this region, resulting in exposure of the Salado salts behind the shaft lining and the consequences discussed in Section 8.4.

Under these circumstances, it appears impossible to prevent ultimate loss of the mine shaft because no shaft decommissioning procedure can totally prevent fluid ingress and the eventual exposure of the Salado salts behind the lining, by one mechanism or another, is inevitable. In order to delay loss of the shafts, the decommissioning procedure should attempt to prevent mine station dissolution without necessarily inhibiting shaft fluid inflow rates.

8.5.3 Leaky Mine Void without Mine Station Dissolution

Another suggested scenario is that shaft water will enter the tailshaft and flow into the underlying salts without causing significant dissolution of the mine station. For this case, the decommissioning procedure would not be required to exhibit any hydrologic control and simple capping of the shaft would be all that should reasonably be done. For this scenario, the eventual loss of the shaft must again be anticipated as the liner deteriorates and the Salado salts are exposed.

It is not considered that the above scenario is realistic. The ability of the mine station area to transmit ingressing water to the underlying salt without local dissolution which would ultimately cause distress to the shafts, could be evaluated however. Water could be pumped into the tailshaft at realistic rates and the conditions observed. It would not be necessary to continue the experiment to the stage where any significant damage occurred but expected behavior would be very quickly demonstrated. High flow rates into the mine station must ultimately be anticipated once the Culebra seal deteriorates, e.g., one grout hole made in excess of 130 gpm during sinking operations.

8.5.4 Shaft Fluid Impacts

As discussed above, the first impact which is likely to result from uncontrolled shaft fluid ingress is the exposure of the Salado salts behind the shaft liner (perhaps as a result of subsidence caused by dissolution of the shaft pillar protection region) and the development of a large dissolution void in the vicinity of the shafts. This may ultimately be expressed as a significant local surface slump. It is considered that, based on the premise that the mine void is water-tight and will eventually fill, this type of impact can be minimized by inhibiting water inflow and protecting the mine station from dissolution. If the mine void is sufficiently leaky that it will never fill and thus prevent continuing shaft water inflow, then the eventual loss of the shafts (together with the associated impacts) appears inevitable, irrespective of the abandonment procedure.

The second impact of shaft fluid ingress is the ultimate existence of a large body of saturated brine within an underground reservoir. Providing the limits of the reservoir are reasonably well defined, such a feature would not appear to impose undue difficulties on future activities in the region. It is considered that containment of the reservoir can be guaranteed if the decommissioning procedure limits inflow rates for a sufficient period of time such that the ultimate volume of brine within the reservoir is limited.

If such action is not taken, then it is likely that preferential carnallite dissolution will result in considerable lateral growth of the void in hydraulic communication with the original mine workings. This development should be evaluated in terms of potential impacts on:

- existing mining operations
- future mining operations
- current and future oil and gas exploration and production.

The nearest present mining operation and regions of current economic potash reserves are in excess of two miles from the Wills-Weaver mine. Although it is feasible that lateral dissolution could extend to these distances, it would not be anticipated that such an event would take place for considerable time, if at all.

A more significant impact on future conventional mining operations is likely, if future cost/pricing structures indicate that potash resources in the near vicinity of Wills-Weaver are economic. Such operations in the vicinity of a large reservoir of brine, of unknown lateral dimensions, are likely to be severely impacted. The real expected impact on future mining operations in the vicinity of Wills-Weaver will be largely determined by the probability of such mining operations taking place.

There are currently three producing oil wells passing though the Wills-Weaver mine and a number of other wells within one-half mile of the mine boundries. All of these operations, together with any future oil and gas activity in the vicinity of the Wills-Weaver mine could be adversely impacted by lateral dissolution action. The pH of mine water sampled from the Wills-Weaver mine (Table 3.5) was measured at 6.15, and such a solution would be expected to present considerable problems of corrosion to well casings, etc.

8.6 SUMMARY

In designing a suitable abandonment procedure for the Wills-Weaver mine and mine shafts, an important consideration is whether or not shaft water should be permitted to migrate down the shaft wall and into the workings. Resolution of this question defines some of the specific requirements that should be satisfied by the abandonment procedure. The following items are considered pertinent:

- If water is permitted to enter the mine workings, it is estimated that from 5 million to in excess of 30 million cubic feet of water would accumulate within the void. This volume refers to the relatively 'fresh' water derived from the overlying Rustler aquifers. The range of values was derived from a consideration of rates of water inflow and rates of mine void closure, together with an assumption about the water-tightness of the undisturbed bedded salt formation. There is considerable indirect evidence to suggest that such formations are very impermeable at depth and the presumption that the mine shafts represent the most significant potential hydraulic connections to the mine void appears to be well founded.

- The dissolution action of the ingressing water would result in a larger volume of saturated brine existing within the mine (from approximately 7 million to in excess of 50 million cubic feet).

- It is estimated that from one-hundred to several hundred years would be required to fill the mine. At flooding, the hydrostatic pressures developed within the brine

would tend to inhibit the rate of void closure. As the void continued to close, the saturated brine would be expelled from the mine and returned either to the surface or more probably to the geologic strata overlying the Salado, at an estimated rate of 10,000 to 50,000 cubic feet per year. While any brine pollution of the relatively brackish aquifers of the Rustler formation is not desirable, such annual volumes of brine discharge are negligible in relation to current discharges to tailings ponds by the potash mining industry of southeast New Mexico.

- The high solubility of carnallite together with its distribution in relation of the existing mine void, suggest that if the upper limits of water enter the workings, then the dissolution front might extend considerable distances (from one to several miles) from the outer limits of the workings. This would provide a relatively large area at the mine horizon level which would be in hydraulic communication with the water reservoir in the mine. It is not anticipated that brine stratification or "blinding" of the mine boundaries by solute precipitation will prevent lateral preferential dissolution.

- Although it is feasible that adjacent existing mining operations could be impacted by the lateral dissolution action, it is not considered likely. The relatively large distances of such mining activity from the Wills-Weaver mine (in excess of two miles), together with the expected slow rate of advance of a dissolution front, should not make this an item of immediate

concern. Future mining in the near vicinity of Wills-Weaver could be adversely impacted by brine and existing and future oil or gas wells within or close to the mine might be impacted by brine attack of the well casings.

- It is considered that the above impacts could be eliminated by abandoning the shafts in such a manner that inflow rates were maintained at a sufficiently low rate for a sufficiently long time in order that the ultimate volume of brine contained within the mine void was limited.

- If the mine void is envisaged as having sufficient hydraulic connectivity to a porous bedded salt formation so as to prevent the void from filling, then the above impacts would not be expected to develop. It is not considered likely that this scenario represents reality.

- The potential for massive dissolution damage around the shafts within the Salado salts is considerable. The linings which are currently protecting these salts from dissolution could be expected to degrade as a result of:

  - development of excessive earth pressures and/or deterioration of concrete strength

  - dissolution of the mine shaft station with subsidence of the strata through which the shafts are constructed (if water enters the mine station).

The deterioration of the shaft lining would expose the Salado salts to the migrating water, leading to loss of the lower portion of the shaft and the generation of a substantial dissolution void. This void might result in strata disturbance to the overlying Rustler sediments, with an hydraulic connection developing directly with the Culebra dolomite aquifer. Rates of inflow would then increase and the situation would deteriorate more rapidly.

- It is possible that local subsidence in the vicinity of the shafts might propagate through to the surface.

- The potential size of a dissolution void in the vicinity of the mine shafts will be determined in part by the quantity of water which enters the mine void and by the degree of exposure of the Salado salts behind the shaft lining to the ingressing water. Since it is anticipated that shaft water inflow will cease when the mine void is filled, the abandonment procedure should protect the mine station from dissolution damage in order that subsidence in this area does not prematurely expose the Salado salts to the ingressing water. Such action will minimize potential damage to the shafts and the dissolution of the Salado salts overlying the mine horizon.

- If it is assumed that the mine void will never fill as a result of seepage into the underlying salt, then the mine shafts will represent a continual source of water discharge into the mine horizon. In this case, the eventual exposure of the Salado salts behind the shaft liners must be anticipated and a large dissolution void is likely to be generated. It is not considered that this represents the most likely condition.

- The potential rate of water inflow from the Rustler aquifers is substantial. Although comparatively limited inflows are presently occurring (20-40 gph), deterioration of the aquifer seals may lead to inflows 2 to 3 orders of magnitude greater than current rates. Evidence for this behavior was provided by high inflow rates through grout injection holes during shaft sinking operations (e.g., in excess of 130 gpm from one hole).

- The above discussion defines the potential impacts of permitting shaft water to migrate into the mine workings. It is considered that for short-term abandonment, where the understanding is that the shafts will be utilized at a later date, it will be necessary to ensure that significant flow rates do not enter the mine station through the shafts. For permanent abandonment, the decision as to whether or not to attempt to hydraulically seal the shafts will be determined by technical feasibility, cost effectiveness, and the 'value' of avoiding the above impacts.

- It is believed that the mine void will eventually fill with saturated brine as a result of shaft fluid ingress. In order to prevent the development of a large volume of brine within the mine void (and hence minimize lateral dissolution propogation, etc.) the technical requirements of the permanent abandonment procedure should be to limit inflow rates for a requisite period of time and to protect the mine station area from dissolution damage. If cost considerations dictate that such criteria cannot be satisfied, the abandonment procedure should permit fluid ingress but should attempt to protect the mine station area from dissolution damage.

This action would minimize the possibility of premature exposure of the Salado salts behind the shaft liners with the subsequent development of a large solution void. When the mine void had filled with brine, shaft inflow would cease and no further dissolution in the area of the shafts would occur.

9.0 MINE SHAFT ABANDONMENT PROCEDURES

9.1 INTRODUCTION

Mine shaft abandonment procedures must be considered for two types of mine abandonment: permanent abandonment of the workings and short-term or temporary abandonment which provides an option for relatively easy recommissioning of the shafts. Short-term abandonment methods will be appropriate if a future utilization of the mine voids or mine shafts is envisaged within a time period sufficiently short that the concrete shaft liners will remain in usable condition. Permanent and temporary abandonment procedures should not be considered in isolation. First, any shaft which is abandoned in a temporary manner will eventually require permanent abandonment procedures and the temporary procedures should not therefore adversely impact the final decommissioning of the shafts. Second, certain procedures which are acceptable for permanent abandonment may also be suitable for temporary abandonment--such procedures may not be optimum from a recommissioning point of view but may still provide the best overall solution; i.e., if the shafts are not used within a specified period of time, there is no need to subsequently apply permanent abandonment procedures to the shafts. Third, it may be be appropriate to apply different abandonment procedures to the two shafts. This might include applying to one shaft an abandonment procedure which is considered permanent in nature and an alternative temporary procedure to the remaining shaft. In addition to maintaining relatively easy access to the workings, this solution to the problem would also enable the efficacy of the permanent abandonment method to be evaluated prior to final decommissioning of the second shaft.

The optimum mine shaft abandonment method(s) should perform satisfactorily in the following general areas:

- Hydrological control
- Structural stability and control
- Chemical stability or durability
- Economic viability.

Each decommissioning method (defined by configuration, installation procedure, materials, etc.) will be evaluated as to acceptable performance in each of the above areas. Technical suitability and cost effectiveness will establish the optimum overall abandonment procedure (temporary and permanent) to be applied to the Wills-Weaver mine shafts.

It is appropriate to consider initially the procedures which might be suitable for permanent abandonment of the Wills-Weaver mine shafts, since such methods must eventually be applied to the shafts. Subsequently, any modifications or alternatives which might be suitable for temporary abandonment will be discussed.

9.2 PERMANENT ABANDONMENT METHODS

Shaft abandonment procedures in relation to Wills-Weaver may be considered in terms of two broad subdivisions:

- Those which inhibit shaft water ingress into the workings

- Those which permit shaft fluid to enter the mine.

The question of shaft water control is fundamental to deciding what constitutes the optimum abandonment method. Section 8.0 has outlined the potential consequences of allowing water to migrate into the mine via the shafts. On the basis of

the above impact study, it is considered technically desirable that the method of shaft abandonment should prevent or inhibit water ingress, and the following discussion is specifically directed towards such shaft abandonment procedures. Optimal solutions from a technical point of view may, however, be unacceptable from a cost consideration. The following analysis therefore details approximate costs associated with abandonment procedures which provide adequate shaft water control. These data will assist in evaluating whether or not shaft water control should constitute an integral part of the abandonment scheme. Subsequently, some considerations for abandonment methods which do not control the shaft water (or provide a lower level of control) are discussed.

It is therefore considered initially that the mine shaft permanent abandonment procedure is required to prevent entry of water into the workings and onto the shaft liner located beneath the Rustler/Salado contact. There are two basic approaches towards achieving this requirement:

- Placement of an impermeable shaft plug within the shaft, with or without backfill

- Prevention of water entry into the shaft by grouting of the aquifers, i.e., the shaft lining and formation grouting provide the water barrier and no impermeable shaft plug is utilized.

With respect to the latter of the above alternatives, the following points should be considered.

- Considerable grouting of the aquifers was carried out during shaft sinking, and this did not completely seal the No. 3 shaft in particular.

- A limited effort was also made to grout off the water inflow following construction of the No. 3 shaft, but this was unsuccessful.

- Over a long period of time, any significant subsidence occurring within the western end of the Main East Entry panel could potentially create new inflow paths into the No. 3 shaft in particular, even if grouting were able to seal the shaft completely.

- The shaft lining presents a very large surface area, a significant portion of which may be connected to water-bearing zones by excavation-induced fractures behind the lining, etc. It is technically a more difficult problem to ensure a permanent, water-tight seal behind a large area of concrete lining than it is to construct a suitable limited-volume water seal at a specific location.

- The concrete forming the shaft liner is subjected to the corrosive action of sulfates. There is some evidence to date to suggest concrete deterioration by sulfate attack over certain parts of the linings, and there are substantial uncertainties about the long-term performance of the concrete as an effective water seal.

- Similar doubts about the long-term durability of cement grouts which may be used in formation sealing exist. The impacts of grout deterioration on hydrological control within an intrinsically low-permeability rock mass will be less than within a relatively high-permeability system. It is not desirable, therefore, that grouting of the high permeability aquifers constitutes the major

hydrologic barrier to water ingress into shafts (for permanent abandonment).

On the basis of the above considerations, any permanent abandonment procedure for the Wills-Weaver mine shafts should incorporate a shaft plug designed to effect an hydraulic seal. The following discussion of appropriate permanent abandonment methods is therefore limited initially in scope to procedures which provide an engineered hydraulic seal or plug. Methods which do not provide such a seal (e.g., simple concrete shaft cap, backfill with permeable material, etc.) are considered later.

Hydraulic shaft seals which might be appropriate to the Wills-Weaver mine shafts (i.e., configuration, installation procedures, materials) are subsequently described and evaluated with respect to hydrologic control, structural stability and control, chemical stability, and cost economy.

9.2.1 Hydrologic Control

9.2.1.1 General Considerations

The prevention of water ingress into a mine shaft and mine workings can be considered in two separate phases. The first phase occurs during shaft construction and its purpose is to enable shaft construction to proceed and to control the entry of water during the operating life of the mine. There are potentially three specific elements comprising this type of barrier to water inflow:

- The shaft lining itself
- The sealing of specific aquifer zones
- General sealing of the interface between the shaft lining and the rock formation.

Monolithic concrete liners such as employed in the Wills-Weaver mine shafts comprise the most common type of shaft liner employed today. With respect to water ingress, the main disadvantage of such liners is that the joints between pours can leak under high water pressures of several hundred psi. Such leaks can usually be controlled by grout injection. Leakage of concrete joints does not appear to be a feature of the Wills-Weaver shaft liners where maximum water pressures behind the lining are generally less than 100 psi.

Sealing of specific aquifer zones (generally ahead of shaft construction) and using grouting or freezing techniques is necessary to enable construction to proceed. Cement grout was used to seal the Magenta and Culebra dolomite aquifers during sealing of the Wills-Weaver shafts (Section 6.0), and this was largely successful, although total seals were not obtained.

During shaft sinking a fracture zone is created around the shaft, and this will generally have largest dimensions for drilling and blasting methods such as were employed at Wills-Weaver. This zone of disturbance provides relatively good hydraulic communication along the outside of the concrete lining and, if it intersects an aquifer which has not been adequately sealed, a reservoir of water can be developed on the exterior of a large portion of the liner. In addition to fractures within the rock mass, a void space may exist between the shaft lining and the rock formation as a result of shrinkage of the liner. In general, no attempt is made to seal the general fracture zone around a shaft liner because of the unfavorable economics and because it is generally not necessary. Instead, the major sources of water, i.e., aquifer zones, are sealed. In accordance with normal practice, no attempt was made to provide a general interface seal at the Wills-Weaver mine shafts.

At the Wills-Weaver mine, the above sealing procedures have enabled safe operation of the mine during its working life. A complete seal within the No. 3 shaft was not achieved, but water inflow was very easily controlled. Inflow rates into the No. 4 shaft were so small that the water was removed by evaporation. The previous discussion has indicated, however, that such a seal is not adequate for permanent abandonment of the workings and a second phase of sealing is required before the mine can be permanently decommissioned. In order to prevent or limit the ingress of water down the shaft and into the workings in the long term, the shaft decommissioning method should include appropriately positioned and adequately designed hydraulic plugs or seals. In general, the overall configuration of such hydraulic seals consists of plugs located at specific intervals, complete backfilling of the shaft, or of some combination of the two. The concept of sealing shafts with a shaft plug embodies the principle of an engineered structure performing to specified design standards and placed in one or more locations down the length of the shaft. The volume between the plugs may then be left open or sealed with natural materials in various forms and probably to less critical standards. Figure 9.1 illustrates the general concept of shaft plugs in achieving hydrologic isolation of various regions of the shaft.

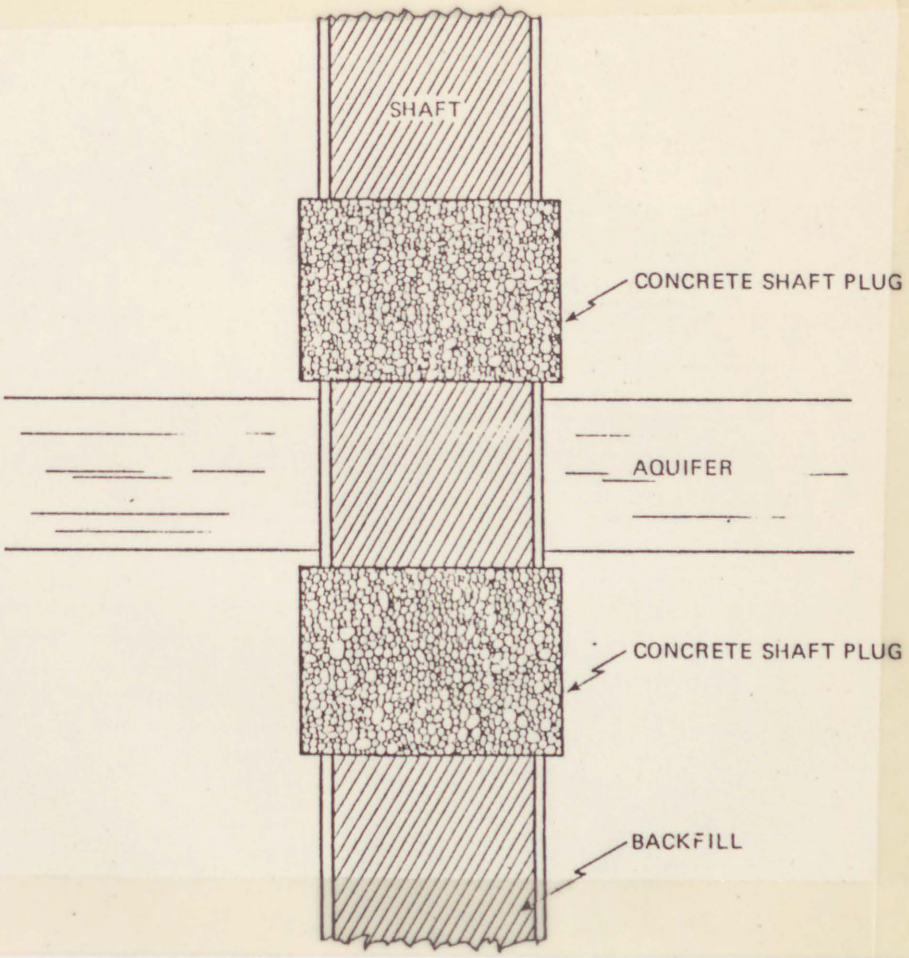
The following discussion presents an evaluation of important factors with respect to hydrologic control exhibited by hydraulic plugs or seals which might be suitable for the Wills-Weaver mine shafts.

9.2.1.2 Hydraulic Seal Location

The economics of shaft sealing will generally dictate that the entire shaft is not sealed to a uniform standard. There

EXAMPLE OF THE USE OF CONCRETE PLUGS IN A SHAFT

Figure 9-1



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will be isolated locations at which adequate hydraulic seals or plugs are sited. These may be engineered structures within an otherwise open shaft, or the shaft may be completely backfilled. The shaft plug locations within a backfilled shaft will be characterized by different materials and by greater attention to sealing the plug/rock interface by appropriate grouting procedures.

The optimum locations for shaft plugs within a shaft will be largely determined by the hydrology and geology of the formations through which the shaft is constructed, together with the function or purpose of the plug itself. A primary function of the Wills-Weaver shaft sealing is to prevent water from migrating down the shaft lining and into the mine voids. The optimum location of the shaft plug designed to achieve this function will be fixed by a number of considerations.

- The plug must be located beneath the lowest point of water ingress into the shaft from the aquifers overlying the Salado salts. At Wills-Weaver, this corresponds to the location of the Culebra dolomite aquifer. The plug should not, therefore, be located above approximate elevation 500 feet beneath the shaft collar, and potential leakage past the plug by hydraulic connection external to the lining must be sealed by grouting.

- The plug should be located so as to minimize imposed hydraulic heads in order that structural loadings and plug leakage be minimized. This condition requires that the plug be located at the highest elevation possible within the shaft, consistent with other requirements. All other things being equal, a shaft plug

located at the mine station would be subjected to substantially more severe loadings than a plug located at a depth of approximately 500 feet.

- The plug should be located within a stable environment in order to minimize the possibility of structural damage to the plug. There are two areas within the Wills-Weaver mine shafts which should be regarded as potentially unstable in the long term.

  - The Salado salts may continue to creep with time, imposing high loads upon the lining and resulting in cracking and deterioration. The plug should not therefore be located within the Salado formation or closer than necessary to the top of the salt.

  - General closure of the mining horizon may eventually result in some disturbance to the strata surrounding the No. 3 shaft in particular. It is estimated (Section 4.0) that for the No. 3 shaft, this potential zone of strata disturbance and subsequent lining distress extends from the shaft collar to an approximate depth of 400 feet.

  It should be noted that this assumes that very little subsidence damage will occur as a result of deformation within the low extraction area surrounding the mine shafts. This will be the case providing dissolution damage of this area does not occur. If dissolution does occur, then there is a substantial potential for significant subsidence damage to the mine shafts.

- The plug should be located so as to prevent water from contacting the lining within the Salado formation. Any

deterioration of the lining would permit access of the water to the highly soluble salts with the potential for significant dissolution damage. This criterion requires that the shaft plug be located above the Rustler/Salado contact.

- It will be necessary to key the plug into or through the shaft lining in order to achieve structural support and/or optimize an hydraulic seal. The depth of the key will be a function of the observed condition of the concrete liner and the expected performance of the concrete in the future. Because the steel reinforcing constitutes a path for potential water migration past the seal, the reinforcing should be removed within the vicinity of the plug. If it is considered that the concrete liner would constitute a weak point in the water seal, it will be necessary to remove the lining completely through to bedrock. The following points should be considered in defining the exact location of the plug:

  - Within the possible range of locations for the shaft plug (i.e., beneath the Culebra aquifer and above the top of the Salado) there exist two types of geological material, i.e., medium soft to medium hard shales and a hard fine-grained anhydrite (the latter is located between depths 556 and 566 feet in the No. 3 shaft).

  - The anhydrite suffers from the disadvantage that it is located close to the top of the Salado. It is considered that the reliability of the formation seal around the plug will be enhanced if the dis-

tance between the plug and the Salado salts is kept to a maximum.

- It is probably preferable, however, to locate a stiff concrete shaft plug within the hard, fine-grained anhydrite rather than within the overlying shales which would offer less satisfactory structural support to the plug. The process of excavating a plug hitch within the competent anhydrite would probably create a significant fracture zone within the material, but this could be sealed by grouting following plug construction.

- A soft clay plug, however, should be preferentially placed within the shales overlying the lower anhydrite (i.e., between depths 500 and 550 feet). The plug hitch should be carefully excavated to minimize disturbance to the rock, and the clay compacted to form a sound plug/rock contact.

- It is also desirable to locate the plug within rock formations of low in situ permeability. This approach will minimize the importance of formation grouting as part of the water seal and hence minimize the significance of potential sulfate deterioration of the grout.

The above criteria fix the location of the shaft plug, designed to prevent water ingress into the workings, within quite narrow limits. The plug should be located beneath the base of the Culebra dolomite aquifer and above the Rustler/Salado contact. A soft clay plug should be located within the shales of the lower Rustler Formation, between approximate depths 500 and 550 feet (Figure 9-2). A relatively stiff concrete shaft plug

requires more competent structural support and should preferentially be located in the hard fine-grained anhydrite which occurs at depth 556 to 566 feet in the No. 3 shaft.

Apart from plugs to prevent mine water ingress, there are other potential shaft plug locations within the Wills-Weaver mine shafts, e.g., above the Culebra and below the Magenta dolomite aquifers, to seal the aquifers from each other, and above the Magenta dolomite to isolate the sediments of the Dewey Lake Redbeds from the brackish waters of the Rustler aquifers. It is not considered necessary to provide shaft plugs at these locations for the following reasons.

- While the Culebra aquifer is somewhat more brackish than the Magenta aquifer, the quality of water in both aquifers is poor (TDS in excess of 10,000 mg/l) and unsuitable for human or stock consumption.

- The Magenta aquifer is poorly developed locally and yields only minor amounts of water in the region of the Wills-Weaver mine site.

- There are no known aquifers within the Dewey Lake Redbed sediments.

It is therefore unlikely that there will be any significant adverse hydrological impact resulting from the limited water mixing which will occur if the aquifers are not sealed from each other and from the overlying sediments. Shaft plugs to provide hydraulic seals at these locations are not, therefore, considered necessary.

9.2.1.3 Shaft Plug Materials

The design of the shaft plug and its method of construction will depend on the anticipated hydraulic pressure that will act on the plug together with the allowable leakage past the plug. The shaft seal may suffer hydraulic failure at the following locations:

- Through the plug
- At the plug/rock contact
- Through the rock adjacent the plug.

Each of these areas should be considered as an integral part of the total seal and the materials which might be used to constitute the plug itself and to seal the adjacent rock formations are subsequently considered with a view to their hydrologic characteristics.

Formation Grouting

The majority of hydraulic seal failures which have occurred within the mining industry are due to flow through the rock surrounding the shaft plug. Consequently, procedures and materials employed to seal the surrounding rock are of utmost importance to the efficacy of the sealing operation. The most practicable solution to sealing the formations surrounding the shaft lining and shaft plug is grout injection. A wide variety of materials may be injected into the rock, but long experience (Garrett et al. 1961) has confirmed that satisfactory results are generally obtained with cement grouts (i.e., normal Portland cements, aluminous cement, and other specialized products). These grouts can be forced into relatively fine cracks and have good capacity to bond with the surrounding rock.

In certain cases where the rock is very finely fractured, it may not be possible to seal effectively with cement. In this situation, chemical sealants (e.g., silica gels) are required. These are water-thin grouts that will enter a porous formation, but have low strength when set up. Consequently, they are most suitable for low permeability rock masses containing no large fissures. High-strength cement grouts can be used in conjunction with chemical grouts to provide adequate strength in specific areas.

Experience with grouting (Garrett et al. 1961) indicates that in hard rock, the grouting pressure must be carried to at least 1.5 to 2.5 times the static water pressure. It is also indicated that the zone around a shaft liner should be treated to a minimum depth of about 20 feet. The permeability of the sealed rock, if adequately grouted, will be quite low and probably within the range 10^{-6} to 10^{-10} cm/sec.

The geological strata in which the shafts' plugs will be located at Wills-Weaver have a low in situ vertical permeability, and installation of the plugs should be carried out in a manner which preserves this situation. The grouting procedure and materials must be carefully designed with regard to the nature of the material to be sealed, e.g., size of fissures, chemical environment.

Concrete

Concrete is an adaptable material which can be engineered through its composition and the use of additives to meet a wide range of characteristics and may provide a suitable shaft plug. The behavior of concrete and the effects of cement additives have been extensively studied and documented in the civil engi-

reering literature. The general procedure for placing a concrete shaft plug would probably include the following:

- Construct bulkhead(s), probably of reinforced concrete
- Pump in concrete (possibly under pressure)
- Pressure grout contact zone between concrete and rock.

The permeability of concrete depends primarily on the type and gradation of the aggregate, the amount of cement, the amount of water, the conditions during curing, and the type of additives. The various types and mixtures of concrete exhibit permeabilities ranging from 10^{-6} to 10^{-10} cm/sec. The lower permeabilities are associated with concrete containing more cement, less water, and nonporous aggregates (Portland Cement Association 1975). Air entrainment procedures can also be effective in reducing the permeability of concrete.

During curing, concrete will shrink about 0.1 percent. However, expanding cements are available to reduce shrinkage and even develop a net swell during curing. In addition, grouting of the plug/rock contact zone can be effective in sealing any shrinkage voids.

Clay

An alternate approach to the use of an engineered material such as concrete to provide a shaft seal is the use of natural materials of low permeability. The use of clay fills may provide suitable hydraulic seals for the shafts. Alternatively, artificial mixtures of sand and bentonite (Karn-Bransle-Sakerket 1977) might be employed.

The permeability of a compacted clay depends on the following:

- Type and percent of clay mineral present
- Compacted density or void ratio
- Compaction procedure, including type of compaction and moisture content.

Figure 9-3 indicates the variability of permeability with void ratio for various types of clay. Figure 9-4 shows the effects of compaction procedures, indicating that permeability decreases for higher compacted densities and high compacted water content. In general, the permeabilities of compacted clays range from about 10^{-5} to 10^{-9} cm/sec. In order to achieve optimum material hydrological properties, mechanical placement of the clay plug to specified engineered standards would be required. Use of suitable filter zones between the clay plug and shaft backfill material would be necessary to prevent internal erosion of the plug.

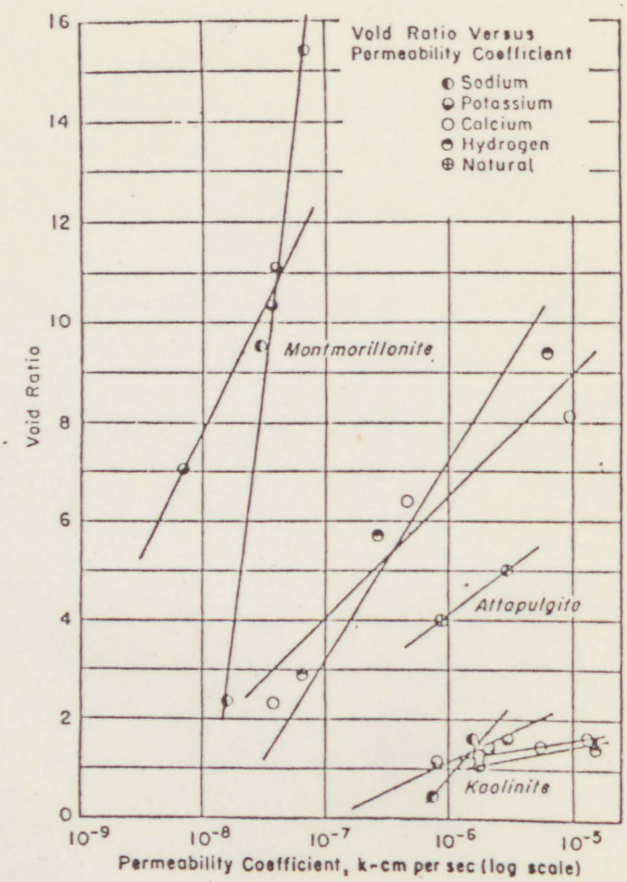
9.2.1.4 In Situ Experience with Sealing Shafts and Tunnels

Experience with plugging underground tunnels and shafts has been gained in sealing against high pressure water inflow (Garrett et al. 1961) and in constructing gas-tight seals for storage of natural gas in abandoned coal or salt mines (Kelly 1961, Buttiens 1977, Buttner et al. 1974).

Substantial experience with the construction of water barriers (plugs) has been obtained in the deep gold mines of South Africa (Garrett et al. 1961). Although the theory of design of such plugs is not well developed, there is general agreement that the most critical factor is the watertightness of the rock

PERMEABILITY OF VARIOUS REMOLDED CLAY SAMPLES

Figure 9-3



Ref. Lambe, 1954

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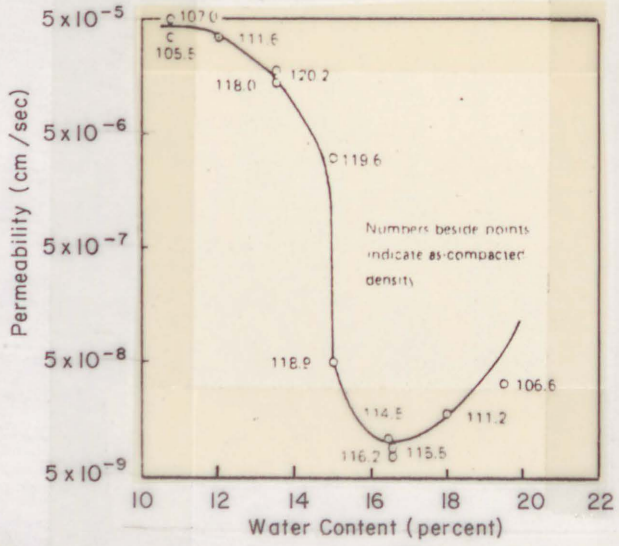
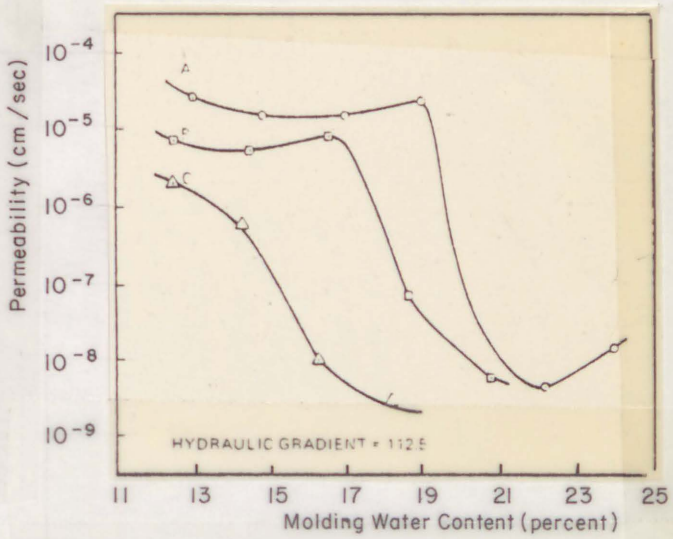
PERMEABILITY AT 100 % SATURATION FOR SAMPLES OF SILTY CLAY PREPARED AT 3 COMPACTIVE EFFORTS

Figure 9-4





(a) PERMEABILITY vs. WATER CONTENT FOR AASHO ROAD TEST SOIL - KNEADING COMPACTION

(b) PERMEABILITY vs. MOLDING CONTENT FOR SAMPLES OF SILTY CLAY PREPARED AT 3 COMPACTIVE EFFORTS

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and the plug/rock interface. Tests have indicated that the rock/concrete contact can be made watertight against applied pressures of 1,750 psi by cement grouting to 3,000 psi. The conclusion is drawn that to minimize water leakage it is necessary to design plugs of greater length than would be required for purely structural reasons. Table 9-1 presents some typical data on plug dimensions and pressure gradients for deep gold mines in South Africa.

For plugs located approximately 200 to 250 feet below the water table such as is recommended at the Wills-Weaver mine shafts, the applied pressure on the upper surface of the plug will be of the order of 100 psi. Referring to Table 9-1 and considering plugs with a total applied pressure of the order of 100 psi, the plug lengths are seen to be generally of similar dimensions to the lateral dimensions of the openings. The applied pressure gradients across these plugs vary from about 4 to 20 psi per foot length. It should be noted that the above experience refers to concrete plugs sealed by cement grouting of the surrounding rock and plug/rock interface.

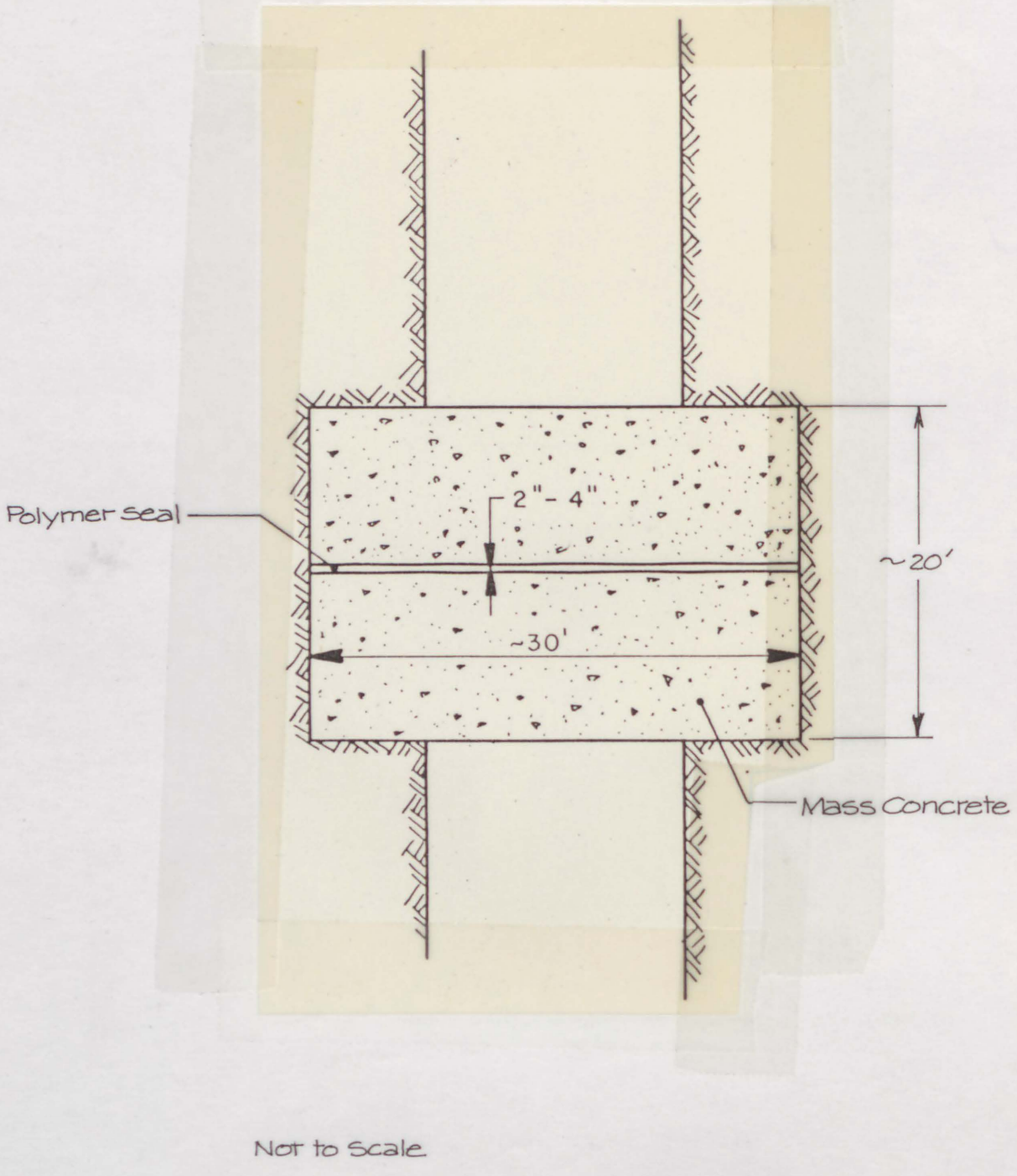
The sealing of large cross-section shafts against high pressure gas leakage generally demands better performance seals, due to higher permeability of the rock formation to gas compared to water. The most common seal design consists of a concrete plug constructed in the shaft. Either reinforced or mass concrete construction is used, depending on permeability and load requirements. Specific sealing designs vary, depending on the nature of the product stored, but seals are generally a variation of the basic concrete bulkhead (Figure 9-5), using chemical sealants, polymer materials, etc. Other types of specialized seals for high pressure containment (such as moving waterbed systems) have no application to the Wills-Weaver problem.

TABLE 9-1
TYPICAL PLUG DIMENSIONS AND PRESSURES

MINE	PLUG LENGTH (FT)	OPENING DIMENSIONS (FT X FT)	PRESSURE (PSI)	PRESSURE GRADIENT (PSI/FT)	LEAKAGE (GPH)
C.M.R. No. 6	17	13 x 12	360	21.2	est. 400
Free State Geduld	27	9 x 8	828	30.7	---
Free State Geduld	100	47 x 11	2,250	22.5	---
East Daggafontein	28	20 x 17	650	23.2	est. 800
West Driefontein	8	4 x 4	6,800	887.0	300/400 heavy grouting
West Driefontein	42	13 x 10	1,827	43.5	est. 200
Virginia	63	14 x 10	1,650	26.2	---
Virginia	36	12 x 11	1,340	39.2	---
Virginia	12	12 x 11	1,340	111.7	est. 1,000
City Deep	12	10 x 12	100	8.3	---
Crown	16	10 x 6	111	6.9	---
Sub Nigel	11	8 x 11	199	18.0	---
Govt. GM Areas	5	8 x 6	100	20.0	---
Durban Deep	4	9 x 9	70	17.5	---
Modder East	6	6 x 11	80	13.3	---
Randfontein Estates	25	14 x 15	93	3.5	---
Rose Deep	20	8 x 11	125	6.3	---

UNDERGROUND CRUDE OIL STORAGE
BULKHEAD AND SEAL

Figure 9-5



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Clay plugs are not conventionally employed for sealing shafts or tunnels against substantial water heads, and there is consequently less relevant field experience. There is no doubt that sufficiently low permeability clay plugs can be constructed to effect an adequate seal, but there are specific problems which deserve special consideration. One such problem is the question of possible internal erosion of the plug caused by relatively high head gradients across the plug. The area in which most experience of this type exists is in the construction of thin clay core dams. Although this subject does not lend itself to theoretical treatment, typical minimum thicknesses of the clay plug would be of the order of 10 percent of the hydraulic head, providing adequate filter zones are employed. At Wills-Weaver, where hydraulic heads of the order of 200 to 250 feet will be exerted on the plug, the plug thickness would be typically 20 feet.

9.2.1.5 Backfill

An alternative approach to sealing a mine shaft with specifically located shaft plugs would be to backfill the entire shaft. A simple extension of the procedures used to construct the shaft plug (i.e., use of concrete or natural earth materials placed to engineered standards) would generally be prohibitively expensive and entirely unnecessary.

In general, therefore, the use of backfill within a mine shaft has little to do with hydrologic control. Control of shaft water is accomplished by shaft plugs, and backfill may be employed to provide structural support of the plug or to control shaft wall stability. Typical materials employed as shaft backfill include waste rock or sand, and these materials have relatively high permeabilities.

At the Wills-Weaver mine site, a most obvious supply of shaft backfill material is the waste stockpile adjacent to the mine shafts. This stockpile consists of gobbed halite, sylvite, and carnallite, and its high solubility would make it a most undesirable material for utilization in an applictaion involving the sealing of relatively "fresh" groundwater.

9.2.2 Structural Stability and Control

9.2.2.1 General Considerations

A mine shaft sealing program may consist of placing two types of structures within the shaft--a shaft plug specifically for hydrologic control and shaft backfill. Each of these components of the shaft seal must satisfy certain structural requirements.

If the shaft plug is constructed of concrete, it must:

- Develop adequate bond strength to support the hydraulic loading
- Develop an allowable stress distribution within the plug so that failure of the plug itself does not occur.

If the impermeable shaft plug consists of compacted earth materials, such materials have no capacity for significant structural action and must be supported either by concrete bulkheads or by underlying backfill.

Shaft backfill, in addition to supporting the shaft plug and its applied hydraulic loads, might also be required to provide some level of support to the shaft lining.

9.2.2.2 Shaft Plugs

Concrete

As mentioned previously, there are few comprehensive and standard mathematical or empirical procedures for the design of underground hydraulic plugs. Hamdi (1975) considers the principal structural design parameter for underground pressure plugs to be the shear stresses that develop inside the plug adjacent to the plug/rock contact. The following parameters are considered by Hamdi to be important in the design of pressure plugs:

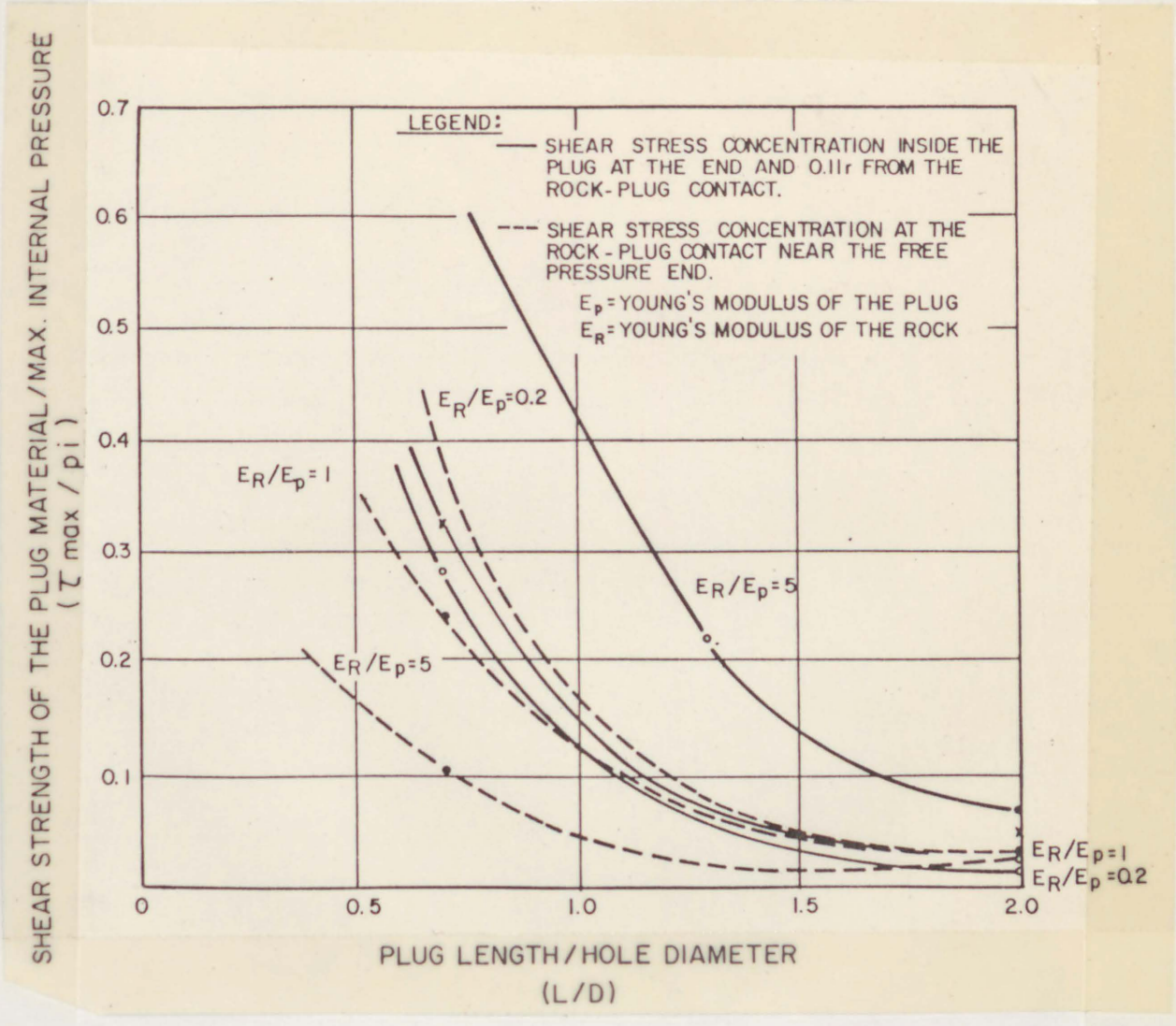
- Plug length
- Diameter of opening
- Pressure applied to plug
- Young's modulus of rock surrounding the plug
- Young's modulus of plug material
- Shearing strength of the plug material

The relationship between these parameters, developed by Hamdi, is shown on Figure 9-6.

Experience with concrete pressure plugs in the South African gold mines (Garrett et al. 1961) suggests that structural design considerations are generally secondary to hydraulic sealing considerations. Examination of the dimensions of and the pressures imposed on these mine plugs indicates that the contact shear stresses between the plug and the rock are low--generally of the order of 20 to 40 psi for low pressure plugs (see Table 9-1). From a structural viewpoint, therefore, it appears as though hydraulic plugs could be of considerably less thickness and still adequately resist the applied water pressures. There are two other factors to consider, however.

DESIGN CURVES FOR LENGTH PROPERTY OPTIMIZATION OF SHAFT PLUGS

Figure 9-6



Ref. HAMDI, 1975

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- Placing a relatively thin plug lowers the probability of effecting a watertight seal in the surrounding formation by appropriate grouting.

- A relatively thin concrete plug would be subjected to substantial bending (tensile) stresses and would require reinforcing. The use of reinforced concrete in a corrosive environment to provide a long-term hydraulic seal appears to exhibit significant disadvantages over the use of an adequately designed mass concrete plug.

As with permeability, the mechanical behavior of concrete varies significantly, depending on its composition and condition during curing. In general, strength and rigidity increase with increasing cement content and curing time. Average values of unconfined compressive strength range from about 2,000 to 6,000 psi, and values of elastic modulus range from about 2×10^6 to 6×10^6 psi. Thus, concrete has deformation characteristics similar to most intact rocks. The relatively good stiffness of the concrete would be effective in resisting any tendency of the shaft lining to close in the vicinity of the plug, and this would help maintain the watertightness of the rock mass surrounding the plug.

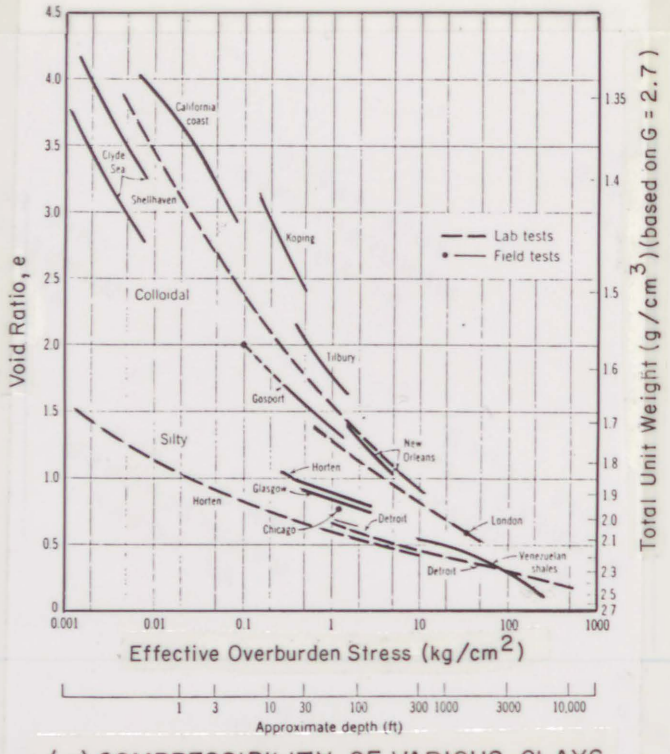
Clay

Since clay has little in situ structural strength, a shaft plug constructed of clay would require structural support, e.g., concrete bulkhead or shaft backfill. Use of a reinforced concrete bulkhead would introduce questions about the longevity of the structual support because of the susceptibility of the reinforcing steel to corrosive attack.

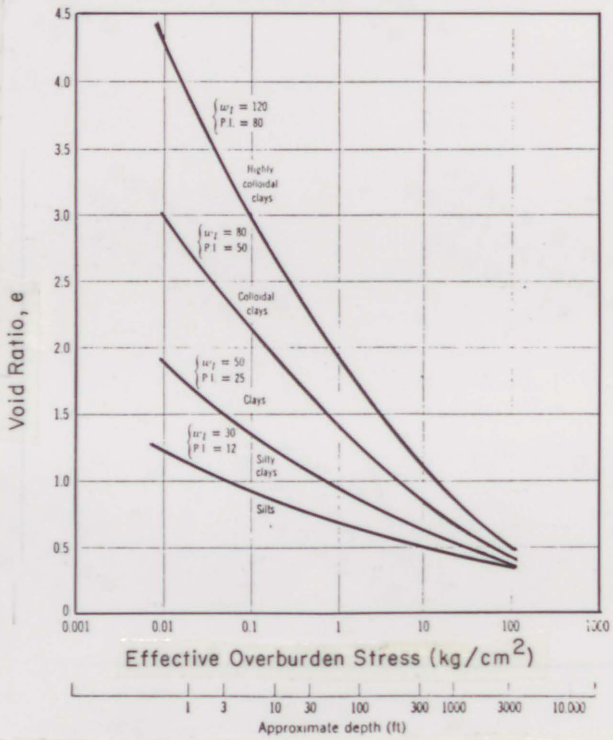
The clay plug would be mechanically placed and compacted in order to achieve a seal of low permeability. As with permeability, the compressibility of clay material is very much dependent on the clay type and placed void ratio. High void ratio, highly plastic clay slurries can compress to less than 20 percent of their initial volume. Conversely, well-compacted clay fill may compress only slightly more than sand fill. The compressibility of clays depends on both clay type and compaction procedures. Figure 9-7a shows the compression of several remolded clay slurries, and Figure 9-7b indicates general ranges for types of clays based on Figure 9-7a. Figure 9-8 shows the compression of a compacted clay for both kneading compaction and static compaction.

Compacted clays will tend to swell when saturated. The swell pressure increases with increasing compacted density, as shown on Figure 9-9. Generally, clays containing bentonite or montmorillonite exhibit high swell pressures, those containing illite exhibit less, and those containing kaolinite exhibit very little. The effects of clay type are shown on Figure 9-10. This swelling characteristic might be utilized to produce a tight contact between the clay plug and the shaft or rock wall.

The relatively high compressibility of a clay plug would prevent it from resisting any tendency towards inward movement of the shaft walls, and it might therefore not be as efficient as a concrete plug in helping to maintain the watertightness of the rock surrounding the plug. The clay plug should be suitably keyed to the shaft walls to ensure a seal and designed for anticipated settlement at the base of the seal if supported by backfill.



(a) COMPRESSIBILITY OF VARIOUS CLAYS



W_L = Liquid Limit
P.I. = Plasticity Index

(b) GENERAL RANGES OF COMPRESSIBILITY OF CLAY TYPES

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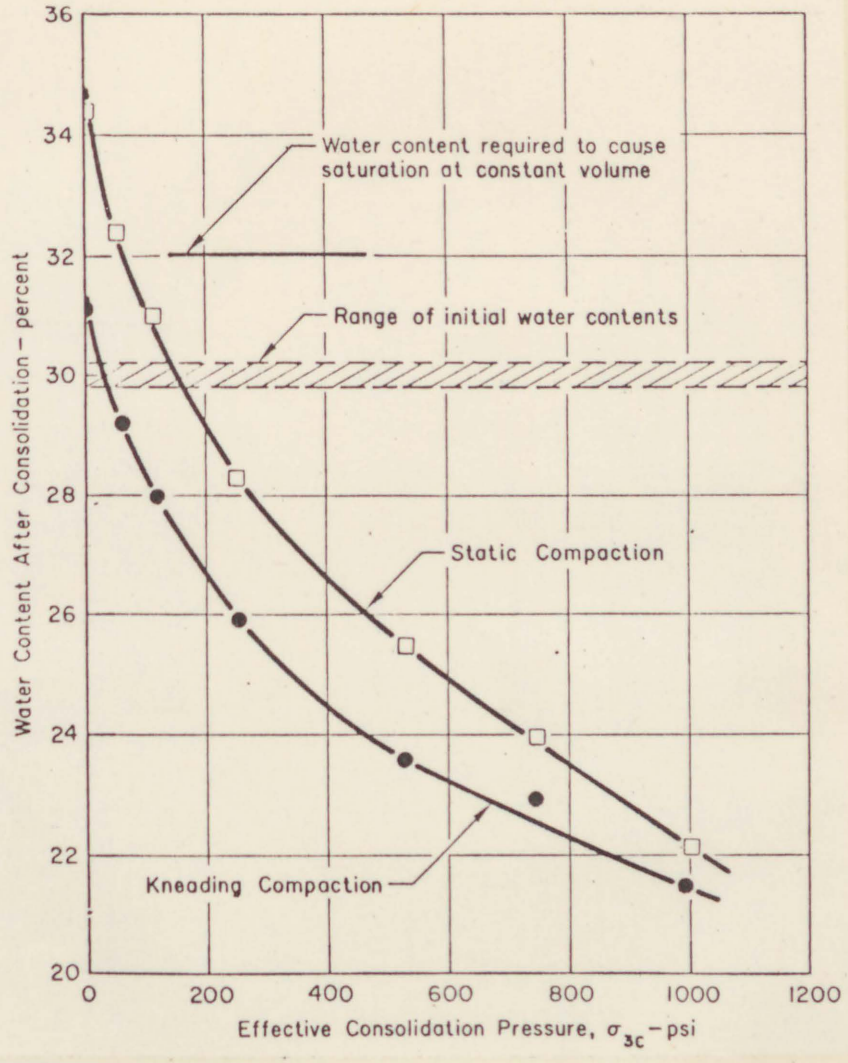
COMPRESSIBILITY OF A COMPACTED CLAY
FOR TWO DIFFERENT COMPACTION METHODS

Figure 9-8



Ref. Lee and Haley, 1968

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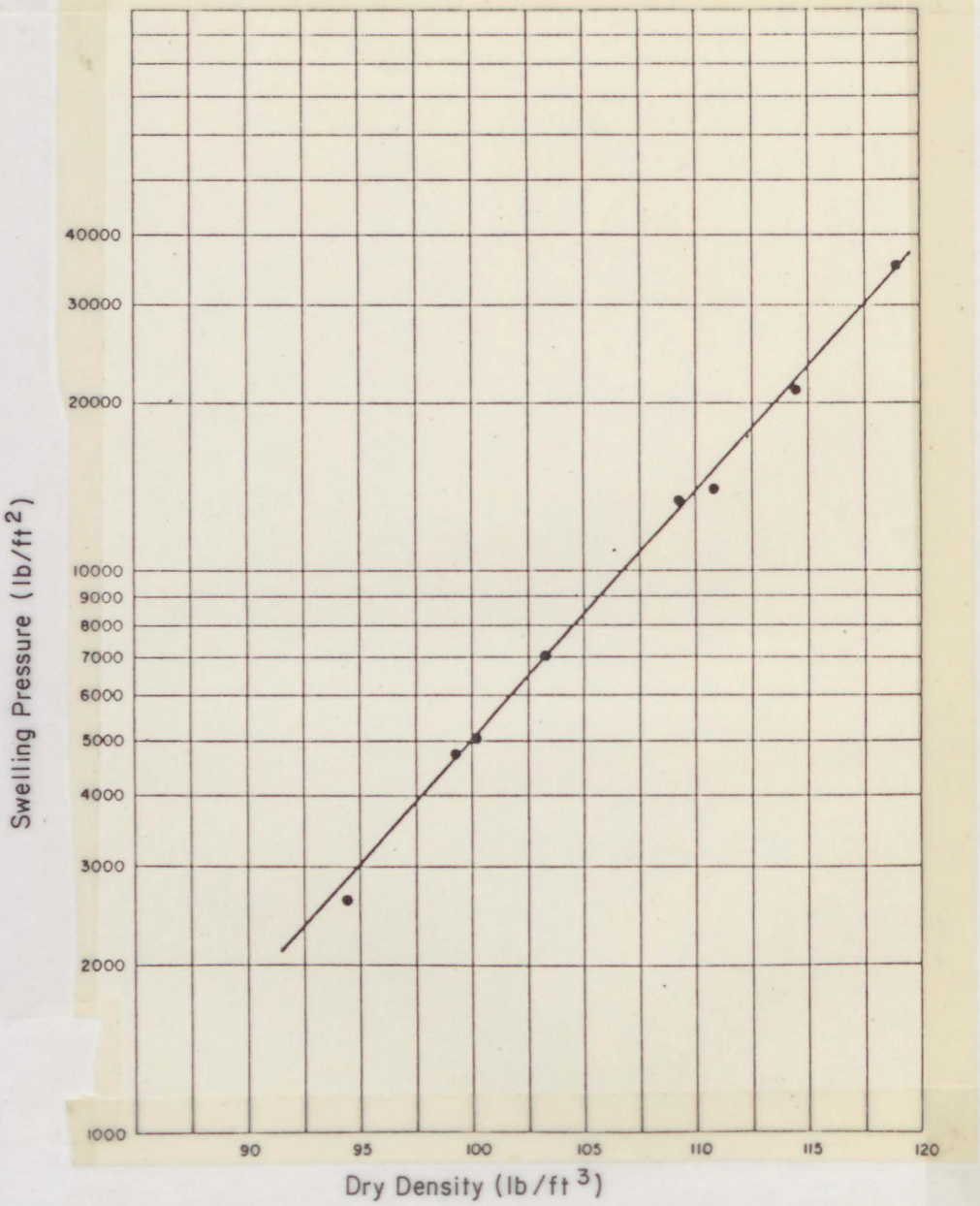
EFFECT OF VARYING DENSITY ON SWELLING PRESSURE FOR CONSTANT MOISTURE CONTENT SAMPLES

Figure 9-9



Ref. Chen, 1975

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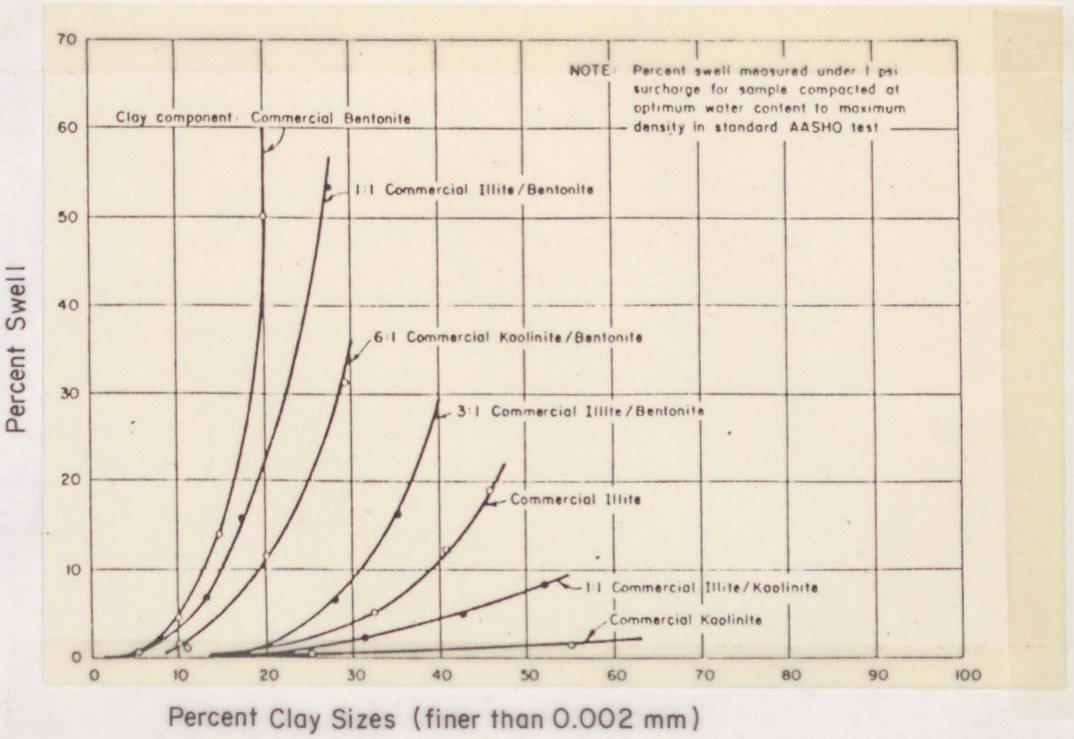
RELATIONSHIP BETWEEN PERCENTAGE OF SWELL AND PERCENTAGE OF CLAY SIZES FOR DIFFERENT TYPES OF SOIL

Figure 9-10



Ref. Chen, 1975

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9.2.2.3 Backfill

Backfill within the shaft may serve a number of purposes:

- General support of shaft lining
- Foundation support for shaft plugs with little structural strength
- Control of surface stability.

The compressibility for non clay fills is primarily a function of the compacted relative density. Figure 9-11 shows the compression under no lateral strain of a clean medium sand compacted to five different densities. The compressibility at a relative density of 39.4 percent was over three times the compressibility at a relative density of 73.1 percent.

The advantages of backfill in terms of shaft and surface stabilization at the Wills-Weaver shafts are considered to be secondary. This position is based on the following considerations.

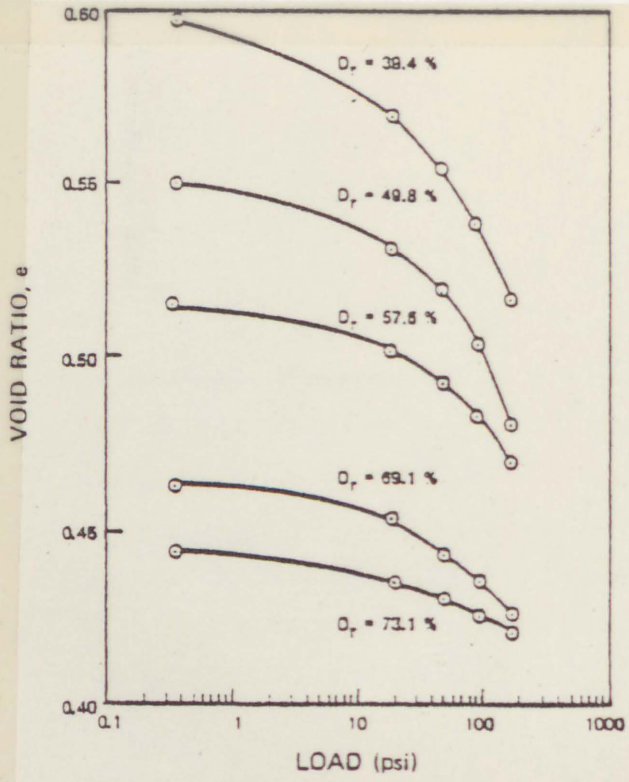
- The compressibility of the backfill is sufficiently high that it will not inhibit lining cracking which might develop in the Salado salts as a result of high earth pressures. It is not considered that such deterioration would impact the plug, providing it is not placed within the Salado salts.

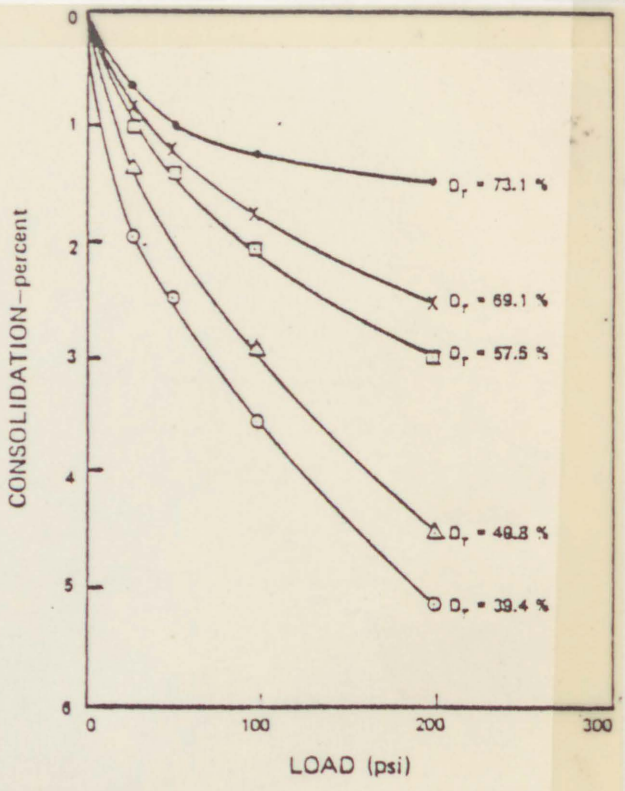
- The presence of backfill will not prevent any cracking which might ultimately develop in the upper part of the No. 3 shaft as a result of subsidence.

- Surface stabilization by backfill is considered a low priority as a result of the limited thickness of

COMPRESSIBILITY OF A CLEAN MEDIUM SAND AT VARIOUS RELATIVE DENSITIES

Figure 9-11





Note: D_r refers to the relative density

Ref. Hilf, 1975

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unconsolidated surface sediments, the shaft lining thickness in the vicinity of the collar, the relatively stable chemical environment at the near surface, and the relatively isolated location of the mine shafts. Shaft backfill to surface would provide some additional long-term security if the surface cap eventually deteriorated and if the backfill were placed properly so that a large settlement void did not develop beneath the cap.

An additional function of shaft backfill is to provide foundation support to the shaft plug. In the case of a concrete plug, backfill support is not necessary and has practical limitations. The concrete material is capable of providing its own structural support, and the relative compressibility of the backfill would result in no base support for a stiff concrete plug which is keyed into a stiff concrete lining or rock mass.

In the case of a natural earth shaft plug, backfill support appears to be a technically attractive approach. The alternative of using a reinforced concrete support poses questions regarding the longevity of the steel rebars in the corrosive shaft environment. The clay shaft plug would have to be designed to accommodate any deformations caused by settlement of the supporting backfill.

Limited backfill placed above a clay plug may also serve a useful purpose. Clay plugs will tend to swell when saturated with water, with the amount of swell being a function of the type of clay and the conditions of placement. This characteristic might be used to achieve a tight seal at the clay/rock interface. For this action to be effective, the clay plug would have to be confined so that swell pressures developed instead of

swell strains which would serve only to result in a less densely placed plug. Backfill above the plug could be effective in providing the required confinement.

In order to prevent loss of shaft backfill into the mine voids, the mine station containment system would require careful design. Limited water ingress through the shaft plug must be anticipated, and it is essential that this water is not permitted to cause dissolution damage in the mine station which would compromise the stability of the shaft backfill.

9.2.3 Chemical Stability or Durability

A basic consideration for shaft sealing is the durability of the materials which form key hydrological barriers or structural supports. Service conditions which should be considered in assessing the durability of the materials are weathering, chemical action, and wear.

Disintegration by weathering is caused mainly by the disruptive action of freezing and thawing and by expansion and contraction under restraint, resulting from temperature variations and alternate wetting and drying. The stable environment of the mine shafts should preclude the effects of weathering as a significant source of problems in the area of material durability. The most significant source of material disintegration in the Wills-Weaver context is the chemical attack of the relative corrosive groundwater environment.

Concrete

Deterioration of concrete by chemical reaction may occur in two ways:

- Chemical reactions between alkalies in cement and mineral constituents of concrete aggregates
- Attack by chemical agents.

Certain types of aggregates will react with the alkalies in cement and cause a rapid deterioration of the concrete. Such deterioration is characterized by the following observations:

- Cracking
- Excessive expansion
- Gelatinous exudations and white amorphous deposits on concrete
- Lifeless, chalky appearance of freshly fractured concrete.

Reactive aggregates include cherts, siliceous limestones and dolomites, rhyolites, dacites, andesites, siliceous shales, and phyllites. The use of such aggregates should therefore be avoided.

Deterioration of concrete also results from contact with certain chemical agents, e.g., acidic water, salt water, and solutions of sodium or magnesium sulfate. Within the Wills-Weaver mine shafts, the agents of principal concern in relation to the chemical attack of concrete are the sulfates of calcium, sodium, and magnesium which are present in the brackish Rustler aquifers and which actively attack unprotected concrete. The stronger the concentrations of these salts, the more active the corrosion. The sulfates react chemically with the hydrated lime and hydrated calcium aluminate in cement paste to form calcium sulfate and calcium sulfoaluminate, respectively, and these reactions are accompanied by considerable expansion and disruption of the material.

The sulfate ion concentrations in the Rustler aquifer waters are in excess of 2,000 ppm, and these concentrations would subject unprotected concrete to severe corrosion effects (Table 9-2). Concrete containing cement with a low content of the vulnerable calcium aluminate can be highly resistant to attack by sulfate-laden waters. While calcium aluminate in Portland cemen reacts more readily with sulfates than any other component, calcium aluminate cement is the most resistant to sulfate attack.

TABLE 9-2
ATTACK ON CONCRETE BY SOILS AND WATERS
CONTAINING VARIOUS SULFATE CONCENTRATIONS

% Water-Soluble Relative Degree of Sulfate Attack	Sulfate (as SO_4) in Soil Samples	P/M Sulfate (as SO_4) In Water Samples
Negligible	0.00 to 0.10	0 to 150
Positive[1]	0.10 to 0.20	150 to 1,500
Severe[2]	0.20 to 2.00	1,500 to 10,000
Very Severe[3]	2.00 or more	10,000 or more

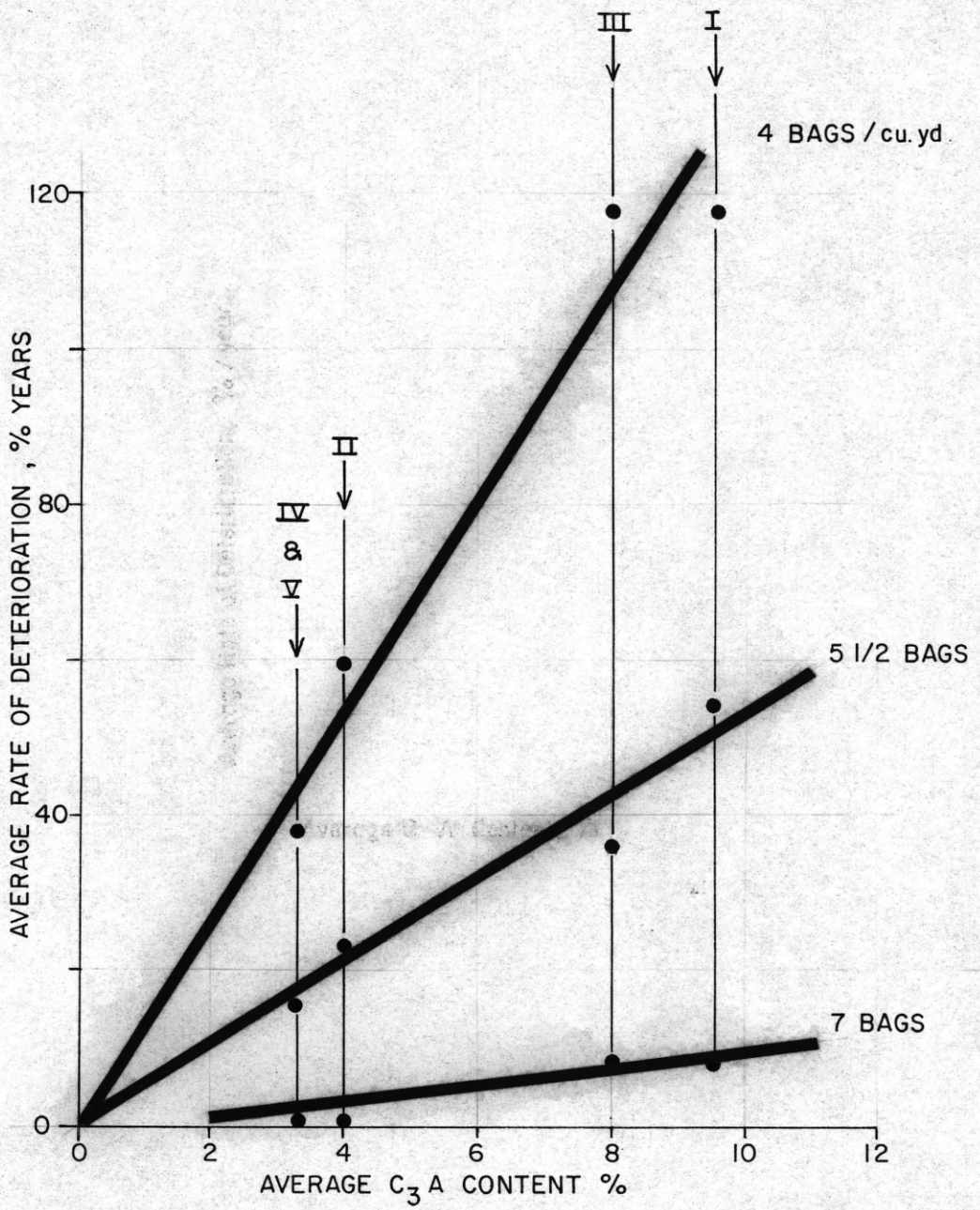
[1]Use Type II cement

[2]Use Type V cement, or approved Portland-pozzolan cement providing comparable sulfate resistance when used in concrete.

[3]Use Type V cement plus approved pozzolan which has been determined by tests to improve sulfate resistance when used in concrete with Type V cement.

The Research and Development Laboratories of the Portland Cement Association have carried out research into the resistance of concretes to sulfate attack over the past several decades. Typical findings are indicated on Figure 9-12. Two parameters which have proved to be of importance in determining sulfate resistance are the cement composition (specifically, the C_3A content) and the cement content (i.e., the richness of the con-

EFFECT OF CEMENT CONTENT OF CONCRETE AND AVERAGE C_3A CONTENTS OF DIFFERENT TYPES OF CEMENT ON RATE OF DETERIORATION IN SULPHATE SOIL EXPOSURE Figure 9-12



(after Verbeck, 1967)

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crete mix). Best results have been obtained with low tricalcium aluminate content cements used in relatively rich mixes.

The cement content is an important parameter, e.g., a Type I or Type III cement in a 5-1/2 bag per cubic yard mix was approximately equal in sulfate resistance to a Type IV or V cement in a 4 bag per cubic yard mix.

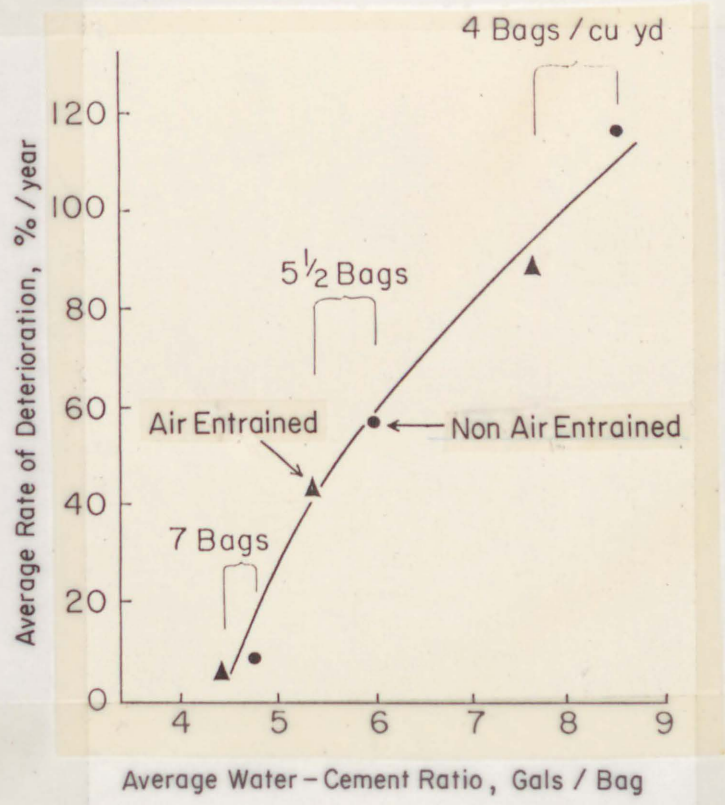
In addition to the cement type and content, the air content of the concrete also has an important influence on sulfate resistance. The beneficial effect of air entrainment is due to a reduction in the water-cement ratio. The role of air is to reduce the water-cement ratio, decrease the permeability of the mature concrete, and thereby retard the ingress of sulfates and consequent sulfate attack. The action of the increased cement content is also related to the associated lower water-cement ratio and subsequent increased sulfate resistance (Figure 9-13).

The question of long-term durability of synthetic materials such as cement concretes cannot be totally answered. Although many variations of basic cementing materials are available, all consist of approximately 90 percent or greater of noncrystalline, meta-stable materials following curing. Material properties such as permeability and strength may be adversely impacted by ultimate crystallization of the meta-stable phases.

Although a type of cement used by the Romans 2,000 years ago has demonstrated good long-term stability, the operating environments of such materials were less severe than those which exist in the Wills-Weaver mine shafts. For the ambient conditions of Wills-Weaver (i.e., high sulfate content groundwater), a Type V Portland cement which exhibits high resistance to sulfate attack is the most suitable material for

RELATIONSHIP BETWEEN RATE OF SULPHATE DETERIORATION AND WATER-CEMENT RATIO AND EFFECT OF AIR ENTRAINMENT

Figure 9-13



(after Verbeck, 1967)

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concrete construction. There is no relevant experience, however, which enables the long-term performance (hundreds of years) of such material to be evaluated.

Formation Grouting

Cement-based grouts have a long experience record in sealing fissured rock masses. From a chemical deterioration viewpoint, the same considerations as discussed above for concrete are also valid for cement grouts. Cement grouts pose special problems which are not inherent to placed concrete. Reducing the water content of cement, for example, can improve its strength, bonding, and durability and reduce its permeability. Since pumpability of grouts must be maintained, use of a lower water/cement ratio may require the use of plasticizers, friction-reducing agents, and retarders.

The use of alternative grouting compounds such as silicate gels also requires a knowledge of the chemistry of the environment. Silicate gels may be acidic or basic. The acid gels are not affected by sodium chloride, but gel rapidly in limestone. The basic gels set up quickly in the presence of salt water. At this time, there is litle evidence to suggest that these materials are superior to the cement grouts in terms of longevity. They do, however, have a different application to cement grouts in that they are most suitable for sealing less permeable materials.

Natural Earth Materials

The question of long-term durability of synthetically produced, noncrystalline, meta-stable materials may be avoided by using natural earth materials (such as clays, which are natural weathering products) placed to specified engineering standards.

9.2.4 Evaluation of Permanent Abandonment Methods

There are a wide variety of mine shaft abandonment schemes which might be applied to the Wills-Weaver mine shafts. Some of these methods need not be considered in any detail because they either

- Fail to satisfy legal requirements
- Fail to meet specific technical requirements.

The state of New Mexico cites legal requirements for abandoning a mine in the New Mexico Mine Safety Code, Article 20 Section 63-20-3, which states:

> Upon abandonment of a mine the owner or operator must effectively close or fence off all surface openings down which persons could fall, or through which persons could enter. Upon or near all such safeguards trespass warnings and appropriate danger notices shall be posted.

In addition, the New Mexico statutes (Sections 63-31-6) require that federal requirements for abandoning mines also be met. Federal legal requirements for abandoning the potash operations are listed in (1) 30 CFR 57.20-20, (2) 30 CFR 57.20-21 and (3) 30 CFR 231.40 and read as follows

Surface only:

57.20-20 Mandatory. Access to unattended mine openings shall be restricted by gates or doors, or the openings shall be fenced and posted.

57.20-21 Mandatory. Upon abandonment of a mine, the owner or operator shall effectively close or fence off all surface openings down which persons could fall or through which persons could enter. Upon or near all such safeguards, trespass warnings and appropriate danger notices shall be posted.

231.40 Surface Openings.

(a) The operator shall substantially fill in, fence, protect or close all surface openings, subsidence holes, surface excavations or workings which are a hazard to people or animals. Such protective measures shall be maintained in a secure condition during the term of the permit or lease. Before abandonment of operations, all openings, including water discharge points, shall be closed to the satisfaction of the mining supervisor.

(b) Reclamation or protection of surface areas no longer needed for operations should commence without delay. The mining supervisor shall designate such areas where restoration or protective measures, or both, must be taken.

On the basis of the study presented in this report, there are a number of technical requirements which any permanent abandonment procedure should satisfy. Such requirements are not necessarily applicable to temporary abandonment of the shafts, and this aspect is discussed later. The important requirements for permanent abandonment are subsequently listed:

- The most important technical requirement for long-term abandonment is considered to be the prevention of water ingress down the portion of the liner which passes through the Salado salts and into the mine station and mine voids. The provision of a shaft cap as an abandonment procedure for a shaft making water, or likely to make water at some time in the future, is not therefore considered to be a technically desirable abandonment method. Economic considerations may, however, override technical preferences.

  It is considered that as a result of possible subsidence damage to the shafts, concrete deterioration (of which there is evidence at this stage), etc., the only

suitable procedure for preventing shaft water ingress over a long time period is the construction of a shaft plug. For reasons outlined previously, grouting behind the lining is not considered to be an acceptable permanent abandonment method.

- The use of soluble material as shaft backfill is not considered acceptable. The presence of brackish water with substantial dissolution capacity implies that any useful function being performed by shaft backfill could be negated if the water contacts the soluble material. The principal requirement for using soluble salts as shaft backfill relates to their presence in the adjacent waste stockpile, i.e., the use of such material is more an expediency for removing surface waste than an action based upon any desirable characteristics which a salt backfill might possess. Such procedures should therefore be avoided.

- It is necessary to prevent access to the workings and it is considered that fencing of the shaft area, without provision for capping or filling the shaft, does not constitute an acceptable abandonment procedure from a surface safety point of view.

- The requirement for the removal of all foreign matter (probably including the shafts' liners) in the vicinity of a shaft plug is obvious. It is also considered that all shaft's fittings should be removed from any regions of the shafts in which compacted backfill is placed, e.g., if backfill is required to provide support for the shaft plug, the presence of shaft steel, etc., would inhibit proper placement of the fill and would

have a detrimental effect on the engineering performance of the fill. Recognizing that any steel buntons, etc., which remain in the shaft are likely to deteriorate rapidly once the shafts are abandoned, it would be prudent to protect the shaft plug from falling debris by the use of a granular fill, for example.

- It is presumed that the shaft pump stations will be filled with concrete when the shafts are permanently abandoned. It is considered prudent to seal such exposed contacts between the salts and the back of the concrete lining. Although it is not considered that dissolution damage behind the lining is presently occuring, good engineering practice requires that such potential dissolution "short circuits" be adequately sealed. Concreting/grouting provides the most versatile procedure for sealing the pump stations.

The following paragraphs discuss a number of permanent abandonment alternatives which satisfy both legal requirements and the technical constraints suggested above. Sealing procedures are discussed in generalized terms only, i.e., hydraulic plugs have not been designed and dimensions, quantities, etc., used to define installation procedures and estimate costs are considered typical but not definitive. The seal configurations will permit design loadings to be evaluated.

Technical requirements for permanent shaft abandonment methods which provide shaft water control limit the options to a few basic configurations (with variations on these possible cases). These are:

Clay Plug (Procedure A - Figure 1)

This basic configuration is shown on Figure 9-14. The main details of this abandonment scheme are as follows.

- Removal of all steel and fittings from the backfilled regions of the shaft, i.e., below shaft plug.

- Fill pump station with concrete.

- Manufactured clay plug located at approximate depth 500 to 550 feet beneath shaft collar, i.e., beneath the Culebra aquifer and above the Rustler/Salado contact, within the shales of the lower Rustler Formation.

- Plug supported by insoluble backfill which is contained within the mine station by concrete bulkheads or equivalent. Plug protected from falling debris by use of a granular backfill above plug (Figure 9-14).

- Plug keyed in by excavating through the liner and into the surrounding bedrock.

- Grouting of strata surrounding the plug to effect an hydraulic seal.

- Surface cap of reinforced concrete to prevent access.

The technical advantages of this method are as follows.

- Clay plug can be designed for a sufficiently low permeability to effect a shaft seal.

- Grouting of surrounding rock can yield a relatively watertight rock mass.

- Natural clay material will exhibit long-term chemical durability and the plug will not therefore deteriorate.

The technical disadvantages of this sealing method are as follows.

- Clay has little structural strength and would require support. A reinforced concrete slab is not recommended because of questions about the longevity of the reinforcing steel in the presence of corrosive groundwater. Backfill placed to engineered design specifications would be technically desirable. The shaft backfill could be expected to compact and settle. The clay plug should be designed for this contingency. The clay composition and method of placement should ensure that the plug has sufficient plasticity to enable settlement without rupturing. The length of the clay plug should also be sufficient to permit the settlement without compromising the seal.

- Cement or chemical grouts used to seal the rock formation are subject to concerns regarding the longevity of the material in a corrosive groundwater environment. Excavation of the plug hitch should be carried out carefully to minimize strata disturbance.

- Water leakage into the mine may erode the bulkhead foundations, leading to loss of support for the clay plug. This contingency should be provided for in the design.

Estimated costs of abandoning the No. 3 and No. 4 shafts are outlined below.

NO. 3 SHAFT

Item No.	Description	Units	Cost
1100	Remove Steel	97 Sets	$ 62,000
2100	Concrete Pump Station	40 CY	4,000
3100	Grouting	1 LS	20,000
4100	Hitch for Shaft Plug	230 CY	43,000
6100	Clay Plug	344 CY	29,000
7100	Backfill Shaft Station and Shaft	4950 CY	137,000
8100	Shaft Cap	14 CY	5,000
			$300,000

NO. 4 SHAFT

Item No.	Description	Units	Cost
1200	Remove Steel	96 Sets	$ 26,000
2200	Concrete Pump Station	40 CY	4,000
3200	Grouting	1 LS	13,000
4200	Hitch for Shaft Plug	172 CY	40,000
6200	Clay Plug	220 CY	24,000
7200	Backfill Shaft Station and Shaft	1100 CY	41,000
	Shaft Cap	7 1/2 CY	3,000
			$151,000
10000	Plant Setup		$ 36,000

SUBTOTAL DIRECT COST	=	$487,000
CONTINGENCIES (20%)	=	97,000
OVERHEAD & PROFIT (25%)	=	146,000
TOTAL ESTIMATED COST	=	$730,000

Mass Concrete Plug (Procedure B - Figure 1)

This configuration is shown on Figure 9-15. The principal details of this abandonment scheme are as follows.

- Fill pump station with concrete
- Mass concrete plug located within the lower anhydrite of the Rustler Formation (depth 556 to 566 feet in Shaft No. 3). Protect plug from falling debris by use of a granular back fill (Figure 9-15).
- Plug keyed in by excavating through the liner and into surrounding bedrock.
- Subsequent grouting of the strata surrounding the plug to effect an hydraulic seal.
- Surface cap of reinforced concrete to prevent access.

The technical advantages of this method are as follows.

- A mass concrete plug can be constructed with sufficiently low permeability to effect a seal.
- Grouting of the surrounding rock can yield a relatively water-tight mass.
- Concrete has sufficient structural strength to resist the applied loads without backfill support. Reinforcing is not required for structural support: the use of structural reinforced concrete for long-term support is not considered desirable.

- The concrete plug would have sufficient stiffness to inhibit any loosening tendency of the rock surrounding the plug.

The main technical disdvantage of a mass concrete plug is as follows:

- There is not a sufficient history of use of sulfate resistant cements to guarantee that serious deterioration of the concrete (strength, permeability) will not occur over several hundred years in the presence of high sulfate content groundwater.

Estimated costs of abandoning the No. 3 and No. 4 shafts are outlined below (cost estimates are detailed in Appendix D and installation procedures outlined in Appendix C).

NO. 3 SHAFT

Item No.	Description	Units	Cost
1100	Remove Steel	8 Sets	$ 5,000
2100	Concrete Pump Station	40 CY	4,000
3100	Grouting	1 LS	20,000
4100	Hitch for Shaft Plug	164 CY	31,000
5100	Concrete Plug	244 CY	16,000
8100	Shaft Cap	14 CY	5,000
			$ 81,000

NO. 4 SHAFT

Item No.	Description	Units	Cost
1200	Remove Steel	8 Sets	$ 2,000
2200	Concrete Pump Station	40 CY	4,000
3200	Grouting	1 LS	13,000
4200	Hitch for Shaft Plug	77 CY	18,000
5200	Concrete Plug	98 CY	6,000
8200	Shaft Cap	7 1/2 CY	3,000
			$46,000
10000	Plant Setup		$36,000

SUBTOTAL DIRECT COST	=	$163,000
CONTINGENCIES (20%)	=	33,000
OVERHEAD & PROFIT (25%)	=	49,000
TOTAL ESTIMATED COST	=	$245,000

Hybrid Clay/Concrete Plug (Procedure C - Figure 1)

A large portion of the costs associated with supporting a clay plug on compacted insoluble backfill (Procedure A) is related to placement of the backfill and removal of shaft steel in the backfilled region of the shaft. An alternative method of supporting the clay plug could achieve substantial cost economies. A most obvious method of providing plug support is the use of an underlying concrete bulkhead.

As noted previously the use of mass concrete in an hydraulic plug is not considered desirable because of questions

regarding the long term durability of the material in the high sulfate environment. If a mass concrete plug were installed as indicated under Procedure B above, the substantial hydraulic gradient across the plug would cause fluid flow through the concrete and the entire volume of the plug material would be exposed to the groundwater. If a mass concrete plug were used to support a clay plug, the clay plug would dissipate the hydraulic pressures and leakage could simply be drained through the concrete plug, i.e., drain holes would be constructed through the concrete plug so that the concrete provided structural support but presented no hydraulic barrier (Figure 9.16).

This arrangement would tend to minimize the volume of concrete exposed to the corrosive groundwater and maximize the life of the concrete (particularly compared with the Procedure B described above).

The main details of this abandonment scheme are described below.

- Removal of all steel and fittings in the vicinity of the shaft plug.

- Fill pump station with concrete.

- Mass concrete plug located below proposed clay plug position. Concrete plug to be keyed into the shaft liner to provide adequate structural support.

- Clay plug keyed in by excavating through the liner and into the surrounding bedrock. Plug protected from falling debris by use of a granular backfill above the plug (Figure 9-16).

- Grouting of strata surrounding the plug to effect an hydraulic seal.

- Surface cap of reinforced concrete to prevent access.

The significant difference between this alternative and that previously described (Procedure A) is the replacement of the fill support by the concrete plug. This results in considerable decommissioning cost economies but the longevity of the concrete plug cannot be guaranteed. It is considered likely, however, that the life of the concrete could be substantially longer than if the concrete plug were used for both structural support and as an hydraulic barrier.

Estimated costs of abandoning the No. 3 and No. 4 shafts are outlined below.

NO. 3 SHAFT

Item No.	Description	Units	Cost
1100	Remove Steel	10 Sets	$ 7,000
2100	Concrete Pump Station	40 CY	4,000
3100	Grouting	1 LS	20,000
4100	Hitch for Shaft Plug	230 CY	43,000
5100	Concrete Plug	95 CY	6,000
6100	Clay Plug	344 CY	29,000
8100	Shaft Cap	14 CY	5,000
			$114,000

NO. 4 SHAFT

Item No.	Description	Units	Cost
1200	Remove Steel	10 Sets	$ 3,000
2200	Concrete Pump Station	40 CY	4,000
3200	Grouting	1 LS	13,000
4200	Hitch for Shaft Plug	172 CY	40,000
5200	Concrete Plug	25 CY	2,000
6200	Clay Plug	220 CY	24,000
8200	Shaft Cap	7 1/2 CY	3,000
			$89,000
10000	Plant Setup		$36,000

SUBTOTAL DIRECT COST	= $239,000
CONTINGENCIES (20%)	= 48,000
OVERHEAD AND PROFIT (25%)	= 72,000
TOTAL ESTIMATED COST	= $359,000

Discussion of Preferred Long-Term Abandonment Method

The most important technical item which differentiates between the abandonment options considered is the longevity of concrete in a high sulfate environment. There is no relevant experience which permits design of concrete seals for the time periods envisaged (several hundred years). Similar comments may also be directed towards cement based grouts with which the rock formations surrounding the plug may be sealed. The shaft plug constitutes, however, a more critical element in that it is required to provide structural strength as well as low permeability. It is considered that the hydrological impact of sulfate deterioration of a cement based grout sealer would be

substantially less than the hydrological impact of sulfate deterioration of the concrete sealing plug (particularly within an intrinsically low permeability rock mass). Since a viable alternative to the concrete plug is available, it is recommended that a natural or manufactured clay plug be adopted to provide the shaft seal.

It is recommended that the plug be keyed through the lining and into the formation rock (i.e., shales). From a construction point of view it will be necessary to grout behind the liner above the intended plug location prior to excavating through the concrete, in order to prevent the possibility of shaft inflows if an hydraulic communication between the overlying aquifers and the concrete/rock interface exists. Based on experience during shaft sinking operations, no serious problems during plug construction are envisaged.

The above procedure will provide an hydraulic plug which will perform to better standards than those which would be exhibited by a plug which was not keyed through the liner. For this latter case, the concrete liner would effectively form part of the hydraulic seal and questions about the long-term chemical stability of concrete in the prevailing environment have resulted in the rejection of concrete as a suitable plug material. It therefore makes little sense to include the liner as part of the hydraulic seal.

Providing the clay plug is properly placed, the opportunity for water migration past the plug will not be increased by keying through the liner. In fact, in the long-term, the reverse is true.

A major disadvantage of the clay plug is the requirement for structural support. The use of compacted soluble backfill to support the clay plug is considered to provide the technically optimum solution, but the costs are expensive. Proper placement of such fill would require the removal of all shafts' fittings. If the steel buntons, etc. were not removed, it is considered highly questionable whether suitably compact fill could be placed. This could result in excessive settlement of the backfill with subsequent rupturing of the plug.

The high estimated decommissioning costs for the above optimum abandonment procedure mitigate against its adoption. Substantial cost economies can be achieved by using a mass concrete plug to support the clay seal, and protecting the concrete from sulfate deterioration by suitable drainage. Although the long-term performance of the concrete cannot be guaranteed, the use of a clay/concrete hybrid plug is recommended as a cost-effective method of shaft abandonment.

Although the present study is concerned principally with the Wills-Weaver mine shafts, it is important to maintain a proper perspective of the total problem of shaft abandonment in the potash fields of southeast New Mexico. Many of the shafts to be ultimately abandoned are not lined through the salt section. Water migrating down these shafts could ultimately cause some dissolution of the shafts' walls. It is considered that a clay/concrete hybrid plug would be less susceptible to damage from this source than would a clay plug supported by backfill. Under these circumstances, therefore, the hybrid plug might be considered technically preferable to the alternative. It could therefore be argued that there is little point in applying more elaborate abandonment procedures to the Wills-Weaver mine shafts than those which might be used to decommission many other shafts in the area.

Additional Considerations

The recommended permanent decommissioning method for the Wills-Weaver mine shafts consists of a clay plug located at approximate depth 500 to 550 feet beneath the shaft collar, supported by either compacted insoluble backfill (surface sand) or by a concrete bulkhead. The shafts above the plug have been assumed not to be backfilled, and the shaft void is isolated from the surface by a reinforced concrete cap.

Advantages and disadvantages of filling the shafts above the plugs with insoluble backfill are listed below.

- The fill would not inhibit liner deterioration. A relatively small amount of fill could be used to protect the shaft plug from falling shaft debris.

- Filling the shaft to surface with backfill may provide some additional long-term security if the shaft cap and liner deteriorate over time. The additional cost of this security would be substantial. In order to provide compact fill, it would be necessary to remove shafts' fittings and place fill to engineered standards. It is estimated that the direct additional cost of removing shafts' fittings above plugs and backfilling the upper halves of both shafts would be $160,000, while the total estimated additional cost would be $240,000.

- If the clay plug is designed to exhibit some swell characteristics in order to effect a tighter clay/rock contact at the seal location, backfill above the plug may be used to provide the confinement necessary to

develop the swell pressures. Depending on the swell pressures to be resisted, this quantity of backfill would be relatively minor.

- The placement of relatively inexpensive uncompacted insoluble fill to surface (i.e., by dumping from the top of the shaft without compaction and without removal of the shafts' fittings) would also be ineffective in improving shaft stability or in enhancing surface safety. Fill placed in such a manner would demonstrate significant settlement and a large void beneath the shaft cap would develop. It has also been suggested that filling the shaft void with even loosely placed fill will prevent certain potential failure modes, e.g., squeezing of compressible portions of the Culebra aquifer into the shaft with an associated slumping of the overlying sediments. It is believed that such potential mechanisms will be self limiting and that mass failure of the overlying consolidated sediments into the shaft void is not likely. Mechanical forces tending to squeeze compressible material into the shaft void will disappear as stress redistribution occurs and erosion by inflowing aquifer water will cease when the shaft fills to the water table and an hydrostatic balance is again achieved. This latter event will occur well before deterioration of the concrete liner permits any unconsolidated aquifer material to enter the shaft void.

- Placing of uncompacted fill in the portions of the mine shafts above the shaft plugs would be a relatively inexpensive exercise and while it is considered that such action would have limited technical advantages, its economic impact on decommissioning costs would be minor.

The use of compacted insoluble backfill, placed from the plug to surface, is not recommended. Although the presence of such material may provide some additional long-term surface security if properly placed, the cost is high. The use of smaller amounts of backfill above the plug to protect it from falling debris and, if necessary, to provide confinement to the plug, is recommended. Uncompacted insoluble backfill may be placed from the plug to surface for relatively low cost but the technical merits of such action are considered minimal.

Consideration should also be given to providing an impermeable plug throughout the aquifer zone in order to prevent mixing and provide hydraulic isolation between the strata (Procedure D - Figure 1). If it is assumed that above the clay plugs the shafts are filled with clay to the water table elevation (approximate depth 300 feet), and then to the surface with insoluble backfill, then an additional direct cost of $320,000 and an additional total estimated cost of $480,000 will be incurred over the option which provides for no backfill above the shafts' plugs. Compared with the procedure which calls for compacted insoluble fill from the plug to surface, the additional direct cost is $160,000 and the additional estimated cost is $240,000. Placement of compacted clay fill through the aquifer zone is not recommended for the following reasons.

- Water quality in both aquifers is poor and the environmental impact of mixing within the shafts will be low.

- No useful aquifers will be adversely impacted.

- The existence of the concrete shaft liner through the aquifer zone, and its probable eventual deterioration,

will provide local hydraulic communication irrespective of whether the shafts voids are clay-filled.

- The additional costs of providing compacted clay fill are substantial.

9.3 SHORT-TERM ABANDONMENT METHODS

The definition of "short term" is given in the following context.

- "Short term" implies a period within which the majority of the concrete shafts' linings might be expected to be usable if the shafts were reclaimed.

- Shafts' fittings (e.g., buntons, ladder roads, shaft guides, etc.) will not be considered to be usable if the shafts are abandoned, even in the short term. To date, the shafts' fittings have suffered to varying degrees and, unless the shafts were to be utilized in the near future and maintained in the meantime, it is considered that all fittings must be replaced when and if the shafts are recommissioned. The existing shafts' fittings will also require removal during either decommissioning or recommissoning.

The general objectives of a temporary abandonment method must remain the same as those pertaining to permanent abandonment, e.g., to prevent the ingress of water down the mine shafts and into the mine voids. From a technical point of view, any abandonment procedure which is acceptable for permanent abandonment is also suitable for temporary abandonment. The benefit of employing a permanent abandonment procedure is that, if the

shaft is not recommissioned, no future action is required and no subsequent costs are incurred. The potential penalties of using a permanent abandonment procedure are determined by the probability of recommissioning the shafts, the differences in cost between long- and short-term abandonment, and the differences in recommissioning costs for shafts that have been abandoned in permanent and temporary manners.

A number of abandonment schemes, suitable for short-term abandonment, are subsequently evaluated.

Clay Plug (Procedure A - Figure 1)

This option is the same as that discussed previously and is the method considered technically desirable for permanent abandonment of the workings. The principal advantage of this method is that it is considered to be the only procedure that is suitable for permanent abandonment of the workings if it is considered necessary to prevent shaft water ingress. If the probability of recommissioning the shafts is low, this option is therefore likely to be the most effective as a current procedure to apply to the shafts.

The disadvantage of the clay shaft plug with backfill support is the relatively high cost of decommissioning. As outlined previously, the estimated cost of decommissioning both shafts is approximately $730,000. The estimated costs of recommissioning both shafts (i.e., mucking out, removing old steel and furnishings, replacing shaft furnishings but not constructing a permanent headframe) are outlined below.

NO. 3 SHAFT

Item No.	Description	Units	Cost
1100	Remove Steel	97 Sets	$ 63,000
11110	Purchase and Install Furnishings	194 Sets	313,000
11120	Muck Out Non-Concrete	5350 CY	63,000
11160	Remove Concrete Bulkhead	166 Cy	19,000
			$458,000

No. 4 SHAFT

Item No.	Description	Units	Cost
1200	Remove Steel	97 Sets	$ 27,000
11210	Purchase and Install Furnishings	193 Sets	79,000
11220	Muck out Non-Concrete	1180 CY	14,000
11260	Remove Concrete Bulkhead	230 CY	26,000
			$146,000
	Plant Setup		100,000

SUBTOTAL DIRECT COST	=	$ 704,000
CONTINGENCIES (20%)	=	141,000
PROFIT + OVERHEAD (25%)	=	211,000
TOTAL ESTIMATED COST	=	$1,056,000

Mass Concrete Plug (Procedure B - Figure 1)

This option was discussed previously as a possible long-term abandonment procedure, but was rejected on the grounds that the long-term durability of the concrete plug could not be guaranteed in the presence of the high sulfate content groundwater. The advantage of this decommissioning method over the clay plug option is its lower cost. It is estimated that the abandonment cost with no shaft backfill is approximately $245,000, compared to approximately $730,000 for the clay plug supported by backfill and $360,000 for the clay plug supported by concrete.

Estimated costs to recommission both shafts which have been abandoned in the above manner are outlined below.

NO. 3 SHAFT

Item No.	Description	Units	Cost
1100	Remove Steel	186 Sets	$120,000
11110	Purchase and Install Furnishings	194 Sets	313,000
11130	Drill and Blast Concrete Plug	80 CY	32,000
11150	Muck Out Concrete	80 CY	14,000
			$479,000

NO. 4 SHAFT

Item No.	Description	Units	Cost
1200	Remove Steel	185 Sets	$ 51,000
11210	Purchase and Install Furnishings	193 Sets	79,000
11230	Drill and Blast Concrete Plug	21 CY	8,000
11250	Muck Out Concrete	21 CY	4,000
			$142,000
	Plant Setup		100,000

SUBTOTAL DIRECT COST	= $	721,000
CONTINGENCIES (20%)	=	144,000
PROFIT + OVERHEAD (25%)	=	216,000
TOTAL ESTIMATED COST	=	$1,081,000

Costs of recommissioning the shafts for this temporary abandonment option are roughly comparable with recommissioning costs for the clay plug method.

The principal disadvantage of this method of abandonment is that it is not considered suitable for permanent abandonment of the mine shafts. Irrespective of whether it was desired to recommission the shaft or not, it would still be necessary to install clay plugs ultimately and bear the costs associated with this.

Hybrid Clay/Concrete Plug (Procedure C - Figure 1)

The hybrid clay/concrete plug abandonment procedure was recommended as a cost-effective permanent decomissioning method. Although the longevity of the concrete cannot be guaranteed, it

is considered that such a procedure will result in a longer plug life than if concrete alone were utilized.

The decommissioning costs for the clay plug supported by a concrete bulkhead are estimated at $360,000. The recommissioning costs for this method of abandonment are:

NO. 3 SHAFT

Item. No.	Description	Units	Cost
1100	Remove Steel	184 Sets	$119,000
11110	Purchase and Install Furnishings	194 Sets	313,000
	Muck Out Non-Concrete	350 CY	4,000
11130	Drill and Blast Concrete Plug	80 CY	32,000
	Muck Out Concrete	80 CY	2,000
			$470,000

NO. 4 SHAFT

Item No.	Description	Units	Cost
1200	Remove Steel	183 Sets	$ 50,000
11210	Purchase and Install Furnishings	193 Sets	79,000
	Muck Out Non-Concrete	220 CY	3,000
11230	Drill and Blast Concrete Plug	21 CY	8,000
	Muck Out Concrete	21 CY	2,000
			$142,000
	Plant Setup		100,000

SUBTOTAL DIRECT COST	= $	712,000
CONTINGENCIES (20%)	=	142,000
PROFIT + OVERHEAD (25%)	=	214,000
TOTAL ESTIMATED COST	=	$1,068,000

Grout Aquifer Zones (Procedure E - Figure 1)

An alternative approach to placing an impermeable shaft plug within the shaft is to carry out additional grouting to inhibit water inflow and to cap the top of the shaft to prevent surface access. The details of this approach are outlined below:

- Fill pump station with concrete
- Grout aquifer zones to seal water inflow
- Construct a surface cap.

Estimated costs of abandoning the No. 3 and No. 4 shafts in this manner are given below (cost estimates are detailed in Appendix D and installation procedures are outlined in Appendix C).

NO. 3 SHAFT

Item No.	Description	Units	Cost
2100	Concrete Pump Station	40 CY	$ 4,000
3100	Grouting	1 LS	67,000
8100	Shaft Cap	14 CY	5,000
			$76,000

NO. 4 SHAFT

Item No.	Description	Units	Cost
2200	Concrete Pump Station	40 CY	$ 4,000
3200	Grouting	1 LS	44,000
8200	Shaft Cap	7-1/2 CY	3,000
			$ 51,000
10000	Plant Setup		$ 36,000

SUBTOTAL DIRECT COST	= $163,000
CONTINGENCIES (20%)	= 33,000
OVERHEAD AND PROFIT (25%)	= 49,000
TOTAL ESTIMATED COST	= $245,000

The costs of recommissioning both shafts for this method of temporary abandonment are as follows.

NO. 3 and NO. 4 SHAFTS

Item No.	Description	Units	Cost
1100	Remove Steel No. 3 Shaft	94 Sets	$125,000
11110	Purchase and Install Furnishings No. 3 Shaft	194 Sets	313,000
1200	Remove Steel No. 4 Shaft	193 Sets	53,000
11210	Purchase and Install Furnishings No. 4 Shaft	193 Sets	79,000
	Plant Setup		$100,000

SUBTOTAL DIRECT COST	=	$ 670,000
CONTINGENCIES (20%)	=	134,000
PROFIT AND OVERHEAD (25%)	=	201,000
TOTAL ESTIMATED COST	=	$1,005,000

It is estimated that this short-term abandonment procedure has decommissioning and minor recommissioning cost incentives over the previous options. Suitable sealing of the aquifers by grouting so that a basically dry shaft is maintained over the period of short-term abandonment has an additional technical advantage over any procedure which relies on a shaft plug for an hydraulic seal and allows water ingress into the shaft above the plug. This advantage is concerned with sulfate deterioration of the concrete shafts' liners. If the aquifers are sealed so as to inhibit access of water to the liners, then chemical deterioration of the concrete will be substantially less than for an abandonment option which permits the shaft void to fill with water above the plug. The maintenance of the condition of the concrete of the shafts' liners is of prime importance for a temporary abandonment method where the understanding is that the shafts are to be utilized at a future date. Prevention of concrete deterioration is of little concern for a permanent abandonment scheme unless the concrete is an integral part of the hydraulic seal.

The main disadvantage of an abandonment option which calls for sealing of aquifers is the uncertainty of being able to achieve an adequate seal. Substantial grouting was carried out within the No. 3 shaft during construction, and a total seal (adequate for abandonment purposes) was not achieved. However, experience within the No. 4 shaft was considerably better and demonstrates that very low inflow rates can be achieved within these formations.

Discussion of Preferred Short-Term Abandonment Method

The most attractive short-term abandonment procedure for the Wills-Weaver mine shafts is considered to consist of a program of aquifer grouting to limit inflow rates into the shafts to acceptable levels and to prevent surface entry with shaft caps. This option provides for maximum protection to the concrete shafts' liners from sulfate attack by the groundwater and also offers decommissioning and recommissioning cost incentives over alternative procedures.

It is desirable that the shafts' liners in the vicinity of the aquifers do not constitute a significant part of the hydraulic seal, because of problems of sulfate deterioration. The aquifers themselves should be sealed with cement grout to a minimum depth of 20 feet from the shafts' walls.

Since small inflow rates will probably continue to occur, it is desirable to construct shaft caps which will allow natural convection within the shafts to continue. The airflow will evaporate moisture in the downcast shaft and carry it into the workings as vapor (to be subsequently absorbed by carnallite present in the mine horizon), rather than simply permit the moisture to migrate down the liner and into the shaft station. This process has been occurring since construction of the shafts, with no significant adverse impacts at the mine station. Within the upcast shaft, shaft wall moisture will again be evaporated and carried out of the workings. The natural convection cycle is therefore an effective method of reducing total inflow and of carrying the moisture which does enter the mine into the workings in a more acceptable form.

Salt concentrations on the shafts' walls suggest that average inflows of 2 to 3 gph can be evaporated by natural convection and the grouting program should therefore aim to reduce inflows to these levels. Within the No. 4 shaft, present water inflows are already of this magnitude and evaporation is maintaining the shaft in a basically dry condition. A drilling and grouting program has still been recommended for the No. 4 shaft in order to ensure the longevity of the seal in the region of the Culebra aquifer. The concrete in this vicinity is considered to be showing evidence of deterioration.

The longevity of the short-term abandonment procedure will be determined by the time period beyond which the seal is likely to be compromised as a result of either grout deterioration under the action of the groundwater or the creation of an hydraulic communication between the shaft and aquifers as a result of closure of the mining horizon and associated subsidence of the overlying strata. Deterioration of the aquifer seal may result in substantial rates of water inflow into the shaft, e.g., during grouting of the Culebra aquifer in the sinking of the No. 3 shaft, one of the grout holes made in excess of 130 gpm.

Based on observations to date, together with the available data on the longevity of cements in the presence of sulfates, it is considered that the expected life of a grouted aquifer seal would be of the order of several decades (possibly 50 years). It is not expected that the No. 3 shaft would suffer significant subsidence damage over this time period providing dissolution damage of the mine horizon does not occur in the vicinity of the shafts.

In order to adopt the preferred short-term abandonment method, it must be considered that there is a very high probability of future utilization of the shafts. If this is not the case, it would be economically more attractive to permanently decommission the shafts.

9.4 ALTERNATIVE SHAFT ABANDONMENT PROCEDURES

The previous sections have discussed permanent and temporary abandonment procedures which are capable of exhibiting a high level of control over shaft water entry, i.e., limit water ingress rates to sufficiently low levels over sufficient periods of time in order to minimize the impacts discussed in Section 8.0. For permanent abandonment, the technically optimum procedure consists of a natural earth impermeable clay plug supported on compacted insoluble backfill. The costs associated with such an abandonment procedure are very high as a result of the tight control of placement procedures necessary to ensure the adequate performance of the plug. Supporting the clay plug by use of a mass concrete plug (largely protected from high pressure water) reduces costs considerably. Although there exists some doubt about the long-term integrity of the concrete, this latter procedure is considered to be a cost-effective alternative. It is considered that protecting the concrete plug from high pressure water (which would undoubtedly seep through the concrete mass) will maximize the life of the concrete.

The costs associated with abandonment procedures which exhibit a high level of control over shaft water ingress are considerable. Alternative procedures which provide a lower level of hydrologic control should therefore be considered. Each method is subsequently discussed, evaluated technically and costed.

Fill Shaft with Uncompacted Permeable Fill
(Procedure F - Figure 1)

Filling the Wills-Weaver mine shafts with uncompacted surface sand, dumped from the top of the shaft without removal of shafts' fittings etc., would provide a relatively economic method of shaft abandonment because of the suitable material available in the immediate vicinity of the shafts and the relatively low placement costs.

The effective permeability of such material would be sufficiently high that it would present virtually no barrier to water ingress. It is estimated that tens of thousands of gallons of water per hour could flow down the No. 3 shaft (filled with uncompacted sand) under gravity drainage. These flow rates are likely to exceed any inflows from the aquifers into the shafts and the shaft fill would therefore have no impact on the rate of inflow into the workings.

As discussed previously, the benefits of non-cemented backfill in terms of shaft stabilization are considered minimal. Such fill will not prevent liner deterioration by subsidence or excessive earth pressures. Filling or partially filling the shaft void would inhibit general collapse of the surrounding strata into the shaft but this is not considered to be a likely failure mechanism for a shaft passing through consolidated sediments. In addition, the limited void volume of the shaft together with the fundamentally stable geometric configuration tend to prevent a significant collapse feature from developing. The existence of an open shaft into which the surrounding strata might collapse is considered to be a second order concern compared with the types of collapse mechanisms which might develop if solutioning occurs in the Salado salts surrounding

the shaft. There is a potential for the creation of a very much larger void than the existing shafts' voids, of a much more unfavorable configuration, as a result of dissolution, and the placement of uncompacted or compacted surface sand within the shafts will do nothing to prevent such an occurrence.

The estimated costs of abandoning both the No. 3 and No. 4 shafts by dumping surface sand down the shafts are outlined below. The main details of this abandonment scheme are:

- Fill pump station with concrete
- Fill shaft with surface sand by dumping from surface
- Surface cap of reinforced concrete to prevent access

The placement of such fill within the Wills-Weaver mine shafts is not considered to achieve any significant useful function.

NO. 3 SHAFT

Item No.	Description	Units	Cost
2100	Concrete Pump Station	40 CY	$ 4,000
12100	Random Fill and Site Preparation	8456 CY	12,000
8100	Shaft Cap	14 CY	5,000
			$21,000

NO. 4 SHAFT

Item No.	Description	Units	Cost
2200	Concrete Pump Station	40 CY	$ 4,000
12200	Random Fill	2715 CY	4,000
8200	Shaft Cap	7 1/2 CY	3,000
			$11,000

SUBTOTAL DIRECT COST	= $32,000
CONTINGENCIES (20%)	= 6,000
OVERHEAD AND PROFIT (25%)	= 10,000
TOTAL ESTIMATED COST	$48,000

The above costs include no provision for protection of the mine station area from dissolution, and since large rates of inflow are ultimately likely there is a high probability of loss of the shafts.

Fill Shaft with Uncompacted Cohesive Fill
(Procedure G - Figure 1)

Placement of an uncompacted manufactured clay fill within the Wills-Weaver mine shafts is in some ways similar to the abandonment method just discussed. Such fill would perform virtually no structural role but would obviously be substantially more expensive than the sand fill option as a result of:

- The cost of procuring, transporting and preparing the fill material
- The more restrictive placement method necessary

The principal potential benefit of a clay fill might therefore be considered to be its ability to exert a satisfactory degree of hydrologic control over shaft water i.e, the method might be presumed to be a more economic equivalent of the clay plug option discussed previously. The following points present a number of important factors in relation to the hydrologic control which such an abandonment method might offer.

- The rate at which water can flow through a shaft plug is a function of the inherent permeability of the material itself together with the thickness of the plug. Thicker plugs will result in lower hydraulic gradients and hence lower flow rates, all other things being equal.

- The clay plug abandonment option discussed previously incorporated a typically 20-foot thick plug, which would result in an hydraulic gradient of about 10 for the proposed configuration. For gravity flow of a fluid through a vertical column of material (e.g., plug) the minimum hydraulic gradient which can exist is unity and this is independent of the thickness of the plug. Filling the shaft with a low permeability material (as opposed to utilizing a limited thickness plug) can therefore reduce the hydraulic gradient (and flow rate) by an order of magnitude at most. For the additional volumes of material involved, this is quite a modest improvement.

- The above scenario assumes that the basic permeability or hydraulic conductivity of the materials is the same in each case. This would, in fact, not be the case. A

finite thickness clay plug would be placed to optimum standards to achieve minimum permeability but such procedures could not possibly be applied to the whole shaft as the cost would be prohibitive. In addition, it would not be necessary to place a continuous plug to such standards as it has already been demonstrated that a limited thickness plug would perform satisfactorily.

- The implication is that continuous filling of the shafts with clay fill would result in the material being placed to significantly lower standards of compaction. As demonstrated on Figure 9-4, the permeability of a clay fill is generally very dependent on the compaction procedure and can vary by several orders of magnitude for comparatively modest changes in the placed void ratio. It is considered that the differences in placed void ratio between clay simply dumped from a bottom dumping skip and allowed to consolidate, and material placed at optimum moisture content with significant compactive effort, would be substantial enough to induce permeability differences of several orders of magnitude.

- The water-tightness of a thick plug placed with limited control is therefore likely to be two to three orders of magnitude lower than that exhibited by a limited thickness plug placed to optimum standards. The improvement in terms of reduced hydraulic gradient is likely to be more than offset by the higher in-place permeability. As a general rule, it is far more cost effective to improve the quality of the hydraulic plug than to increase the quantity of material within the plug.

- Placement of an approximate 20-foot thick clay plug with a permeability of 10^{-6} cm/sec would limit water ingress through the No. 3 shaft to a few gallons per hour. Filling the shaft with uncompacted clay fill would possibly permit a few hundred to a few thousand gallons per hour to pass through the plug and as discussed in Section 8.0, this does not represent any significant hydrologic control in terms of this particular problem.

- In order to fill the shaft with a clay backfill it would be necessary to transport the material down the shaft and place it at the working level. Any attempt to dump such material from surface down 1000-feet of shaft containing all the original fixtures (buntons, ladder roads etc.) would likely result in the existence of large voids within the fill. The fill would be placed in uncompacted condition and the probability of it providing an hydraulic seal adequate to prevent the impacts outlined in Section 8.0, is considered doubtful. Significantly reduced permeability of the material as a result of consolidation under its own weight would also be doubtful because of the limited size of the shafts and the presence of shaft steel, etc., (it is considered that adhesive forces generated at the shaft walls and fittings would tend to inhibit in situ compaction and consolidation).

The broad details of this abandonment scheme are outlined below.

- Fill pump station with concrete.
- Fill shaft with manufactured clay fill by dumping at the working level.
- Surface cap of reinforced concrete to prevent access.

The estimated costs of abandoning the No. 3 and No. 4 shafts in this manner are given below.

NO. 3 SHAFT

Item No.	Description	Units	Cost
2100	Concrete Pump Station	40 CY	$ 4,000
13100	Clay Fill	8456 CY	177,000
8100	Shaft Cap	14 CY	5,000
			$186,000

NO. 4 SHAFT

Item No.	Description	Units	Cost
2200	Concrete Pump Station	40 CY	$ 4,000
13200	Clay Fill	2715 CY	62,000
8200	Shaft Cap	7 1/2 CY	3,000
			$ 69,000
10000	Plant Set Up		$ 36,000

SUBTOTAL DIRECT COST	= $291,000
CONTINGENCIES (20%)	= 58,000
OVERHEAD AND PROFIT (25%)	= 87,000
TOTAL ESTIMATED COST	= $436,000

The costs are very substantial and could be significantly higher depending on the local availability of suitable clay fill. The method of abandonment provides negligible structural control and hydrologic control considered dubious at best. The above procedure is not recommended as a cost-effective abandonment alternative for the Wills-Weaver mine shafts.

Shaft Cap (Procedure H - Figure 1)

The provision of shaft caps is certainly the minimum abandonment requirement that would be necessary for the Wills-Weaver mine shafts. The purpose of the caps is simply related to surface safety aspects and this abandonment method would provide for no structural or hydrologic control. Such an abandonment method would not avoid the impacts discussed in Section 8.0. Methods which are considered capable of providing an hydraulic shaft seal over sufficient time periods have been demonstrated to be expensive.

If measures are not taken to protect the mine station from dissolution damage caused by potential high inflow rates through the shafts, premature exposure of the Salado salts behind the shaft liner must be anticipated. It is desirable to prevent this event from occurring for as long as possible in order to minimize the impacts associated with solutioning of the Salado salts in the vicinity of the shafts e.g., it is conceivable that the mine void could fill with water without loss of the shafts providing the salts behind the liners were not exposed to shaft water during filling. The potential impacts of shaft fluid ingress would then be limited to lateral migration by preferential carnallite dissolution, and the probability of development of large solution voids at the shafts would be minimized.

If cost considerations dictate that shaft plugging is an unacceptable method of abandoning the Wills-Weaver mine shafts, the impacts of shaft water ingress can be minimized by protecting the mine station area from dissolution so that the potentially high rates of inflow do not cause premature exposure of the salt behind the shaft liner. This could be achieved by constructing clay blankets in the vicinity of the shafts in order to conduct the shaft water inflow out of the shaft pillar region. The estimated costs of decommissioning both shafts by capping, concreting the pump stations and protecting the mine stations from dissolution, are outlined below.

NO. 3 and NO. 4 SHAFTS

Item No.	Description	Units	Cost
2100	Concrete Pump Station (No. 3)	40 CY	$ 4,000
2200	Concrete Pump Station (No. 4)	40 CY	4,000
14000	Clay Floor Linear	825 CY	55,000
8100	Shaft Cap (No. 3)	14 CY	5,000
8200	Shaft Cap (No. 4)	7 1/2 CY	3,000
$71,000			
10000	Plant Set Up		$36,000

SUBTOTAL DIRECT COST= $107,000
CONTINGENCIES (20%)=21,000
OVERHEAD AND PROFIT (25%) = 32,000

TOTAL ESTIMATED COST $160,000

A large portion of the above costs is related to the provision of mine station protection. While this action could prove effective at the Wills-Weaver mine (in terms of preventing loss of the lower regions of the shafts and subsequent dissolution of the Salado salts behind the liners), there are many other shafts to be abandoned which are unlined through the salt. If shaft plugs are not provided for these latter shafts, then dissolution of the unlined salt walls of the shafts will occur.

If the ultimate abandonment procedure is not to prevent entry of shaft water into the mine workings, (e.g., shaft plugs), it is recommended that the shafts be capped and the pump stations concreted. At Wills-Weaver, provision of mine station dissolution protection could be effective in mitigating some of the impacts outlined previously (i.e., premature exposure of the Salado salts behind the liners). However, since there are a number of shafts which are unlined through the salt section, it is doubtful whether the additional expenditure of providing mine station protection at Wills-Weaver could be justified, particularly when such action at many other shafts would have no benefit.

Injection of Shaft Water into Formations Underlying the Mine

A somewhat radical method for dealing with shaft water entering the mine is to attempt to discharge the fluid into the formations underlying the mine. The envisaged system would encompass a structure collecting the incoming shaft water which would be subsequently drained to the underlying formations via a well.

As discussed previously, the underlying Salado salts are not considered sufficiently permeable to accommodate shaft water discharges. According to Brokaw, et al., (1972)

> "The Salado Formation has an intergranular porosity and permeability that ranges from low to virtually none. Locally, fractures and solution openings impart a spotty formational permeability. In the potash mining area, the Salado Formation is dry except for water in the leached zone at the top of the formation and small pockets of water or water and gas encountered occasionally during mining".

Underlying the Salado Formation in the potash mining area, the Capitan Limestone and associated dolomitic part of the Yates and Tansill Formations form the topmost aquifer. The Wills-Weaver mine is located north of the Capitan Limestone reef and the Tansill and Yates Formations underlie the Salado salts in this region. At Wills-Weaver, these water-bearing sediments are located approximately 500 feet beneath the mine level.

East of the Pecos River, water in the Capitan Limestone and in the Yates and Tansill Formations is under strong artesian pressure. The potentiometric surface of water in these units is largely above the land surface in this area (Brokaw et al., 1972).

These hydrologic conditions make the aquifers underlying the Salado Formation most unsuited to any attempt to discharge shaft water into them.

9.5 SUMMARY OF ABANDONMENT RECOMMENDATIONS

The following items summarize recommended procedures for short-term and permanent abandonment of the Wills-Weaver mine shafts, together with the important factors which identified the appropriate procedures.

- The most important technical consideration for abandonment of the Wills-Weaver mine shafts (in both the short term and long term) is whether or not it is necessary to inhibit shaft water ingress to the workings. Abandonment methods which control shaft water are considered initially.

- An hydraulic shaft seal may be achieved either by placement of an impermeable plug within the shaft or by aquifer grouting behind the lining, with no seal as such within the shaft.

Permanent Abandonment

- Cement grouting of the aquifer zones is not recommended for permanent abandonment for the following reasons.

  - The No. 3 shaft in particular is susceptible to subsidence damage in the long-term which could create an hydraulic communication between the Rustler aquifers and the shaft. The Culebra aquifer is capable of providing very large flow rates to the shafts (e.g., hundreds of gallons per minute potentially).

  - The long-term durability of concrete and cement based grouts cannot be guaranteed in the presence of the high sulfate groundwater which occurs within the Rustler dolomite aquifers. It is therefore desirable that formation grout seals within high permeability zones such as the Culebra and Magenta aquifers do not form the major hydraulic barrier to water ingress into the shafts.

- The method of permanent abandonment recommended for the Wills-Weaver mine shafts therefore consists of a shaft plug, designed to achieve an hydraulic seal between the mine voids and the overlying aquifers of the Rustler Formation.

- Two types of material can be considered for the construction of the shaft plug--concrete and clay fill. Both materials can provide a seal of adequately low permeability, if properly designed and placed. Concrete offers the advantage of adequate structural strength to provide its own support and hence no requirement for structual support of the plug. Clay material, on the other hand, must be adequately supported. The technically optimum method of support consists of compacted surface sand fill, as it is considered that this procedure would provide a shaft seal of long durability. The cost of decommissioning the shafts in this manner is high as a result of the large quantity of compacted backfill to be placed. The use of a concrete hydraulic plug is not recommended because of doubts about the longevity of the concrete under the action of high sulfate content groundwater. The technically optimum abandonment procedure is therefore considered to consist of a compacted clay plug supported by compacted insoluble backfill. The high cost of this abandonment procedure may mitigate against its adoption.

- The optimum location for the shaft plug is determined by factors such as the point of fluid ingress into the shafts, the location of soluble salts behind the lining, susceptibility of the shaft environment to mechanical damage (excessive earth

pressures, subsidence cracking) and the mechanical competence and permeability of the geological strata behind the lining. These considerations indicate that the clay shaft plug, designed to prevent access of groundwater into the mine voids via the shaft, should be located in the shales of the lower Rustler Formation, approximately 500 to 550 feet beneath the shaft collar.

- It is not considered necessary to provide impermeable plugs at other locations within the shafts or to seal through the aquifer zone in order to isolate the aquifers from each other and from the overlying Dewey Lake Redbed sediments. Water mixing within the shafts as they fill above the plugs will have a negligible hydrological impact.

- The clay plug should be keyed through the concrete lining and into the surrounding rock. The concrete lining should not constitute part of the hydraulic plug because of doubts regarding its longevity. The rock key should be carefully excavated to minimize disturbance and loosening of the rock and to maintain its in situ low permeability.

- The clay plug should be mechanically placed to provide an adequately low permeability. The use of swelling additives (e.g., bentonite) should be considered to assist in the development of a tight seal at the clay-rock interface. The surrounding rock should be grouted to effect a seal.

- Considerable settlement of the backfill and plug can be expected within the shaft, e.g., several feet. The

plug must therefore be designed to withstand these movements without compromizing the seal (e.g., the clay should be placed wet of optimum and of such a composition so as to be highly plastic, and the plug should be of adequate thickness).

- The mine station should incorporate appropriate (clay) sealers and drain holes through the bulkheads in order to prevent small amounts of water which might enter the workings from contacting salts at the base of the fill or at the foundations of the bulkheads. Such ingressing water should be conducted away so that its dissolution capacity does not ultimately compromise the backfill retaining system.

- The use of compacted insoluble backfill above the shaft plug is not considered necessary. The potential benefits of such fill are minimal and the costs are substantial. Limited fill should be placed above the plug to protect it from falling debris. The cost of placing uncompacted fill above the plug would be minimal but such action would have limited technical merit.

- Soluble surface material (e.g., waste stockpile) should not be used as backfill within the shafts.

- The presence of fittings in backfilled areas would prevent adequate placement of compacted fill and would impair the engineering performance of the fill. Such fittings should therefore be removed.

- The shafts' pump stations should be filled with concrete as part of the abandonment procedure.

- Mine stations' excavations (e.g., tailshafts, ore bins, etc.) should be backfilled and all fittings in the regions to be backfilled should be removed.

- The major features of the technically optimum abandonment procedure for the Wills-Weaver mine shafts are summarized on Figure 9-14. The estimated decommissioning cost for both shafts is approximately $730,000.

- An alternative more cost-effective method of abandoning the Wills-Weaver mine shafts consists of a compacted clay plug, keyed into the surrounding rock strata as previously described, but supported by a mass concrete bulkhead. Although the longevity of the concrete cannot be guaranteed, its life can be maximized by ensuring that the volume of concrete exposed to groundwater is kept to a minimum, e.g., by suitable drainage. The basic concept behind this clay/concrete hybrid plug is that the clay is employed to effect an hydraulic seal and the concrete provides structural support but no hydraulic barrier. In addition, the use of a concrete bulkhead to support the clay plug might be technically preferable to backfill support for other mine shafts which are unlined through the salt section.

- The major features of this abandonment procedure are summarized on Figure 9-16. The estimated decommissioning cost is $360,000 and the method is recommended as a cost-effective procedure for abandoning the Wills-Weaver mine shafts.

Temporary Abandonment

- The most suitable method of short-term abandonment of the Wills-Weaver mine shafts is considered to be the grouting of aquifer zones to seal the shafts against water inflow.

- The pump stations should be filled with concrete, the aquifer zones grouted to minimize inflow, and the surface security of the shafts maintained with reinforced concrete caps.

- This method of abandonment has certain technical and cost advantages over alternative techniques which call for the placement of a shaft plug to prevent water ingress into the mine.

- Decommissioning costs for both shafts are estimated at approximately $245,000. Recommissioning costs are estimated at about $1 million for both shafts (not including construction of permanent head frames).

- The most significant technical advantage of this method of temporary shaft abandonment over methods which incorporate a shaft plug and permit the shaft to fill with water above the plug, is the less significant exposure of the liner concentrate to the high sulfate groundwater.

- Inflow rates within the No. 3 shaft should be reduced to levels low enough (2 to 3 gph) such that the natural convection will prevent water entering the mine station in liquid form. The shaft caps should be designed to

prevent the ingress of rainwater, etc., but to permit a natural convection cycle to continue. The existence of adequate air flow within the shafts to evaporate all incoming water will result in the following desirable conditions:

- Water ingress to the upcast shaft will be removed by evaporation and exhausted to the atmosphere, i.e., it will not enter the workings.

- Water ingress to the downcast shaft will enter the workings as vapor. This process has been occurring since commissioning of the shafts without any adverse impacts. Dissolution damage in the vicinity of the mine station is therefore avoided.

- The longevity of a seal of this type is considered to be of the order of several decades (approximately 50 years) providing adequate grouting of the aquifers is carried out.

- Unless there is a high probability of future utilization of the mine shafts, it will be economically more attractive to permanently decommission.

Additional Comments

- The above recommended abandonment strategy for the Wills-Weaver mine shafts is based upon the premise that it is necessary to limit water entry into the mine voids. The impacts of such water ingress are outlined in Section 8.0.

- The recommended procedures are expensive, and it will be necessary to evaluate whether the consequences of permitting mine water ingress are worth the expense. This evaluation must consider factors such as future utilization of the surface and subsurface regions of the potash mining district, procedures which can reasonably be applied to other shafts in the area, consequences of water ingress into other mine voids, etc.

- If it is decided that restriction of shaft water ingress is not practical then the following comments and recommendations are offered in relation to abandonment methods which provide limited hydrologic control.

  - The optimum abandonment procedure for the Wills-Weaver mine shafts under the above conditions consists of providing shaft caps to maintain surface security together with the provision of mine station protection to prevent dissolution damage in this area by shaft water. The estimated cost of abandoning both shafts in this manner is $160,000.

  - Protection of the mine station from dissolution would have no significant impact in mine shafts which are unlined through the salt section. It is therefore questionable whether the additional cost of providing mine station protection at Wills-Weaver can be justified, as such procedures can only be usefully applied to some of the shafts of the region. If shaft plugging is not desirable or

practical then the optimum method of abandonment consists of simple capping of the shafts.

- The use of uncompacted sand backfill within the shafts would contribute little to the structural integrity of the shafts and would provide no hydrologic control.

- Placement of uncompacted clay fill within the shafts would be very expensive, and would provide dubious hydrologic control. This procedure is not recommended as an abandonment method.

- Hydrologic conditions are not suited to schemes involving the discharge of shaft water into the aquifers underlying the Salado salts.

- The Wills-Weaver mine must be placed in correct perspective in relation to the total problem of mine abandonment within the potash leases. Unless a uniform strategy is to be applied to all properties, then elaborate measures at a few properties will not be effective in an overall sense. In this regard it would be useful to apply a ranking system to all of the mine shafts and voids in the area. Comparisons should be made in terms of criteria that are pertinent to abandonment rather than those which might be pertinent to an operating mine. For example, the rate of water inflow might be a good index as to likely problems within operating shafts. In terms of abandonment, however, it may be no more difficult to seal a shaft making much water than it is to seal a shaft which is comparatively dry. In addition, the impacts of

providing an inadequate seal may be less in the former than in the latter depending on factors such as the volume of the mine void, the nature and distribution of the salts present, etc.

Hence, the necessity for proper abandonment procedures should be determined by an abandonment impact analysis rather than by operational criteria such as whether the shaft is "dry" or "wet."

- Since there is no precedent for abandonment of mine shafts in the potash fields of southeastern New Mexico, it would be desirable to evaluate the efficacy of any abandonment procedure applied to the Wills-Weaver shafts. Procedures to evaluate the performance of the shaft abandonment method will depend on the type of abandonment adopted.

  If the abandonment method permits shaft water ingress, a factor of principal concern is the condition of the mine station. The only way of evaluating whether mine shaft station protection procedures (if applied) have been effective is to visually examine these areas. Since the recommended abandonment procedure under these circumstances does not require filling of the shafts, the mine stations would be accessible via the shafts. It is recognized that the present hoists and headframes will not remain on site, and special provisions will have to be made to enter the workings. Similar visual inspections would also provide the best method of evaluating the shafts' conditions if they were abandoned in a temporary manner (e.g., checking the condition of the aquifer grout seal).

If a shaft plug is installed as part of the abandonment procedure (as recommended), it would be useful to monitor the performance of the plug. If the plug were completely impermeable, the shaft would fill with water above the plug to the water table. Significant leakage past or through the plug would be indicated by monitoring water levels within the shaft. In order to investigate the performance of the plug under full hydraulic heads, it might be convenient to fill the shaft to the water table rather than to wait for natural drainage into the shaft to achieve this. If the shaft were open above the plug, the water level could be monitored directly. In the case of fill above the plug, the probe would have to be inserted through casing.

Settlements of the plug should also be monitored to provide an indication of possible distress as a result of differential movements. This would again be a relatively straightforward exercise if the shafts were left open above the plugs. Displacement monitering would be of concern only if the clay plug was supported by fill.

The condition of the mine stations is also of interest for abandonment procedures which attempt to prevent water ingress. This is best evaluated by visual inspection. In order to check the performance of any hydraulic seal installed at Wills-Weaver, it would be useful to delay permanent decommissioning of the No. 4 shaft so that access to the No. 3 shaft station was available. Once again, it is recognized that existing hoist facilities might not be available.

10.0 POTENTIAL ALTERNATIVE USES FOR WILLS-WEAVER MINE

10.1 INTRODUCTION

Lease abandonment is the final phase of a potash mining operation. As part of this procedure, it is appropriate to review possible uses for the shafts and underground workings of the Wills-Weaver Mine.

Proposals for some alternative uses have previously been made to potash operators in the Carlsbad potash area by both industry and government (BLM 1975), and it is evident that there is considerable interest in using the underground space created by the mining operations.

This is a complex issue that has been debated for several years among those with legitimate vested interests in the area, and the impact of any future use of the area's resources must be carefully considered. Presently exploited resources that may be affected include potash, oil and gas, and groundwater.

The potash mines in the Carlsbad area are the main source of domestic potash production. Large contiguous reserves remain (BLM 1975) and it appears that potash production will continue to be a major priority for this area, providing that competitive mining costs can be maintained.

Production of oil and gas from formations beneath the potash horizon in this area is increasing, and future demand for additional domestic production will undoubtedly intensify exploration activities. A significant number of producing wells are located within or adjacent to active mining areas, and a procedure to determine which future oil and gas wells may be drilled within

the potash area was established in 1974 (BLM 1975). This allows drilling only under specific conditions and therefore protects the extraction of identified potash reserves.

The four major cities in the general area--Carlsbad, Hobbs, Artesia and Lovington--all obtain their domestic water supply from water wells. The possibility of pollution of sub-surface waters as a result of future use of underground mines must therefore be considered.

Clearly, a determination of the economic factors that may favor one or more potential uses of the Wills-Weaver mine is beyond the scope of this report. The following sections of this report do, however, provide a description of potentially feasible alternative uses and definition of the more important criteria that govern each case. Thus, a ranking of potential uses may be possible. The impact of each usage on existing resources is also briefly described.

10.1.1 Possible Alternatives

The following listing of possible alternative uses for the Wills-Weaver mine is derived largely from existing publications (BLM 1975) and has been divided into three basic categories.

I. Mineral Extraction
   - Potash (sylvite and langbeinite)
     - conventional exploitation
     - solution mining of remaining pillars or thin seams
   - Oil and Gas

II. Storage of Materials

- Short-Term Storage
  - high pressure air
  - natural gas
  - LPG
  - Oil
  - warehousing
- Long-Term Storage
  - chemical waste
  - radioactive waste
  - septic waste

III. Other

- Personnel Shelters

These alternatives will be discussed in terms of application to the Wills-Weaver mine. They may, however, also be applicable to the other mines in the potash area east of Carlsbad, as the total reserves are exhausted.

The assessment of use should include, in addition to the absolute criteria essential for the success of each scheme, other pertinent factors such as the following.

- Need:
  The need for each considered use at the specific site must be evaluated, including consideration of available alternative sites or other design options.

- Duration of Use:
  The general time requirement to develop the site and the likely duration of each use must be evaluated.

- Revocability:
  This considers whether the commitment to the use is irreversible once the use has begun.

- Impact:
  Each use will have an impact on the immediate surrounding human and natural environment. Those factors typically included in EIS preparation would be considered.

10.1.2 Present Status of Wills-Weaver Mine

The Wills-Weaver mine is located at the northern extremity of the Carlsbad potash mining district, some 30 miles northeast of the city of Carlsbad.

The areas of mine workings in the district and the distribution of known economic potash resources (BLM 1975; Jones 1975) are shown on Figure 10-1. It is noted that the Wills-Weaver mine workings are presently isolated and distant from other contiguous ore deposits or interconnected mine workings.

The Wills-Weaver mine covers approximately 720 acres of room and pillar workings. Mining started in 1960 and, during 11 years of operation, some 5 million tons of potash were mined from the First Ore Zone of the McNutt potash zone.

The mining horizon lies at a depth of approximately 1,000 feet below surface in the shaft area and dips to the east at an average of 1 degree. Mine development generally proceeded away from the shafts in the northeastern quadrant. Initial extraction in production panels was about 60 percent at a mined height of 52 inches. This was followed by pillar mining to a final extraction

DISTRIBUTION OF KNOWN POTASH RESOURCES

Figure 10-1



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of between 70 and 85 percent, except in the Main East entry west of 3rd North Panel. As a result of high extraction, considerable closure has occurred and the majority of the mine workings are inaccessible. The area currently considered safe to enter is shown on Figure 5-4.

Within the areas in which high extraction (85 percent) pillar mining has been completed, deduced pillar closure rates and observed subsidence suggest that the majority of these areas have largely closed (sections 4.0 and 5.0). It is also deduced that the overlying strata have suffered comparatively limited disturbance. The three producing oil wells located within the mine site are protected by relatively small salt pillars (150-foot radius) in a area of 75 percent extraction. Closures in these areas have not apparently been sufficient to cause problems.

10.1.3 Storage Volume

The mine provides a potentially large storage capacity, but its usefulness is limited by the closure and loss of access that has occurred. The total available storage volume is uncertain and can be estimated only from the limited data available on underground closure and subsidence.

The maximum volume is estimated to be 60 million cubic feet, as follows.

Mine Region	Mined Extraction (%)	% of Mined Area	Est. Room Height (ft)	Est. Volume (10 ft)
West of 2nd North Panel North of Main East Entry	85	25	0	0
Main East Entry to 3rd North Panel	60	10	4.0	7.5
Other Areas	75	65	3.5	53.5

The extraction in most areas is sufficiently high that the mine cannot be considered stable, and continued room closure is expected. The mine void is continuing to close with an estimated volume loss of 100,000 to 600,000 cubic feet per year.

10.1.4 Maximum Storage Pressure

The maximum permissible storage pressure is an important factor in assessing the mine storage capacity for pressurized fluids or gases. This maximum pressure will be determined by the following:

- Depth of the mine below surface
- Structural stability of the mine
- Leakage through natural or man-made features.

The pressure exerted on the roof, walls, and floor of the mine will be hydrostatic and, at constant pressure, would tend to improve the long-term stability of the mine. In normal storage operations, however, pressures will be cyclic, and this may cause accelerated local failure of roof and pillars, resulting in a reduction in storage capacity.

As the rock overlying the mine is relatively impermeable and contains few structural weaknesses, the mine opening is expected to contain pressures up to a limit required to induce hydro-fracturing. While this limiting value could be as high as the pre-mining overburden pressure of 900 psi, a reasonable working assumption for a storage operation would be about one-half of this value (assume 500 psi).

The storage pressure that can be maintained will also depend on uncontrolled leakage that occurs. This is discussed in the following section. Normal practice requires that the mine is pressure tested to an acceptance pressure before full-scale storage is started (Cobbs Engineering 1975) to ensure that significant leakage does not occur.

10.1.5 Leakage Potential

Two types of leakage are considered possible:

- Direct leakage of pressurized fluid or gas through man-made or natural pathways

- Dissolution of liquids or solids in groundwater and migration to aquifer systems.

The mine workings are contained within the massive halite beds of the Salado Formation and are overlain by approximately 250 to 400 feet of halite. This contains additional thin potash deposits within the McNutt potash zone, minor clay and sand beds, and numerous thin anhydrite beds. The Rustler Formation, containing two aquifer zones, overlies the Salado and is composed of massive anhydrite with interbeds of siltstone, shale and dolomite/limestone.

Natural fracturing in the halite beds is sparse and the dominant structure is sub-horizontal bedding. The water permeability of the intact salt is extremely low (less than 10^{-10} cm/sec), while bedding or clay seam permeability is expected to be less than 10^{-6} cm/sec. It is therefore extremely doubtful that significant leakage of fluids or gas, at pressures comparable to existing hydraulic heads, could occur through the natural formations in the life of a typical storage facility.

A limited number of exploratory boreholes and dry oil wells have been drilled into or through the mine horizon and have subsequently been sealed. Three producing oil wells are adequately protected by 150-foot radius pillars on the mining horizon. There are also two mine shafts. The seals in these non-producing boreholes and shafts must be adequate to prevent leakage of material at storage pressure or of fluid waste material that may be pressurized by continued closure of the mine area. A design value in excess of overburden pressure should therefore be used. Resealing of the boreholes may be required, based on inspection of existing seal quality.

Leakage by dissolution and groundwater migration is a long-term problem of particular importance to the safe disposal of toxic waste material. The natural impermeability of the surrounding rock means that the rate of escape of the toxic material will be very low, and it may take several hundred years to reach the biosphere. This is a problem that is presently being evaluated for the disposal of high level nuclear waste, and the arguments extend beyond normal engineering practice. While recognized as significant, it will not be discussed in detail here. Obviously, if groundwater recharge of the mine can be prevented, the problem does not exist.

10.2 DISCUSSION OF POTENTIAL USES

10.2.1 Mineral Extraction

10.2.1.1 Conventional Potash Mining

Mining at Wills-Weaver ceased in 1971 due to the exhaustion of economic potash reserves, based on the prevailing production cost/pricing structure. There are two possibilities for future conventional exploitation at this property, namely:

- Lower grade exploitation of the First Ore Zone
- Exploitation of other ore zones in the McNutt potash.

The viability of continued mining at Wills-Weaver will depend on several market factors, including the future potash price structure, import/export demand, the reserves of higher grade potash within the Carlsbad district, and projected future production costs. A detailed evaluation of the alternatives is outside the scope of this contract, but some of the factors that should be considered in such an analysis are discussed in this section.

Potash demand is expected to increase worldwide at an annual rate of about 5 percent, and increased supply will come from both expanded production in North America and by large-scale incursion into Western markets of potash from the Soviet Union (BLM 1975; Mining Engineering 1979). How will this affect production from the Carlsbad district?

World potash production statistics are shown in Table 10-1. (Mining Engineering 1979).

TABLE 10-1
WORLD POTASH PRODUCTION
(Thousand Short Tons K_2O; Year Ending June 30)

	1976	1977	1978
Canada	5,337	6,234	6,842
U.S.	2,420	2,399	2,495
North America	7,757	8,633	9,337
France	1,911	1,727	1,840
West Germany	2,037	2,444	2,523
Italy	155	150	149
Spain	558	617	605
U.K.	37	50	136
Israel	789	711	789
Congo	306	269	89
Western Europe, Israel, & Congo	5,793	5,698	6,131
China (Mainland)	331	331	331
East Germany	3,328	3,484	3,559
USSR	8,757	9,160	9,201
Communist countries	12,416	12,975	13,091
Total World	25,966	27,578	28,559

The potash mines in the Carlsbad district produce about 85 percent of annual domestic production (Brokaw 1972; Jones 1975), which has not increased markedly in the past 5 years (2,232,000 tons in 1973). Approximately 72 percent of North American potash production is delivered to U.S. and Canadian markets, and this proportion also holds true for U.S. production alone. In 1978, 73 percent of U.S. domestic demand was obtained from Canada.

By far, the largest production increase (worldwide) in the past 2 years has been achieved by Canadian producers, who have the potential to maintain continued strong growth with a predicted annual production capacity of 7.4 million short tons K_2O (Jones 1975). In comparison, the predicted capacity for the Carlsbad district was 2.6 million short tons (loc. cit.).

Increased worldwide demand will tend to stimulate production in all present producers, but more immediately so in those operations with developed or proven reserves and which are currently producing economically. This would certainly include Canadian producers (very large reserves) and, to a lesser extent, existing operations in the Carlsbad area.

It is anticipated that additional export markets would be required to absorb significantly increased domestic production. Rising production costs, related to increasing natural gas prices and environmental expenditures, may make U.S. potash uncompetitive on world markets (Mining Eng. 1979).

Low Grade Exploitation, First Ore Zone

For Wills-Weaver to resume production of lower grade ore, it is essential that:

- Substantial reserves must be identified
- Increases in potash prices must be assumed
- Lower production costs must be realized.

Known resources of sylvite and langbeinite are show on Figure 10-1 (Jones 1975 from BLM 1975). This indicates substantial reserves to the east of the present mining operations in the district, while the Wills-Weaver mine is located in an isolated zone to the north of the district. It is noted that at least 15 exploratory holes have been drilled south of the Wills-Weaver mine in T18S, R50E; and T18S, R31E (BLM 1975). While the definition of resources will undoubtedly be updated, it appears unlikely that substantial economic reserves will be identified in the vicinity of the mine in the near future.

As previously stated, increases in potash prices will benefit all potential and operating producers alike and will provide little immediate incentive to low grade producers. Production costs at Wills-Weaver are not likely to be less than those experienced at other operations in the Carlsbad area, and again this tends to favor expansion of existing operations.

It is concluded, therefore, that the prospects of reopening the Wills-Weaver mine for low-grade mining based on existing information appear to be rather poor in the foreseeable future.

Exploitation of Other Ore Zones

The potentially economic deposits of potash occur in a number of mineralized halite beds ("ore zones"), of which only five contain sylvite and langbeinite as the potash ore mineral in proven economic values (Jones 1975). The bulk of the sylvite ore is contained in the 1st, 7th, and 10th zones. Langbeinite has been exploited only in the 4th and 5th zones, with some economically significant amounts in the 1st and 10th zones. Sylvite and langbeinite are the only minerals that have a clearly established economic value and industrial recovery process.

At Wills-Weaver, only the First Ore Zone has been exploited, and there is currently no information to suggest that economic potash deposits occur in any of the other zones above the First Ore Zone. Should such mineralization exist, the existing shafts could be used to mine the deposits. Mining of several ore zones has been successfully practised elsewhere in this district (Brokaw 1972).

Exploitation of sub-economic deposits, under present conditions, would suffer from the same constraints as outlined in the previous section.

10.2.1.2 Solution Mining

It has been suggested that recovery of the potash resource within the Wills-Weaver mine may be improved by solution mining of the potash remaining in the unmined pillars (approximately 2.0 to 2.5 million tons) and in other thin contiguous potash seams.

Solution mining of potash poses the particular problem that the potash minerals are seldom pure. The major potash minerals, sylvite and langbeinite, are often found associated with complex sulphate salts containing magnesium and calcium. At Wills-Weaver, thin seams of carnallite are found above and below the sylvite ore zone in particular.

The only known potash solution mining operations in deep deposits are:

- Kalium Chemicals Ltd., Belle Plain, Saskatchewan, where operations started in 1964 (Husband 1971).

- Texasgulf Inc., Moab, Utah, where operations started in 1972 (E/MJ 1972).

The Kalium Chemicals operation is producing from an unmined potash bed. Details of the operation have not been obtained, but the process is presumably similar to existing rock salt solutioning practice.

The Texasgulf operation is, however, of more immediate interest to the Wills-Weaver operation. Both mines have produced in excess of 400,000 tpy by conventional methods. Conversion of the Cane Creek mine to solution mining was made when underground mining became economically uncompetetive (E/MJ 1972). This con-

version involved sealing the mine (note: unspecified method) and filling it with some 700 million gallons of water pumped through wells drilled to the mining horizon. The resulting brine is pumped to solar evaporation ponds after which the salts are slurried in spent brine and processed by flotation in the original mill. Production (in 1972) was two-thirds that achieved by conventional mining at about three times the productivity per man.

Feasibility

The efficiency of the solutioning process is dependent on several factors, including:

- The thickness of the potash
- The exposed surface area
- The solubility of the rock above and below the potash zone
- The potash grade
- The temperature and salt concentration of the feed brine
- The residence time of the feed brine in the potash zone
- The pumped flow rate.

The ore zone at Wills-Weaver averaged 40 inches thick, containing 24 inches of high grade sylvite with mixed carnallite/polyhalite/sylvite above and below the heart. The potash bed is contained within a thick halite zone.

The limited height of the potash bed is offset by the large surface area exposed by pillar mining. However, for an average 75 percent extraction and assuming no closure occured, the ratio of pillar wall to combined roof and floor areas would be approximately 1 to 15. With the closure that has occurred, this ratio may be effectively decreased due to pillar creep.

The solutioning process for a zone comprising halite, sylvite, and carnallite is complex and time dependent. Given unlimited sources of these minerals and sufficient time, the solution will always reach an equilibrium condition in which the dominant salt would be magnesium chloride with lesser quantities of sodium and potassium chlorides. This is of particular importance for Wills-Weaver, where significant quantities of carnallite (containing magnesium chloride) are associated with the sylvite and would preferentially go into solution. Additional amounts of sylvite could in fact be precipitated in the solid phase, due to the excess potassium chloride contributed from the carnallite.

The rate of dissolution of the in situ halite and potash minerals is obviously quite important to the success of a solution mining operation, and there is no theoretical method of determining the intermediate stages of solutioning. As the mine workings will be flooded with water as the first step in solution mining, initial pumping may recover a brine containing low potassium concentrations. Thus, unless the temperature and residence time of the solution in the potash zone and its concentration of ions can be critically controlled, the economic recovery of potash may not be possible.

Early attempts at solution mining in virgin potash (Husband 1971) failed due to low production rates, dissolving roof salt, accumulation of sediments in the cavity, and thickness of the potash bed. The economic viability of solution mining cannot be properly determined until an operating scheme has been engineered. However, by considering only the potash contained in the pillars, it is estimated that Wills-Weaver could produce potash (at an assumed rate of 300,000 tons per year) for some 6-1/2 years. This would be a lower bound value for the life of a potential operation.

Impact of Solution Mining

A solution mining operation will have several impacts on the Wills-Weaver Mine and the surrounding potash district.

- Evaporation Ponds: Solar evaporation may be an economic way to concentrate sodium and potassium salts as the initial step in processing, as it is suited to a dry, hot climate. Based on the Cane Creek operations, it is estimated that an area of several hundred acres may be required to provide sufficient pond capacity for solar evaporation. This should be located in the general area of the mine and the treatment plant.

- Water Consumption: A considerable volume of water would be required to sustain a solution mining operation, and this may impact other water users (domestic and agricultural). Apart from the initial requirement to fill the mine workings (est. 400 million gallons), the volume of makeup water would be quite considerable, approximately 1,000 gpm for an annual production rate of 250,000 tpy of potash (Husband 1971).

- Effect of Solutioning: The extent of the mine workings (some 720 acres) is such that the degree of control that could be exercised over the solutioning process may be quite limited, even with multiple injection and pump wells. It is therefore probable that:

  - The shaft pillar will be effectively removed by solutioning
  - Halite will be removed from the floor and possibly the roof of the mined area

- Solutioning will occur at the boundary of the mine workings.

We are presently unable to estimate with any reliability the rate and extent to which this will occur during solution mining.

- Brine Reservoir: At the completion of solution mining operations, the underground cavity will contain a large volume of brine. This material could be largely pumped out of the workings and the mine void abandoned so as to prevent water ingress. If the brine was not pumped out or the mine void was not adequately sealed against water inflow, consequences similar to those described in Section 8.0 could result.

10.2.1.3 Oil and Gas

The oil and gas industry maintains an ongoing interest in the potential for new production from fields below the NcNutt potash zone. Procedures have been established (BLM 1975) to ensure that the development of both potash and oil and gas resources can proceed in a consistent and orderly manner. These procedures include the following.

- The preparation of maps to be updated annually, showing active and completed mining operations, minable reserves, and known barren zones within these areas. This is to be done for both leased and unleased land within the designated potash area.

- Permits to drill oil and gas tests from surface locations within the above areas would only be granted:

- From barren areas within the identified minable reserves.

- From a "drilling island" within the identified minable reserves when the target cannot be reached from either barren areas or by a well directionally drilled from outside the reserve area.

- No "island" will be established within 1 mile of any area where approved mining operations will be conducted within 3 years.

A minable reserve is considered (loc. cit.) to include "those areas where potash ore is known to exist in sufficient thickness and quality to be mineable under present day technology and economics." Based on existing information, no minable reserves are located adjacent to the Wills-Weaver mine. Unrestricted drilling for oil and gas is therefore possible outside the present mine workings.

Should such a minable reserve be identified due to improved technology and economics, all producing wells would have to be protected by pillars on the mining horizon. There are currently three producing wells within the Wills-Weaver mine, protected by 150-foot radius pillars. Well operation apparently has not been affected by mine closure and subsidence of the surrounding area. It is concluded that, while oil and gas wells located in the vicinity of Wills-Weaver mine may reduce flexibility in any future mine planning, they do not unreasonably limit future potash exploitation in the area of the mine.

10.2.2 Storage of Materials

The mined workings of the Wills-Weaver mine may be considered for the storage of several types of material either in transient, short-term storage or long-term disposal. The distinction between short- and long-term storage is related to the criteria that may be required for facility operation.

In the former, the facility may be used typically for several decades. The lower limit would be determined by the time needed to recoup setup costs; the upper limit would be determined by need. After storage operations ceased, abandonment procedures for the mine would have to be determined, as it is unlikely in most cases that it would be re-opened for mining at that point.

Long-term disposal, however, is based on dedication of the facility to storage for time periods that are likely to exceed several hundred years. Any plan for long-term disposal must therefore include provision for secure abandonment as an integral part of the proposed operation.

Underground storage of materials can be desirable for one of several reasons.

- Economic - the cost of using available underground space can be considerably less than the construction of surface storage facilities

- Isolation - disposal of hazardous waste materials by underground burial can minimize their impact on the environment.

- Security - access to strategic or otherwise important materials can be readily restricted by underground storage. Terrorism or sabotage can be resisted.

10.2.2.1 Short-Term Storage

This category includes all forms of energy storage, including compressed air, natural gas, LPG, and crude oil, plus warehousing storage.

High Pressure Air

Compressed air energy storage (CAES) is used as a power source for peak shaving electrical energy generation. This may be coupled with thermal energy storage, in which the temperature of the compressed air is boosted with hydrocarbon fuel, allowing operating storage pressures to be reduced.

CAES is considered feasible where significant electrical demand over baseload exists. Expansion of the potash mining industry may provide this demand. Potash mining is projected to continue for at least 80 years, possibly 200 years, depending on maximum possible resource utilization (BLM 1975). Use of CAES is therefore likely to be a long-term commitment, if it is considered at all.

- Storage Concepts: The storage concept most applicable to a cavern in salt would be dry storage at constant volume, in which the air pressure fluctuates between maximum and minimum limits determined by the operating characteristics of the turbines. The alternative concept, compensated or wet storage, in which air pressure is maintained at an operating level by water pressure,

is not feasible for a cavern in a soluble media unless saturated brines are employed.

- Design Considerations: Typical operating pressures would be in range of 250 to 800 psi. For the dry storage concept, however, the pressure loading consists of a constant minimum loading with a superimposed cyclic surcharge. With a suggested operating pressure of 500 psi, it is possible that a CAES facility could be operated at Wills-Weaver.

  The air temperature entering the mine during the storage phase may be of the order of 50°C (122°F), which is in excess of ambient rock temperature at that depth. This will also be cyclic in nature. The increased temperature may increase the rate of closure of the mine, but it is unlikely to cause severe structural damage.

  The successful operation of a CAES facility depends on both the stability of the cavern and the absence of major leaks. Continued closure of the mine cannot be prevented; it will be essential, however, that inflow of groundwater through the shafts is prevented to avoid structural damage by solutioning.

- Impact: Construction of appropriate surface plant would be required, replacing all of the mining-related facilities. The mine shafts would be stripped and sealed. Pressurized inlet and exhaust piping would be installed to the mining horizon. This could be done either through the shafts or by cased boreholes.

The pressure within the cavern can be easily controlled, as the medium is non-toxic and can be vented to the atmosphere if necessary. If by some circumstance leakage should occur to the adjacent mine workings, no adverse effects should be observed.

Natural Gas and LPG

The storage of natural gas and liquified propane gas (LPG) in underground caverns is common practice and has been developed over the past 40 years (Cobbs Engineering 1975). It is successfully carried out in a variety of rock types, including granites, shales, and limestone, as well as in salt.

The location of the Wills-Weaver mine is considered attractive for gas storage, and the thickness and permeability of the salt beds are likely to provide secure gas containment.

- Storage Concepts: Natural gas may be stored freely under low pressure or in liquified form (LNG). Natural gas storage caverns are typically quite large, often in excess of 10 billion ft^3 (Duffaut 1979). Storage of LPG generally is carried out in smaller caverns, due to the higher storage pressures required. For LPG caverns, it is essential that sufficient hydrostatic head exists to counterbalance the vapor pressure of the stored product. Saturated brine is frequently used to displace the LPG during withdrawal, particularly in caverns at shallower depths.

- Design Considerations: The successful operation of a gas storage facility depends on the stability of the cavern and the absence of major leaks.

At the storage pressures considered appropriate for Wills-Weaver (about 500 psi), the overall stability of the mine is not likely to be worsened. However, continued closure due to the high extraction cannot be prevented. Pressure within the mine during storage operations will be cyclic, and cyclic temperature changes of up to 30°C may be experienced. While these changes are unlikely to have a major influence on overall mine stability, the surface of the mine roof is likely to deteriorate by induced tensile fracturing.

Inflow of groundwater from the shafts should be prevented. While this does not create problems for the storage of petroleum products, which are immiscible, the water will cause solutioning and is likely to hasten the closure of the mine by weakening pillars. Gradual closure of the mine, or slabbing of loose rock, is unlikely to significantly affect gas storage operations other than by loss of storage volume.

Leakage may occur through either natural or man-made features. It is essential, therefore, that all boreholes and the mine shafts are adequately sealed for the proposed storage operating pressure.

The potential for leakage through the salt itself is quite small. Major structures such as faults and breccia pipes are not commonly found, and the salt permeability is quite low. The water permeabilty of the pervasive bedding features such as clay seams is also expected to be less than 10^{-6} cm/sec.

- Impact: Minimal surface construction would be required. The mine shafts would be stripped and sealed, and pressurized inlet and exhaust piping would be installed to the mining horizon.

  Normal practice would require a pressure acceptance test before storage operations began. Due to the proximity of adjacent mining properties (approximately 2 miles), it would be considered prudent to install a gas monitoring system within those mines. It is extremely unlikely, however, that significant quantities of gas would permeate over such a long distance through a low permeability material. Only major, uncontrolled flooding and dissolution could modify this condition.

Crude Oil

Considerable attention has been recently focused on the underground storage of crude oil as a strategic reserve (Davies and Willett 1977). Emphasis has been placed on storing imported oil, so storage cavern locations close to Gulf or East Coast ports have been considered most desirable. The Louisiana salt domes are obviously prime candidates.

The Wills-Weaver mine is located in a major oil producing area, and it is therefore possible that it may be considered as a future oil storage facility for either domestic or foreign production if it can deliver oil at acceptable quality and cost during import interruptions.

- Storage Concepts: Siting factors that have been established as important for oil storage in existing mines (loc. cit.) include the following:

- A minimum oil storage capacity of 20 million barrels
- On-site conversion costs
- Cost of interconnection with the oil distribution system
- Simple geometrical shape
- Geotechnical/structural conditions.

The maximum available storage volume at Wills-Weaver mine could provide a credible storage facility. The 60 million ft^3 (1.7 million m^3; 10.5 million barrels) compares reasonably well with a typical range of 5 to 10 million m^3 (Duffaut 1979), particularly when the possible addition of other worked-out mines to a storage complex is considered. However, the storage volume is likely to be reduced by closure that will undoubtedly occur, and some loss of oil may result by entrapment.

- Design Considerations: The room and pillar workings and the regular dip angle provide a suitable geometry for storage operations. Oil pumped into the mine will flow downdip to the lowest mine elevation, and submersible pumps used to withdraw the oil must be installed in large dimater boreholes located at the eastern boundary of the mine to avoid loss of oil in low level sections of the mine. For maximum recovery of oil, the pumps should be located in sumps excavated below the minimum oil level. Oil filling may be accomplished through lines located in the mine shafts.

  All shaft openings must be sealed by concrete plugs designed to withstand explosion overpressure of 10 atmospheres (150 psi). Vapor pressure build-up would be reduced by venting to a vapor recovery system.

Normal practice in preparing a storage cavern would involve ensuring long-term stability of the excavation. This encompasses structural support and securing loose or potentially floating material that could damage pumps and lines. It is unlikely that the Wills-Weaver mine could be properly prepared to acceptable standards due to loss of access.

Successful operation of an oil storage facility depends on the cavern stability and the absence of major leaks. Continued closure of the mine cannot be prevented, but the low permeability of the salt and the minimal distortion anticipated to the overlying beds would tend to ensure that no oil leakage or groundwater contamination would occur. Inflow of groundwater through the shafts should be prevented.

- Impact: Construction of surface pumping and associated facilities would be required. Oil storage at Wills-Weaver may not be possible on the grounds of structural instability and high cost. Such an operation, however, is not likely to adversely impact adjacent potash mining operations.

Warehousing

The use of underground space as a warehousing facility may be considered primarily for security reasons.

- Storage Concepts: This category of materials includes supplies, records, strategic materials, etc., which may be contained in cartons, drums, or on pallets. These would be either stacked or otherwise laid out in storage

rooms, separated by aisles to allow loading equipment access. Some of these materials may be perishable, requiring particular conditions of temperature and humidity for successful storage.

- Design Considerations: The following important requirements must be met:

  - Adequate shaft dimensions and duty
  - Adequately sized access and storage areas
  - Long-term stable underground openings
  - Control of groundwater inflows
  - Control of temperature and humidity.

  Shafts - The Wills-Weaver main shaft and hoist is adequately sized to allow typical palletized materials to be lowered. Rehabilitation and re-equipping of shafts and hoists would, however, be required.

  Underground Openings - Of the total mine area, approximately 8×10^6 ft^3 of total storage in the Main East entry could be considered for packaged storage and associated access. Even in this area, however, the original entries were mined to a height of only 52 inches. These entries are not large enough to operate conventional forklift equipment adapted from the mining industry. It is unlikely that re-mining the entries to a more appropriate height could be justified on the basis of cost.

  The Main East entry was mined at 60 percent extraction, and closure and deterioration may be expected to occur progressively. Systematic ground support would have to

installed to ensure a minimum acceptable level of safety before storage operations could be started.

Control of Groundwater/Temperature - It would be essential that any groundwater entering the mine through the shafts be sumped at the shafts and pumped to surface. Drainage into storage areas, following the natural slope of the mine, would not be permissible.

The temperature of the mine air could be maintained at 70 to 80 degrees Fahrenheit by normal ventilation, and even cooler (60°F) if refrigeration were used. With adequate flow of ventilating air, the humidity in the storage areas should be very low.

- Impact: Surface handling facilities such as receiving and temporary storage areas and shaft loading facilities may be readily adapted to handle warehouse materials. The impact of this operation is therefore quite small.

On the basis of the restricted working height, however, it is concluded that the Wills-Weaver mine is unlikely to be suitable for the storage of warehouse materials.

10.2.2.2 Long-Term Disposal

This category includes the permanent disposal of toxic wastes, some of which are presently injected into natural formations, and other waste products.

Industrial/Chemical Waste

- Storage Concepts: The number of toxic chemical wastes is very large, and the economic feasibility of storing and managing these products has not been clearly established (Stone et al. 1975). While the waste may be injected or pumped into the formation in liquid form, concern over long-term contamination of the environment has led to consideration of waste treatment, containerization, or encapsulation. These procedures will also minimize the chemical interaction between various wastes, with possible byproduct formation, and between the waste and the host rock.

- Design Considerations: Factors that must be considered for chemical waste storage include the following:

  - Stability of the storage cavern
  - Hydrologic and surface isolation
  - Chemical compatability of the stored wastes with the host rock and each other.

  A rock salt or potash environment is chemically compatible with the majority of industrial wastes considered for storage (loc. cit.), and the depth of the Wills-Weaver mine is within a range determined to be acceptable for waste isolation (600 to 1,200 feet).

  The migration of the waste, either to surface or to aquifer systems, may occur either by direct leakage or by dissolution in groundwater. Direct leakage would be more likely to occur if the waste is placed in fluid form; sealing the mine against groundwater inflow will reduce the potential for dissolution.

Containerized Waste - The poor structural stability and lack of access in the Wills-Weaver mine is such that the Main East entry is likely to be the only acceptable storage space. The shafts are adequately sized to handle palletized waste containers, but rehabilitation and re-equipping would be necessary. The height of the mine workings is inadequate for conventional forklift equipment, and underground handling could be achieved only by using modified mine loading equipment. Systematic ground support would have to be installed to ensure a minimum level of safety.

Containerized waste should be completely isolated in order to inhibit container corrosion and waste dissolution. It is therefore important to minimize the entry of water into the mine at completion of storage operations, and proper shaft sealing techniques would be required. Over time periods of hundreds of years, ultimate containment will be provided by the low permeability of the surrounding salt beds. Treatment of waste before containerization may reduce its toxicity.

Liquid Waste - The quality of the mine workings is not important for the injection of liquid waste, and the total potential storage volume is available. The facility would, however, be limited to the disposal of one or possibly a few non-reactive waste forms.

Migration of the waste in groundwater flow is obviously a potential problem should groundwater re-entry occur. Over long time periods, however, there is probably little significant difference compared with treated containerized waste. This is because of the very long time

required for groundwater flow through the impermeable salt compared to the likely dissolution time.

Continued closure of the mine workings may pressurize the fluid waste and induce leakage past shaft and borehole seals. This effect could be reduced by filling only a fraction of the total potential storage volume--say 25 percent or less. In any case, the seals should be designed for the maximum anticipated fluid pressures to ensure that leakage does not occur.

- Impact: The long-term disposal of toxic industrial waste in underground storage is not a proven concept, and absolute security against leakage cannot be guaranteed. However, this would not be likely to occur during the life of the local potash mining industry.

Storage of industrial waste at Wills-Weaver may not be considered for economic reasons, including distance from industrial centers. The short-term impact of such an operation, other than sterilizing the mine for future mining, is considered to be minimal.

Radioactive Waste

- Storage Concepts: Storage of nuclear waste in underground geologic repositories is a very controversial issue. The heat-producing, high level waste will be subject to disposal criteria, for both site selection and repository design, that will never be met by an abandoned mine such as Wills-Weaver. The only waste form that could therefore be considered for disposal at this site would be low level waste.

Low level wate consists of non-heat producing, alpha emitting waste with some very small quantities of long half-life actinides. Such waste is presently disposed of in surface land burial facilities under carefully monitored conditions to prevent long-term contamination of the environment.

- Design Considerations: The major design issues are as follows:

  - Security of waste handling from surface to the underground storage area
  - Stability of the storage area
  - Control of groundwater inflow.

  A considerable amount of work would be required to adapt the shaft handling facilities for nuclear waste handling, primarily in relation to safety features such as controls and interlocks. While the low level waste containers can be safely handled by mine operators, accidental spillage should be avoided.

  The present mine workings are inadequately sized and poorly supported for a low level waste storage facility. Again, security against accidental spillage is of major concern. It is unlikely that the openings could be re-excavated or supported in a manner that would meet the design criteria currently indicated for an operating repository. Extraction ratios of approximately 45 percent are considered an upper limit for low level waste storage. Long-term release of radioactive material is likely to occur through dissolution in groundwater and migration to the biosphere. Total control of groundwater inflow would therefore be required, and present

minor inflows through the mine shaft would have to be sealed off.

- Impact: It is unlikely that the Wills-Weaver mine in its present form would satisfy design criteria indicated for underground disposal of low level nuclear waste. The incentive to modify the facility, even if it were possible, does not appear to be strong, as licensed surface disposal facilities are adequate for present waste disposal needs.

- Shaft Sealing Trials: A critical design feature of a high level nuclear waste repository is the method of sealing the access and ventilation shafts at decommissioning. The seal must prevent entry of water from above and escape of contaminated water under pressure from the resaturated repository. The shaft seal must be effective for hundreds of thousands of years, a time period for which no field experience exists.

  Monitored field trials of potential shaft sealing techniques will therefore be required to develop and verify seal design criteria. The Wills-Weaver mine would provide a satisfactory site for these trials in a rock salt environment and would be particularly useful for the proposed WIPP site repository.

Septic Waste

- Storage Concepts: Where sewage treatment facilities are inadequate, raw sewage or primary treatment effluents could be pumped into underground disposal ponds. The volume of septic waste and the distance to be pumped must be considered.

- Design Considerations: The four major population centers are located 30 miles or more from the Wills-Weaver mine, and all have sewage treatment facilities servicing 95 percent or more of the population. The plant capacities are all greater than peak loads by an adequate margin (BLM 1975).

  The nearest city, Carlsbad, has a trickling filter type system with 6 million gpd capacity and peak load of 4.7 million gpd. The system discharges into the Pecos River. The city's solid waste disposal is a sanitary landfill.

  Design features considered would be similar to those for liquid industrial wastes discussed previously. Contamination of water supplies by direct leakage would be possible as mine closure continued, and appropriate shaft sealing procedures would have to be adopted.

- Impact: The disposal of septic wastes in underground storage may lead to groundwater contamination if shaft seals are inadequate.

The existing treatment facilities in all nearby population centers are adquate for continued growth, and would be the preferred method of processing septic waste.

10.2.3 Other Uses

Other uses for the Wills-Weaver mine would include those where personnel access only is maintained. The value of the space for living or for commercial use is very much limited by the remote location of the site, and the only use considered here is for personnel shelters.

10.2.3.1 Personnel Shelters

- Design Considerations: The important factors considered for personnel shelters are:

  - Security of access
  - Long-term stable underground openings
  - Control of groundwater inflows
  - Control of temperature/humidity.

  Access - The Wills-Weaver shafts and hoists may be inadequately sized for personnel use where ability to quickly transport large numbers of people is important. Rehabilitation and re-equipping would be required in any case to ensure safety. This would include the following:

- Separation of ventilation intake/exhaust and personnel travel ways by shaft dividers.

- Installation of automatic, elevator type hoists, and controls in both shafts.

  Underground Openings - Of the total mine area, only the shaft pillar area and perhaps portions of the western part of the Main East entry could be considered. Systematic ground support of a high quality would have to be installed to ensure personnel safety, and the rate of closure in the area should be small to minimize structural damage.

  Control of Groundwater/Temperature - It would be essential that any groundwater entering the mine through the

shafts is sumped at the shafts and pumped to surface. Drainage into storage areas, following the natural slope of the mine, would not be permissible.

The temperature of the mine air could be maintained at 70 to 80 degrees Fahrenheit by normal ventilation, and even cooler (60°F) if refrigeration were used. With adequate flow of ventilating air, the humidity in the storage areas should be very low.

- Impact: Surface facilities would require considerable adaptation for personnel use but the impact on the area would be small.

On the basis of the restricted working height and poor accessibility, it is concluded that the mine is unlikely to be suitable for personnel shelters.

10.3 SUMMARY

The principal potential uses of the Wills-Weaver mine and mine shafts can be considered as belonging to one of two general categories--mineral extraction and storage of materials. The following items summarize current considerations regarding future utilization of the Wills-Weaver property.

- Based on existing data, the prospects of re-opening the Wills-Weaver mine for low grade potash exploitation within the First Ore Zone appear to be rather poor in the forseeable future.

- There is currently no information to suggest that economic potash deposits occur above the First Ore Zone at Wills-Weaver.

- Extraction of potash by solution mining in the potash fields of southeast New Mexico has not yet been proven to be a viable operation. In particular, the presence of highly soluble magnesium chlorides might inhibit dissolution of the potassium salts. The availability of adequate supplies of water for such a mining technique in this region is doubtful.

- Based on existing information, no minable potash reserves are located adjacent to the Wills-Weaver mine. Unrestricted drilling for oil and gas should therefore be permissible outside the present mine workings. While the presence of such wells might reduce flexibility in any future mine planning, they do not unreasonably limit future potash exploitation in the area of the mine.

- The use of the Wills-Weaver mine for compressed air storage (to be used as an energy storage for peak load electrical generation) is considered technically feasible. The requirement for such an energy store is dictated by significant electrical demand over baseload. The present need for such a facility in the potash mining district would determine the likelihood of such a utilization of the mine.

- Utilization of the mine for natural gas storage is considered attractive. It is understood that the need for such storage exists and the Wills-Weaver mine appears to be technically suited to such an operation. The geology of the area is likely to provide secure gas containment but it would be necessary to adequately seal the shafts and to check the suitability of the corehole and well seals. It is believed that the probability of gas

leakage to adjacent properties (approximately 2 miles away) through the highly impermeable Salado Formation is low. However, gas monitoring both within the adjacent mines and at intermediate locations would be necessary.

- The Wills-Weaver mine might not be suitable for crude oil storage as a result of the following factors.

  - The volume of the working is substantially less than the usual minimum storage volume which is considered for such an operation.

  - The lowest point of the workings at which withdrawal pumps, etc., must be located is in an area of poor stability. It is very unlikely that the mine could be prepared to acceptable structural standards for such an operation.

- The Wills-Weaver mine is unlikely to be suitable for storage of warehouse materials (e.g., supplies, records, strategic materials) because of its unfavorable geometric configuration.

- The long-term disposal of toxic industrial waste material in underground storage is not a proven concept. Concern over long-term contamination of the environment has led to considerations of containerization of waste prior to disposal. The geometrical configuration of accessible areas of the Wills-Weaver mine is not considered suitable for underground placement of waste containers.

The distance from industrial centers which produce toxic wastes might also adversely impact economic considerations.

- Underground storage of radioactive waste materials is a subject of much interest at present. Disposal criteria for high level (heat producing) wastes would never be satisfied by an abandoned mine such as Wills-Weaver. In addition, the mine workings are inadequately sized and supported for even a low level waste storage facility, and there appears to be little incentive for significant expenditure in modification of a presently unacceptable facility. The mine would, however, provide a very useful location for monitored field trials of proposed shaft sealing techniques and designs, which will be required by DOE to demonstrate satisfactory long-term seal performance.

- The disposal of septic wastes in the mine may lead to groundwater contamination, and the method is not considered appropriate. Existing waste treatment facilities are adequate for present demand.

- The Wills-Weaver mine is unlikely to be suitable for use as a personnel shelter because of its restrictive geometry and poor accessibility.

11.0 WASTE PILE RECLAMATION PROCEDURES

11.1 INTRODUCTION

In the near vicinity of the Wills-Weaver mine shafts there is a waste stockpile of approximately one-half million tons of gobbed halite, sylvite and carnallite. This material is solid waste produced during mine development. Refining of the ore was carried out elsewhere and consequently there are no refinery tailing stockpiles or brine effluent ponds at the Wills-Weaver site.

In order to put the above quantity of material into the correct perspective, the Preliminary Regional Environmental Analysis Record - Potash Leasing in Southeastern New Mexico reports that there is an estimated 200 million tons of salt tailings piled adjacent to refineries in the area, and this is increasing at a rate in excess of 14 million tons per year. The refineries discharge three parts of halite for each part of sylvite produced. In addition to the tailings piles, the refineries discharge approximately 10,000 acre feet of brine effluent each year and this brine is collected in tailings ponds.

The surface waste deposits at the Wills-Weaver site therefore constitute a very small fraction of the total surface waste within the potash mining area. In recommending waste reclamation procedures for the Wills-Weaver site, it is important to recognize that the total problem of surface waste reclamation should be considered.

The first step in determining what mitigating measures might be applied to the surface stockpiles, is to evaluate the environmental impacts for which the stockpiles are responsible. The information contained in the following discussion has been drawn from the Preliminary Regional Environmental Analysis Record

for the potash mining area of southeastern New Mexico.

11.2 ENVIRONMENTAL IMPACTS OF SURFACE STOCKPILES

Potential impacts to the environment caused by surface salt stockpiles may be classified into the following major groups:

- air quality
- water quality
- land usage

Air Quality

Potash mining oeprations in general have a significant adverse impact on air quality in the potash mining area. The primary air pollutant emissions are particulate in nature (salt dust) and emanate from the refining operations. Federal and state ambient air quality standards for particulates are currently being exceeded but drop to acceptable levels within approximately three miles of a refinery.

The salt stockpiles tend to form into a hard cohesive mass and would not be a significant contributor to a lowering of the air quality. In addition, the area of land denuded of vegetation and thus exposed to wind erosion with associated reduced air quality as a result of proximity to surface waste piles, is minimal.

Water Quality

Potash mining operations in southeastern New Mexico are a source of groundwater pollution. As mentioned previously, refining operations discharge approximately 10,000 acre feet of brine effluent each year into tailings ponds and these structures can leak. The impact on the hydrology of the area is considered

to be small because of the dominantly poor quality of the groundwater in the area. Only small areas of aquifers useful for stock watering exist within the Rustler Formation and the Dewey Lake Redbeds are not known to yield useful supplies of water.

At present there is no evidence of salt pollution of the Pecos River from refinery operations but some pollution may ultimately reach the river.

The other potential source of groundwater pollution is rainfall runoff from the salt stockpiles. The regional impact from this source is considered negligible as a result of:

- limited rainfall
- low humidity and high rate of evaporation
- comparatively small surface area of existing tailings piles.

Land Usage

Existing potash activities in Eddy and Lee Counties of southeastern New Mexico occupy a total of approximately 4,000 acres of surface land. This area of land could support livestock forage for approximately 55 to 60 head of cattle. Loss of grazing land due to potash mining operations (which include stockpiling waste material on surface) is therefore minimal.

Greatest impacts on land usage occur in the immediate vicinity of the refineries. Due to particle dust emissions, all natural vegetation to within 1/4 to 1/2 mile from the refineries is dead. In the vicinity of the tailings piles and brine disposal areas, the excessive salt presence has rendered the underlying and immediately adjacent land permanently sterile. The region adjacent to surface waste which has been destroyed is very limited - approximately 10 to 25 feet from the base of the pile.

While refining and surface waste stockpiling operations have a severe local effect on surface vegetation, the regional impact is negligible due to the small areas of land affected.

It has also been recognized that mining operations have a negligible impact on recreational activities and a very limited impact on visual appearances of the area. The scenic quality of the region has been evaluated as 'top of class C' on the Bureau of Land Management scenic quality scale.

11.3 WASTE PILE RECLAMATION PROCEDURES

The waste stockpiles of the potash mining area of southeastern New Mexico are considered to have a small to negligible regional impact on air and water quality and on land use optimization. One possibility that has been suggested, however, is that over a period of time, the action of rainwater will tend to spread the waste piles over a much larger area of land, destroying vegetation and enhancing wind erosion. The rapidity with which this event might take place will be determined by the annual rainfall, the topography of the waste pile and surrounding surface and the permeability of the material surrounding the waste. The low annual rainfall of southeast New Mexico ensures that dissolution of the salt piles will proceed at a relatively slow rate. If the stockpile is contoured so as to limit runoff, much of the rain will evaporate and the salt will not be transported away from the pile. If the waste is not adequately protected however, dissolution will tend to reform the stockpile configuration over a long period of time so that runoff tends to be maximized, i.e., topographic lows on the surface of the salt will initially act as sinks and will eventually be filled with salt as inflowing brine is evaporated. Salt will be carried into the groundwater system by runoff down the sides of

the stockpile. Since the stockpile itself forms into a relatively impermeable mass, the quantity of brine which enters the groundwater through the base of the stockpile will be negligible. A relatively permeable surface material will inhibit spreading of the waste material by encouraging infiltration of the brine flowing from the waste piles. The very limited spreading of waste piles that has taken place to date - as evidenced by the existence of natural vegetation almost to the edges of the piles suggests that infiltration is tending to inhibit any lateral surface migration of the salt.

There are, however, a number of mitigating measures which could be applied to prevent lateral spreading of the waste and at the same time to inhibit groundwater pollution caused by brine infiltration. Neither of these impacts is considered very significant.

Underground Disposal of Waste

Underground disposal of surface tailings cannot be recommended as a viable alternative for a number of reasons:

- The present mining system of first mining followed by pillar mining does not easily permit placement of tailings underground as part of the mining operation.

- With the situation as it exists at present, there are insufficient underground voids in the potash area to accommodate the total volume of surface tailings. At the Wills-Weaver mine there are probably sufficient accessible mine voids to accept the surface waste, as a result of the substantial area of comparatively low extraction and the limited volume of surface salt. As

indicated previously however, any action taken in relation to surface waste at Wills-Weaver will be inconsequential unless it is part of an overall strategy applied to all surface waste.

- The surface salt, under the action of rainfall, has welded into a cohesive mass which would require remining before it could be transported (underground or elsewhere). This type of operation would not be economically feasible.

Containment of Surface Waste

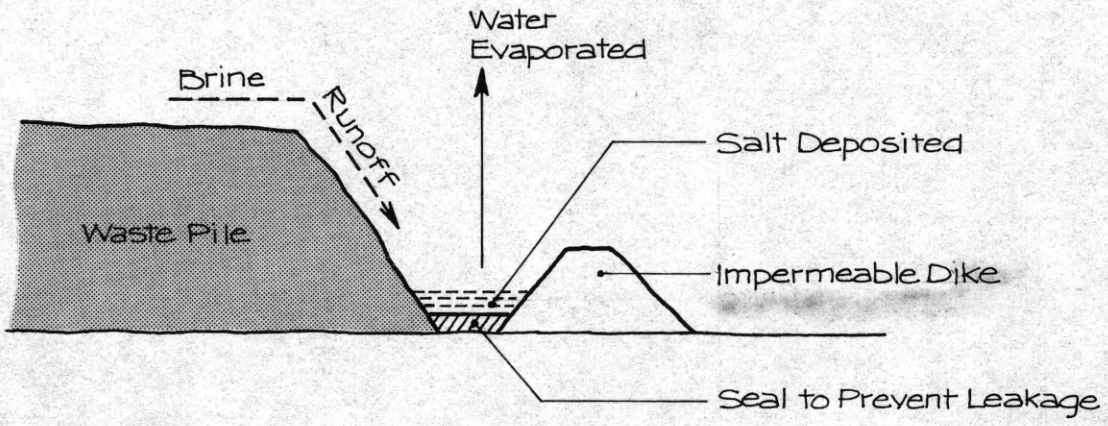
The "Preliminary Regional Environmental Analysis Record - Potash Leasing in Southeastern New Mexico" recommends that, where feasible, a dike should be constructed around the edge of waste disposal areas to restrict the flow of runoff. The following comments are offered in relation to this type of solution.

- Construction of a containing dike around the edge of a waste disposal area is shown diagramatically on Figure 11.1. It would be necessary to seal the ground surface between the stockpile and the dike, otherwise the brine runoff would not be contained.

- As water was evaporated and salt deposited behind the retaining dike, the storage volume would eventually be filled with salt and the dike would ultimately be unable to impound the runoff. The retaining dike concept would therefore delay the escape of the salts from the waste stockpile but would not prevent it unless the dikes were built to quite massive proportions (considering the geometrical configurations of the existing stockpiles).

RETENTION OF WASTE STOCKPILE WITH IMPERMEABLE DIKE

Figure 11-1



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- It is believed that properly constructed impermeable dikes could contain the salt for a long period of time but this is attributed to the very limited average rate of brine runoff.

- The construction of retaining dikes to contain waste stockpile brine runoff is not recommended because:

  - it is not a permanent solution unless massive dikes are formed.
  - it is not a cost-effective solution.

Contouring of Surface Waste

In order to prevent lateral spreading of salt from the stockpile or the addition of brine to the groundwater system, the stockpile could be contoured so as to minimize runoff and maximize evaporation. This abandonment alternative is shown diagrammatically on Figure 11.2. By proper contouring of the surface of the pile, it is possible to limit runoff to that solution originating from the side-slopes of the stockpile.

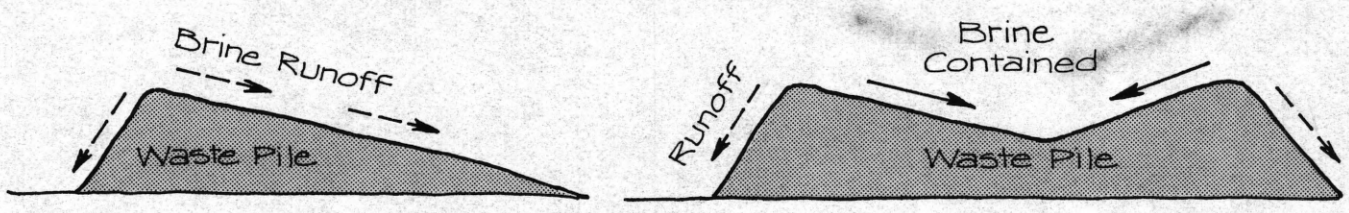
With time, however, the dissolution-transportation-evaporation cycle will tend to destroy the original contour and eventually reform the configuration of the stockpile to once again maximize surface runoff. Because of minimal rainfall, this process would take a very long time.

11.4 OBSERVED CONDITIONS AT WILLS-WEAVER

The waste stockpile at Wills-Weaver lies to the east of the No. 3 shaft. It is approximately 400 feet in width (east-west) and 1000 feet in length (north-south). The eastern side of the stockpile has a maximum height of about 50 feet with roughly 40 degree side slopes. The surface of the waste-pile slopes gently

STOCKPILE CONTOURING TO MINIMIZE BRINE RUNOFF

Figure 11-2



MAXIMUM RUNOFF

MINIMUM RUNOFF

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to the west from the eastern extremity, to the south from the northern extremity and to the north from the southern extremity so that all surface runoff flows to the central region of the west boundary of the pile and then onto the ground surface. Because of the ground surface topography and the formation of the waste-pile, slope heights on the western side of the waste pile vary from 0 to 15 feet.

Virtually all runoff from the stockpile is to the west. Runoff to the north, south and east is limited to the precipitation which falls on the respective sideslopes which do not present a very significant catchment area.

The following conditions were observed at the Wills-Weaver waste stockpile:

- To the north, south and east of the stockpile, where brine runoff is limited by the contour of the waste pile surface, vegetation appears to be adversely impacted to within about 30 feet of the base of the pile (Plate 16).

- To the west, where virtually all runoff from the waste-pile drains, the vegetation is less sparse. This area corresponds, however, to the mine surface station and the affected area appears to be no larger than that which would be influenced by the mine surface facilities alone. There is little evidence, therefore, that a significant area of vegetation is being adversely impacted by brine runoff even to the west of the stockpile, where virtually all runoff occurs.

11.5 RECOMMENDED PROCEDURES FOR WILLS-WEAVER WASTE PILE

It is recommended that no action be taken with respect to the Wills-Weaver waste pile. This recommendation is based on the following considerations:

- The waste material does not significantly adversely impact the regional air or water quality, or land usage.

- The lateral extent of the stockpile is not expected to increase significantly with time, leading to an increase in the area of surrounding land that will be rendered sterile by the salt. This position is based on the following observations and facts:

  - The area of land surrounding the stockpile that has been influenced to date is limited, despite the non-optimum configuration of the waste pile from a drainage point of veiw.

  - Runoff is limited because of the low annual precipitation, the high rate of evaporation, and the pitted and irregular nature of the surface which tends to retain water for evaporation rather than permitting runoff (Plate 17)

  - The surface blow sand is sufficiently permeable that runoff quickly infiltrates the ground instead of spreading laterally.

In situations where it might be considered that brine runoff from the stockpile is causing the area of sterile land to increase significantly, the most cost-effective solution is considered to consist of appropriate surface contouring of the waste pile in order to inhibit runoff.



Plate 16

Vegetaion located close to base of waste pile



Plate 17

Irregular nature of waste pile surface which tends to inhibit runoff

Respectfully submitted,

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APPENDIX A
SCOPE OF WORK

2.0 TECHNICAL DISCUSSION

The overall objective of the current program is to provide proper abandonment procedure recommendations for the Wills-Weaver mine and shafts, and reclamation procedures for the salt stockpile currently located in the vicinity of the shafts. A program plan has been formulated which identifies a number of individual tasks or activities which must be performed in order to achieve the required objectives. The following technical discussion identifies each of the separate tasks and identifies for each task:

- Purpose of the task and relation to objectives of the study
- General method of approach
- Anticipation of difficulties
- General supporting sevices required
- Man hours by labor category required for each task (see program plan).

2.1 REVIEW OF MINE AND SHAFT GEOLOGY

Purpose: To establish geological conditions in the vicinity of the shafts and across the mine site generally (including the geology of the foundation of the salt stockpile). The geology provides important baseline data with respect to anticipated hydrogeological conditions, stability of mine openings and shafts, etc.

Approach: The geology will be evaluated by reviewing available shaft sinking records, corehole data, oil and gas tests and water well logs, together with any other records of Wills-Weaver mine and surrounding geology.

Support Required: Provision of raw data pertaining to site geology, from USGS/Duval records, would be necessary.

2.2 REVIEW OF MINE AND SHAFT HYDROGEOLOGY

Purpose:

1. To establish hydrogeological conditions in the strata overlying the mine and underlying the waste stockpile. The hydrogeology of the strata overlying the mine constitutes fundamental baseline data in a study such as this. In particular, the location of all significant water-bearing aquifers, the hydrogeological characteristics of those aquifers, water quality, etc., constitute necessary base data for consideration of factors such as potential water inflow rates through shafts, wells, cored holes and strata disturbance resulting from mine subsidence, the potential impact of water ingress on mine and shaft stability through dissolution, and potential environmental consequences related to aquifer contamination.

2. To determine shaft water inflow rates and water quality. The shafts constitute a significant potential pathway for water ingress into the mine workings. In addition, the principal factor in determining future shaft stability is the quantity and quality of water migrating within the strata surrounding the concrete lining.

3. To establish water inflow rates into the mine workings through other pathways, e.g., cored test holes, gravity seepage through overlying strata, etc. The rate of water ingress into the mine workings (together with the quality of the water) from all potential pathways is an important

factor in establishing expected life of the underground voids. Apart from filling the underground cavities, the presence of water may also hasten mine instability through dissolution.

Approach:

1. The hydrogeology of the Wills-Weaver Mine Area will be established essentially by a review of existing data. This will include pertinent regional and local hydrogeological studies carried out by USGS or by USGS in conjunction with local agencies, existing environmental impact studies, together with site specific information obtained from shaft sinking records, water well logs, etc.

2. The shaft hydrology definition requires that regions of significant water inflow be established, that flow rates for each of those regions be determined, and that corresponding water quality parameters be defined. It is the presence of flowing water with a substantial dissolution capacity that is potentially dangerous to shaft stability, rather than the presence of water itself. It is known that substantial data on water inflow rates down the shafts exist, both prior to and subsequent to cessation of mining activities. Various collector drains have been installed within the Number 3 shaft lining, and water from these drains was initially piped to collector tanks located in the shaft pump station. Since the completion of mining, this water has simply been piped into the mine workings.

   The shaft hydrology will be initially established from a review of existing records. This will be complemented by visual observations obtained during detailed inspections of the Number 3 and Number 4 shafts. It is not known what water quality

sampling has been undertaken previously, and it will probably be necessary to collect appropriate water samples for chemical analysis.

3. The most direct approach towards establishing total water inflow into the current mine openings would be to locate and quantify existing mine water. It is not presently known whether this is a feasible option and the suitability of this approach will be assessed by considering factors such as the location of natural mine sumps, the stability of the backs, etc.

   Procedures for protecting existing oil wells and abandoning oil wells and cored test holes will be reviewed. This data will be obtained from the appropriate available records.

   If it is not possible to determine total mine inflow directly, predictive methods will be employed based on our best understanding of the mine hydrology.

Support Required:

1. Provision of raw data pertaining to site hydrogeology from USGS/Duval records.

2. Provision of raw data pertaining to shaft hydrology from Duval records.

3. Provision of mine plans, mine level contours, oil well protection and abandonment records, core test hole abandonment records.

2.3 DOCUMENTATION OF PRESENT STABILITY OF MINE

Purpose: To review the current condition of the mine workings from a stability viewpoint. An understanding of the present condition of the mine constitutes a fundamental starting point in any study of the potential life of the existing mine voids. This is of particular interest in the case of future possible utilization of the mine for energy storage, etc. The Wills-Weaver mine has generally employed room and pillar extraction methods with an extraction ratio of 65 percent. In the vicinity of the shafts, a safety zone of somewhat lower extraction (40 percent) has been established, and secondary pillar recovery has been achieved in some areas of the mine. The life of the mine voids will be determined by the rate of water ingress, the dissolution capacity of the incoming water, together with the rate of closure of the mine as a result of pillar creep/failure and caving of the back. The prediction of the life of the mine voids is also ultimately related to the rate of ground subsidence, with its consequent implications on shaft stability and hydrogeological environmental impact.

Approach: It is proposed that the review of current mine stability be achieved principally by geotechnical inspection of the workings. On the basis of existing mining records, different zones within the mine will be identified according to extraction ratio, age, etc., and for each of these zones the following geotechnical data will be collected:

- Average pillar and room dimensions
- Appearances of pillars - spalling, cracking, etc.
- Appearances of back - sagging
- Condition of floor - heaving

- Support (if any)
- Review of conditions causing roof and pillar distress, e.g., shale bands, etc.

The most considerable shortcoming is liable to be the extent of the mine workings which are sufficiently stable to enable visual examination. The shaft protection zone, which is of particularly critical importance in protecting the shaft from general mine subsidence, should be in adequate condition for fairly detailed inspection.

Support Required: Mine development plans and records (Duval); and access to the underground workings where appropriate.

2.4 DOCUMENTATION OF SUBSIDENCE DEVELOPMENT

Purpose: To document the current subsidence of the su[illegible] overlying the Wills-Weaver mine and to review subsidence patterns on adjacent properties which use high extraction ratio mining methods. An assessment of mine subsidence is of interest for two principal reasons:

1. It has a surface settlement and possible hydrogeological environmental impact.
2. It is directly related to an understanding of the shaft stability.

Present mining methods within the potash leases of southeast New Mexico employ high extraction ratio (90 percent) mining methods over substantial areas. Surface settlements associated with these methods are typically 60 to 70 percent of the thickness of the mined zone. Because of the lower extraction ratios generally employed at the Wills-Weaver mine, future subsidence

and overlying geological strata disturbance associated with closing of the underground openings at Wills-Weaver would be expected to be less than that presently induced on other properties as a result of the mining method. This problem must however be considered in a somewhat broader framework than the present study, and is not, therefore, included as part of the scope of work.

Subsiding ground can result in substantial damage to shafts, if they are not adequately protected by a sufficiently large zone of nonsubsiding ground. In addition to defining the conditions under which the mine voids will close, it is also necessary to understand the character of the overlying ground movements, if their potential influence on the stability of the shafts is to be predicted.

Approach: In addition to a review of subsidence data for the Wills-Weaver lease, substantial information relevant to describing the character of subsiding ground (e.g., magnitude of movements, angle of "draw", etc.), is available from surface subsidence induced by high extraction ratio mining. It is proposed to review all accessible records relating to this problem. In addition to data developed from the southeast New Mexico potash leases, a literature review will also be undertaken to develop an appropriate data base which will enable an assessment of potential shaft stability problems resulting from geological strata disturbance, to be made.

Support Required: Provision of subsidence records for the Wills-Weaver mine (Duval), and provision of similar data for selected areas of the southeast New Mexico potash leases (various mine operators).

2.5 EVALUATION OF ROCK MECHANICS DATA RELEVANT TO MINE AND SHAFT STABILITY

Purpose: To provide a data base which will enable the stability of the mine openings to be established. As already discussed, the life of the mine openings is of particular interest for abandonment procedures which require that consideration be given to future use of the mine voids. In addition to a review of the current condition of the mine, it is also necessary to provide a technology for assessing future stability. The stability of the mine openings may also have direct relevance to shaft stabilities, which may be adversely influenced by ground subsidence which accompanies closure of the mining horizon.

Approach: Two basic approaches are applicable to the development of procedures for assessing mine stabilty. Initially, stability may be assessed on the basis of an understanding of the mechanical properties (strength, rate of creep, etc.) of the appropriate materials, as determined by laboratory testing programs. Secondly, a similar body of data may be assembled from the results of observation of the performance of existing in-situ mines. The latter approach has particular advantages in that significant extrapolations are often not required and long-term behavior can be assessed from the performance of old mines, whereas it is generally impractical to determine similar data within the laboratory.

It is proposed to assemble pertinent data of both types. The data will consist of any site specific information which describes the mechanical properties of the material constituting the major structural elements of the mine, together with a literature review to supplement the above data. The extent and nature of such site specific information is not presently known.

Support Required: Provision of available rock mechanics/ mechanical property data pertinent to mine stability considerations (Duval, other operators).

2.6 DOCUMENTATION OF PRESENT CONDITION OF SHAFTS

Purpose: To review the present state of the Wills-Weaver shafts. The stability of the shafts can be influenced in two ways. Firstly, general ground subsidence resulting from collapse of the underground workings can result in destruction of the shafts. Secondly, shaft failure can occur as a result of factors which can be regarded as purely local to the shaft, e.g., shaft buckling caused by loss of bond to the rock as a result of dissolution, etc., shear failure of the concrete lining caused by excessive earth pressures. An assessment of this latter aspect of shaft stability requires that the present condition of the shafts be assessed.

Approach: It is proposed initially to review all available shaft sinking records in order to establish the construction and sealing details of the shafts. This will be complemented by a detailed shaft inspection which will report on the present condition of the guides, timbering, pump stations, sumps, gathering rings, etc.

The condition of the concrete linings will be established by obtaining in-situ compressive strength profiles for each lining. This is economically, conveniently and effectively achieved by running Schmidt hammer profiles down the length of the shaft. Areas of particular concern include:

1. Shaft lining in the region of the Magenta and Culebra Dolomite aquifers. Preliminary inspection of Number 3 shaft indicated a gravelly appearance to the lining in these regions, possibly caused by inadequate sealing during construction.

2. Shaft lining in the lower regions, where the potential for relatively high circumferential loadings exist, as a result of creep of the salt material.

In addition to the condition of the concrete lining itself, the state of the rock material behind the lining is of paramount concern to the shaft stability. This is particularly so for shafts within evaporite deposits, where the potential for dissolution is always present. One aspect of this problem will be considered by reviewing the overall shaft hydrology. It is also desirable that the presence of any air or water filled voids at the lining-rock interface be established. This is most conveniently achieved by the use of various radar methods which are able to define location and dimensions of air and water cavities behind a shaft lining. Preliminary discussions have been held with Geophysical Survey Systems of Hudson, New Hampshire. As yet, there remain a number of technical problems to be reviewed in order to assess the suitability of this approach, e.g., resolution of equipment for filled and unfilled voids, distribution of reinforcing steel within the lining, feasibility of providing equipment set-up on the top of the cage for relatively undisturbed traverses. Discussions with the above organization are continuing. The equipment is relatively compact and transportable.

The condition of the reinforcing steel can be most effectively established by exposing the steel in a number of selected locations. Typical worst locations would be expected to be regions of the shaft suffering higher rates of water ingress. This activity could be most effectively undertaken using portable hand excavators which require no service hook-ups.

Support Required: Provision of shaft sinking records (Duval); and access to both shafts--nature of access to be compatible with outlined testing requirements.

2.7 PREDICTION OF MINE AND SHAFT LIVES

Purpose: To evaluate the expected lives of the existing mine and shaft voids. The anticipated lives of the mine and shaft openings are of particular relevance to the decision whether to abandon the workings permanently or to retain an option for future utilization of the site. A short expected life would mitigate the possibility of long-term utilization, and favor the employment of permanent abandonment procedures.

Approach: The approach employed for the solution of the problem would utilize current geotechnical technology in conjunction with the data base developed as part of Phase 1 of the program. Part of that data base comprises current technology (review of in-situ pillar performance and stability at various potash mines throughout the world, study of ground subsidence characteristics associated with void closure), i.e., technology which utilizes past field experience rather than theoretical model predictions.

The life of the mine opening is influenced by geomechanical and hydrogeological factors. The rate of void closure will initially be determined by the competence of the pillars and backs. A review of the current in-situ state of these elements, together with an understanding of the performance of such materials (through available laboratory and field experience), will enable the future mechanical performance of the mine to be established. The life of the mine voids is also directly

related to water ingress rates and quality. Ingress of water will obviously occupy mine voids, and may hasten instability through dissolution. The hydrogeological study carried out as part of Phase 1 will enable the significance of this problem to be assessed.

The lives of the mine shafts may be determined by local conditions (e.g., shaft failure through dissolution, etc.), or by subsidence associated with closure of the mine voids. The data evaluated during Phase 1 of the program will be sufficient to allow the above consideration to be examined.

2.8 REVIEW OF MINE AND SHAFT ABANDONMENT PROCEDURES

Purpose: To provide a data base of mine and shaft abandonment procedures relevant to the Wills-Weaver mine. Since the prime purpose of the scope of work outlined in the RFP is to develop recommendations for proper abandonment of Wills-Weaver mine and shafts, the generation of a data base of possible decommissioning procedures is obviously a principal requirement.

In order to consider only those abandonment procedures which have possible relevance to the Wills-Weaver mine, it is necessary to establish principal decommissioning objectives. With respect to the mine openings (excluding shafts), two extreme scenarios can be examined:

- No significant action
- Backfilling.

Backfilling is used as an integral part of many high extraction ratio mining methods, and its prime function is to provide improved mining conditions and to limit surface subsidence. In the case of the Wills-Weaver mine, there is evi-

dently no strong reason for using backfill to inhibit surface subsidence as part of a permanent abandonment procedure, since current mining methods on the potash leases of southeast New Mexico already induce maximum subsidence over substantial areas. In the case of temporary abandonment procedures (i.e., access to mine relatively easily re-established), backfilling would obviously defeat the whole purpose of the exercise, i.e., to maintain the underground voids.

It is not likely, therefore, that any potential permanent or temporary abandonment procedure for Wills-Weaver mine will involve significant activity in the area of providing improved mechanical support for a substantial region of the present underground openings. Principal concern is liable to relate to the development of suitable shaft abandonment recommendations, consistent with anticipated behavior of the mine workings generally.

The decommissioning objectives with respect to the shafts will vary, according to whether permanent or temporary abandonment is required. Permanent abandonment procedures would be indicated by a short anticipated life for the mine voids or a low expectation of any future activity for the mine workings. Under these circumstances, the primary objective would be to eliminate the possibility of future environmental impact of the shaft (possibly by appropriate backfilling). The indication of potential future uses for the mine, coupled with an estimate of adequate mine life, would indicate that temporary abandonment procedures were more appropriate. Under these circumstances, the principal objectives of the shaft decommissioning program would include control of water ingress which was potentially damaging to both shafts and mine openings, maintenance of the shaft to preclude the occurrence of instabilities arising from causes other than general strata subsidence, and the provision of acceptable recommissioning procedures.

Approach: It is intended that an extensive literature review of appropriate mine and shaft abandonment procedures be undertaken. There is extensive literature available on the subject, and much of the impetus for the current interest in decommissioning procedures relates to the increasing trend of using underground storage for energy and wastes of various forms. The technology of shaft and borehole sealing is one area which has received substantial attention in recent years, since man-made connections between underground voids and overlying aquifers have been shown to be generally highly significant hydrologic elements.

The various abandonment procedures will be evaluated with respect to cost, materials, procedures, shaft stability control, hydrologic control, and ease of recommissioning.

2.9 ANALYSIS OF IMPACT OF FLUID INGRESS

Purpose: To evaluate whether fluids should be permitted to migrate down the shafts and into the workings. This study will initially consist of an impact analysis to establish the influence of shaft fluid migration on mine and shaft stability. The decision as to whether or not it is advisable to permit water ingress will be based on the results of the impact analysis together with the type of abandonment procedure considered, e.g., the impact of shaft fluid ingress on permanent and temporary abandonment procedures may be significantly different.

The fluid migration problem is of particular interest for abandonment schemes which retain the option to recommission the mine openings. There are a number of factors which will influence the expected life of the mine voids, but as discussed

previously, the control of potential water ingress problems is the activity which is most liable to represent a cost-effective engineered solution.

Approach: The impact analysis will be based on data which has been collected and evaluated as part of Phase 1 of the program. This data includes projected mine and shaft lives, quantity and quality of shaft water ingress and the hydrology of the mine workings generally. The study will analyze the sensitivity of the anticipated mine and shaft lives to shaft water ingress, and this is expected to be a function of:

- Mechanical stability of mine and shafts
- Significance of shaft water inflow in terms of total mine inflow
- Dissolution capacity of the mine inflow.

The final recommendations regarding shaft water ingress will be based on the results of the impact analysis, and referenced to the category of abandonment procedure (i.e., temporary or permanent).

2.10 POTENTIAL ALTERNATIVE USES FOR WILLS-WEAVER MINE

Purpose: To list potential feasible alternative uses for the Wills-Weaver mine and mine shafts. The use of underground space for waste and energy storage has received considerable attention in recent years, as studies have shown that this provides a most acceptable and environmentally suitable solution to many problems which arise using more conventional surface or engineered structures. Waste forms which may be stored underground include both chemical and radioactive forms, and subsurface energy storage may include gas, compressed air and oil.

Approach: It is obviously not the purpose of this task to provide a detailed appraisal of the alternative uses to which the Wills-Weaver mine might be placed. There are, however, various site suitability criteria, either existing or in the process of being formulated, for many of the potential uses to which underground space might be put. The approach to be followed in this task will consist of categorizing required site suitability criteria, in order to provide a broad overview of the Wills-Weaver site potential.

2.11 EVALUATION OF ABANDONMENT OPTIONS

Purpose: To evaluate abandonment options for both permanent and temporary abandonment.

Approach: The data base describing currently available practical methods of mine and shaft abandonment will provide the basic information required for this evaluation. Identification of preferred abandonment procedures will be based principally on cost and effectiveness studies. The effectiveness of a decommissioning method must be assessed in terms of the requirements of the decommissioning program, and these requirements will be determined by:

- Whether the abandonment is temporary or permanent
- The nature of future feasible operations within the mine openings, if recommissioning options are desirable.

2.12 EVALUATION OF OPTIMUM ABANDONMENT OPTION

Purpose: To determine the optimum abandonment option and evaluate the associated consequences.

Approach: This evaluation requires that essentially all of the data and analysis results of the entire program be considered. The previous task has evaluated optimum abandonment methods for both permanent and temporary abandonment, and the real purpose of this task is to evaluate the relative desirabilities of the two methods of abandonment. This evaluation must therefore be based on the following considerations.

- Likelihood of requirement for recovery of remaining potash reserves
- Suitability of mine for alternative uses such as waste disposal or energy storage.

Support Required: Provision of data relating to estimated existing potash reserves, together with their geographical distribution, etc.

2.13 WASTE PILE RECLAMATION PROCEDURES

Purpose: To recommend reclamation procedures for approximately a one-half million ton waste pile, consisting of gobbed halite, sylvite and carnallite, which is located adjacent to the Wills-Weaver shafts. The conventional requirements for waste pile reclamation include prevention of stream and aquifer pollution, dust control, stability and erosion control, together with general aesthetic factors. The critical factors in assessing the consequences of failure to reclaim are mineralogy of the waste pile, topography, hydrogeology and meteorology of the area. Operations which are often associated with waste reclamation programs are landscaping (shaping of the surface of the waste to control erosion and leaching), soil amelioration and vegetation, together with waste stabilization procedures using chemical and physical methods.

Approach: The chemical composition of the waste pile ensures that more conventional reclamation procedures such as landscaping and vegetation will be inappropriate. The proposed approach to the solution of this problem consists of:

- A review of the potential environmental impact of the waste dump (this data will include hydrogeological information for the site)
- Recommendation of reclamation procedures which are in accordance with the magnitude of potential environmental problems and the aggressive nature of the stockpiled material.

APPENDIX B
INVENTORY OF DOCUMENTS RESEARCHED

INVENTORY OF DOCUMENTS RESEARCHED

In addition to the articles itemized in the Reference List, the following documents were researched as part of the Phase I program. Each of the documents provided data required to fulfill the work program outlined in the contract. The source of each document is also given.

Document	Source
No. 3 Shaft - General Arrangement (Drawing P-1)	Duval Corporation
No. 3 Shaft - Skip Loading Station General Arrangement (Drawing P-30)	Duval Corporation
No. 3 Shaft - Skip Loading Station Excavation Plan View (Drawing P-31A)	Duval Corporation
Shafts Nos. 3 & 4 - 14' Diameter Shaft Reinforcing (Drawing P-6)	Duval Corporation
Shafts Nos. 3 & 4 - 9' Diameter Shaft Reinforcing (Drawing P-17)	Duval Corporation
Shafts Nos. 3 & 4 - Shaft Sinking Log (Drawing P-45)	Duval Corporation
Shaft No. 3 - Grouting Record at 365 Level (Drawing P-23)	Duval Corporation
Shaft No. 3 - Grouting Record at 454 Level (Drawing P-24A)	Duval Corporation

Document	Source
Shaft No. 4 - Grouting Record at 445 Level (Drawing P-24B)	Duval Corporation
Wills-Weaver Mine Contour Map (Drawing W-4-A-1)	Duval Corporation
Sundry Notices and Reports on Wells (see Table 3-3)	USGS, Carlsbad
Geological Logs of Coreholes and Wells (see Table 3-3)	USGS, Carlsbad
Drillers Logs of Coreholes DSP 209, DSP 215	Duval Corporation
Water Quality Analyses from DSP 209	Duval Corporation
Water Quality Analyses from Shaft No. 3	USGS, Carlsbad
Surface Subsidence Records - Wills-Weaver Mine	Duval Corporation
Room Convergence Monitoring Data - Wills-Weaver Mine	Duval Corporation

APPENDIX C

CONSTRUCTION METHODS

ITEM 1000 - REMOVE SHAFT STEEL

Commencing from the bottom of the shaft, or from the lowest steel set to be removed, remove sets and transport steel to surface in the cage. Acetylene torches are used to burn off the heads of the anchor bolts and to cut up the ladder road into small enough sections to be hoisted.

It has been assumed that the material removed from the shaft will be piled on surface on the Wills-Weaver property and that a scrap dealer would haul the material away at no extra cost.

ITEM 2000 - CONCRETE IN PUMP STATIONS

The pump stations in the No. 3 and No. 4 shafts will be backfilled with mass concrete. The following construction method has been assumed.

A form is built flush with the concrete shaft liner. Mass concrete is poured through an elephant trunk arrangement into the pump station excavation, gradually raising the form planks as the concrete level rises in the pump station. When the final lift is poured it is necessary to use high slump concrete to fill in the upper part of the pump station.

ITEM 3000 - GROUTING

For purposes of estimating cost, it has been assumed that the same area around the shaft wall will be regrouted as was grouted in the original program in order to seal the shafts. It has been assumed that the grout take will be the same as in the original program.

The grouting method involves drilling horizontal rings of holes through the shaft walls and injecting cement grout into these holes. It will be necessary to install a drilling platform in the shaft and a safety station should be located near the bottom of the grouting zones.

Grouting costs for sealing the formations around a shaft plug have been estimated by assuming 30 percent of the grouting costs to seal the shafts' liners.

ITEM 4000 - HITCHES FOR PLUGS

The following assumptions have been made regarding the construction of hitches:

- In the No. 3 shaft, the hitch for the concrete plug will be 14 feet in the vertical dimension, will be cut on a horizontal plane through the concrete liner at the upper and lower ends of the plug and will taper towards the outside of the hitch (see Figure 9.15). For the clay plug, the hitch will be 20 feet high.

- The hitch for the concrete plug in the No. 4 shaft was assumed to be nine feet high and constructed to a similar shape. For the clay plug the hitch is again assumed to be 20 feet high.

- The construction method proposed is:

  - Moil out four 12" x 12" x 12" pockets in the shaft wall a distance of 18 inches below the bottom of the proposed plug.

- Set wide flange beams in these pockets and grout in.

- Install a concrete form on top of beams and pour a reinforced concrete floor with four 12" x 12" blockouts. This floor is to be 18 inches thick.

- Drill presplit holes in horizontal rings at the top and bottom of the proposed hitch. A timber staging will be rigged to facilitate drilling of the upper ring. Presplit holes are on one-foot centers at their collars and are loaded at the rate of 1/4 to 1/2 pound per L.F. using presplit powder.

- After blasting the presplit rounds, drill holes through the concrete such that there is a hole for each two square feet of face. Blast these pattern holes using dynamite and delay caps.

- Dig out the concrete and burn off the rebar in the concrete. Broken concrete will be dumped to the shaft bottom through the blocked out windows.

- Using moils and clay spades, excavate the rock material to form the hitch. Broken rock will be disposed of through the blockouts.

- Cover the blockouts in preparation for the placement of the plugs.

For plugs supported by backfill, the shaft hitch can be excavated from the top of the fill and the construction of the reinforced concrete support is not required. Because of the limited construction cost, no allowance has been made for this saving. If the shaft is backfilled, the above saving will be offset by the necessity to haul the concrete and rock muck to surface.

ITEM 5000 - CONCRETE PLUG CONSTRUCTION

The following construction procedures have been assumed:

- The blockouts in the reinforced concrete slab are covered using steel plates torch-welded to the steel at the rim of the blockout.

- Concrete is then lowered in 5 CY concrete buckets and placed by the underground crew. High early-setting agents would be added to the concrete so that the lower portion of the pour would develop some strength before the upper portion of the pour began, thus preventing the total load of the concrete from bearing on the slab.

- Use of an electric or pneumatic vibrator will be necessary to assure that no air pockets remain in the irregularities of the hitches.

ITEM 6000 - CLAY PLUG CONSTRUCTION

It has been assumed that satisfactory clay can be bought at a site within 20 miles of the Wills-Weaver mine. It has been further assumed that a suitable clay plug can be built with the addition of 10 percent by weight of bentonite.

The plug material would be hauled to the site in end dump or belly dump trucks. A 4 C.Y. front-end loader would stockpile the material at the site.

The clay would be subsequently reclaimed by the 4 C.Y. FEL and loaded into a transit mix truck of about 6 C.Y. capacity.

Bentonite and water would be added at this point and the material blended within the transit mix truck. The prepared clay would then be delivered to the shaft collar and dumped into a skip (or into a concrete bucket in the case of No. 4 shaft).

The plug material would then be lowered to the plug area and placed from the skip or concrete bucket by the workers in the shaft. It will be necessary to modify the skip to use it in this manner.

The clay is to be spread and compacted to engineered standards using pneumatic tampers.

ITEM 7000 - BACKFILL SHAFTS (COMPACT FILL)

The material used to backfill the shafts will consist of surface sand obtained from the vicinity of the shafts. This material will be retained in the respective mine stations by concrete bulkheads.

At the No. 3 shaft mine station it will be necessary to pump out the No. 3 tailshaft prior to placing fill. The pumping could be accomplished by using a pneumatic spinner pump lifting the brine into a sealed skip for hoisting to surface. Alternatively the brine could simply be pumped into the mine workings.

In order to provide for long-term security of the backfill support, all fittings would have to be removed from the tailshaft and storage bins in the floor of the No. 3 shaft mine station and voids would have to be backfilled.

Mass concrete bulkheads would then be constructed within the mine stations in order to contain the shaft fill. In the No. 3

shaft mine station, the bulkheads could be placed at the entries leading to the mine workings and to the No. 4 shaft. Due to the distance from the shaft, it would be necessary to use a placer to place the concrete. The bulkheads would have a typical minimum thickness of 10 feet and would be formed in conventional manner. The concrete could be lowered from surface using a concrete bucket.

In the No. 4 shaft mine station the bulkhead could be formed adjacent to the shaft.

Surface sand backfill would be trammed to the shaft collar using a 4 C.Y. F.E.L., and transported underground using the ore skip or concrete bucket. Water would be added by the shaft workers and pneumatic tampers employed to bring the material to optimum density. On the station level it would be necessary to make extensive use of a slusher to place the fill material into the upper portions of the station void. Extensive handmucking would be necessary to obtain good fill to the top of the station.

ITEM 8000 - SHAFT CAPS

Shaft caps for the No. 3 and No. 4 shafts are assumed to be 18 inches thick and to extend two feet from the inside edge of the shaft. It has been assumed that reinforcing will consist of #6 bar on 9-3/4" centers both ways on top and bottom.

The construction method consists of moiling pockets into the shaft collar concrete for setting wide flange beams to support the forms. The edge forms would be set, the rebar installed and the cap poured using ready-mix concrete.

ITEM 10000 - PLANT SETUP

The cost covered in this item includes mobilization and demobilization of contractor's equipment for performing the work in Items 1000 through 8000 and 13000 to 14000.

Also included is the cost of installing (but not purchasing) doors to cover the shafts for protection of men working in the shaft. Repair to existing shaft utilities is included so that pipes can be utilized to deliver air and water to the shaft bottom.

No provision is made for installation of a hoist or headframe.

ITEM 11000 - RECOMMISSIONING SHAFTS

This item covers the estimated cost for setting up a temporary hoisting plant on surface, clearing of plug materials from the shaft and installation of shaft furnishings so that hoisting operations could commence (assuming the existence of a sound liner). It has been assumed that one hoist could be set up that would serve either shaft.

Shaft backfill is assumed to be excavated using an air operated Cryderman mucker loading into a sinking bucket of about 5 C.Y. capacity. The Cryderman could work in one compartment of the No. 3 shaft or could be pinned to the shaft wall within the No. 4 shaft.

Shaft furnishings will be installed as excavation progresses so that the need for a Galloway stage and stage winches is eliminated.

Muck from the shaft would be hoisted in the sinking bucket using a cross-head arrangement. The bucket would be dumped into a temporary chute in the head frame using a dingleball arrangement on the bucket. The muck would then go to an end-dump truck for haulage to disposal.

Concrete plugs would be blasted out using presplit holes along the shaft walls and four-foot lifts to prevent large pieces from falling to the bottom of the shaft. Due to the limited volume of concrete to be handled, a mechanical mucker could not be justified, so hand-mucking has been assumed. The last cut could be burned through and then blasted a few holes at a time to prevent large volumes of concrete from falling down the shaft.

ITEM 12000 - BACKFILL SHAFTS WITH RANDOM FILL

The material used to backfill the shafts is assumed to consist of surface sand excavated from a borrow pit located about 200 feet from the shafts. The pit is assumed to be on the order of 4 feet deep and is excavated by a FEL without assistance from a dozer.

The surface sand is dumped down the shafts from surface and mine station retaining structures are assumed to be unnecessary.

ITEM 13000 - BACKFILL SHAFTS WITH UNCOMPACTED CLAY

Purchase, haulage and stockpiling of clay fill is identical to that described under item 6000. Reclamation and mixing of the clay would be similar to that previously described, but it has been assumed that there will be no addition of purchased bentonite as was the case for the clay hydraulic plug. The clay

fill would again be delivered to the working level by skip or concrete bucket as it is not recommended that such material be dumped from the top of the shaft.

The shaft fill has been assumed to be placed uncompacted and mine station retaining structures are considered unnecessary.

ITEM 14000 - PROTECT MINE STATION AREA

The mine station area has been assumed to be protected from dissolution damage by placing clay blankets to conduct ingressing water out of the shaft pillar region. The tailshafts would be filled with surface sand and clay fill would be delivered to the mine stations by skip or concrete bucket as previously described.

A 10 CF Bobcat would then be used to place a clay blanket from each shaft mine station into the region of the workings external to the shaft pillar. The clay blanket would be profiled to conduct ingressing water out of the pillar area so that no dissolution ocurred in this region.

APPENDIX D

COSTING DATA

LABOR COST: A labor rate of $15 per hour has been assumed for all types of labor.

SUPPLIES: The following unit costs have been used for supplies:

Common Lumber	$700/MBF
Structural Steel	$0.50/LB
Cement for grout	$3.50 per sack
Drill bits & steel-small holes	$0.15 per ft.
Presplit explosives	$1.00 per lb.
Dynamite	$0.60 per lb.
Caps.	$0.75 ea.
Rebar	$0.50 per lb.
Bentonite	$156 per Ton
Ready-mix concrete	$50 per C.Y.
Selected clay	$7 per LCY

EQUIPMENT HOURLY COST: The following hourly costs have been assumed for equipment:

Hoist & shaft doors	$15.00 per hr.
5T Flatbed Truck	$10.00 per hr.
4-1/2" Bore Drifter	$5.00 per hr.
Grout pump & remix	$4.25 per sack
600 CFM Portable Diesel Compressor	$15.00 per hr.
Pneumatic moil	$3.00 per hr.
Jackleg Drill	$3.00 per hr.
Ready-mix truck	$30.00 per hr.
4 CY FEL	$25.00 per hr.
Pneumatic Tamper	$0.50 per hr.
Pneumatic spinner pump	$1.00 per hr.
20T End Dump Truck	$20.00 per hr.
Cryderman Mucker	$30.00 per hr.
Crosshead & 5CY Buckets	$10.00 per hr.
Hand-Held Sinker Drill	$5.00 per hr.

CREW A - EQUIPPING

	Labor	Plant	Equip.	Supplies	Total
Surface labor (2)	$30				
5T truck (1)			$10		
Hoist & Doors (1)		$15			
Hoistman (1)	$15				
Small Tools & Sup.				$15	
UG Labor (3)	$45				
Cost/Hr.	$90	$15	$10	$15	$130
Cost/Shift	$720	$120	$80	$120	$1040

CREW B - MUCKING

	Labor	Plant	Equip.	Supplies	Total
Surface Labor (2)	$30				
20T End Dump (1)			$20		
Hoist & Doors (1)		$15			
Hoistman (1)	$15				
Small Tools & Sup.				$10	
600 CFM Compr. (1)			$15		
UG Labor (3)	$45				
Cryderman (1)			$30		
Crossheads & Buckets (2)			$10		
Cost/hr.	$90	$15	$75	$10	$190
Cost/Shift	$720	$120	$600	$80	$1520

CREW C - CONCRETE DRILLING

	Labor	Plant	Equip.	Supplies	Total
Surface labor (2)	$30				
5T truck (1)			$10		
Hoist & Doors (1)		$15			
Hoistman (1)	$15				
Small Tools & Sup.				$15	
UG Labor (3)	$45				
Drill bits & steel				$15	
Hand-held sinkers			$15		
Cost/hr.	$90	$15	$25	$30	$160
Cost/Shift	$720	$120	$200	$240	$1280

OPERATION Remove Shaft Steel ITEM NO. 1000

JOB Wills-Weaver

QUANTITY 387 UNIT sets EST BY W. Brown DATE 1/6/79

		LABOR AND INSURANCE		PERMANENT MATERIALS		PLANT		EQUIPMENT		SUPPLIES		SUBCONTRACT		TOTAL DIRECT COST	
		UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT
Item 1100 No. 3 Shaft	194 Sets														
UG Labor (3)	387 MS	120	46,440												46,440
Surface Labor (2)	258 MS	120	30,960												30,960
Hoistman (1)	129 MS	120	15,480												15,480
Hoist & Shaft Doors	1032 EH					15	15,480								15,480
5 Ton Flatbed Truck	1032 EH							10	10,320						10,320
Small Tools & Gas	129 SH									50	6,450				6,450
SUBTOTAL ITEM 1100			92,880				15,480		10,320		6,450				125,130
Item 1200 No. 4 Shaft	193 Sets														
UG Labor (3)	165 MS	120	19,800												19,800
Surface Labor (2)	110 MS	120	13,200												13,200
Hoistman (1)	55 MS	120	6,600												6,600
Hoist & Shaft Doors	440 EH					15	6,600								6,600
5 Ton Flatbed Truck	440 EH							10	4,400						4,400
Small Tools & Gas	55 SH									50	2,750				2,750
SUBTOTAL ITEM 1200			39,600				6,600		4,400		2,750				53,350
TOTAL ITEM 1000			132,480				22,080		14,720		9,200				178,480

OPERATION Concrete In Pump Stations ITEM NO. 2000

JOB Wills-Weaver

QUANTITY 80 UNIT CY EST BY W. Brown DATE 7/22/79

		LABOR AND INSURANCE		PERMANENT MATERIALS		PLANT		EQUIPMENT		SUPPLIES		SUBCONTRACT		TOTAL DIRECT COST	
		UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT
Item 2100 No. 3 Shaft															
2110 Forms	42 SF														
Crew A	2 CS	720	1,440			120	240								
Lumber	.084 MBF							80	160	120	240			1040	2,080
Concrete	40 CY			50	2,000					700	60				60
															2,000
SUBTOTAL ITEM 2100			1,440		2,000		240		160		300				4,140
Item 2200 No. 4 Shaft (same as item 2100)			1,440		2,000		240		160		300				4,140
TOTAL ITEM 2000			2,880		4,000		480		320		600				8,280

OPERATION Grouting ITEM NO. 13000

JOB Wills-Weaver

QUANTITY 2500 UNIT sacks EST BY W. Brown DATE 7/6/79

		LABOR AND INSURANCE		PERMANENT MATERIALS		PLANT		EQUIPMENT		SUPPLIES		SUBCONTRACT		TOTAL DIRECT COST		
		UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	
Item 3100 No. 3 Shaft	1500 SK															
3110 Setup for Grouting	6 EA															
Labor (3)	36 MS	120	4,320												4,320	
Lumber	1.2 MBF									700	840				840	
Steel	2000 LB									0.50	1,000				1,000	12 Shifts
3120 Drill Grout Holes	2500 LF															
Labor (2) 200 FT/CS	25 MS	120	3,000												3,000	
Drill 4½" Bore Drifters	200 EH							5.00	1,000						1,000	
Drill Steel & Bits	2500 LF									.15	375				375	13 Shifts
3130 Inject Grout & Test																
OG Labor (2) 50 SK/SH	60 MS	120	7,200												7,200	
Grout Pump	1500 SK							4.25	6,375						6,375	
Cement	1500 SK									3.50	5,250				5,250	30 Shifts
3110 Surface Plant																
Surface Labor (2)	110 MS	120	13,200												13,200	
Hoistman (1)	55 MS	120	6,600												6,600	
Hoist & Shaft Doors	440 EH					15	6,600								6,600	
600 CFM Compressor	440 EH							15	6,600						6,600	
5 Ton Flatbed Truck	440 EH							10	4,400						4,400	
SUBTOTAL ITEM 3100	1500 SK		34,320				6,600		18,375		7,465				66,760	55 Shifts

OPERATION: Grouting
ITEM NO. 3000
Wills-Weaver
Page 2 of 2

		LABOR AND INSURANCE		PERMANENT MATERIALS		PLANT		EQUIPMENT		SUPPLIES		SUBCONTRACT		TOTAL DIRECT COST		
		UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	
Item 3200 No. 4 Shaft																
3210 Setup for Grouting	3 EA															
Labor (3)	18 MS	120	2,160												2,160	
Fabricated Platform	2 EA									3000	6,000				6,000	6 Shifts
3220 Drill Grout Holes	1000 LF															
Labor (2) 200 FT/CS	10 MS	120	1,200												1,200	
Drill 4½" Bore Drifter	80 EH							5	400						400	
Drill Steel & Bits	1000 LF									.15	150				150	5 Shifts
3230 Inject Grout & Test	1000 SK															
Labor (2) 50 SK/SH	40 MS	120	4,800												4,800	
Grout Pump	1000 SK							4.25	4,250						4,250	
Cement	1000 SK									3.50	3,500				3,500	20 Shifts
3240 Surface Plant																
Surface Labor (2)	62 MS	120	7,440												7,440	
Hoistman (1)	31 MS	120	3,720												3,720	
Hoist & Shaft Doors	248 EH					15	3,720								3,720	
600 CFM Compressor	248 EH							15	3,720						3,720	
5 Ton Flatbed Truck	248 EH							10	2,480						2,480	
SUBTOTAL ITEM 3200	1000 SK		19,320				3,720		10,850		9,650				43,540	31 Shifts
TOTAL ITEM 3000	2500 SK		53,640				10,320		29,225		17,115				110,300	86 Shifts

OPERATION Hitches for Plugs ITEM NO. 4000

JOB Wills-Weaver

QUANTITY ____ UNIT ____ EST BY W. Brown DATE 1/6/79

		LABOR AND INSURANCE		PERMANENT MATERIALS		PLANT		EQUIPMENT		SUPPLIES		SUBCONTRACT		TOTAL DIRECT COST		
		UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	
Item 4100 No. 3 Shaft	167 CY															
4110 Moilout 4 pockets 12"x 12"x 12"	4 MS	120	480												480	2 Shifts
Moils	32 EH							3	96						96	
WF Steel	1250 LB									.50	625				625	
4120 Place Rebar	2 MS	120	240												240	1 Shift
Rebar	4000 LB									.50	2,000				2,000	
Place Forms	2 MS	120	240												240	1 Shift
Lumber	0.6 MBF									700	420				420	
4130 Place Concrete	2 MS	120	240												240	1 Shift
Concrete	9 CY									50	450				450	
4140 Drill & Blast Concrete	70 CY															
Labor 300 Ft/MS (2)	4 MS	120	480												480	2 Shifts
Jack Leg Drill	23 EH							3	69						69	
Drill Bits & Steel	1184 LF									.15	178				178	
Presplit Powder	100 LB									1.00	100				100	
Dynamite	250 LB									.60	150				150	
Primers	310 EA									.75	232				232	
4150 Moilout Slot in Rock	91 CY															
Labor 2 CY/MS (2)	47 MS	120	5,640												5,640	24 Shifts
Moils	376 EH							3	1,128						1,128	
4160 Muckout Rock & Concrete	164 BCY															
Labor (2)	10 MS	120	1,200												1,200	5 Shifts

OPERATION: Hitches for Plugs
ITEM NO. 4000
Wills-Weaver
Page 2 of 2

		LABOR AND INSURANCE		PERMANENT MATERIALS		PLANT		EQUIPMENT		SUPPLIES		SUBCONTRACT		TOTAL DIRECT COST		
		UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	
4170 Surface Plant	36 SH															
Hoistman (1)	36 MS	120	4,320												4,320	
Hoist & Shaft Doors	288 EH					15	4,320								4,320	
Surface Labor (1)	36 MS	120	4,320												4,320	
600 CFM Compressor	288 EH							15	4,320						4,320	
SUBTOTAL ITEM 4100			17,160				4,320		5,613		4,155				31,248	36 Shifts
Item 4200 No. 4 Shaft	77 BCY															
4210 Loilout 4 Pockets 12"x 12"x 12"	4 MS	240	480												480	2 Shifts
Moils	32 EH							3	96						96	
WF Steel	800 LB									.50	400				400	
4220 Place Rebar	2 MS	120	240												240	1 Shift
Rebar	1709 LB									.50	850				850	
Place Forms	2 MS	120	240												240	1 Shift
Lumber	0.25 MBF									700	175				175	
4230 Place Concrete	2 MS	120	240												240	1 Shift
Concrete	4 CY									50	200				200	
4240 Drill & Blast Concrete	30 CY															
Labor 300 Ft/MS (2)	2 MS	120	240												240	1 Shift
Jackleg Drill	13 EH							3	39						39	
Drill Bits & Steel	508 LF									.15	76				76	
Presplit Powder	50 LB									1.00	50				50	
Dynamite	65 LB									.60	39				39	
Primers	200 EA									.75	150				150	
4250 Moilout Slot in Rock	47 Cy															
Labor 2 CY/MS (2)	24 MS	120	2,880												2,880	12 Shifts
Moils	192 EH							3	576						576	
4260 Muckout Rock & Concrete	77 BCY															
Labor (2)	5 MS	120	600												600	3 Shifts
4270 Surface Plant	21 SH															
Hoistman (1)	21 MS	120	2,520												2,520	
Hoist & Shaft Doors	168 EH					15	2,520								2,520	
Surface Labor (1)	21 MS	120	2,520												2,520	
600 CFM Compressor	168 EH							15	2,520						2,520	
SUBTOTAL ITEM 4200			9,960				2,520		3,231		1,940				17,651	21 Shifts
TOTAL ITEM 4000			27,120				6,840		8,844		6,095				48,899	57 Shifts

OPERATION Concrete Plug ITEM NO. 5000

JOB Wills-Weaver

QUANTITY 452 UNIT CY EST BY W. Brown DATE 7/23/79

		LABOR AND INSURANCE		PERMANENT MATERIALS		PLANT		EQUIPMENT		SUPPLIES		SUBCONTRACT		TOTAL DIRECT COST	
		UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT
Item 5100 No. 3 Shaft	244 CY														
Crew A 15 CY/CH	2 CS	720	1,440			120	240	80	160	120	240			1040	2,080
Ready-Mix Concrete	244 CY			50	12,200										12,200
Additives	244 CY			5	1,220										1,220
SUBTOTAL ITEM 5100	244 CY		1,440		13,420		240		160		240				15,500
Item 5200 No. 4 Shaft	98 CY														
Crew A 15 CY/CH	1 CS	720	720			120	120	80	80	120	120			1040	1,040
Ready-Mix Concrete	98 CY			50	4,900										4,900
Additives	98 CY			5	490										490
SUBTOTAL ITEM 5200	98 CY		720		5,390		120		80		120				6,430
TOTAL ITEM 5000			2,160		18,810		360		240		360				21,930

OPERATION Clay Plugs ITEM NO. 6000

JOB Wills-Weaver

QUANTITY ____ UNIT ____ EST BY W. Brown DATE 7/9/79

		LABOR AND INSURANCE		PERMANENT MATERIALS		PLANT		EQUIPMENT		SUPPLIES		SUBCONTRACT		TOTAL DIRECT COST		
		UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	
Item 6100 No. 3 Shaft	344 CCY															
6110 Purchase Select Clay	460 LCY									7.00	3,220				3,220	
6120 Haul to Site 2 x 20 CY Trucks	460 LCY											9.00	4,140		4,140	
6130 Stockpile, 4CY FEL	16 EH	15	240					25	400						640	
6140 Reclaim, Mix, Load Skips																
Mixer Truck	56 EH	15	840					30	1,680						2,520	
Reclaim, 4 CY FEL	56 EH	15	840					25	1,400						2,240	
6150 Place & Compact																
Labor (3) 2 CCY/MH	172 MH	15	2,580												2,580	7 Shifts
Tampers (3) 2 CCY/EH	172 EH							.50	86						86	
6160 Purchase Bentonite	60 T									156	9,360				9,360	
6170 Surface Plant	7 SH															
Hoistman (1)	7 MS	120	840												840	
Hoist & Shaft Doors	56 EH					15	840								840	
Surface Labor (2)	14 MS	120	1,680												1,680	
600 CFM Compressor	56 EH							15	840						840	
SUBTOTAL ITEM 6100			7,020				840		4,406		12,580		4,140		28,986	7 Shifts

OPERATION: Clay Plugs
ITEM NO. 6000
Wills-Weaver
Page 2 of 2

		LABOR AND INSURANCE		PERMANENT MATERIALS		PLANT		EQUIPMENT		SUPPLIES		SUBCONTRACT		TOTAL DIRECT COST		
		UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	
Item 6200 No. 4 Shaft	220 CCY															
6210 Purchase Select Clay	290 LCY									7.00	2,030				2,030	
6220 Haul to Site	290 LCY											9.00	2,610		2,610	
6230 Stockpile, 4 CY FEL	12 EH	15	180					25	300						480	
6240 Reclaim, Mix, Load Buckets																
Mixer Truck	72 EH	15	1,080					30	2,160						3,240	
Reclaim, 4 CY FEL	72 EH	15	1,080					25	1,800						2,880	
6250 Place & Compact																
Labor (2) 1½ CCY/MH	148 MH	15	2,220												2,220	9 Shifts
Tampers (2) 1½ CCY/EH	148 EH							.50	74						74	
6260 Purchase Bentonite	35 T									156	5,460				5,460	
6270 Surface Plant	9 SH															
Hoistman (1)	9 MS	120	1,080												1,080	
Hoist & Shaft Doors	72 EH					15	1,080								1,080	
Surface Labor (2)	18 MS	120	2,160												2,160	
600 CFM Compressor	72 EH							15.	1,080						1,080	
SUBTOTAL ITEM 6200			7,800				1,080		5,414		7,490		2,610		24,394	9 Shifts
TOTAL ITEM 6000			14,820				1,920		9,820		20,070		6,750		53,380	16 Shifts

OPERATION Backfill Shafts ITEM NO. 7000

JOB Wills-Weaver

QUANTITY____ UNIT____ EST BY_____ DATE 7/16/79

		LABOR AND INSURANCE		PERMANENT MATERIALS		PLANT		EQUIPMENT		SUPPLIES		SUBCONTRACT		TOTAL DIRECT COST		
		UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	
Item 7100 No. 3 Shaft																
7110 Pumpout Tailshaft	3500 Gal															3 Shifts
UG Labor	6 MS	120	720												720	
Surface Labor	6 MS	120	720												720	
Spinner Pump	24 EH							1	24						24	
600 CFM Compressor	8 EH							15	120						120	
Hoistman	3 MS	120	360												360	
Hoist & Shaft Doors	24 EH					15	360								360	
SUBTOTAL ITEM 7110			1,800				360		144						2,304	
7120 Remove Tailshaft & Equipment	1 LS															5 Shifts
UG Labor	15 MS	120	1,800												1,800	
Surface Labor	10 MS	120	1,200												1,200	
5 Ton Flatbed Truck	40 EH							10	400						400	
Gas & Small Tools	1 LS										500				500	
Hoistman	5 MS	120	600												600	
Hoist & Shaft Doors	40 EH					15	600								600	
SUBTOTAL ITEM 7120			3,600				600		400		500				5,100	
7130 Construct 2 Bulkheads	166 CY															
Crew A Forms 150 SF/CS	900 SF	720	4,320			120	720	80	480	120	720			1040	6,240	6 Shifts
Form Materials	900 SF									1.00	900				900	
Concrete 10 Ft. Thick	166 CY			50	8,300										8,300	
SUBTOTAL ITEM 7130		26.02	4,320	50	8,300	4.34	720	2.89	480	9.76	1,620			93.01	15,440	

OPERATION: Backfill Shafts
ITEM NO. 7000
Wills-Weaver
Page 2 of 2

		LABOR AND INSURANCE		PERMANENT MATERIALS		PLANT		EQUIPMENT		SUPPLIES		SUBCONTRACT		TOTAL DIRECT COST		
		UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	
7140 Backfill Shaft Station	2500 CY															52 Shifts
Crew A 48 CY/CS	52 CS	720	37,440			120	6,240	80	4,160	120	6,240			1040	54,080	
4 CY FEL	416 EH	15	6,240					25	10,400						16,640	
Tampers	1248 EH							.50	624						624	
SUBTOTAL ITEM 7140			43,680				6,240		15,184		6,240				71,344	
7150 Backfill Shaft	5525 CY															75 Shifts
UG Labor	225 MS	120	27,000												27,000	
Surface Labor	150 MS	120	18,000												18,000	
4 CY FEL	600 EH	15	9,000					25	15,000						24,000	
Tampers	1800 EH							.50	900						900	
Hoistman	75 MS	120	9,000												9,000	
Hoist & Shaft Doors	600 EH					15	9,000								9,000	
600 CFM Compressor	600 EH							15	9,000						9,000	
SUBTOTAL ITEM 7150		11.40	63,000			1.63	9,000	4.51	24,900					17.54	96,900	
TOTAL ITEM 7100			116,400		8,300		16,920		41,108		8,360				191,088	
Item 7200 No. 4 Shaft																
7230 Construct Bulkhead (same unit costs as for Item 7130)	230 CY	26.02	5,985	50.00	11,500	4.34	998	2.89	665	9.76	2,245			93.01	21,393	8 Shifts
7250 Backfill Shaft (same unit costs as for Item 7150)	2382 CY	11.40	27,160			1.63	3,881	4.51	10,736					17.54	41,777	32 Shifts
TOTAL ITEM 7200			33,145		11,500		4,879		11,401		2,245				63,170	

OPERATION Shaft Caps ITEM NO. 8000

JOB Wills-Weaver

QUANTITY 2 UNIT EA EST BY W. Brown DATE 7/16/79

		LABOR AND INSURANCE		PERMANENT MATERIALS		PLANT		EQUIPMENT		SUPPLIES		SUBCONTRACT		TOTAL DIRECT COST		
		UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	
Item 8100 No. 3 Shaft	14 CY															
8110 Mollout 4 pockets 12"x 12"x 12"	4 MS	120	480												480	2 Shifts
Moils	32 EH							3	96						96	
WF Steel	1250 LB									.50	625				625	
8120 Place Rebar	2 MS	120	240												240	1 Shift
Rebar	4000 LB									.50	2,000				2,000	
Place Forms	2 MS	120	240												240	1 Shift
Lumber	.6 MBF									700	420				420	
8130 Place Concrete	2 MS	120	240												240	1 Shift
Concrete	14 CY									50	700				700	
600 CFM Compressor	16 EH							15	240						240	
SUBTOTAL ITEM 8100			1,200						336		3,745				5,281	5 Shifts
Item 8200 No. 4 Shaft	7½ CY															
8210 Mollout 4 pockets 12"x 12"x 12"	4 MS	120	480												480	2 Shifts
Moils	32 EH							3	96						96	
WF Steel	800 LB									.50	400				400	
8220 Place Rebar	2 MS	120	240												240	1 Shift
Rebar	1700 LB									.50	850				850	
Place Forms	2 MS	120	240												240	1 Shift
Lumber	.25 MBF									700	175				175	
8230 Place Concrete	2 MS	120	240												240	1 Shift
Concrete	7½ CY									50	375				375	
600 CFM Compressor	16 EH							15	240						240	
SUBTOTAL ITEM 8200			1,200						336		1,800				3,336	5 Shifts
TOTAL ITEM 8000			2,400						672		5,545				8,617	10 Shifts

OPERATION Plant Setup ITEM NO. 10,000

JOB Wills-Weaver

QUANTITY ____ UNIT ____ EST BY W. Brown DATE 7/17/79

		LABOR AND INSURANCE		PERMANENT MATERIALS		PLANT		EQUIPMENT		SUPPLIES		SUBCONTRACT		TOTAL DIRECT COST	
		UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT
10100 Mobilization/Demobilization	100 MS	120	12,000												12,000
10200 Shaft Doors	25 MS	120	3,000								10,000				13,000
10300 Repair Pipes in Shafts	50 MS	120	6,000								1,000				7,000
10400 Plant & Equipment	15 ES														
5 Ton Flatbed Truck	120 EH							10	1,200						1,200
Hoist & Shaft Doors	120 EH					15	1,800								1,800
10500 Contract Haulage	500 LDMi											2.00	1,000		1,000
TOTAL PLANT SETUP			21,000				1,800		1,200		11,000		1,000		36,000

OPERATION Recommissioning Shafts ITEM NO. 11000

JOB Wills-Weaver

QUANTITY ____ UNIT ____ EST BY W. Brown DATE 7/10/79

		LABOR AND INSURANCE		PERMANENT MATERIALS		PLANT		EQUIPMENT		SUPPLIES		SUBCONTRACT		TOTAL DIRECT COST		
		UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	
Item 11100 No. 3 Shaft	191 Sets															
11110 Purchase & Install Furnishings																
Crew A	129 SH	720	92,880			120	15,480	80	10,320	120	15,480			1040	134,160	
Dividers	106000 LB			1.00	106,000										106,000	
Ladder with Cage	1000 LF			33	33,000										33,000	
Expanded Metal Landings	175 SF			3.68	639										639	
Clip Joints	1800 EA			5	4,000										4,000	
Guide Brackets	400 EA			5	2,000										2,000	
Utility Brackets	67 EA			20	1,340										1,340	
Landing Frame Angle	5000 LB			1	5,000										5,000	
4" Victaulic Pipes	1000 LF			6	6,000										6,000	
1½" Pipe	2000 LF			1.50	3,000										3,000	
1" Pipe	1000 LF			1	1,000										1,000	
Anchor Bolts	3500 EA			1.50	5,250										5,250	
Guides	11667 BF			1000/MBF	11,667										11,667	
SUBTOTAL ITEM 11110			92,880		178,896		15,480		10,320		15,480				313,056	129 Shifts
11120 Muckout - Non Concrete	8000 CY															
Crew B 130 CCY/SH	62 CS	720	44,640			120	7,440	600	37,200	80	4,960			1520	94,240	62 Shifts
11130 Drill & Blast Concrete Plug	80 CY															
Crew C 100 LF/CS	24 CS	720	17,280			120	2,880	200	4,800	240	5,760			1280	30,720	24 Shifts
Dynamite	400 LB									.60	240				240	
Presplit Powder	165 LB									1.00	165				165	
Caps & Det. Cord	500 EA									.75	375				375	
SUBTOTAL ITEM 11130			17,280				2,880		4,800		6,540			393.75	31,500	

OPERATION: Decommissioning Shafts
ITEM NO. 11000
Wills-Weaver

		LABOR AND INSURANCE		PERMANENT MATERIALS		PLANT		EQUIPMENT		SUPPLIES		SUBCONTRACT		TOTAL DIRECT COST		
		UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	
11140 Muckout - Concrete	80 CY															
Crew B 80 BCY/SH	1 CS	720	720			120	120	600	600	80	80			1520	1,520	1 Shift
11150 Handmuck Concrete Plug	80 CY															
Crew A 6 CY/CS	13 CS	720	9,360			120	1,560	80	1,040	120	1,560			1040	13,520	13 Shifts
11160 Remove Concrete Bulkhead	166 CY															
Crew C	15 SH	720	10,800			120	1,800	200	3,000	240	3,600			1280	19,200	15 Shifts
Item 11200 No. 4 Shaft	193 Sets															
11210 Purchase & Install Furnishings																
Crew A	50 Sh	720	36,000			120	6,000	80	4,000	120	6,000			1040	52,000	50 Shifts
Guide Brackets	386 EA			5.00	1,930										1,930	
Utility Brackets	64 EA			20	1,280										1,280	
1" Pipe	2000 LF			1	2,000										2,000	
3" Victaulic Pipe	1000 LF			3.25	3,250										3,250	
6" Victaulic Pipe	1000 LF			10.40	10,400										10,400	
Anchor Bolts	1675 EA			1.50	2,512										2,512	
Guides	5833 BF			1000/MBF	5,833										5,833	
SUBTOTAL ITEM 11210			36,000		27,205		6,000		4,000		6,000				79,205	
11220 Muckout - Non Concrete	2400 CY															
Crew B 130 CCY/SH	19 CS	720	13,680			120	2,280	600	11,400	80	1,520			1520	28,880	
11230 Drill & Blast Concrete Plug (same as for item 11130)	21 CY													393.75	8,270	
11240 Muckout Concrete	21 CY															
Crew B 21 CY/SH	1 CS	720	720			120	120	600	600	80	80			1520	1,520	1 Shift
11250 Handmuck Concrete Plug	21 CY															
Crew A 6 CY/CS	4 CS	720	2,880			120	480	80	320	120	480			1040	4,160	4 Shifts
11260 Remove Concrete Bulkhead	230 CY															
Crew C	20 CS	720	14,400			120	2,400	200	4,000	240	4,800			1280	25,600	20 Shifts

OPERATION Backfill Shafts with Random Fill ITEM NO. 12000

JOB Wills Weaver

QUANTITY____ UNIT____ EST BY W. Brown DATE 10/23/79

		LABOR AND INSURANCE		PERMANENT MATERIALS		PLANT		EQUIPMENT		SUPPLIES		SUBCONTRACT		TOTAL DIRECT COST	
		UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT
Item 12000 Random Fill															
12100 Random Fill, No. 3 Shaft	8456 CY														
966C FEL (81 CY/EH)	104 EH	15	1,560					16	1,664	72	7,488				10,712
12110 Site Preparation															
966C FEL	16 EH	15	240					16	256	72	1,152				1,648
12200 Random Fill, No. 4 Shaft	2715 CY														
966C FEL (81 CY/EH)	34 EH	15	510					16	544	72	2,448				3,502

OPERATION Backfill Shafts With Uncompacted Clay ITEM NO. 13000

JOB Wills Weaver

QUANTITY ____ UNIT ____ EST BY W. Brown DATE 10/23/79

		LABOR AND INSURANCE		PERMANENT MATERIALS		PLANT		EQUIPMENT		SUPPLIES		SUBCONTRACT		TOTAL DIRECT COST	
		UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT	UNIT COST	AMOUNT
Item 13100 No. 3 Shaft	8456 CY														
13110 Purchase Select Clay	9300 CY									7.00	65,100				65,100
13120 Haul to site	9300 CY											9.00	83,700		83,700
13130 Stockpile, 4CY FEL	310 EH	15	4,650					25	7,750						12,400
13140 Reclaim, mix, load skips															
Mixer Truck	85 EH	15	1,275					30	2,550						3,825
4CY FEL	85 EH	15	1,275					25	2,125						3,400
13150 Place															
Labor (2) 50CY/MH	169 MH	15	2,535												2,535
13160 Surface Plant															
Hoistman	12 MS	120	1,440												1,440
Hoist & Shaft Doors	96 EH					15	1,440								1,440
Surface Labor (2)	24 MS	120	2,880												2,880
SUBTOTAL ITEM 13100			14,055				1,440		12,425		65,100		83,700		176,720
Item 13200 No. 4 Shaft	2715 CY														
13210 Purchase Select Clay	2986 CY									7.00	20,900				20,900
13220 Haul to site	2986 CY											9.00	26,874		26,874
13230 Stockpile, 4CY FEL	100 EH	15	1,500					25	2,500						4,000
13240 Reclaim, mix, load skips															
Mixer Truck	54 EH	15	810					30	1,620						2,430
4CY FEL	54 EH	15	810					25	1,350						2,160
13250 Place															
Labor (2) 25CY/MH	109 MH	15	1,635												1,635
13260 Surface Plant															
Hoistman	8 MS	120	960												960
Hoist & Shaft Doors	64 EH					15	960								960
Surface Labor (2)	16 MS	120	1,920												1,920
SUBTOTAL ITEM 13200			7,635				960		5,470		20,900		26,874		61,839