Underground Disposal of Retorted Oil Shale for the Paraho Retorting Process

Prepared For

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By

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Western Division
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16. Abstract (Limit: 200 words) This report covers the results of a two-phase study to determine the feasibility of underground disposal of retorted oil shale. Two mining methods, chamber and pillar mining and sublevel stoping, were specified for backfilling. Phase I of the study covered the evaluation of hydraulic, mechanical, and pneumatic transport and stowing systems, individually and in various combinations, and the selection of the most feasible methods. Conveyor transport and stowing and a combination conveyor transport and stowing with pneumatic topfilling were selected for detailed study of phase II. Phase II activities included detailed engineering analysis and design of the selected systems, an engineering economic analysis, and a cost comparison with total surface disposal of retorted oil shale. An extensive literature search was conducted and the bibliography is included in the report. Research done under Contract No. J0265052 by the Cleveland-Cliffs Iron Co. Available for reference at Bureau of Mines facilities in Denver, Colo., Twin Cities, Minn., Bruceton and Pittsburgh, Pa., and Spokane, Wash.; U.S. Department of Energy facilities in Carbondale, Ill., and Morgantown, W. Va.; National Mine Health and Safety Academy, Beckley, W. Va.; and National Library of Natural Resources, U.S. Department of the Interior, Washington, D.C. Order ONLY from NTIS:

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Oil shale	Surface disposal	Underground support
Retorted oil shale	Mining	Belt conveyors
Waste disposal	Chamber and pillar	Hydraulic transport
Underground backfilling	Sublevel stoping	Pneumatic transport
Mass flow	Materials handling	Hydrology
Cooling		

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FOREWORD

This report was prepared by The Cleveland-Cliffs Iron Company, Western Division, P.O. Box 1211, Rifle, Colorado 81650 under U. S. Bureau of Mines Contract No. J0265052. The contract was initiated under the Bureau of Mines Program for Advancing Mining Technology--Oil Shale. It was administered under the technical direction of the U. S. Bureau of Mines, Spokane Mining Research Center, with Mr. John R. M. Hill acting as the Technical Project Officer. Mr. David J. Askin was the contract administrator for the Bureau of Mines.

This report is a summary of the work recently completed as part of this contract during the period July 1976 to April 1978. This report was submitted in May 1978.

No patents or inventions have resulted from this study.

Reference to specific brands, equipment, or trade names in this report is made to facilitate understanding and does not imply endorsement by the U. S. Bureau of Mines.

EXECUTIVE SUMMARY

The objective of this Contract was to determine the most desirable systems for the underground disposal of retorted oil shale from the Paraho retorting process based on the mining methods most likely to be employed in the deeper oil shale deposits. Two mining methods, chamber and pillar mining and sublevel stoping, were specified for backfilling. A deep underground mine in the central part of the Piceance Creek Basin of northwestern Colorado was assumed for this study.

The contract work was divided into two separate phases. Phase I involved the investigation of several backfilling methods and the selection of the most feasible methods for more detailed study. Phase I activities and results are described in Section 7.0 of this report. Three basic methods for transporting and stowing retorted shale were studied individually and in combinations of two or three to determine the most desirable system for the prescribed mining methods. Hydraulic, pneumatic, and mechanical (belt conveyors and trucks) methods were studied. Limited laboratory work was done to determine the physical and chemical characteristics of hydraulically transported retorted shale and the effects of cementing additives on dry (limited moisture) retorted shale. On the basis of a subjective and objective technical and economic analysis, conveyor transporting and stowing was selected as the most feasible underground disposal method. The principal advantages of the conveyor transport and stowing method, when compared to the other systems studied, are as follows:

- Highest fill density
- Most retorted shale stowed underground
- Greatest increase in potential resource recovery
- Highest potential strength for pillar support
- Least surface disruption
- Lowest ground water contamination potential
- Least environmental degradation
- Lowest manpower requirements
- Lowest energy requirements
- Lowest capital and operating costs
- Safest system

The main disadvantages of the conveyor transporting and stowing method are dependence upon a single conveyor and system routing is inflexible.

Hydraulic transport and stowing causes severe degradation of retorted shale. Resulting slimes must be removed if any degree of dewatering is to be achieved, and their disposal on the surface may be more adverse environmentally than total surface disposal of retorted shale. In addition, water requirements are high.

Pneumatic transport is inefficient and requires excessive amounts of energy to move the material. Equipment requirements and operating conditions are extreme. Dust is a major problem in the pneumatic stowing activity.

The work covered in Phase II included detailed engineering analysis and design of the selected disposal system. The system includes the following facilities:

- Cooling facilities to reduce retorted shale temperature from 400°F to 100°F prior to placing it in the mine.
- Surface conveyors for transporting the retorted shale to the cooling facility and ultimately to the borehole.
- A large diameter borehole for moving the retorted shale vertically from the surface to the underground backfilling levels.
- Underground conveyor network for distributing the retorted shale from the borehole to the backfilling areas.
- Backfilling facilities and equipment for the actual placement of retorted shale in the mined-out chambers and stopes.
- Dust control and environmental monitoring systems for the entire backfilling operation.
- Surface disposal facilities for that portion of the retorted shale that cannot be placed back in the mine because the volume to be disposed of exceeds the original in-place volume of the oil shale prior to mining.

Limited laboratory work was done to determine the mass flow and cooling characteristics of the retorted shale. An engineering economic analysis was done for the selected system with all operating and capital costs being discounted to present worth. A discounted cash flow analysis was not performed since the determination of mining and retorting costs and shale oil pricing were outside the Scope of Work for the contract.

SUMMARY OF CONCLUSIONS AND RECOMMENDATIONS

Results of the study indicate that 70 to 85 percent of the retorted shale can be placed back in the mine with the remainder being disposed of on the surface. Total underground and surface retorted shale disposal costs will be about $0.80 per ton of retorted shale or $1.10 per barrel of shale oil produced. This was based on fourth quarter 1977 dollars and a deep underground mine and surface retorting facility producing shale oil at a rate of 50,000 barrels per day. Total surface disposal costs would be $0.40 per ton of retorted shale or $0.55 per barrel of shale oil produced.

Underground backfilling can reduce the surface environmental impact of retorted shale disposal by reducing the land area required for surface disposal to 15-30 percent of that required for total surface disposal. Surface subsidence can also be reduced by backfilling. Backfilling can increase resource recovery by about 15 percent by allowing thinner pillars to be left in the backfilling areas.

The study has shown the need for additional work. The hydraulic characteristics of retorted shale should be determined under conditions more closely approximating expected field conditions. The effects of retorting temperature on the self-cementing properties of retorted shale warrant additional work. The Bureau of Mines has awarded Contract No. J0285001, "Natural Cementation of Retorted Oil Shale," for investigation of this property.

Modified in situ retorting facilities will, in most cases, employ surface retorts for processing the oil shale mined to provide expansion space for rubblization. The retorted shale from these surface retorts could be disposed of in the depleted in situ retorts. However, the concept is new and untested. Both the technical feasibility and the economic ramifications of such a disposal system must be considered but will not be known until more investigative work has been completed.

Utilization of waste heat from the cooling system for preheating the retort feed and for power generation should be studied further.

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1.0 INTRODUCTION

This report presents the results of a two-phase contract study of underground disposal of retorted oil shale from the Paraho retorting process. Two mining methods, chamber and pillar mining and sublevel stoping, which are amenable to mining the deeper oil shale deposits of the Piceance Creek Basin in northwestern Colorado were stipulated in the Contract. Phase I of the study covered the evaluation of hydraulic, mechanical, and pneumatic transport and stowing systems individually and in various combinations and the selection of the most feasible methods for detailed analysis. Phase II included detailed engineering analysis and design of the selected system, an engineering economic analysis, and a comparison with total surface disposal.

1.1 Scope of Work - Phase I

Work under Phase I of the contract began in July 1976 and was completed in February 1977 with an Interim Report of results being submitted to the Bureau of Mines. The results of Phase I work indicated that conveyor transport and stowing was the most feasible system for underground disposal of retorted oil shale for either of the prescribed mining methods. An alternative method using conveyor transport and stowing with pneumatic topfilling was also selected for more detailed study.

Seventeen combinations of hydraulic, mechanical, and pneumatic transport and stowing systems were investigated and ranked prior to selecting the most feasible system for underground retorted shale disposal. Laboratory work included a determination of physical and chemical characteristics of hydraulically transported shale and the effects of cementing additives on dry retorted shale.

1.2 Scope of Work - Phase II

Phase II work started in February 1977 and was completed in April 1978 with the publication of this Final Report. The underground backfilling methods selected in Phase I were studied further for detailed engineering analysis and system design. Equipment and operating costs were refined and an engineering economic analysis was completed. Since mining and retorting costs and shale oil pricing were not within the Scope of Work, the final economic evaluation did not include a discounted cash flow analysis. Instead, it was limited to a present worth evaluation of all costs for backfilling over the assumed project life.

Limited laboratory work was done to determine the mass flow and thermal characteristics of the retorted shale.

2.0 CONDITIONS AND ASSUMPTIONS

2.1 Physical Properties and Conditions:

2.1.1 Physical Properties:

Most of the retorted oil shale used in physical properties determinations was produced by the Paraho direct-heating mode semi-works retort. It should be noted that the characteristics of retorted shale from other retorting methods may vary significantly from those used in this study.

Retorted shale leaves the retort at about 400°F (14)*. The size of the material varies from approximately 2-1/2-inch-diameter fragments to extremely fine silt, as shown in Table 2.1.1-1.

TABLE 2.1.1-1

SCREEN ANALYSIS

RETORTED SHALE FROM RETORT

Screen Size	Range Percent Passing (14)	Range Percent Passing (2)
3"	100 - 100	100 - 100
1-1/2"	96 - 96	88 - 97
3/4"	76 - 77	64 - 76
3/8"	48 - 57	36 - 54
4M	34 - 45	27 - 44
8M	32 - 44	27 - 44
16M	29 - 43	26 - 44
30M	24 - 37	25 - 43
50M	18 - 33	23 - 41
100M	15 - 27	20 - 39
200M	13 - 22	18 - 36
325M	12 - 21	13 - 23

M = U. S. Standard Sieve

Variations in physical properties reported by different sources are attributable to sample handling prior to analysis and differences in conditions within the retort during the retorting phase. Loose density of freshly retorted shale varies from 60 pounds to 75 pounds per cubic foot (13). Seventy pounds per cubic foot is a representative value for the material used in this report (14).

*(14) Designates reference listed in Section 10.1.

2.1.2 Physical Conditions:

The site selected for this study is the same as for the proposed Bureau of Mines deep underground oil shale research facility in Piceance Creek Basin in northwestern Colorado. The surface facilities will be located at an elevation of approximately 6,200 feet above sea level. Mining activity will be 2,000 feet below the surface in the saline zone of the Green River Formation. Two aquifers are anticipated above the backfilling levels. The ambient temperature at a depth of 2,000 feet, as taken from the log of the shaft pilot hole, will range from 90°F to 100°F (12). Surface temperatures range from -40°F to 90°F with an average annual temperature of 40°F (1).

2.2 Economic Parameters:

2.2.1 Capital and Operating Costs:

All capital and operating costs are based on current 1977 dollars. Discounting for present worth is considered in the economic analysis of the transport and stowing methods. Labor costs used are representative of mining activities in Western Colorado. No allowances are made for inflated wage rates common to most energy-related impact areas.

All equipment and manpower are assumed to be used solely for disposal operations. No attempt has been made to place a dollar value on the environmental effects of surface disruption resulting from surface disposal of retorted oil shale.

2.3 Basic Assumptions:

2.3.1 Production Parameters:

A mine-retort facility which produces shale oil at a rate of 50,000 barrels per day was selected for use in comparing the prescribed transport and stowing methods. Paraho direct-heating-mode retorts have been assumed because the retorted shale that was available for testing was produced in the semi-works version of this retort. An adequate number of retorts will be available to permit 50,000 barrels per day to be produced 365 days per year. A retort efficiency factor (percent available oil recovered) of 0.95 has been assumed. Eighty-two percent, by weight, of the dry retort feed will report as retorted shale.

The average grade of the retort feed will be 28 gallons of oil per ton of ore. In determining the amount of material to be transported, it has been assumed that the retorted shale will contain five percent residual moisture resulting from cooling and dust suppression processes. The amount of moist, retorted shale for disposal is calculated as follows:

$$\text{Tons/Day} = 50{,}000\ \frac{\text{bbl}}{\text{day}} \times 42\ \frac{\text{gal}}{\text{bbl}} \times \frac{\text{ton}}{28\ \text{gal}} \times \frac{1}{0.95} \times 0.82 \times 1.05 = 67{,}974$$

Use: 68,000 tons per day material to disposal system.

Surplus retorted shale that cannot be placed underground must be disposed of on the surface. This surplus material is the result of the time lag between development and mining to provide sufficient stopes for backfilling, plus the inherent greater volume of retorted shale as compared to in-place raw shale.

An additional source of waste material from the Paraho process, which was outside the Scope of Work for this contract, is the rejection of fine raw shale prior to retorting. The fines rejected represent approximately five percent of the oil shale mined.

2.4 Mining Methods:

Chamber and pillar mining and sublevel stoping are the two mining methods for which underground disposal of retorted oil shale are being developed. The methods were specified in the contract for this study and both are amenable to mining the deeper deposits in the Piceance Creek Basin and to backfilling.

The word "stope" is used in the report to signify either a chamber or a sublevel stope where the description applies to both. However, in cases where one method is being discussed explicitly, the appropriate term is used.

2.4.1 Chamber and Pillar Mining With Backfill:

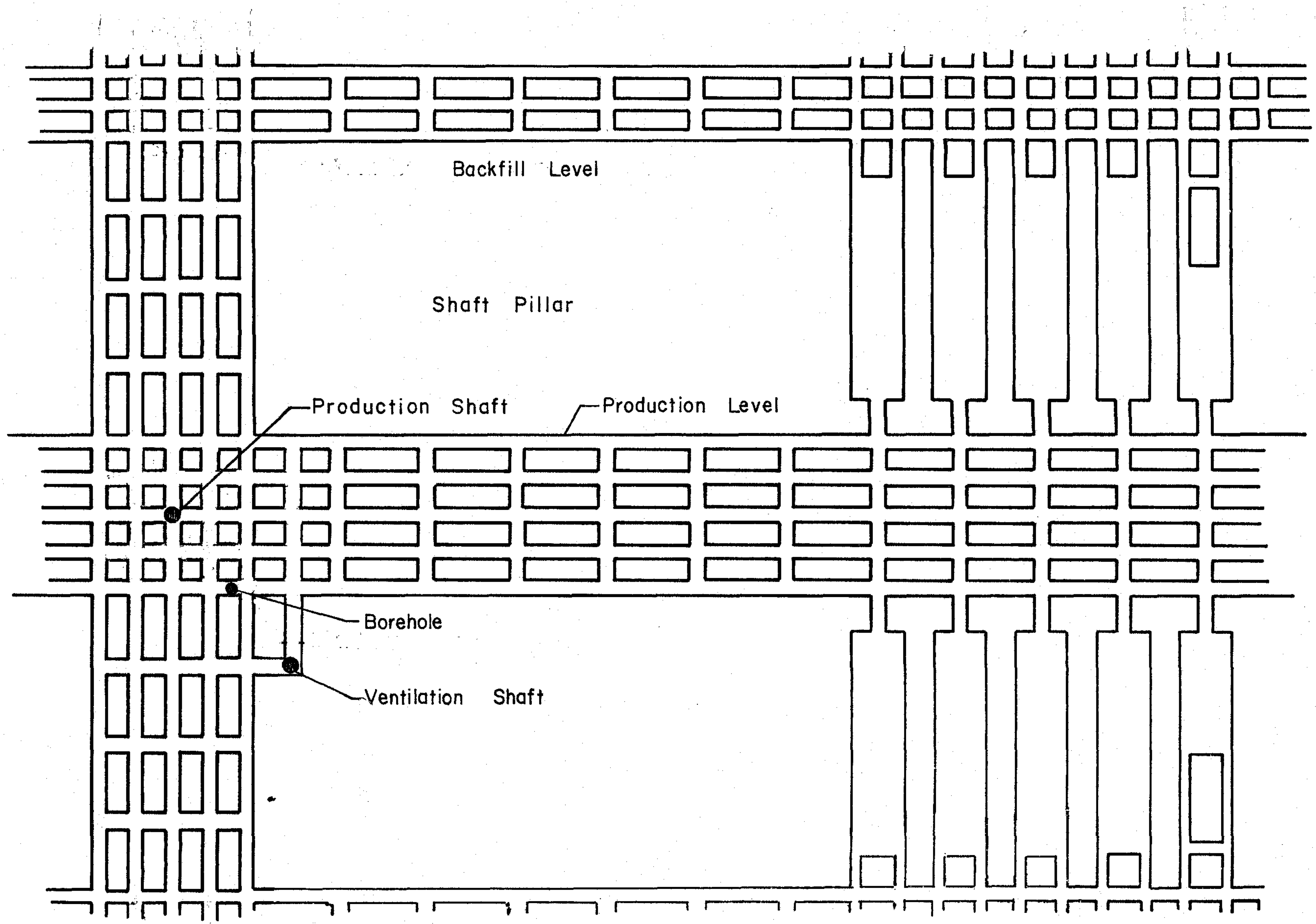
Chamber and pillar mining is a modification of room and pillar mining in which drifts, driven normal to the main entries, are enlarged into chambers by fan drilling and blasting. The major advantage of this system is that it facilitates backfilling which has a stabilizing effect on the existing pillars and thus increases the overall extraction ratio. Chamber preparation consists of developing LHD entries, chamber drilling drifts, and an initial slot. Chamber preparation will be followed sequentially by chamber mining and backfilling (Figures 2.4.1-1 through 2.4.1-3).

2.4.1.1 LHD Entry Development:

As part of each chamber development, a drift will be driven perpendicular to the main entries, along the projected centerline of each chamber, through a barrier pillar, and into the mining zone at the nearest end of the chamber area. Driven at the chamber floor elevation, the opening will serve as an LHD travelway until excavation of the chamber has been completed. The opening will also provide access to the mining area for scaling and roof-bolting equipment.

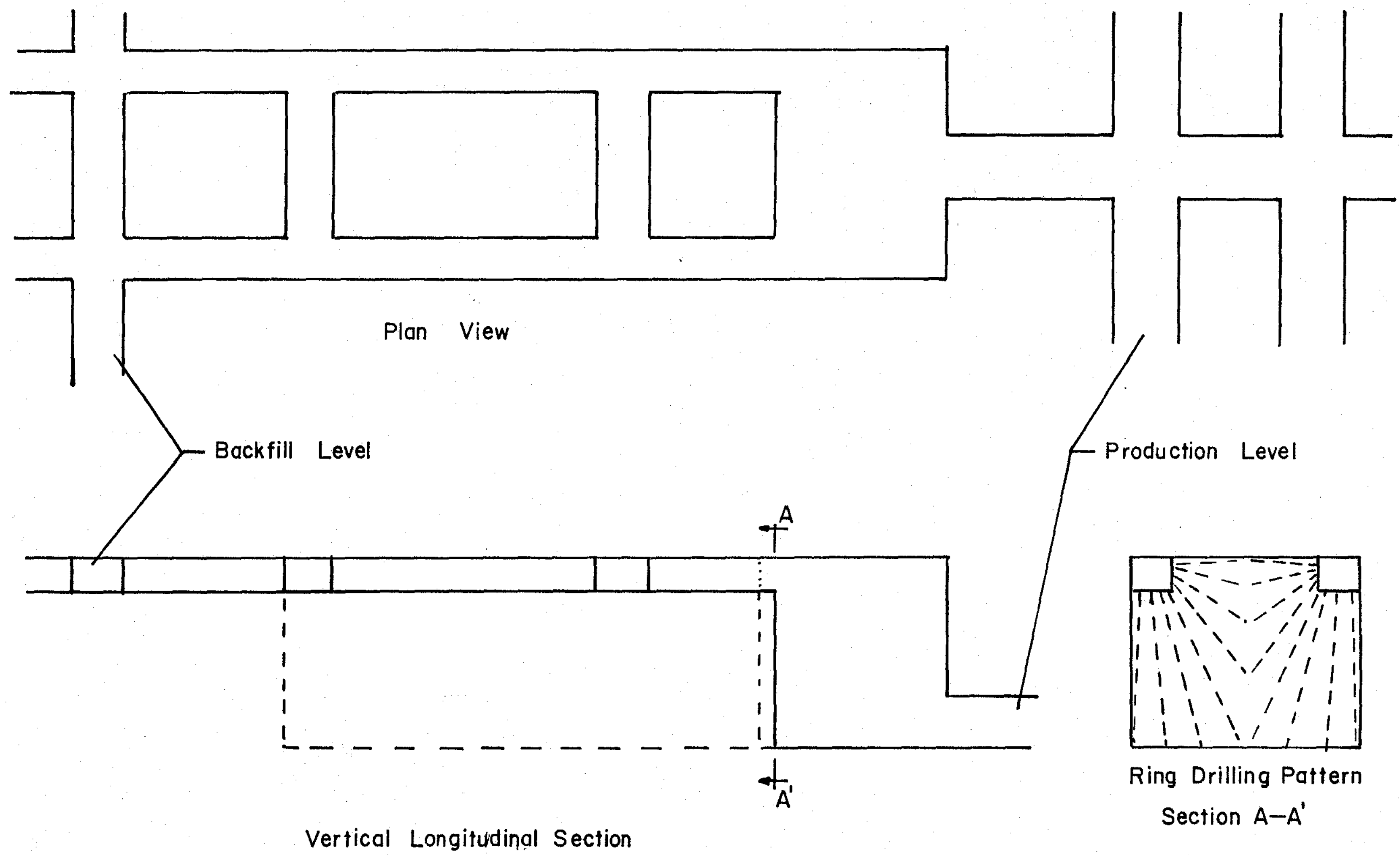
2.4.1.2 Drilling Drift Development:

A gassy environment requires that multiple headings must be driven in pairs with connecting crosscuts no more than 100 feet apart (23). For this reason, two ring-drilling drifts will be required for each chamber, with crosscuts at 100-foot intervals. The drifts will be driven at the chamber roof elevation and will extend from entries running behind the chambers. These drifts will provide access and ventilation for



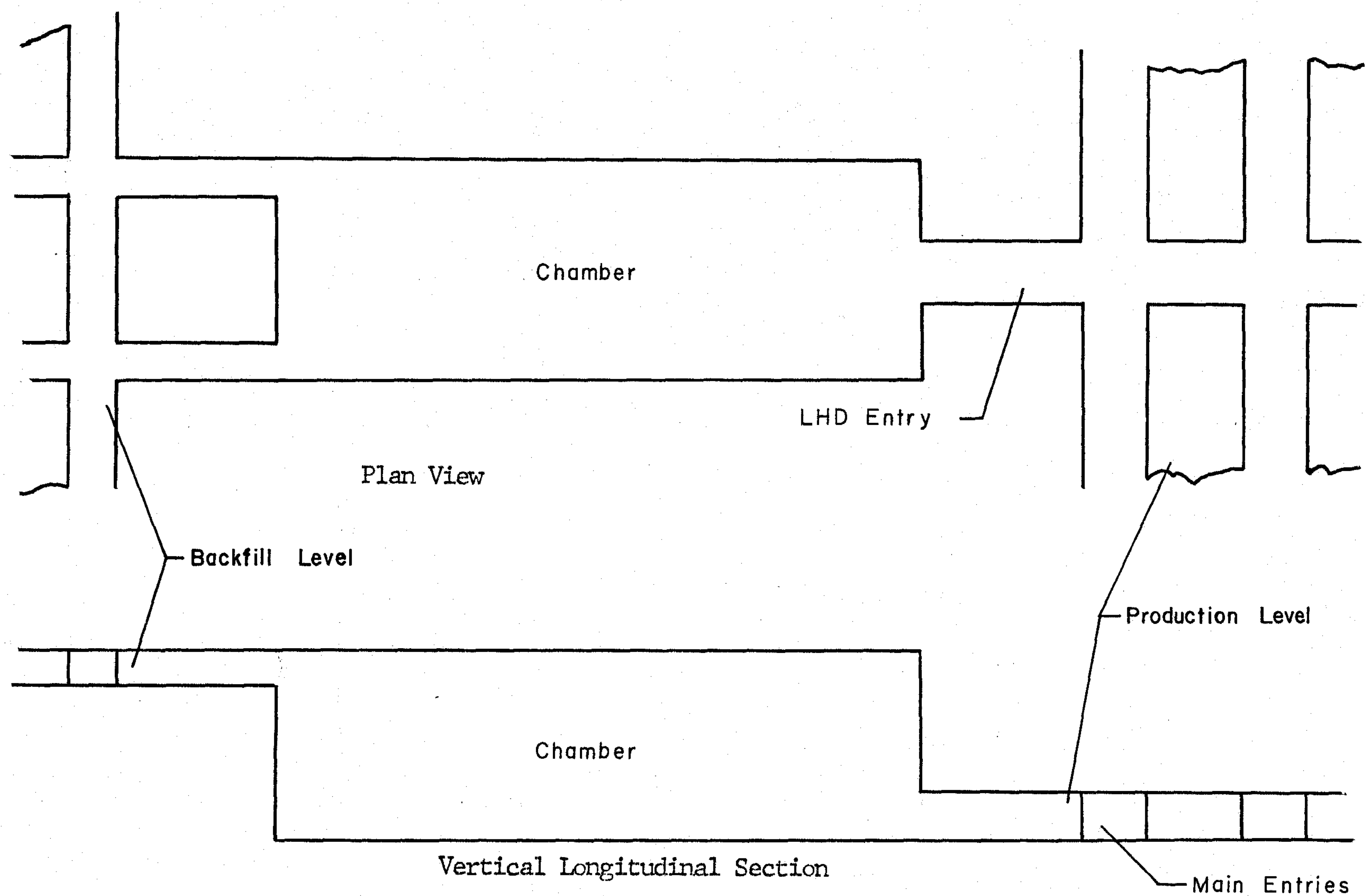
MINE LAYOUT - CHAMBER AND PILLAR MINING
(BACKFILLING LEVEL SUPERIMPOSED OVER PRODUCTION LEVEL)

FIGURE 2.4.1-1



CHAMBER DEVELOPMENT - CHAMBER AND PILLAR MINING

FIGURE 2.4.1-2



TYPICAL CHAMBER - CHAMBER AND PILLAR MINING

FIGURE 2.4.1-3

the backfilling operation. A final ventilation crosscut will connect the ends of the two drilling drifts.

As the roof in both ring-drilling drifts and ventilation crosscuts will ultimately become part of the chamber roof, they will be bolted as they are exposed.

2.4.1.3 Slot Development:

A raise will be blind bored vertically upward from the LHD entry to intersect the crosscut that joins the ends of the ring-drilling drifts at the front of each chamber. The raise will provide blast-hole relief as the opening is expanded to a slot by subsequent blasting of vertical holes drilled from the crosscut above. When completed, the slot will extend to the outer limits of the chamber profile. The opening will provide a free face for larger-scale production blasting as the remainder of the chamber is mined.

2.4.1.4 Chamber Mining:

Initially, a single row of holes, drilled downward in an asymmetrical fan pattern from each drilling drift, will be blasted to start production in each chamber. Thereafter, two successive fans of holes will be detonated for each blasting cycle. The blasted material will be loaded with LHD's and hauled to the main haulage entry where it will be discharged into a feeder-breaker. The broken ore is then conveyed to the shaft for hoisting to the surface.

The relatively thin rib pillars that separate the chambers are designed for maximum resource extraction and will require that chambers be mined in an alternating sequence. Intermediate chambers will not be excavated until the mined-out chambers on either side have been backfilled, thus stabilizing the intervening rib pillars.

2.4.1.5 Advantages and Disadvantages:

Chamber and pillar mining is a variation of room and pillar mining, modified specifically to facilitate backfilling. The advantages and disadvantages of this method are as follows:

ADVANTAGES:

1. It is a relatively flexible system, since it can be modified in a number of ways to suit varying geologic or economic conditions.

2. It permits a high degree of mechanization and some degree of selectivity in mining.

3. It facilitates backfilling which, in turn, will allow an increased overall extraction ratio.

4. The amount of preproduction development is relatively small and most of the development work can be confined to the ore zone itself.

DISADVANTAGES:

1. Production will come from a number of working areas. As a result, supervision will be difficult and productive time will be lost in repeated movement of equipment and supplies.

2. Compared to sublevel stoping, the extraction ratio may be relatively small, especially in thick ore bodies.

3. Costs for ventilation and roof control are higher compared to other mining methods. However, with a high degree of mechanization using large-size equipment, the overall cost per ton may prove to be competitive with other methods.

4. This method requires a competent layer of rock above the mining zone.

2.4.2 Sublevel Stoping With Backfill:

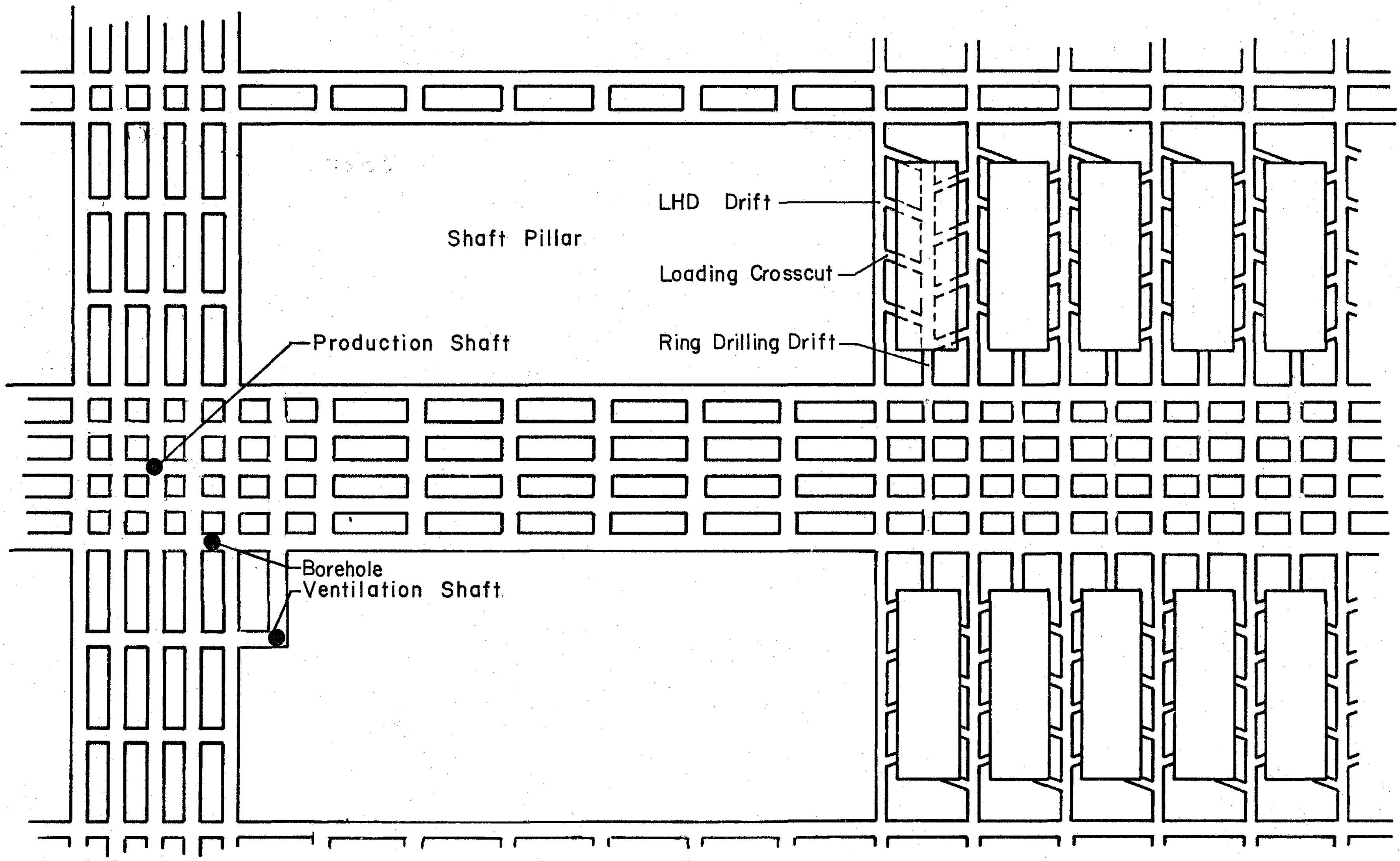
Sublevel stoping is a large-production, low-cost open stoping method which is well suited to fairly regular ore bodies having both competent ore and host rock. Open stope production utilizes long-hole drilling from levels and sublevels, and blasting in successive slices. Backfilling improves ground control in a sublevel stoping system. However, when compared to sublevel stoping with full subsidence, it results in a reduced total resource recovery due to unmined pillars (Figures 2.4.2-1 through 2.4.2-4). Mining the lowest stopes first in a multilevel operation eliminates the need for sill pillars and precludes the hazards of working beneath filled stopes which may become saturated before the lower areas are mined out.

2.4.2.1 Haulage Level Development:

Preproduction development will begin with the driving of LHD drifts and an exhaust entry at the floor level of the stopes. From each LHD drift, loading crosscuts will be driven to connect with the drawpoints. A ring-drilling drift in the center of each stope will be driven at the stope floor level and will become the stope floor. These drifts will connect the various drawpoints in each stope. The roofs of the LHD drifts, loading crosscuts, and exhaust entry will be roof bolted.

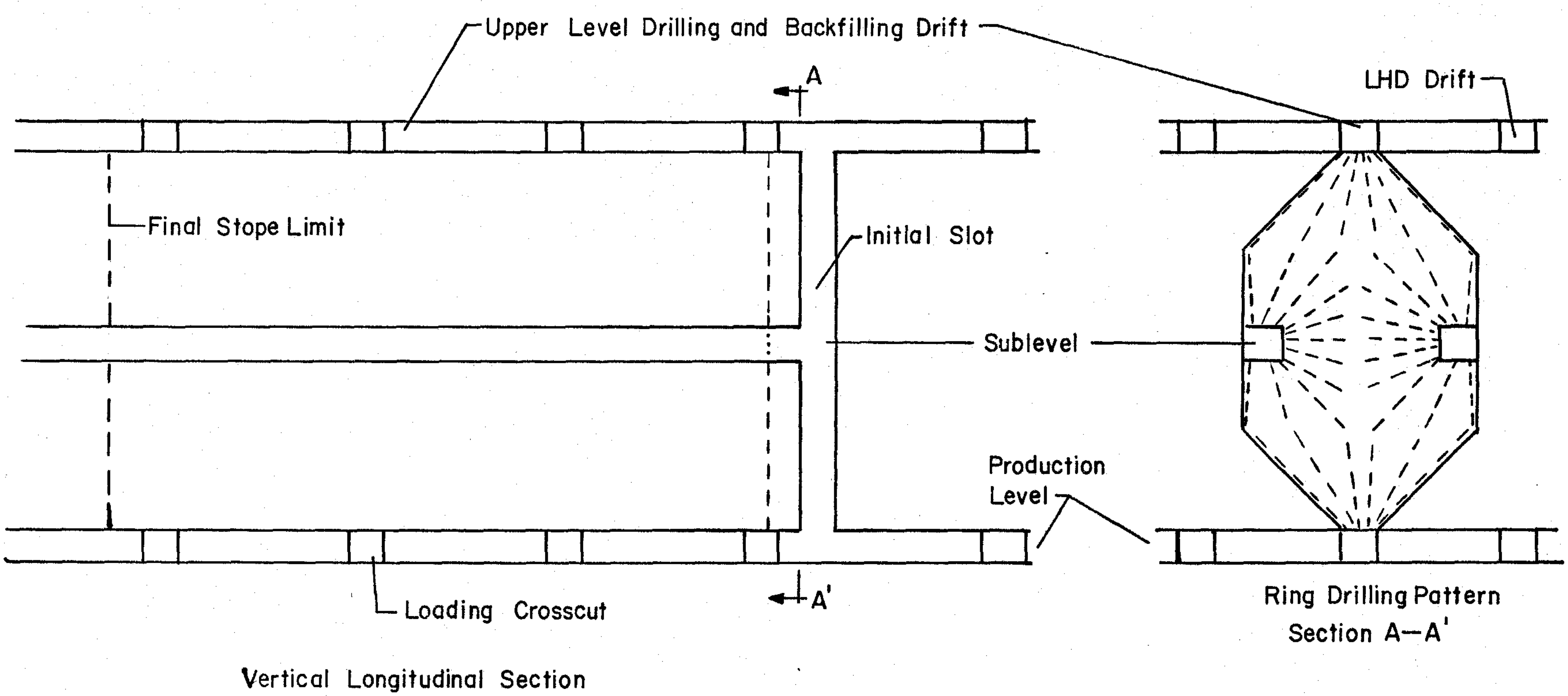
2.4.2.2 Sublevel Development:

One or more sublevels, driven at approximately 100-foot intervals above the stope floor, will be developed by driving ring-drilling drifts. Two drifts will be driven for each sublevel, one along each longitudinal boundary of each stope. The drifts will extend the full length of the stope and will require ventilation crosscuts on approximately 100-foot centers to connect the two drilling drifts within each stope (23).



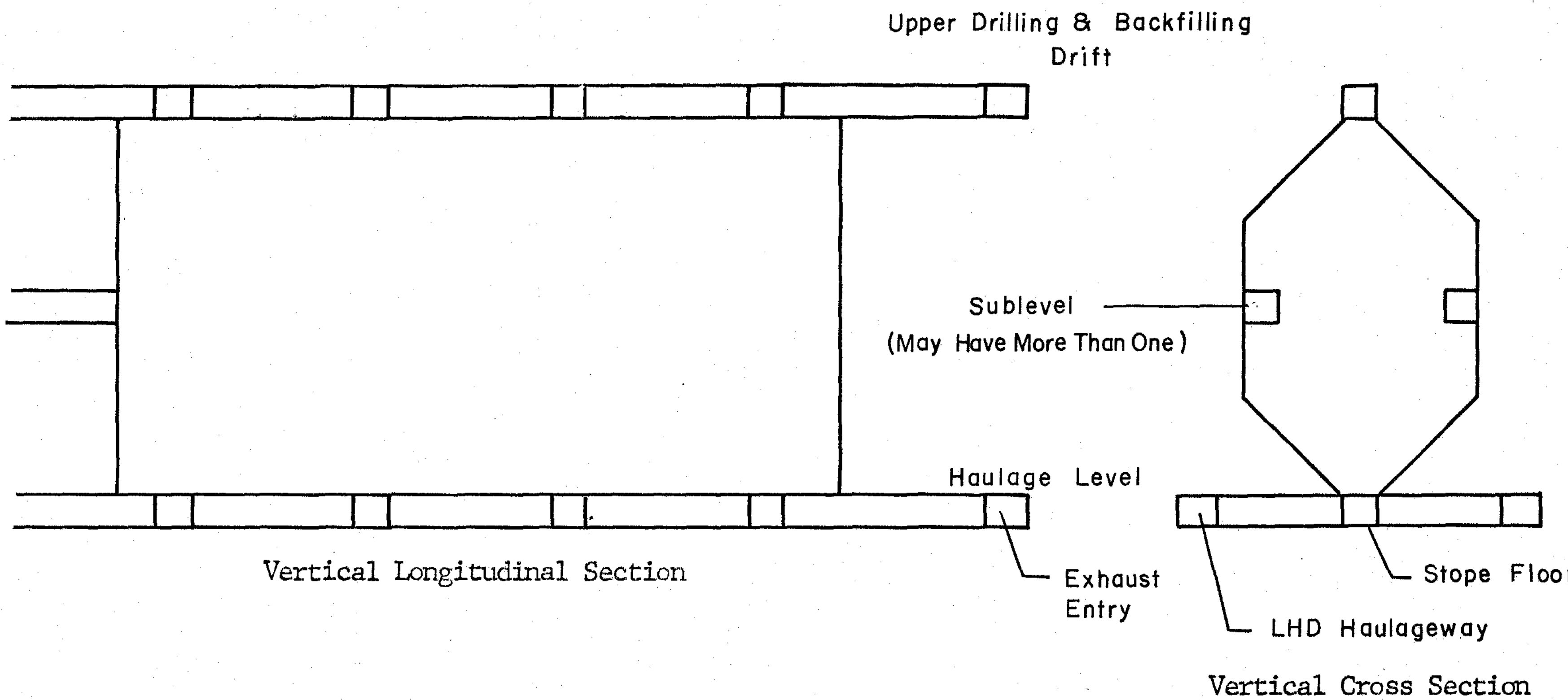
MINE LAYOUT - SUBLEVEL STOPING

FIGURE 2.4.2-1



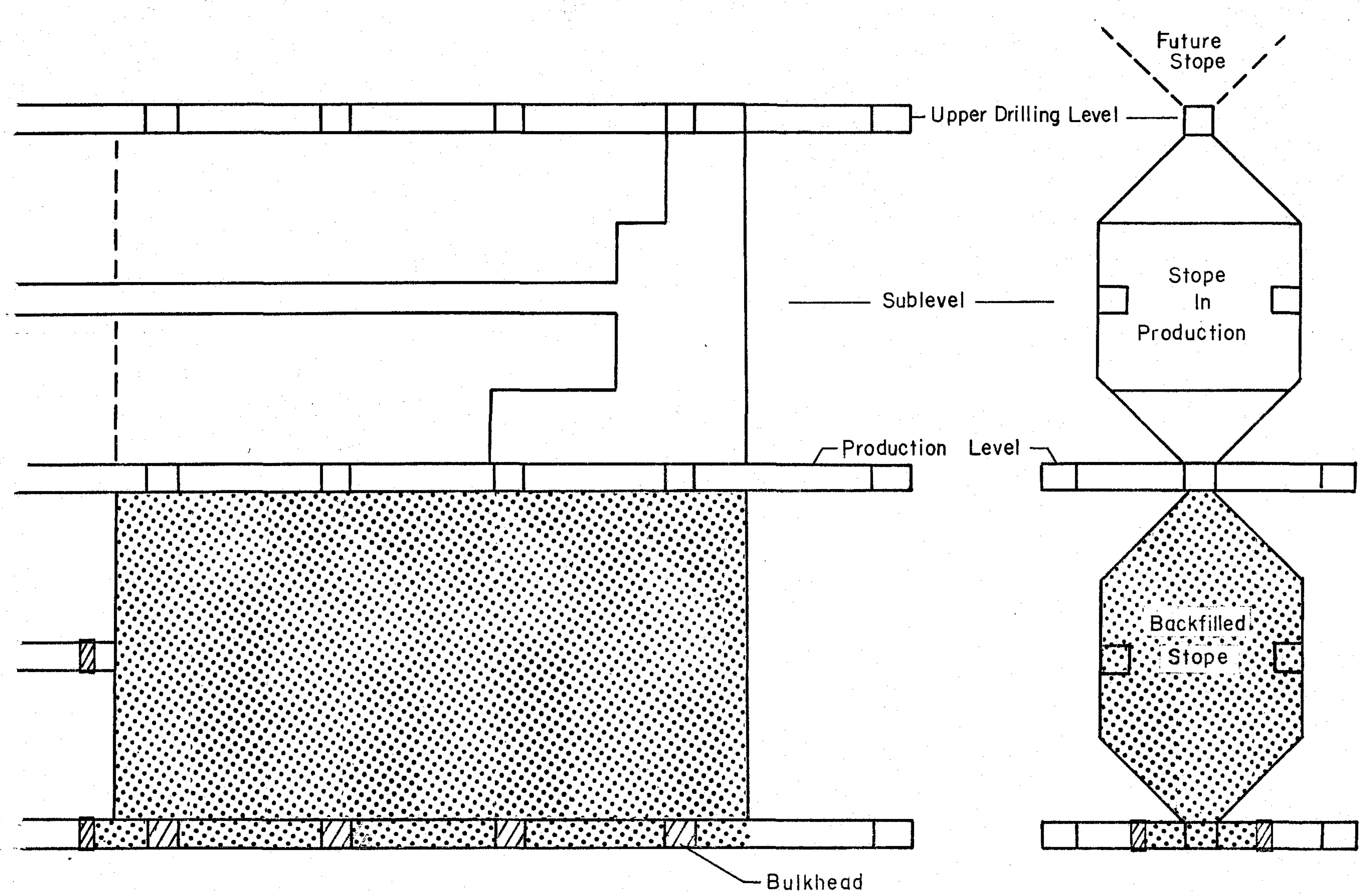
SUBLEVEL STOPE - INITIAL SLOT DEVELOPMENT

FIGURE 2.4.2-2



TYPICAL SUBLEVEL STOPE

FIGURE 2.4.2-3



MULTILEVEL SUBLEVEL STOPES

FIGURE 2.4.2-4

2.4.2.3 Upper Level Development:

The upper level will be developed by driving ring-drilling drifts at the stope roof level, one in the center of each stope. Upper level development will duplicate the design of the underlying haulage level and will then serve as the next production area when the lower level is mined out.

2.4.2.4 Slot Development:

A raise will be blind bored along the rear wall of each stope from the stope floor to the upper level. It will provide blasthole relief during initial slot development.

Following completion of the slot raise, the slot will be opened to the full height and width of the stope in two or more steps. The first step will be enlargement of the bottom 100 feet or more of the raise. This will be accomplished by drilling holes upward from the stope floor and drilling holes both upward and downward from the sublevel. The second step will be to open the slot to the full height and width of the stope. The lower part of the slot will be blasted initially to provide expansion for the remaining blasts. Subsequently, the slot will be excavated by drilling and blasting from the sublevels. Excavation of the upper part of the slot, including enlargement of the bored raise, will be accomplished from the upper level. The slot will provide expansion room for the large-scale production blasting of the remainder of the stope.

2.4.2.5 Stope Mining:

The relatively thin rib pillars that separate the stopes are designed for maximum resource extraction and will require that the stopes be mined in an alternating sequence. The stopes will be mined by ring drilling from the stope floor, sublevels, and upper level drilling drifts. Mining will start at the slot at the rear of each stope and retreat toward the main entries. Ring drilling will be a continuing activity and the rings will be blasted as needed. Excavation from the stope floor will precede excavation from the overlying sublevel to prevent muck from filling the stope floor drift. Following backfilling of two alternate mined-out stopes, the intermediate stope will be mined and subsequently backfilled. The muck is loaded from the loading crosscuts and the loaders never work on the fill material in the lower stopes.

2.4.2.6 Advantages and Disadvantages:

Sublevel stoping with backfill method is designed to prevent fracturing of the strata above the stope and thus reduce significantly the potential for surface subsidence. The advantages and disadvantages of the method are as follows:

ADVANTAGES:

1. Increased resource recovery by eliminating the sill pillars.

2. After stope development is completed, large tonnages are produced at a low cost per ton.

3. Miners work in small openings and are not exposed to wide roof spans; hence, the method is safe.

4. Effective ground control is possible.

5. A high degree of mechanization can be employed.

6. Fragmentation is adequate, especially with the use of delay detonators.

7. Backfilling improves both extraction ratio and ground control and provides a means of underground disposal of retorted shale.

DISADVANTAGES:

1. A large amount of stope development and consequent high capital outlay is required.

2. Resource recovery is lower than in sublevel stoping with full subsidence due to unmined pillars that are left for support.

3. The surrounding rock must be structurally stable to prevent premature ground movement as stoping progresses.

3.0 DESCRIPTION OF SELECTED METHODS

The three general methods stipulated in the Contract for transporting to and stowing retorted oil shale in underground disposal areas are hydraulic, mechanical, and pneumatic. These methods were investigated in sufficient depth to rank the technical and operating feasibilities, relative costs, and environmental effects, and select the most suitable method. Phase I work discussed in Section 7.0 describes the alternative methods and the rationale for selecting conveyor transporting and stowing as the most efficient system for placing retorted oil shale underground that is compatible with the stipulated mining methods.

Since retorted shale will leave the retort at about 400°F and the ambient mine temperature will be approximately 95°F, the retorted shale must be cooled prior to placement. A surface cooling facility will be an integral part of each underground transport method.

A large diameter vertical borehole was chosen for transporting the retorted shale from the surface to the underground disposal levels. The depth to the backfilling level, the possible existence of one or more aquifers between the surface and the backfilling level, and the volume of material to be moved influenced this selection.

3.1 Cooling:

Elevated temperatures present several problems for an underground retorted oil shale disposal operation. Initially, it was felt that placement of hot, retorted shale in the mined out areas of the mine would result in reduced pillar stability due to accelerated creep and thermal stresses. These concerns are valid and are being investigated under other USBM contracts. However, temperatures high enough to affect physical rock characteristics will create far more severe environmental problems within the mine.

The ambient rock temperature at the planned mining depth is about 95°F. This fact in itself creates a potential health and productivity problem for the work force. To avoid drastic aggravation of an existing problem, the retorted shale will be cooled prior to transport into the mine. This cooling will also negate the possiblity of creep and thermal stress in the pillars.

Additional benefits of cooling the retorted shale may be found in the area of waste heat utilization. Preheating retort feed or power generation may be possible. These possibilities were not studied for this report.

3.1.1 Cooling Methods and Design Parameters:

Major factors adversely affecting cooling efforts are as follows:

1. Water is not readily available in the project area.

2. To avoid flow problems of the retorted oil shale in the eight-foot-diameter borehole, the material

must be dry and degradation must be kept to a minimum.

3. Dust must be effectively suppressed or controlled.

Laboratory tests were conducted to determine the cooling characteristics of the retorted oil shale between 400°F and 100°F. The results of the tests provided the following parameters which were used in the design of the cooling units.

Specific heat capacity: 0.28 BTU/LB/°F

Cooler retention time: 8 Minutes

Air velocity at cooler exits: 500 fpm

Due to the possible lack of sufficient quantities of water in the project area available for cooling, the methods investigated utilized air only, or a combination of air and water, as the cooling medium.

3.1.1.1 Air Swept-Rotary Tube Cooler (Alternate #1):

This alternate cooling method utilizes only direct air contact as the cooling medium. The process involves a rotating drum equipped with lifters which causes the material to cascade and mix as it passes through the cooler. The cascading effect provides for direct contact of the material with the air. The air is drawn through the cooler in a counter current direction which is opposite to the direction of material flow in the cooler. As the material passes through the cooler, approximately six percent will be drawn off as dust by the airflow. The dust passes out of the cooler and into a cyclone which captures approximately 90 percent of the dust, and then into a baghouse which captures 99.5 to 99.9 percent of the remaining dust before the air is released into the atmosphere.

This system requires 32 air swept-rotary tube cooling units, each unit being 17 feet in diameter and 90 feet long. Each unit requires a 250-horsepower drive unit, a cyclone system, which consists of four 84-inch-diameter cyclones in a cluster, a baghouse, and a 500-horsepower induced draft fan.

ADVANTAGES:

1. Minor water requirement.

DISADVANTAGES:

1. Comparatively high capital, operating, and maintenance costs.

2. Large land area required for the total operation.

3.1.1.2 Air Swept-Water Tube Cooling (Alternate #2):

Assuming that water is available to assist in cooling, either from a mine or other source, this system, which uses a combination of water and air as the cooling medium, is applicable to this project.

The process involves a rotating drum which has four concentric rings of water pipes on the inside. These pipes carry the counter current flow of cooling water which provides indirect water cooling. The water pipes also provide for the mixing and cascading of the material as it passes through the cooler. The flow of cooling air is drawn through the cooler in a counter current direction and into a cluster of cyclones and baghouse before being released to the atmosphere.

A total of 500 gallons per minute of in-tube cooling water per cooling unit is required. The water cooling portion of this system provides an indirect cooling method in which the water flows through tubes and, therefore, does not come into direct contact with the retorted oil shale. After the cooling water passes through the cooler it flows through a cooling tower so that it can be reused. Approximately 50 gallons per minute of the cooling water evaporates to the atmosphere in the cooling tower for each cooling unit and must be made up with fresh water. To assist the cooling process, 50 gallons per minute of water is sprayed directly on hot retorted oil shale as it enters each cooler. Direct application of water provides some high efficiency initial cooling at the entrance to the cooler. This water totally evaporates into the cooling airflow and is not recoverable. Total water loss per cooling unit is 100 gallons per minute. Therefore, the total volume of required makeup water for the entire cooling system, which is made up of eight individual cooling units, is 800 gallons per minute. Each unit is 17 feet in diameter by 145 feet long and includes two 400-horsepower-rated main-drive units, a 700-horsepower-rated induced-draft fan, a cluster of four 84-inch-diameter cyclones, and a baghouse.

ADVANTAGES:

1. Low comparative capital, operating, and maintenance costs.

2. Small land area requirements.

DISADVANTAGES:

1. Water dependency.

2. System complexity (pumps and sumps, etc. to handle the water).

3.1.1.3 Air Swept-Water Tube Cooling (Alternate #3):

The cooling equipment and the cooling method for this system is identical to that in Alternate #2. The difference between the two is in the dust collection system.

Instead of baghouses as in Alternates #1 and #2, scrubbers are used for final dust removal. Scrubber slurry solids concentration is kept in the five to ten percent range by continually drawing off a quantity of sludge. This alternate increases the total makeup water required from 800 to 1,650 gallons per minute.

ADVANTAGES:

1. Lowest capital, operating, and maintenance cost.

2. Small land area required.

DISADVANTAGES:

1. Large water requirements.

2. System complexity (pumps and sumps, etc. to handle the water).

3. Scrubber slurry disposal requirements.

3.1.2 Cooling System Selection:

The three cooling systems were analyzed and ranked using Least Total Divisor Ranking Analysis. This ranking method is described in Section 7.6. Based on the subjective and objective technical analyses and economic considerations, Alternate #2 "Air Swept-Rotary Water Tube Cooling System" was selected for this study. Tables 3.1.2-1 through 3.1.2-4 show the results of the ranking analysis. The general cooling facility layout and cooler configuration are shown in Figures 3.1.2-1 through 3.1.2-3.

3.1.3 Surface Conveying System:

The retorted oil shale will be transported from the retort facility to a transfer point near the cooling facility. At this point, the material to be sent to the surface disposal area is separated from the cooler facility feed. Output from the coolers will be sent to the borehole for transport to the underground backfilling level. In an emergency situation, the cooled shale can be routed to the surface disposal area. The general surface conveyor layout is shown in Figure 3.1.2-1.

All conveyor transfer points will have dust collectors to control the dust generated. The captured dust will be ducted to the nearest cooler cyclone-baghouse unit for removal from the air.

3.1.3.1 Surface Disposal Conveyor:

The surface disposal conveyor will pick up the retorted shale at a transfer point near the cooling facility and deliver it to the surface disposal load-out bin. Normally the retorted shale will not be cooled prior to surface disposal. This conveyor will be capable of carrying the total retort output in the event that the cooling facility is

TABLE 3.1.2-1

COOLING SYSTEM

SUBJECTIVE TECHNICAL ANALYSIS

	Surface Environmental Effects	Air Quality	Sludge-Dust Disposal	Safety	Maintenance	Factor	Rank
#1 Air Swept Rotary	401.5	180.9	227.9	129.0	235.0	1,174.3	1.27
#2 Air Swept Rotary-Water Tube	275.0	126.9	202.1	99.0	220.9	923.9	1.00
#3 Air Swept Rotary-Water Tube and Scrubber	258.5	89.1	270.9	90.0	235.0	943.5	1.02

TABLE 3.1.2-2

COOLING SYSTEM

OBJECTIVE TECHNICAL ANALYSIS

	Energy Consumption RI=2*	Water Usage RI=2	Surface Area RI=7	Crew Size RI=7	Units Required RI=5	Total	Rank
#1 Air Swept Rotary	4.0	2.0	17.2	16.2	20.5	59.4	1.06
#2 Air Swept Rotary-Water Tube	2.0	32.0	7.0	7.0	8.0	56.0	1.00
#3 Air Swept Rotary-Water Tube and Scrubber	2.0	66.0	7.0	7.0	8.0	90.0	1.61

* RI = Relative Importance Factor

TABLE 3.1.2-3

COOLING SYSTEM

ECONOMIC ANALYSES

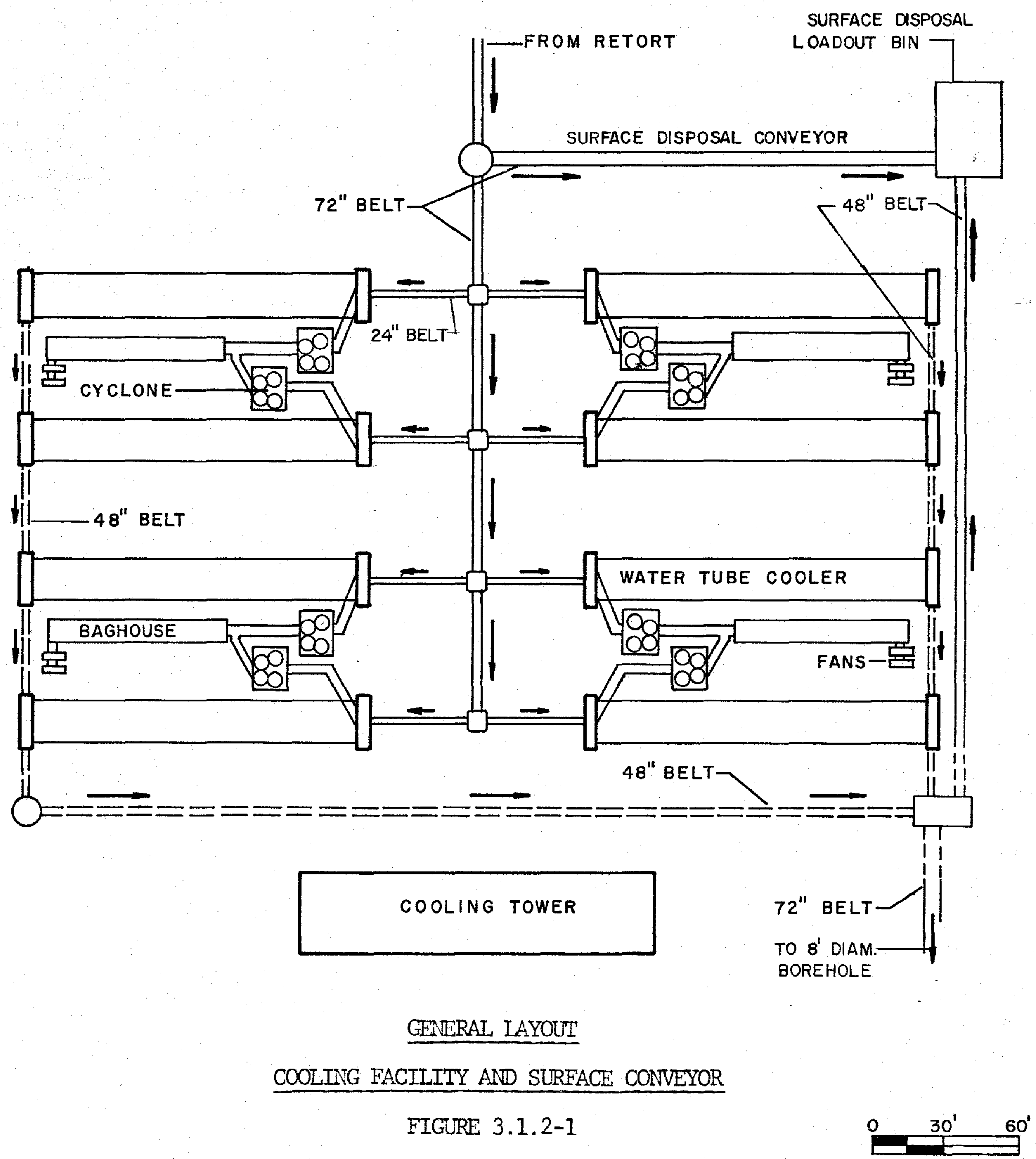
	Capital Costs		Operating Costs	
	$\$ \times 10^6$	Rank	\$/Ton	Rank
#1 Air Swept Rotary	43.0	2.81	0.3857	1.87
#2 Air Swept Rotary-Water Tube	16.0	1.05	0.2061	1.00
#3 Air Swept Rotary-Water Tube and Scrubber	15.3	1.00	0.2221	1.08

TABLE 3.1.2-4

COOLING SYSTEM

FINAL SELECTION

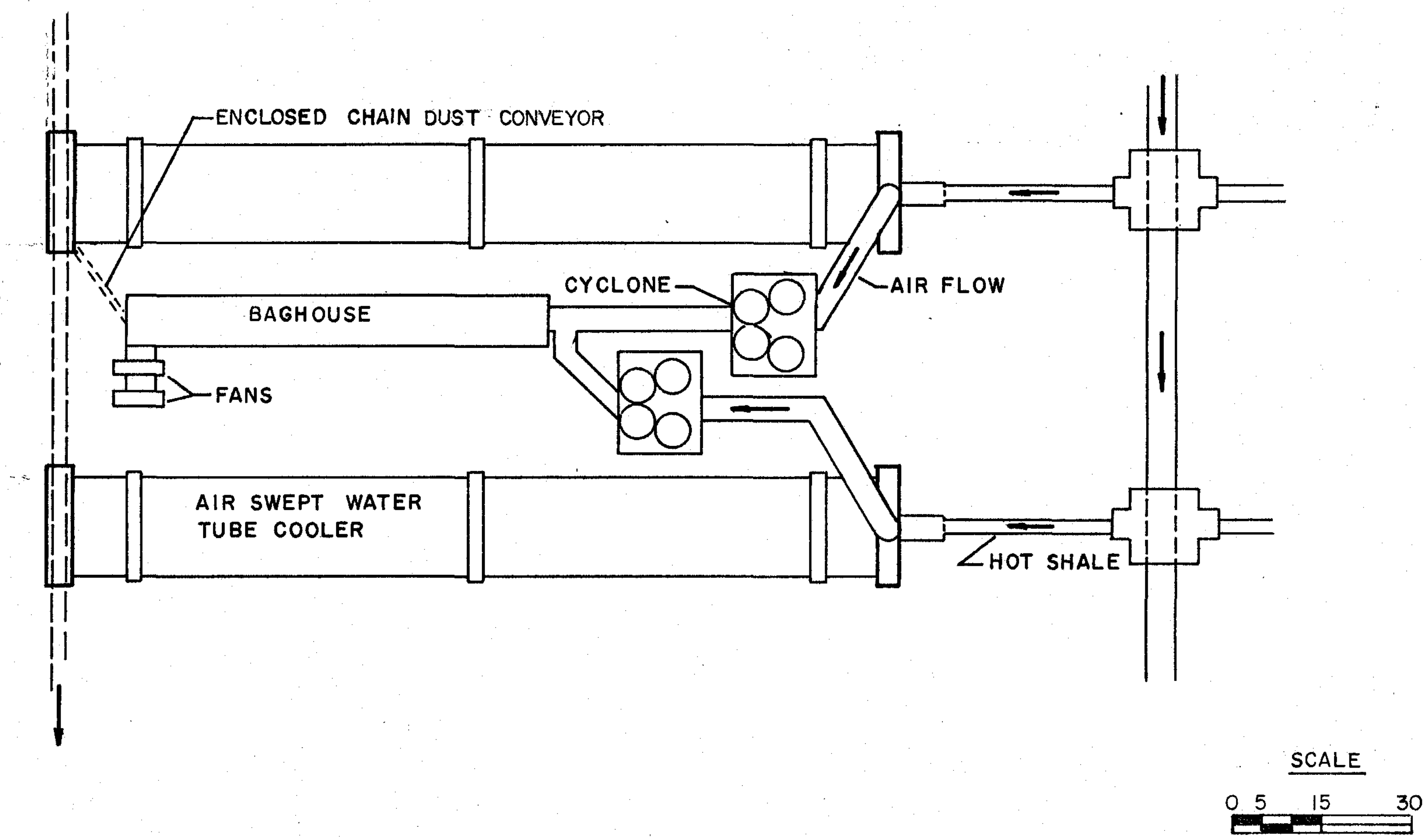
Alternative	Subjective Technical Analysis RI=6	Objective Technical Analysis RI=3	Capital Costs RI=3	Operating Costs RI=1	Total	Final Rank
#1	7.62	3.18	8.43	1.87	21.10	1.60
#2	6.00	3.00	3.15	1.00	13.15	1.00
#3	6.12	4.83	3.00	1.08	15.03	1.14



GENERAL LAYOUT

COOLING FACILITY AND SURFACE CONVEYOR

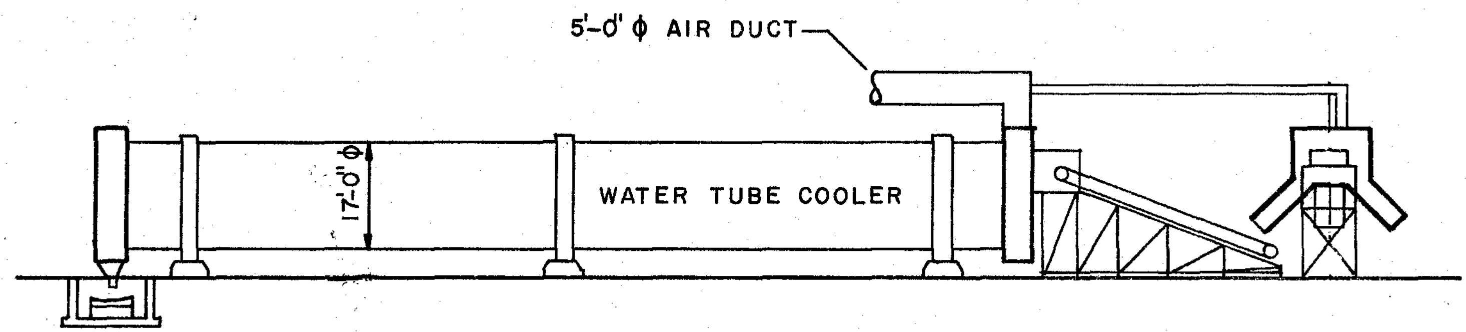
FIGURE 3.1.2-1

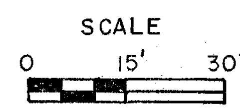


PLAN VIEW

TYPICAL COOLER LAYOUT

FIGURE 3.1.2-2





ELEVATION VIEW

COOLER

FIGURE 3.1.2-3

inoperative. This 72-inch conveyor will be of rigid-frame construction with a dust hood along its entire length of 230 feet.

The lagged drive pulley will be 24 inches in diameter and will also serve as the head pulley. The face of all pulleys on this conveyor will be 76 inches wide. The takeup will be a vertical gravity type and both the tail and takeup pulleys will be 16 inches in diameter.

A 150-horsepower wound rotor electric motor will drive the conveyor. The drive pulley will turn at 55 rpm and belt speed will be 350 feet per minute. A motor starter controller will be used to minimize belt stress during start up. This conveyor is rated at 3,000 tons per hour.

Since the retorted shale will leave the retort at about 400°F, a high temperature service belting similar to Goodyear Plylon Super Thermo-Flo Belting will be used. This belting is designed to withstand temperatures up to 400°F.

3.1.3.2 Main Cooling Supply Conveyor:

The main cooling supply conveyor will deliver the hot retorted shale to the individual cooler feed conveyors. This 230-foot-long conveyor will have rigid-frame construction with a continuous dust hood. A stationary tripper and splitter chute will be installed at each of the first three pairs of coolers. The fourth pair of coolers will be fed through a splitter chute at the discharge end of the conveyor. The belting will be 72-inches wide and will be similar to the high temperature service belting described in Section 3.1.3.1.

The single drive-head pulley will be 30 inches in diameter and will be lagged to improve the coefficient of friction between the belt and pulley. All pulleys on this conveyor will have a face width of 76 inches. The tail, tripper, and takeup pulleys will be 20 inches in diameter and a vertical gravity takeup will be used.

A 200-horsepower wound-rotor electric motor will drive the conveyor. The drive pulley will turn at 45 rpm and belt speed will be 350 feet per minute. A motor starter-controller will be used to minimize starting stresses and to control the steady-state operation of the conveyor. This conveyor is rated at 3,000 tons per hour.

3.1.3.3 Individual Cooler Supply Conveyor:

Each of the eight coolers will be fed by a 24-inch-wide supply-belt conveyor. These conveyors will transport the retorted shale from the splitter chutes to the rotary coolers and will be hooded to minimize dusting. All pulleys will be 12 inches in diameter with a face width of 26 inches and all drive pulleys will be lagged. High-temperature belting, as described in Section 3.1.3.1, will be used. Each conveyor will be 50 feet long and will be equipped with an automatic hydraulic takeup.

A 10-horsepower squirrel-cage electric motor will drive the conveyor. The drive pulley will turn at 145 rpm and belt speed will be 450 feet per minute. Each conveyor is rated at 375 tons per hour.

3.1.3.4 Cooled Shale Collector Conveyors:

Two 48-inch-wide cooled-shale collector-belt conveyors, each 220 feet long, carry the cooled material away from the coolers. These conveyors will be installed in underground conduits to allow vehicular access to the cooler area. The installation will feature rigid-frame construction without dust hoods. The lagged drive-head pulley diameter will be 16 inches, the tail and takeup pulley diameters will be 12 inches, and all pulleys will have a face width of 51 inches. An automatic hydraulic takeup will be installed on each conveyor. The belting for carrying the cooled shale will be similar to Goodyear's Plylon Style B Hot Material Belting which is rated for temperatures up to 200°F.

A 50-horsepower squirrel-cage electric motor will drive each conveyor. The drive pulley rotational speed will be 95 rpm and the belt speed will be 400 feet per minute. Each of these conveyors is rated at 1,500 tons per hour.

3.1.3.5 Cooled Shale Transfer Conveyor:

The cooled shale transfer-belt conveyor will carry the retorted shale from one of the collector conveyors to a transfer point that is common to the other collector conveyor. This conveyor, 48-inches wide and 380-feet long, will be installed in an underground conduit. It will be a rigid-frame conveyor with no dust hood. The lagged drive-head pulley will be 16 inches in diameter. All other pulleys will be 12 inches in diameter and all pulleys will have a face width of 51 inches. An automatic-hydraulic takeup will be installed. The belting will be the same as that described in Section 3.1.3.4.

A 60-horsepower squirrel-cage electric motor will drive each conveyor. Drive pulley speed will be 95 rpm and the belt will travel at 400 feet per minute. This conveyor is rated at 1,500 tons per hour.

3.1.3.6 Cooled Shale Emergency Diversion Conveyor:

A conveyor to transport cooled shale to the surface disposal load-out bin, in case of a short-term emergency in the borehole area, is included in the surface conveying system. This conveyor will be 48 inches wide and 275 feet long. The conveyor will be rigid frame with a dust hood along the section which is above ground level. The lagged drive-head pulley will have a 16-inch diameter and all other pulleys will be 12 inches in diameter. All pulley face widths will be 51 inches. An automatic-hydraulic takeup will be installed. The conveyor belting will be the same as the description in Section 3.1.3.4.

A 60-horsepower squirrel-cage electric motor will drive the conveyor at 500 feet per minute. Drive pulley speed will be 120 rpm. The conveyor is rated at 2,000 tons per hour.

3.1.3.7 Borehole Feed Conveyor:

The borehole feed conveyor is the same as the surface disposal conveyor described in Section 3.1.3.1 with the exception that the lower temperature Goodyear Plylon Style B Hot Material Belting will be used. The conveyor discharge and borehole opening will be enclosed and connected to a baghouse for dust control.

3.2 Transport of Retorted Oil Shale to the Backfill Level:

Based on the scale of mining and retorting which was assumed, retorted oil shale has to be transported from the surface to the backfill level at the rate of 3,000 tons per hour. In order to design a system which will not interfere with productive operations, borehole transport of the material from surface to the backfilling level was considered. Compared to alternate methods, the use of a borehole transport system will result in significant capital and operating cost savings.

A vertical eight-foot-diameter borehole was designed to transport the retorted oil shale from the surface to the backfilling levels. The bottom hopper was designed for mass flow onto the conveyor system. Borehole and hopper design were based on Jenike & Johanson's (16) work on solids flow.

3.2.1 Material Properties:

The physical properties of retorted oil shale that affect gravity flow in a borehole and hopper were investigated. Since fine sized particles (minus eight mesh) significantly affect gravity flow of granular material, the flow characteristics of this size fraction of retorted oil shale were determined. Gradation analysis of the sample used for flow characteristics testing is as shown in Table 3.2.1-1. This prepared sample was scaled to the laboratory test equipment and does not infer that all material passing through the borehole will be this fine. Detailed descriptions of the laboratory procedures and results are found in Appendix E.

TABLE 3.2.1-1

SIZE ANALYSIS OF RETORTED OIL SHALE SAMPLE FOR FLOW TESTS
(Prepared Sample)

Sieve Size	% Passing
8 M	100.0
16 M	79.7
30 M	62.1
50 M	48.7
100 M	39.1
200 M	30.5

1) Flow of solids neglecting the gaseous phase:

The flow function along with the wall frictional and adhesive properties allow the determination of mass flow hopper configurations and minimum outlet sizes for flow without arching.

Test results indicate a minimum outlet diameter for mass flow of 1.3 feet and maximum hopper wall slope angles from the vertical of 19° and 30° for a circular cone and a wedge, respectively, for a hopper made of carbon steel. Details of the results obtained are summarized in Tables 3.2.1-2 and 3.2.1-3.

2) Flow of solids considering the gaseous phase:

The gaseous phase (in this case, air) affects the flow of the solid. As air is entrained with the retorted shale moving down the borehole, and as the solid compacts under increasing solids pressure in the borehole, pore size is reduced and air pressure increases. At the outlets onto the belt feeder, the shale expands and air pressure drops. The amount of air entrained and the air pressure at the exit depend on the surface density, the compressibility, and the permeability of the solid.

(a) Compressibility is measured by placing a sample of solid in a shallow cylinder and compressing it with a loaded piston. The bulk density is determined as a function of the effective head of solid. The function

$$\gamma = f(h)$$

is plotted in Figure 3.2.1-1.

(b) Permeability is measured by placing a sample of solid in a cylinder and determining the airflow rate for a given air pressure drop as a function of the bulk density of the solid. A combination of the results of these tests with the tests of compressibility yields the following constants:

$\sigma_s = \sigma_o = 0.114$ psf

σ_s = solids pressure below which theory does not apply

TABLE 3.2.1-2

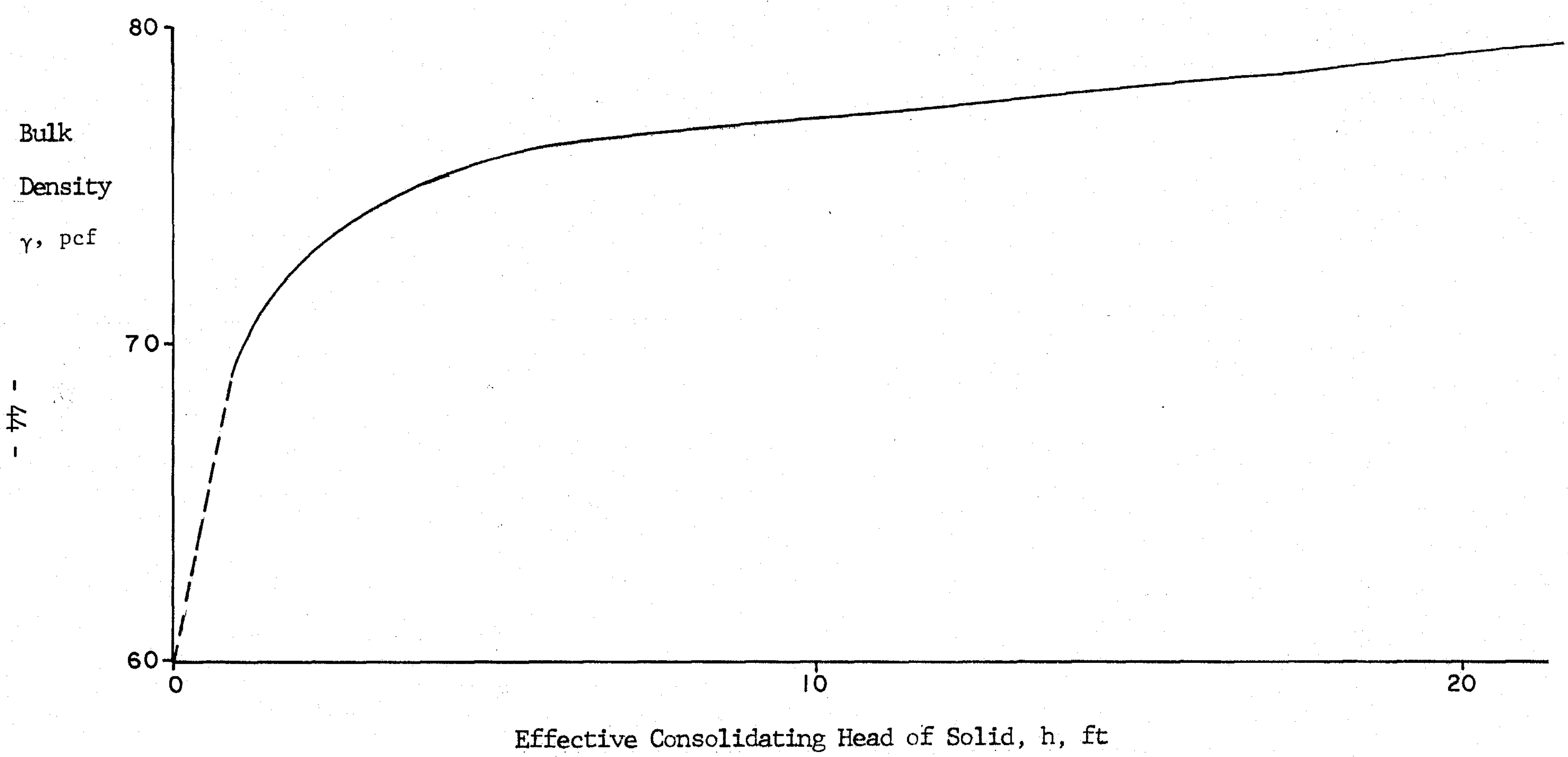
MINIMUM OUTLET DIMENSIONS FOR DEPENDABLE GRAVITY FLOW RETORTED OIL SHALE

Storage Time At Rest	Water Content (% Wet Wt.)	Temperature (°F)	Mass Flow: Circular Outlet Diameter (B_c, Ft.)	Mass Flow: Oval Outlet Width (B_p, Ft.)	Funnel Flow: Rectangular Outlet Width (B_f, Ft.)	Funnel Flow: Critical Rathole Diameter (D_f, Ft.)	Funnel Flow: Effective Consolidating Head (h_e, Ft.)
Continuous flow		100	1.3	0.6	0.6	5'	5'
						11	10
						24	20
						49	40
	0.4						
4 Days		100 to Amb.	1.3	0.6	0.6	6	5
						12	10
						25	20
						50	40

TABLE 3.2.1-3

FRICTION TEST RESULTS FOR VARIOUS WALL MATERIALS AND MAXIMUM RECOMMENDED HOPPER ANGLES (measured from the vertical) FOR MASS FLOW RETORTED OIL SHALE

Effective consolidating head, feet		0.25	0.5	1.0	2.0	4.0	5.0
Width (B_p) of oval outlet, feet		0.25	0.5	1.0	2.0	4.0	5.0
Diameter (B_c) of circular outlet, feet		0.5	1.0	2.0	4.0	8.0	10.0
Wall Material	Angle Degree						
Carbon Steel (Continuous Flow)	ϕ'	31	26	24	23	23	22
	θ_c	8	15	18	19	19	20
	θ_p	18	25	28	29	30	30
Carbon Steel (4 day time test)	ϕ'	31	26	24	23	23	22
	θ_c	8	15	18	19	19	20
	θ_p	18	25	28	29	30	30
304 - 2B finish Stainless Steel (Continuous Flow)	ϕ'	18	18	18	18	18	18
	θ_c	25	25	25	25	25	25
	θ_p	36	36	36	36	36	36
304 - 2B finish Stainless Steel (4 day time test)	ϕ'	18	18	18	18	18	18
	θ_c	25	25	25	25	25	25
	θ_p	36	36	36	36	36	36



BULK DENSITY AS A FUNCTION OF THE EFFECTIVE HEAD OF SOLID

FIGURE 3.2.1-1

$\beta = 0.0289$

$u_o = 0.00437$ feet per second

$\gamma_o = 60.0$ pcf

$n = -7.88$

(σ_o, β, u_o, γ_o , n - experimentally determined constants)

(c) Surface density is defined as the lowest density at which the solid forms a distinct surface. This density was measured at $\gamma_s = 60$ pcf.

(d) Particle density was measured at $\Gamma = 165$ pcf.

3.2.2 Design and Operation of Borehole and Hopper System:

The design and operation of the borehole and hopper system is based on the need to transport retorted oil shale from the surface to the underground disposal levels at a rate of 3,000 tons per hour, and on the material flow characteristics determined in the laboratory.

3.2.2.1 Design Parameters:

Dry retorted oil shale is to flow down a 2,000 foot-deep borehole and onto a belt feeder at a rate of up to 3,000 tons per hour. In order for the flow onto the belt to be uniform and controlled, it is necessary that:

a) The flow of retorted shale in the regions of the hopper above the outlets must be steady. This means that movement of material in the hopper must be mass flow, i.e., all the material must be in motion whenever any of it is withdrawn.

b) The pressure of air in the pores of the retorted shale discharging onto the belt feeder must be close to the ambient air pressure. If pore pressure is too high, shale will flush uncontrollably and flood the belt; if pore pressure is too low, flow will be intermittent with arching followed by flushing and flooding of the belt.

c) The area of the outlets must be sufficiently large to assure unobstructed flow at the specified rate.

d) The chute and skirt design at the outlet must ensure fully live outlets.

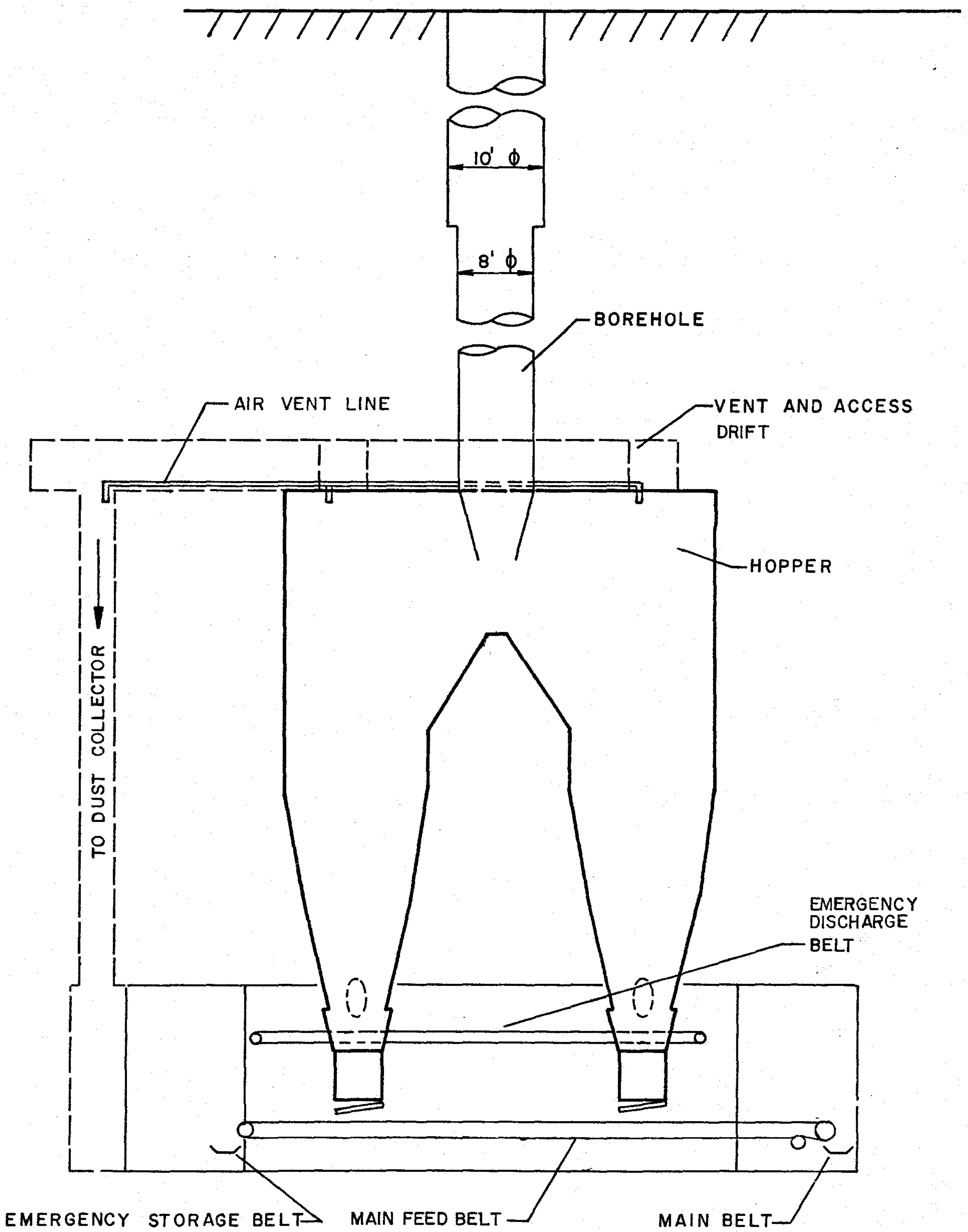
In addition to assuring controlled flow under continuous flow conditions, it is also necessary to provide for start up during the filling of the borehole and for restarting after a stoppage of flow.

Particulate solids tend to flow in a pulsating motion. Coarse (permeable) solids pulsate more, fine (impermeable) solids pulsate less. Pulsation is particularly pronounced in tall, vertical channels of constant cross section, like the borehole under consideration. Due to lack of an appropriate theory, the magnitude of likely pulsation in the borehole cannot be predicted. It is, therefore, necessary to provide a disengaging region between the borehole and the hopper outlets so that borehole pulsations do not affect the feed on the belt. Disengagement is obtained by providing a space in the hopper where the solid can form a free fluctuating surface. Through that surface air can also be introduced or evacuated, as needed, to maintain pore air pressure close to ambient air pressure at the outlets.

The layout of the borehole and hopper is shown in Figure 3.2.2.1-1; the hopper is shown in greater detail in Figures 3.2.2.1-2 and 3.2.2.1-3. The top 100 to 200 feet of the borehole is 10 feet in diameter and the remainder, which discharges into a hopper, is eight feet in diameter. Retorted shale flows down the mass-flow hopper into two three-foot-diameter outlets which feed onto a 72-inch conveyor belt. Since the belt needs to be reversible, the chutes have pivoted skirts to permit each outlet to discharge approximately the same layer of material on the belt, one on top of the other, in either direction of belt movement.

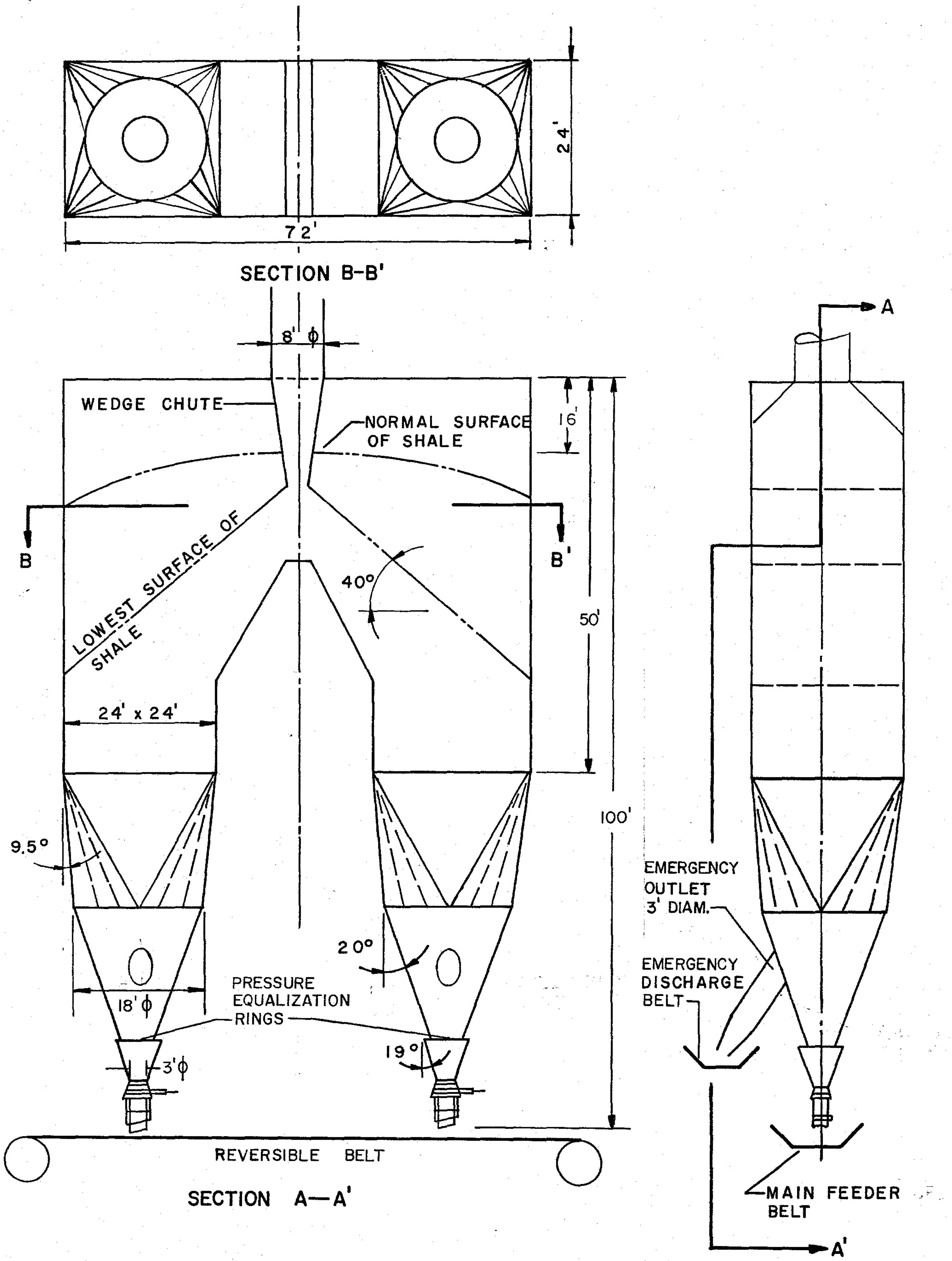
Calculations indicate that an excess of air is likely to be entrained within the retorted shale in the borehole. The excess will be evacuated at the free surface of the hopper by maintaining lower air pressure at the top of the hopper than the pressure in the pores of the retorted shale issuing from the borehole. As the flow pattern of solids expanding from the borehole into the hopper cannot lead to uniform deaeration, the proposed hopper design aims at the prevention of gross nonuniformities. In addition, hopper ring-expansions are utilized. These rings provide a passage through which air pressure equalization can take place across a hopper prior to discharge. This will help prevent flushing of solids through the side of the outlet with an excess of air, while the other side flows sluggishly because of air deficiency.

A plot of pressure versus depth is presented in Figure 3.2.2.1-4. The pressure distribution is almost independent of the solids flow rate for rates between 500 and 4,000 tons per hour. The airflow rate in the borehole is 180 psia x cubic feet per second. The ambient air pressure at a depth of 2,000 feet is 11.6 psia, and the airflow rate at the hopper outlet has to be 146.9 psia x cubic feet per second for ensuring this pressure and obtaining controlled flow. Thus, excess air in the amount of 33.1 psia x cubic feet per second must be evacuated through the surface at



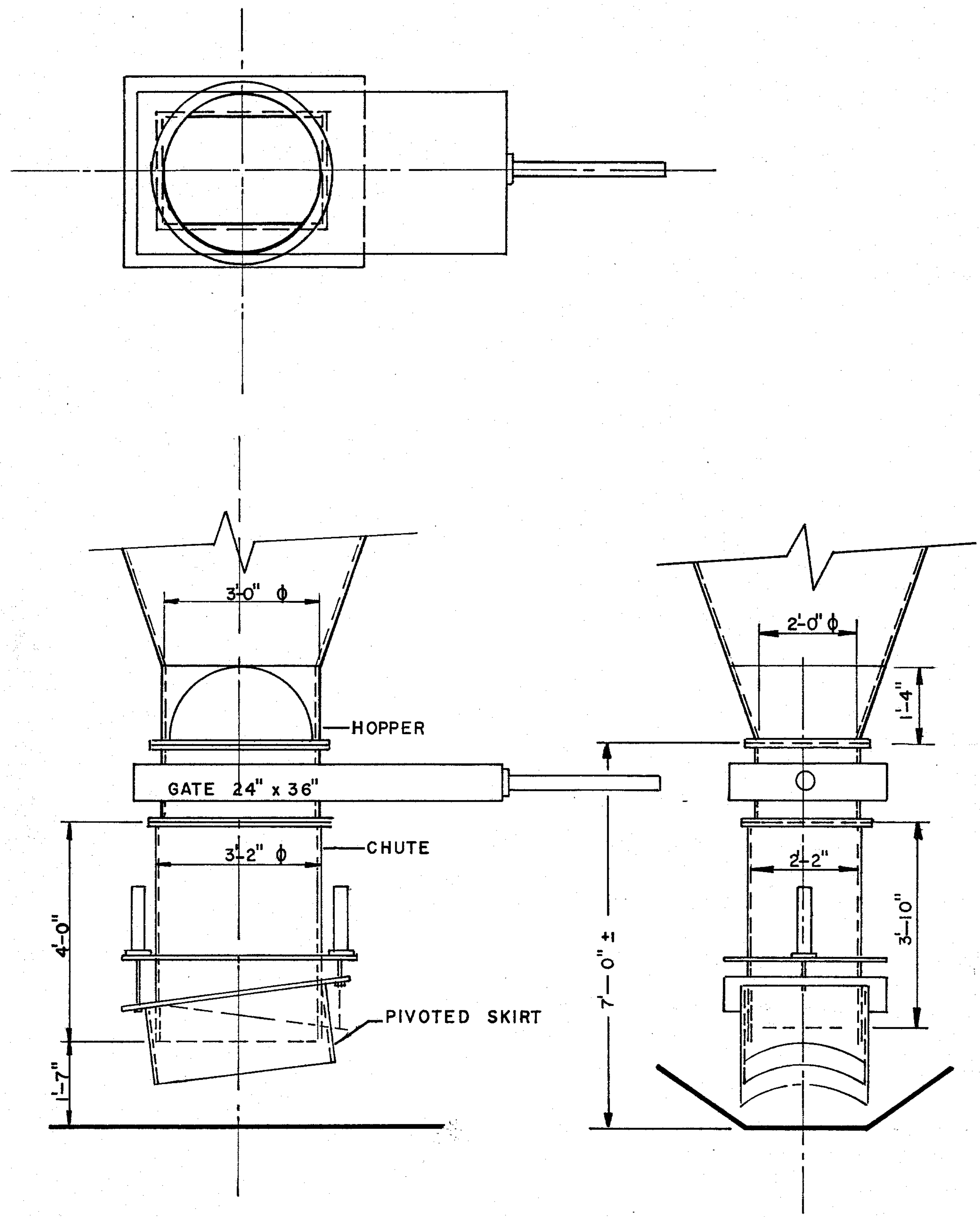
BOREHOLE AND HOPPER LAYOUT

FIGURE 3.2.2.1-1



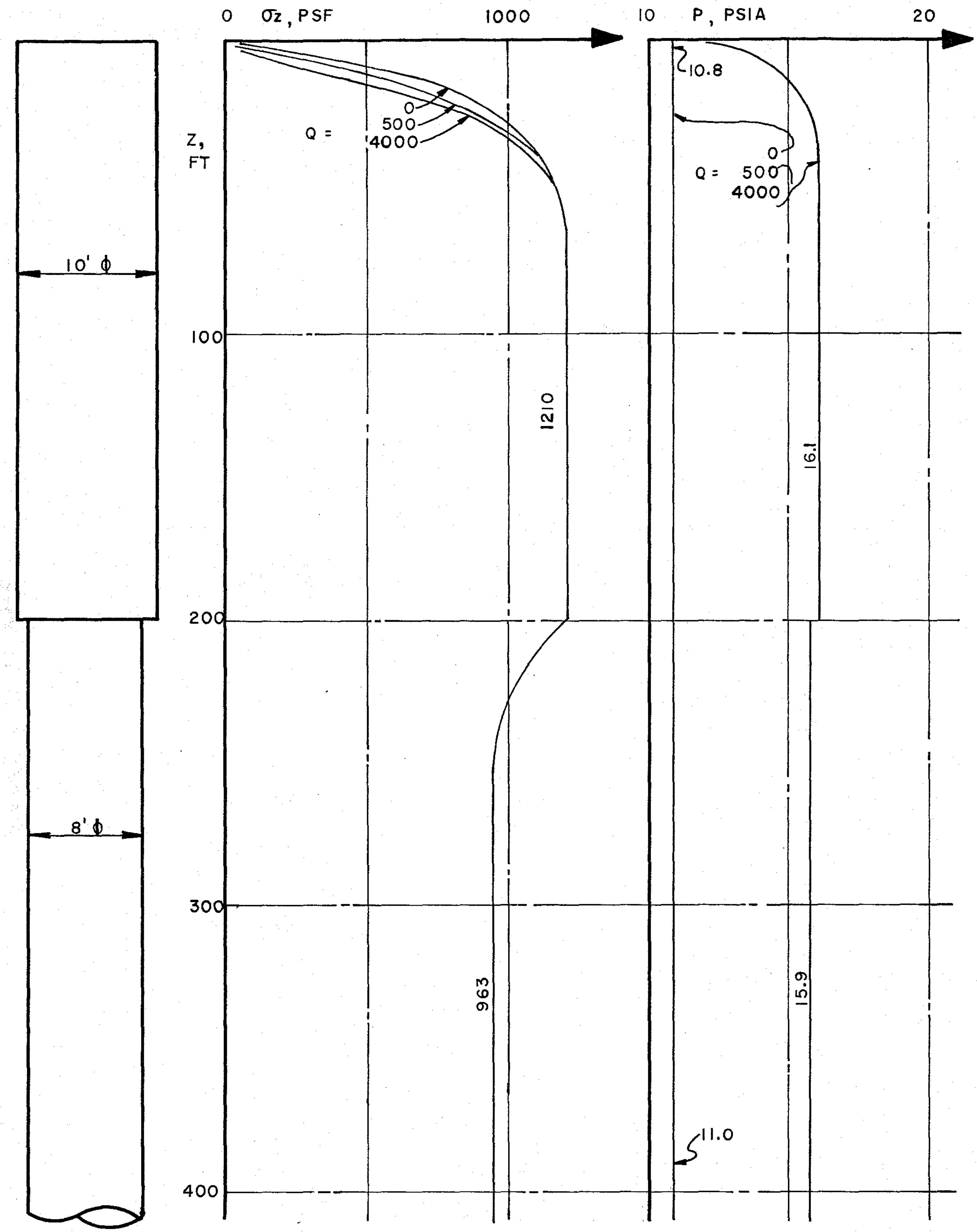
HOPPER DETAIL

FIGURE 3.2.2.1-2



FEED CHUTE DETAIL

FIGURE 3.2.2.1-3



SOLIDS PRESSURE VERSUS DEPTH IN BOREHOLE

FIGURE 3.2.2.1-4

the top of the hopper. A surface area of 477 square feet is necessary to evacuate air at the required rate. However, due to uncertainties involved in the flow at the surface, a surface area of 1,728 square feet has been provided. The large surface area will ensure that air is evacuated at very low velocities, thus minimizing dust escape into the mine. A duct has been provided for air evacuation (Figure 3.2.2.1-1).

Since the gravity flow rate of solids at the hopper outlet is 15,551 tons per hour (five times the required rate) a uniform flow rate is assured. The belt speed required to move 3,000 tons per hour is 425 feet per minute, assuming a surcharge angle of five degrees.

A flow sensor will be installed at the bottom of the borehole and a bin level detector will be installed at the top of the borehole. If the borehole should become plugged, flow of retorted shale to and from the borehole would be stopped. Incoming material would be diverted to the surface disposal system. No material will be drawn from the borehole until the plug is cleared. It is imperative that an extended length of the borehole not be evacuated below the plug point. The potential air blast that would accompany the clearing of the plug could be extremely hazardous to the men and equipment underground. The plug will be cleared with a modified rotary drill using high pressure air and a bit that directs the air radially to dislodge the hung-up material.

3.2.2.2 Operation:

Single-Level Operation: Initial filling of the borehole will proceed at a low charging rate in order to minimize impact loads on the hopper. After the initial 500 tons have been dropped, the belt will be started and the gates opened. The withdrawal will proceed through both outlets at approximately one-fourth of the rate of charge into the borehole. The top of the hopper will be vented at the ambient pressure in the mine. When the operating level of solids in the borehole has been reached, all the rates will be increased to normal operating values.

In order to assure smooth start up, restart after a stoppage and steady-state operation, it will be necessary to monitor and control the following:

a) Level of solids in the borehole: While the borehole will provide a substantial surge capacity, the level of solids will be monitored and the in- and outflow rates controlled to maintain the retorted shale level within specified high-low limits.

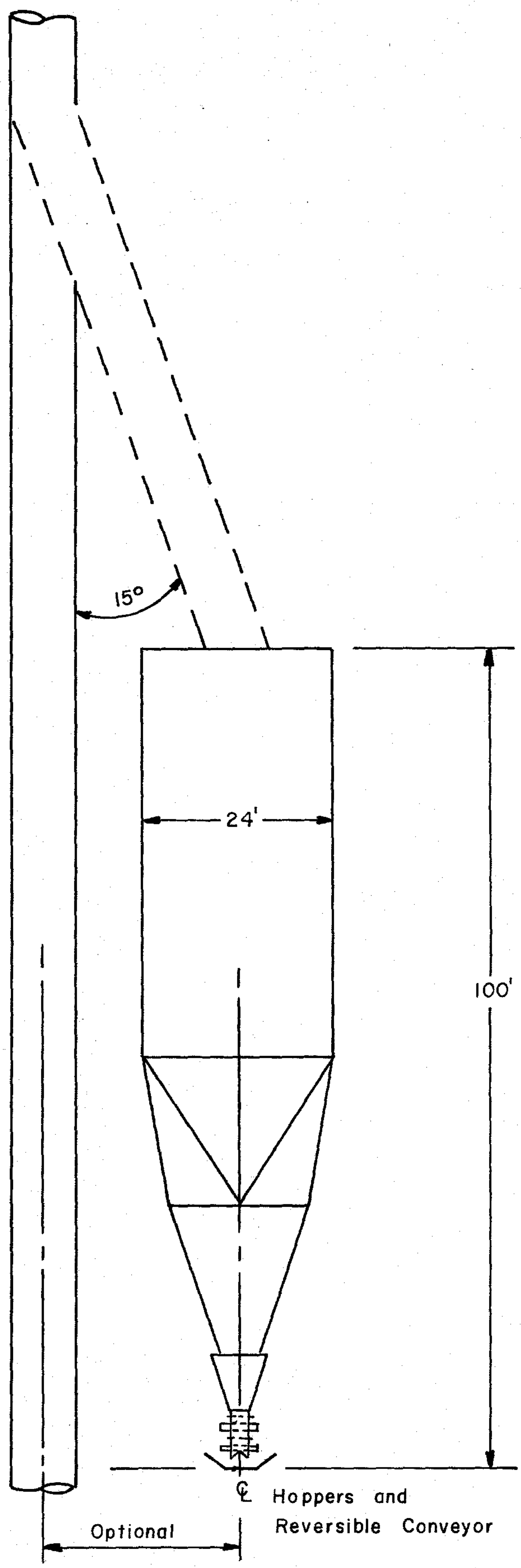
b) Air presssure at the top of the hopper: The amount of air entrained into the borehole depends on the bulk density of the retorted shale at the surface of the borehole. That density cannot be accurately predicted from a one-dimensional analysis and experiment because, in fact, the problem is two-dimensional with, to date, no available solutions. Similarly, the amount of air expelled through the top surface in the hopper is uncertain because of the lack of a two-dimensional solution of the flow from the borehole to the hopper. However, since the rate of air evacuation is directly proportional to the air pressure at the top of the hopper, that pressure can be used effectively to control the uniformity of flow on the belt. Provision will be made to hold air pressure p_2 at any required level between 10 and 12 psia.

c) Feed rate from hoppers:
The gross feed rate may be controlled by varying the height of the chutes above the belt. Fine adjustment will be best achieved with a variable speed drive. The slide gates at the hopper outlets will not be used for a rate control because, in a partly open position, they would prevent mass flow from developing and would lead to non-uniform flow with likely flushing.

In order to provide for emptying the borehole in the event of a breakdown in the main feeder belt, a side chute has been provided (Figure 3.2.2.1-2). Retorted shale will be discharged from this chute onto a secondary belt, and conveyed to an emergency surge area. If retorted shale is not withdrawn continuously from the borehole during an emergency, air pressure in the borehole will be monitored. If the air pressure drops below 13 psi, air at that pressure will be supplied to the borehole. This will permit rapid return to the normal rate of borehole flow when the emergency ends.

Multilevel Operations: Backfilling will start at the lowest level of the mine and, as that level is filled, a new hopper will be constructed at a higher level. In order to maintain continuity of the backfilling when operations are shifted from one level to the next, the construction of the new hopper should be completed prior to completion of backfilling at the lower level.

The completed hopper is built at the next level to be backfilled, offset from the borehole as shown in Figure 3.2.2.2-1. Prior to commencing backfilling operations at the higher level, the top of the hopper would be connected to the borehole by means of a steel-lined eight-foot-diameter shaft inclined at an angle of less than 15° to the vertical.



HOPPER POSITION FOR UPPER LEVEL OF MULTILEVEL BACKFILLING OPERATION

FIGURE 3.2.2.2-1

3.3 Conveyor Transport:

All underground conveyor belts, except the main feeder belt, will be supported on wire rope suspended from the mine roof. Pairs of chain hangers will be bolted to the roof with roof bolts installed on 10-foot centers. Rope tensioning anchors will be bolted to the roof every 300 feet along all belt lines to maintain tension in the wire rope and minimize sag in the line conveyor. Each mainline and panel conveyor will be equipped with a fire detecting and alarm system along its entire length. Every conveyor will also have an automatic water deluge spray system at the belt drive and head pulley area.

All transporting belts will be 72 inches wide and will be carried on heavy duty, 35° offset-roll troughing idlers mounted on five-foot centers. Return idlers will be heavy duty, six-inch-diameter rollers suspended from the wire rope by drop brackets on 10-foot centers.

Belting will carry a minimum tension rating of 420 pounds per inch width. Special compounds capable of handling material at temperatures up to 200°F will be used in the belt covers. All main-line and main-panel supply belts will have vulcanized splices, while piggyback and chamber or stope belts will be joined with mechanical splices. The typical conveyor layout is shown in Figure 3.3-1. All conveying drive motors will be rated at 4,160 volts for extended life and smaller physical size.

3.3.1 Main Feeder Belt:

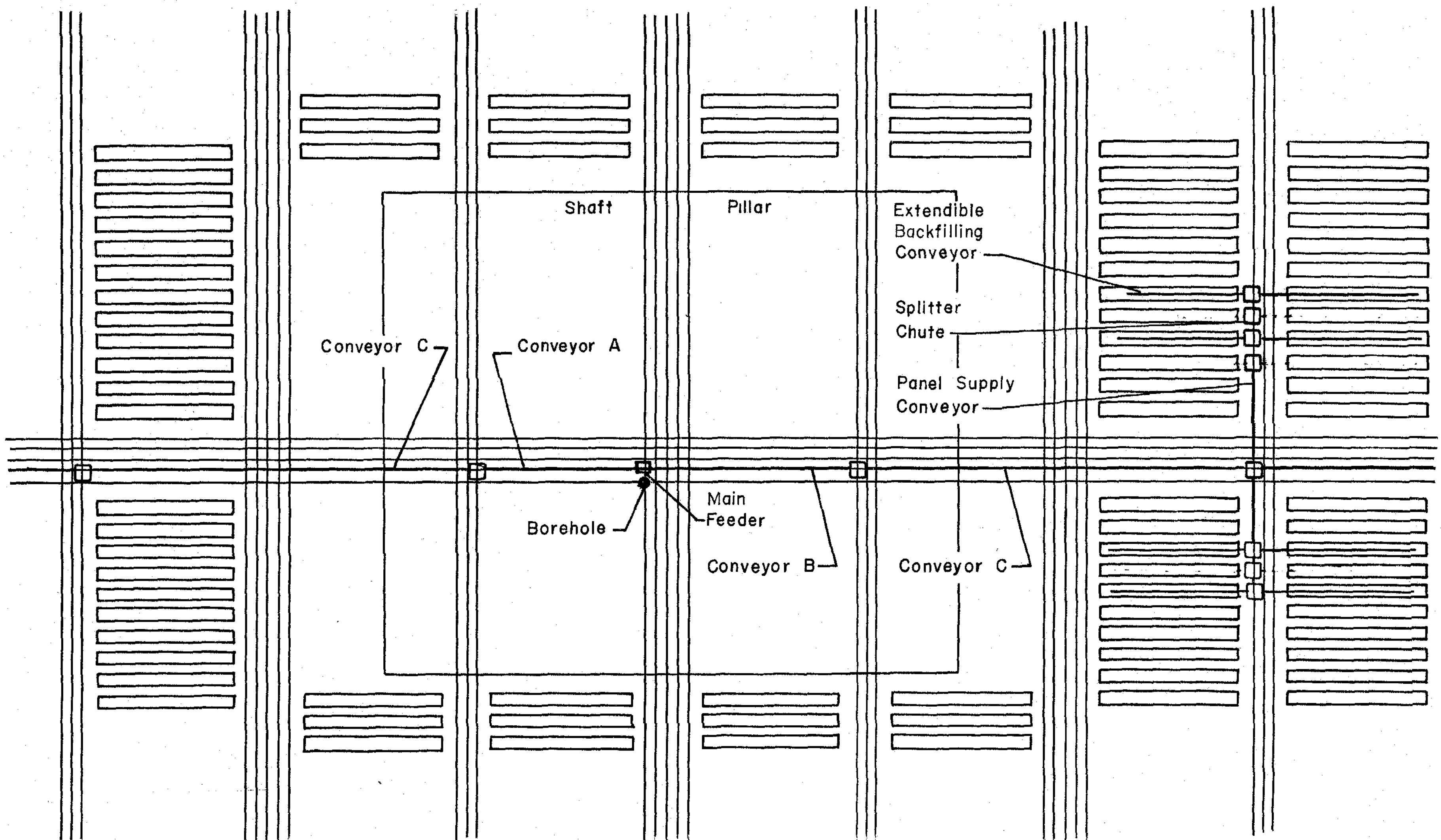
Material will be discharged from the borehole onto a reversible feeder belt. This conveyor will be constructed on a rigid frame as opposed to wire rope suspension used elsewhere in the system. Under normal operating conditions, the main feeder belt will discharge onto the first of the main transport belts. In the event of a failure of the main transporting system, mass flow in the borehole will be maintained by reversing the direction of the feeder belt to discharge material into an emergency surge area.

The main feeder belt will be 72 inches wide and will be carried on 35° equal-roll troughing idlers. Carrying idlers will be mounted on five-foot centers while return idlers will be spaced on 10-foot centers.

The head pulley will be 30 inches in diameter with a 78-inch-wide face and will be covered by a one-half-inch-thick rubber lagging grooved in a herringbone pattern. The tail pulley will be a 20-inch diameter pulley with a 78-inch-wide face.

The head pulley will also serve as the drive pulley and will be coupled to the motor through a speed reducer with a final output of 65 rpm. The power unit will be a 100-horsepower squirrel-cage motor monitored by a solid-state motor starter-controller.

If the main feeder belt should become nonoperational for any length of time, the borehole will be evacuated through the emergency outlets and the material will be discharged onto a back-up conveyor. This conveyor will transport material to the emergency surge area at a rate just great enough to maintain flow in the borehole.



GENERAL UNDERGROUND CONVEYOR LAYOUT

FIGURE 3.3-1

The back-up conveyor will be 36 inches wide and will have a capacity of 1,000 tons per hour. The conveyor will be supported by wire rope suspended from the roof. Carrying idlers will be 35° troughing idlers spaced on five-foot centers and return idlers will be mounted on 10-foot centers.

The drive-head pulley will be 10 inches in diameter with a 40-inch-wide face and will be lagged with one-half-inch-thick grooved rubber. The tail pulley will be an eight-inch-diameter wing-type pulley with a 40-inch-wide face.

The power unit will consist of a 15-horsepower squirrel-cage motor coupled to the head pulley through a speed reducer producing a final drive speed of 153 rpm. The takeup will be of the manual, screw-type, mounted on the tail pulley shaft.

3.3.2 Main Transport Belts:

Mining and backfilling will proceed from the borehole in two directions simultaneously. Material will be conveyed from the crossbelt at the borehole to the nearest panel on either side on a single roof-hung conveyor. (Conveyor A, Figure 3.3-1). The head pulley will be 42 inches in diameter with a 78-inch-wide face and will be supported by a steel A-frame attached to the roof. The tail pulley will be a 24-inch-diameter by 78-inch-wide face wing pulley mounted as part of the loading hopper.

The drive unit will be a single-motor tandem drive installed behind the head discharge pulley. The drive pulleys will be 42 inches in diameter with 78-inch-wide faces and will be covered by a vulcanized, one-half-inch-thick rubber lagging with herringbone grooving. The power unit will be a 250-horsepower wound rotor electric motor and a reducer producing a final drive rate of 45 rpm. A motor starter-controller will be used to minimize belt stress during start up as well as monitoring motor performance during steady-state operation.

Belt tension will be maintained by a hydraulic take up unit located behind the drive unit. A 24-inch-diameter pulley mounted on a rolling take up carriage will be positioned automatically for proper tension by a six-inch-diameter hydraulic cylinder. The take up carriage will have a travel capacity of 20 feet.

The longer of the two initial main belts (Conveyor B, Figure 3.3-1) will be made up of the same components specified for Conveyor A. The only exception will be the use of a 300-horsepower wound rotor electric motor that is required due to the greater length of the conveyor.

As the mine expands, new panels will be mined and ultimately backfilled. As each panel is filled, the main supply conveyor system will be extended to intersect the new panel belts. Each extension will be accomplished through the addition of 2,050 feet of main line conveyor for the chamber and pillar system and 1,850 feet of belting for the sublevel stoping system (Conveyor C, Figure 3.3-1). This 72-inch belt line will be mounted on wire rope suspended from the mine back on chain hangers. The hangers will be secured by roof bolts installed on 10-foot centers. Rope tensioning anchors will be mounted to the roof every 300 feet.

These main line panel-to-panel belts will have the same general configuration in both mining systems. Head pulleys will be roof-mounted on steel A-frames and tail pulleys will be the self-cleaning wing-type mounted in tail frame loading hoppers. All pulleys will have a 78-inch-wide face.

Drive units for both mining systems will be powered by two electric motors. The tandem drive pulleys will be lagged with one-half-inch-thick vulcanized rubber having a herringbone groove pattern. Motor starter-controllers will be used to minimize belt stress during start up and to monitor steady-state operation.

Belt tension will be maintained by take up units consisting of a pulley mounted on a rolling carriage positioned automatically by a six-inch-diameter hydraulic cylinder.

Individual pulley and power unit specifications for each of the belt lines are listed in Table 3.3.2-1. A typical drive unit, take up, and transfer point are shown in Figure 3.3.2-1.

3.3.3 Panel Transport:

Material will be carried into the working panels by a combination of conveyor systems. A roof-hung conveyor will deliver the spent shale to the current backfilling area in the panel. A series of roof-hung, piggyback conveyors will then carry the material to the individual chambers.

The main panel supply belt will be a permanent installation for the life of the panel (Figure 3.3-1). It will receive material from the main line belt, carry it into the panel, and deliver it to the vicinity of the chambers being currently backfilled.

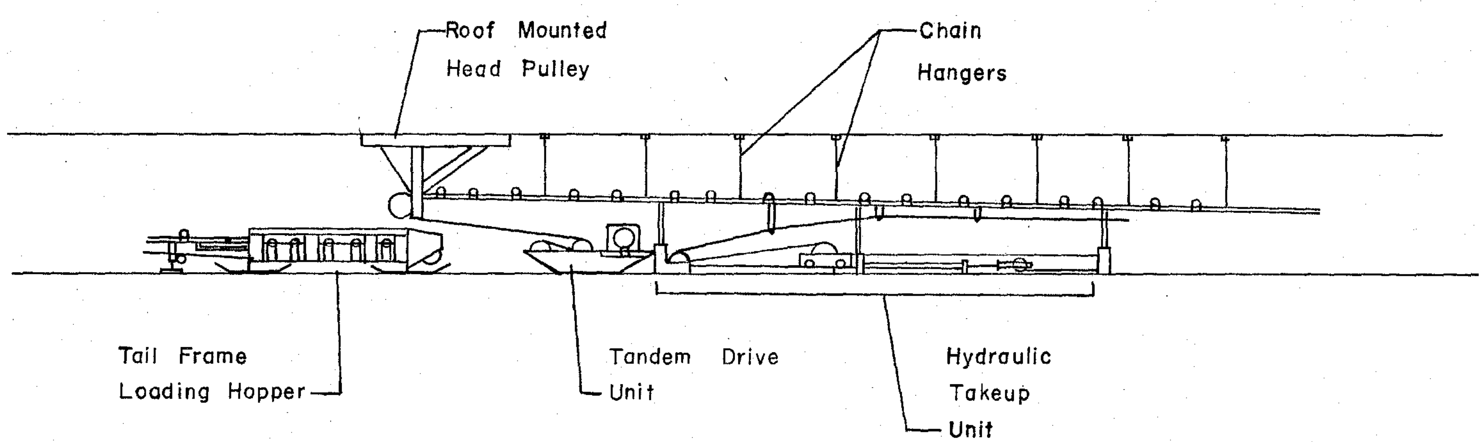
The main panel supply belt will discharge onto the first of a string of piggyback conveyors near the backfilling site (Figure 3.3.3-1). The length of each piggyback conveyor will be equal to the center-to-center distance between chamber entries. A transfer chute at the discharge end will direct all or part of the material either into the chambers or onto the next piggyback conveyor for transport further down the line.

Installation crews working in advance of the backfilling operations will install additional piggyback conveyors ahead of the working belts. This will allow operations to shift immediately to the next chamber down the line as the chambers become filled. As the backfilling process continues and the number of piggyback conveyors in use grows, the main panel supply belt will be extended to replace the piggyback conveyors beyond those chambers already backfilled. The roof attachment angles and chain hangers used for the indivdual piggybacks will be utilized for supporting the main panel belt to keep extension downtime to a minimum. During this time, material will be diverted to the other working panels until the new extension is completed.

TABLE 3.3.2-1

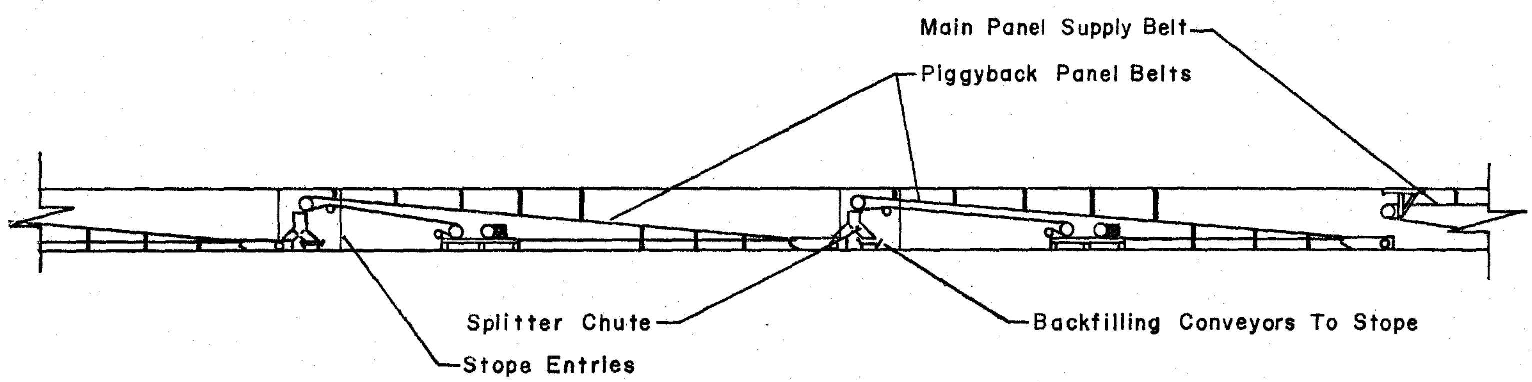
MAIN LINE PANEL-TO-PANEL BELT SPECIFICATIONS

	Chamber & Pillar System	Sublevel Stoping System
Conveyor Length	2,050 ft.	1,850 ft.
Head Pulley Diameter	48 in.	42 in.
Tail Pulley Diameter	30 in.	24 in.
Drive Pulley Diameter	48 in.	42 in.
Power Unit	2 motors @ 250 hp each	2 motors @ 200 hp each
Final Drive Rate	39 rpm	45 rpm
Take-up Pulley Diameter	30 in.	24 in.



TYPICAL CONVEYOR DRIVE, TAKEUP, AND TRANSFER POINT

FIGURE 3.3.2-1



PANEL AND PIGGYBACK CONVEYORS

FIGURE 3.3.3-1

3.3.3.1 Main Panel Supply Belt:

In the chamber and pillar mining system, each panel has a maximum length of 7,800 feet. The maximum length in the sublevel stoping system is 4,050 feet. Rather than cover these distances with a single belt requiring massive drives and pulleys, main panel belts will be installed in approximately 2,050 feet long units in the chamber and pillar system and 1,850 feet long units in the stoping system. Since these lengths are the same as the distances between panel entries on the main belt line, they will be the maximum single conveyor lengths in each system. Such standardization of conveyor sizes will allow the use of interchangeable equipment between belt lines. All drives, pulleys, and other conveyor components for the main panel belts will be identical to those specified in Section 3.2.4 in the description of the main line panel-to-panel conveyors.

3.3.3.2 Piggyback Panel Belts:

Each piggyback conveyor will have a center-to-center pulley spacing equal to the center-to-center distance between chamber or stope entries. The head pulley, which will be mounted on a roof-hung steel A-frame, will be 30 inches in diameter and will have a 78-inch-wide face. The tail pulley will be a 20-inch-diameter by 78-inch-wide face wing pulley mounted in a tail frame loading hopper.

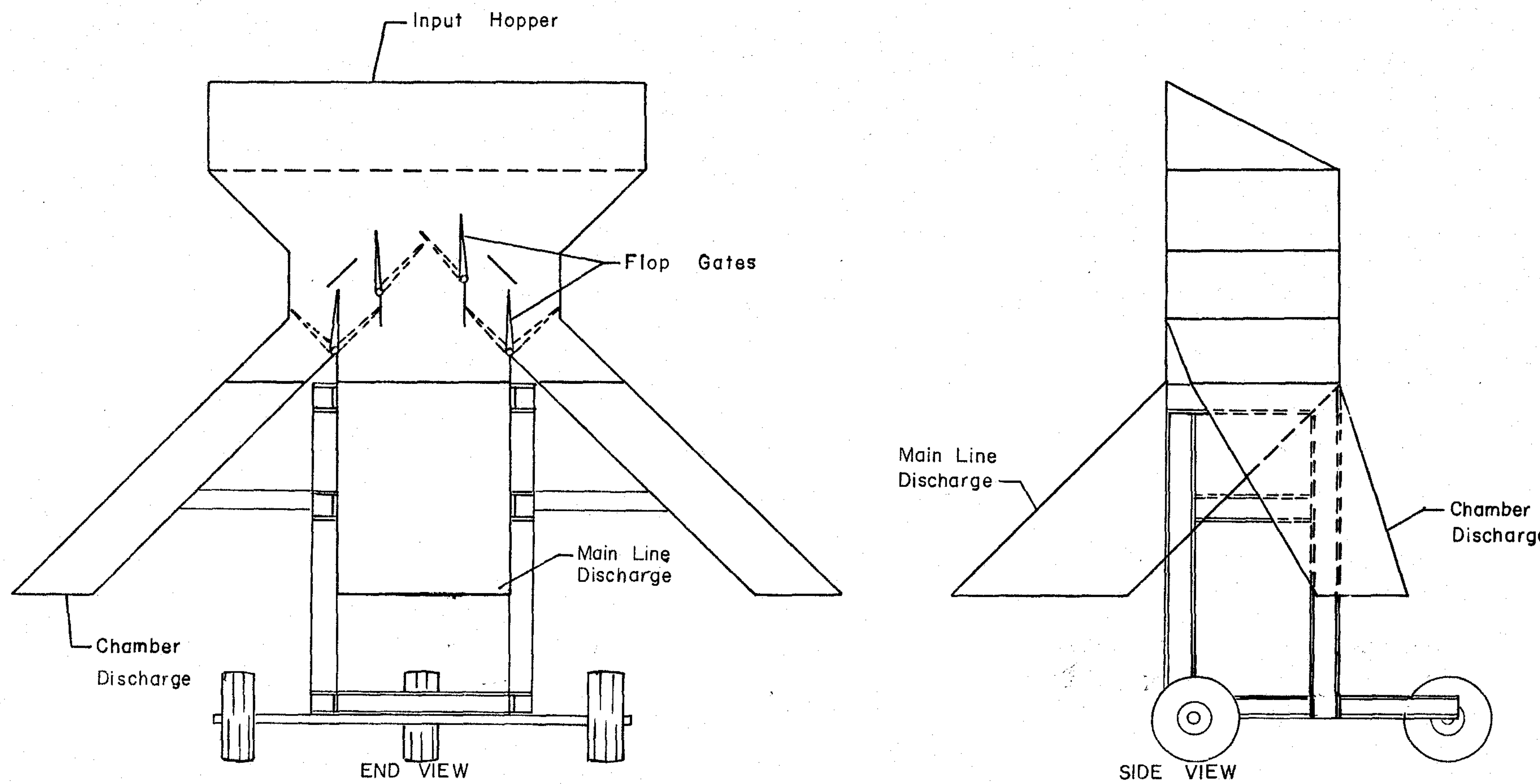
The tandem drive unit will employ two 30-inch-diameter by 78-inch-wide face pulleys covered by one-half-inch-thick vulcanized lagging with a herringbone groove pattern. The power unit will be a single 100-horsepower squirrel-cage motor coupled with a speed reducer yielding a final output of 65 rpm. The entire drive unit will be skid mounted for ease of relocation.

The intermediate section of the piggyback belt will be suspended from roof bolts and chain hangers spaced in such a way that the same bolts and hangers can be used when the main panel supply belt is extended.

For ease of relocation, each piggyback conveyor will discharge into a splitter chute mounted on a carrier (Figure 3.3.3.2-1). Hydraulic cylinders will actuate a series of flop gates controlling the material's movement through the chute. All of the material, or any fraction of it, can be diverted to any of the three loading hoppers positioned below the splitter by varying the positions of the flop gates. Two of the hoppers will feed conveyor belts running at right angles to the piggyback belt and entering the backfilling chambers. The third hopper will feed another piggyback conveyor carrying the material farther down the line (Figure 3.3.3-1). A series of piggyback belts and splitter chutes, along with their associated chamber backfilling belts, provides extended flexibility to the backfilling process in that chambers can be brought "on line" or shut down immediately through control of the splitter chutes.

3.4 Backfilling:

Chambers or stopes will be backfilled using extensible conveyor belts to carry the retorted shale from the main panel supply belt to the



SPLITTER CHUTE

FIGURE 3.3.3.2-1

point of deposition. Mechanical compaction will be a part of the chamber and pillar mining system, but not the sublevel stoping system because of the high unsupported ribs in the open stopes. The fill may settle beneath the conveyor during sublevel stope backfilling. If this occurs, the extensible conveyor will be moved back and the sunken areas will be filled to the desired level.

An alternative system, using a combination of conveyor stowing with pneumatic final topfilling, is also presented. In all cases, backfilling will be done from an access level at the top of the chambers or stopes.

3.4.1 Conveyor Backfilling:

Extensible belt conveyors will be used to carry the retorted oil shale into the chamber or stope to be backfilled. As backfilling progresses, the conveyor will be advanced into the stope. Final topfilling will be done as the conveyor retreats from the partially filled stope. Final placement to the roof is done with a modified track-type loader.

Each conveyor will be extended from a minimum of 100 feet to a full length of 850 feet. The conveyor will be extended as backfilling progresses, with additional belts being added after each 135 feet of advance. The conveyors have been designed to handle 1,500 tons per hour at peak capacity with 1,000 tons per hour as the expected normal load. Two backfilling systems, conveyors alone and conveyors combined with pneumatic topfill, have been studied.

3.4.1.1 Chamber Backfilling:

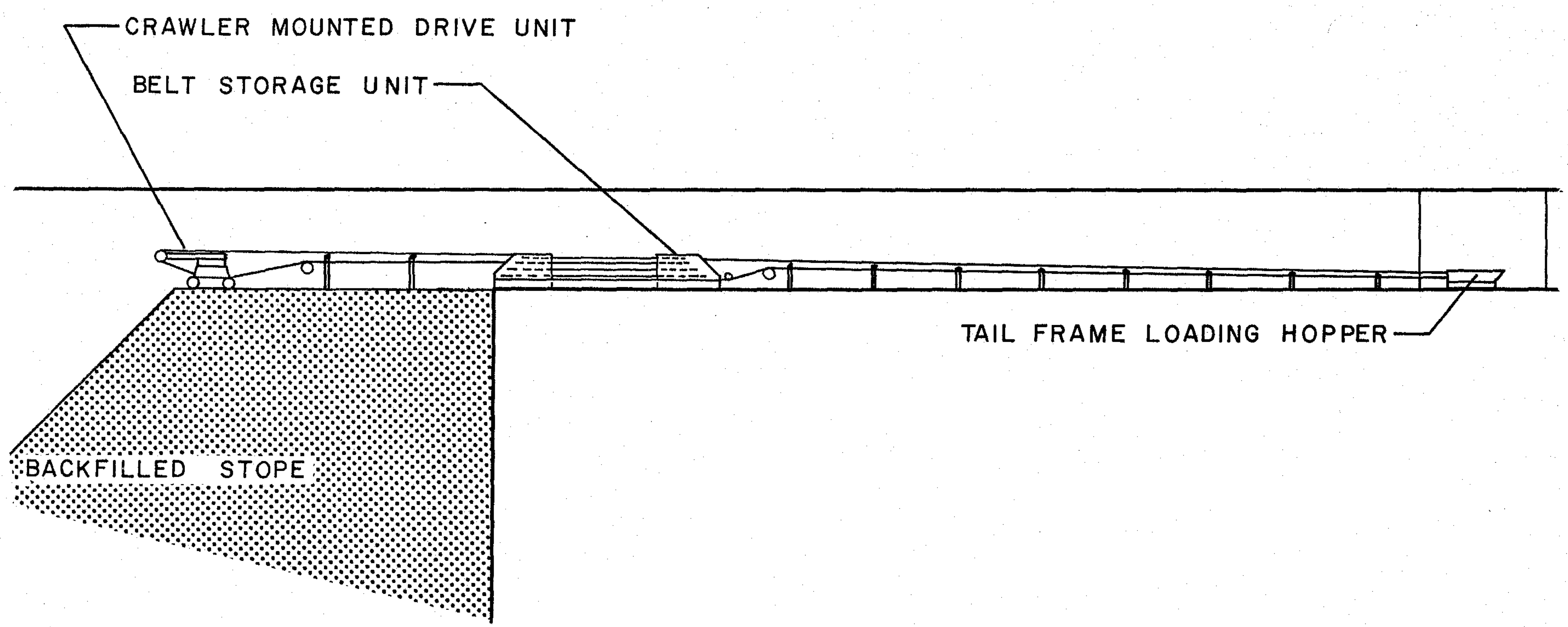
Access for backfilling is provided by a series of drifts at the rear of the chambers. These drifts are located at the top of the chamber so that backfill material can be discharged and placed by gravity.

3.4.1.2 Backfilling Conveyors:

A 36-inch-wide belt will be used for the extensible conveyors. The belt will have a maximum tension rating of 315 pounds per inch width and will be constructed from a heat and oil resistant cover.

Material from the piggyback panel belts will be placed on the extensible belt by the splitter chutes. A tail frame placed to receive spent shale from the chutes will include a 10-inch-diameter wing-type tail pulley, impact idlers, a belt plow for cleaning the return belt, and an integral hopper and skirt boards to provide for even loading of the belt. The tail frame will be skid mounted for ease of relocation.

After loading, the material will be carried into the stope or chamber and discharged over the head pulley. The head pulley and drive unit will be crawler mounted so as to be self-advancing. As the stope fills, the crawler unit will move forward onto the fill, thereby extending the belt line until the entire length of the stope is filled (Figure 3.4.1.2-1). The head pulley will be 16 inches in diameter with a 40-inch-wide face. The drive pulley will be a single motor tandem drive with two



EXTENSIBLE CONVEYOR IN STOPE

FIGURE 3.4.1.2-1

16-inch-diameter neoprene-lagged driving pulleys. The 75-horsepower, squirrel-cage motor will be coupled with a speed reducer to produce a final drive rate of 120 rpm. A remotely located starter unit will be used to control belt sequencing and slippage.

An automatic-hydraulic belt storage unit will be used to hold excess belting in the system and to feed it out as the crawler unit advances. A unit designed to hold at least 270 feet of belting, providing 135 feet of advance, will be used. When the belt in the storage unit has been exhausted, the system will be shut down, another length of belt will be spliced into the line and extended to the storage unit, and operations will resume. Mechanical hinge-pin fasteners will be used on this belt line so as to reduce downtime due to adding or removing belt. The belt storage unit also will act as a take up unit, eliminating the need for any other tensioning devices. In addition, it can be broken down into component pieces for ease of relocation.

The belt itself will be supported by wire rope. Support stands will be floor mounted on 10-foot centers and tensioning anchors will be installed every 300 feet. The rope will carry 35° equal-roll troughing idlers spaced on five-foot centers. Return idlers will be mounted on the rope support stands.

Self-propelled segmented-tamping foot compactors will be used in the chambers to compact the retorted shale to a density of 90 pcf. The compactors will be equipped with a dozer type blade for leveling the retorted shale into uniform lifts prior to compacting. An enclosed pressurized and air conditioned cab with rollover and falling object protection will provide the compactor operator with a safe and comfortable operating environment. It is not known whether backfilling and compacting will be done concurrently in a given chamber. Dust conditions may require that compacting not be done during actual backfilling.

3.4.2 Pneumatic Topfilling:

An alternative to final topfilling by mechanically packing the retorted shale to the roof is to place the final topfill pneumatically. Upon completion of backfilling with a conveyor to the backfill supply level, the conveyor system would be removed from the chamber or stope (in a multilevel sublevel stoping operation only the uppermost stope will be topfilled). A pneumatic stowing system would then be installed to complete the topfilling. The discharge line would extend to the inby end of the chamber or stope and topfilling would be accomplished by retreating toward the access entry as filling progresses. The in-place density for pneumatic topfilling will be about 80 pcf.

The pneumatic system will require a feeder-blower installation at the conveyor transition point. This system will include a surge bin, feeder, 17,000 cfm blower rated at 15 psig and powered by a 1,400-horsepower motor, and a 16-inch-diameter placement pipe and nozzle. A total of three units, each rated at 400 tons per hour, are required for each panel being worked. Water sprays and an exhausting dust collector will be used at the transfer point.

Pneumatic stowing will create a large amount of dust which will be partially alleviated by water injection near the discharge end of the pipeline. Although fugitive dust will be vented to the return air course, men will not be permitted in the stope or in the immediate return air course during stowing operations.

The major advantages and disadvantages of pneumatic top-filling are listed below:

ADVANTAGES:

1. Complete stope filling

DISADVANTAGES:

1. Severe dust conditions
2. Potential for ground-water saturation of fill
3. High energy consumption for stowing activity
4. Large number of units needed
5. High operating costs
6. Greater surface disposal area required compared to mechanical backfilling due to lower in-place fill density

3.5 Operational Considerations:

An underground backfilling operation can be affected by ventilation requirements, hydrologic conditions, environmental monitoring, and the need or the potential for modification of the retorted shale physical properties.

3.5.1 Modification of Physical Properties:

Several methods of modifying the physical properties of retorted oil shale were investigated and the effects of compaction and chemical stabilization additives were studied. The results varied from substantial improvement to no change, and to deterioration of retorted oil shale with respect to its strength and load-carrying capacity.

3.5.1.1 Compaction:

Most of the work relating to the effects of compaction of retorted oil shale from the Paraho process was done by others under contract for the U. S. Bureau of Mines. Table 3.5.1.1-1 shows the results of compaction tests performed on retorted oil shale from the Paraho semi-works retort (14).

As can be seen from Table 3.5.1.1.-1, added moisture has little effect on the density of retorted oil shale after compaction. This is significant in that acceptable densities can be obtained with only enough water added for dust control and cooling. A maximum five to six percent moisture content for stowed shale has been assumed except for retorted shale that is transported or stowed hydraulically.

TABLE 3.5.1.1-1

COMPACTION TESTS - RETORTED OIL SHALE

PARAHO SEMI-WORKS RETORT

Compaction Effort (Ft-Lb/Cu Ft)	Optimum Moisture (%)	Dry Density Compacted At Optimum Moisture (Lb/Cu Ft)	Density Compacted At Air Dry Moisture (Lb/Cu Ft)
-0-	-	70.0	70.0
6,200	23.7	88.0	85.8
12,375	22.0	92.5	89.9
56,250	22.0	98.7	96.4

Compaction will be accomplished by a compactor with a blade for spreading the material. No rollers or additional compaction devices are envisioned.

The permeability of retorted oil shale decreases as the amount of compaction increases. At a density of 90 pcf, the permeability will be less than 10 feet per year (14). As the fill material assumes the lateral load from the pillars, the permeability around the periphery of the stope should decrease.

Minor effects of self-cementation of retorted shale have been reported in connection with compaction testing (14). More investigation of these phenomena is needed.

3.5.1.2 Additives:

The effects of additives on dry retorted shale were investigated by performing a series of laboratory tests. The primary purpose of the tests was to identify beneficial trends and not to obtain statistically significant results. Two different ratios of cementing agents to retorted shale were investigated as to their effect on unconfined compressive strength. Details regarding test material, test equipment, experimental procedure, and results obtained are given in Appendix D.

3.5.1.2.1 Dry Retorted Shale:

A mixture of 50 percent plus 4-mesh material and 50 percent minus 4-mesh material, with a maximum particle size of

0.5 inch, was used to simulate mechanical backfilling conditions. Results obtained from laboratory tests on these materials are as follows:

(1) Increasing moisture content from 15 percent to 25 percent produced significant increases in compressive strength when portland cement is used.

(2) With an eight-day cure, a 5 to 1 retorted shale-cement mixture produced a large increase in compressive strength, whereas a 30 to 1 mixture did not have any effect. A mixture containing equal amounts of cement, lime, and flyash produced noticeable improvements in compressive strength at the same shale-cement ratios.

(3) Distribution of moisture, cementing agent, and fines affect compressive strength. Curing period and curing environment are two other factors that exert a significant effect on compressive strength and deserve further study.

3.5.1.2.2 Other Tests:

The tests mentioned above were conducted to supplement the results obtained under USBM Contract No. J0255004. Under that contract, only dry retorted shale was tested and the major results obtained are as follows (14):

(1) Treatment with five percent hydrated lime promotes cementing, thus, increasing compressive strength and decreasing permeability rate.

(2) Accelerated curing times promote reactions of calcium and magnesium oxides present in Paraho retorted shale, increasing compressive strength from 17 psi to 125 psi with a 28-day curing period at 125°F.

3.5.1.3 Summary:

Compaction of stowed, retorted shale improves its strength and resistance to saturation by ground water, and increases the amount that can be stored underground. Some self-cementing is possible when moisture is added prior to compaction. However, more work is needed to determine the ultimate degree of self-cementation and the retorting conditions that will produce this and other favorable characteristics. Adequate compaction is possible using a compactor with attached blade to spread the material into one-foot lifts as it is compacted.

In contrast, the amounts of moisture and cementing agent that are added to dry retorted shale have a distinct effect on the strength characteristics of the fill material.

3.5.2 Ventilation and Dust Control:

Underground disposal of retorted oil shale will increase mine ventilation requirements by 110,000 cfm for chamber and pillar mining and 50,000 cfm for sublevel stoping. The ventilation system must be able to cope with increased amounts of dust and noxious gases as well as higher humidity and higher ambient temperatures which are associated with a retorted oil shale disposal operation. The severity of these conditions is dependent on the transport and stowing methods used.

The ventilation system will consist of exhaust fans located on the surface, auxiliary fans in the working areas, overcasts, stoppings, regulators, and curtains. Each backfilling site will receive its own split of air. A bleeder system is planned for the backfilled areas to prevent the accumulation of explosive or noxious gases. Figures 3.5.2-1 through 3.5.2-3 show typical mine ventilation systems for the chamber and pillar, and sublevel stoping mining methods used for this study.

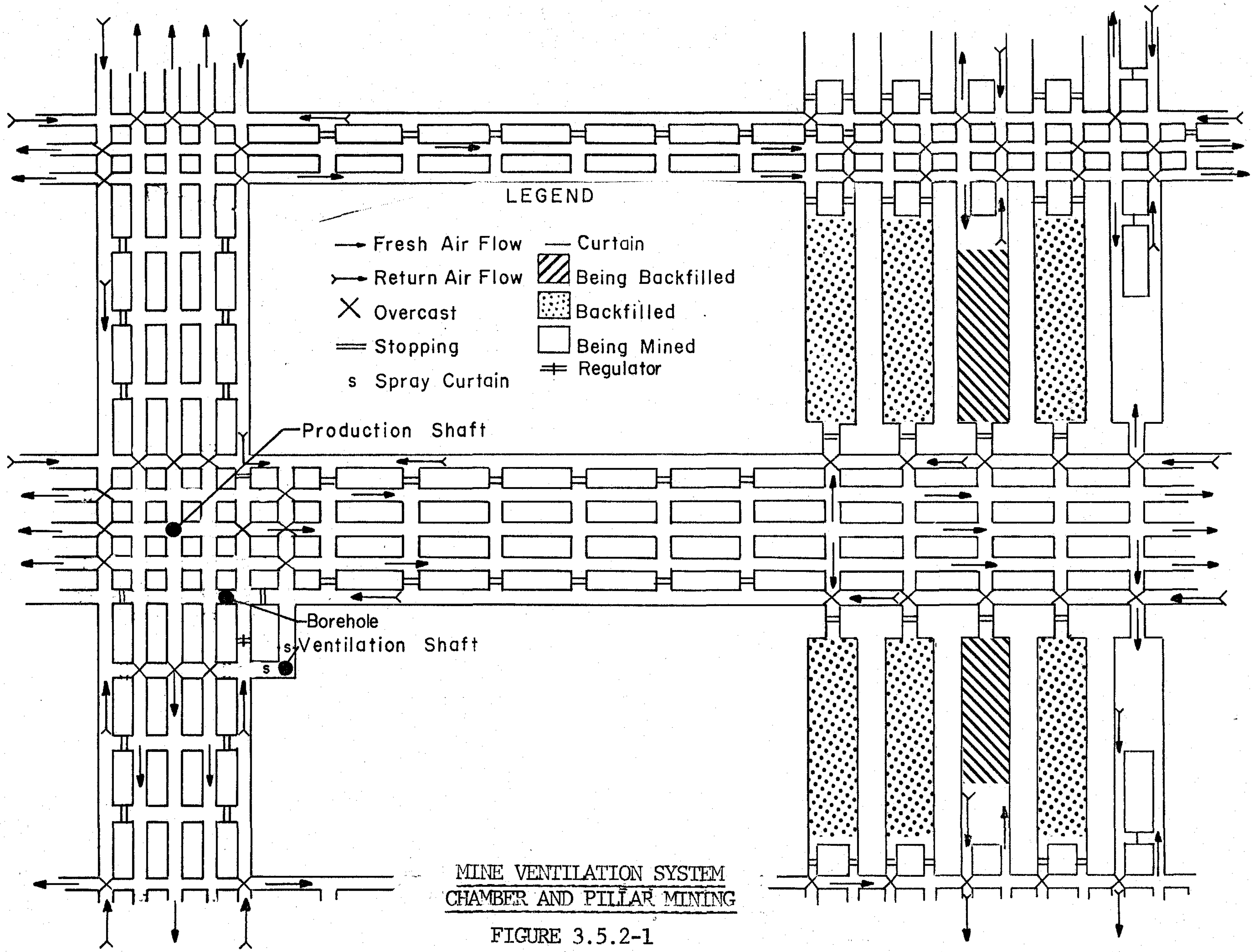
Ventilating air within a chamber will be coursed using brattice lines and vent tubing along with blowing or exhausting auxiliary fans. Pneumatic stowing methods will require as much as 65,000 cfm additional air which must be exhausted through the normal ventilation system. Figure 3.5.2-4 shows a typical chamber ventilation layout. A vent tube for exhausting air from the chamber during backfilling will be hung from the roof during chamber development. The ventilation system for a sublevel stope is simplified because equipment will not be used within the stope for compaction and the return air course will be located at the inby end of the stope. Figures 3.5.2-5 and 3.5.2-6 present a typical sublevel stope ventilation layout.

A high ambient mine temperature, coupled with the introduction of warm retorted shale into the mine, may require some type of air cooling system to provide an acceptable working environment.

3.5.2.1 Noxious Gases:

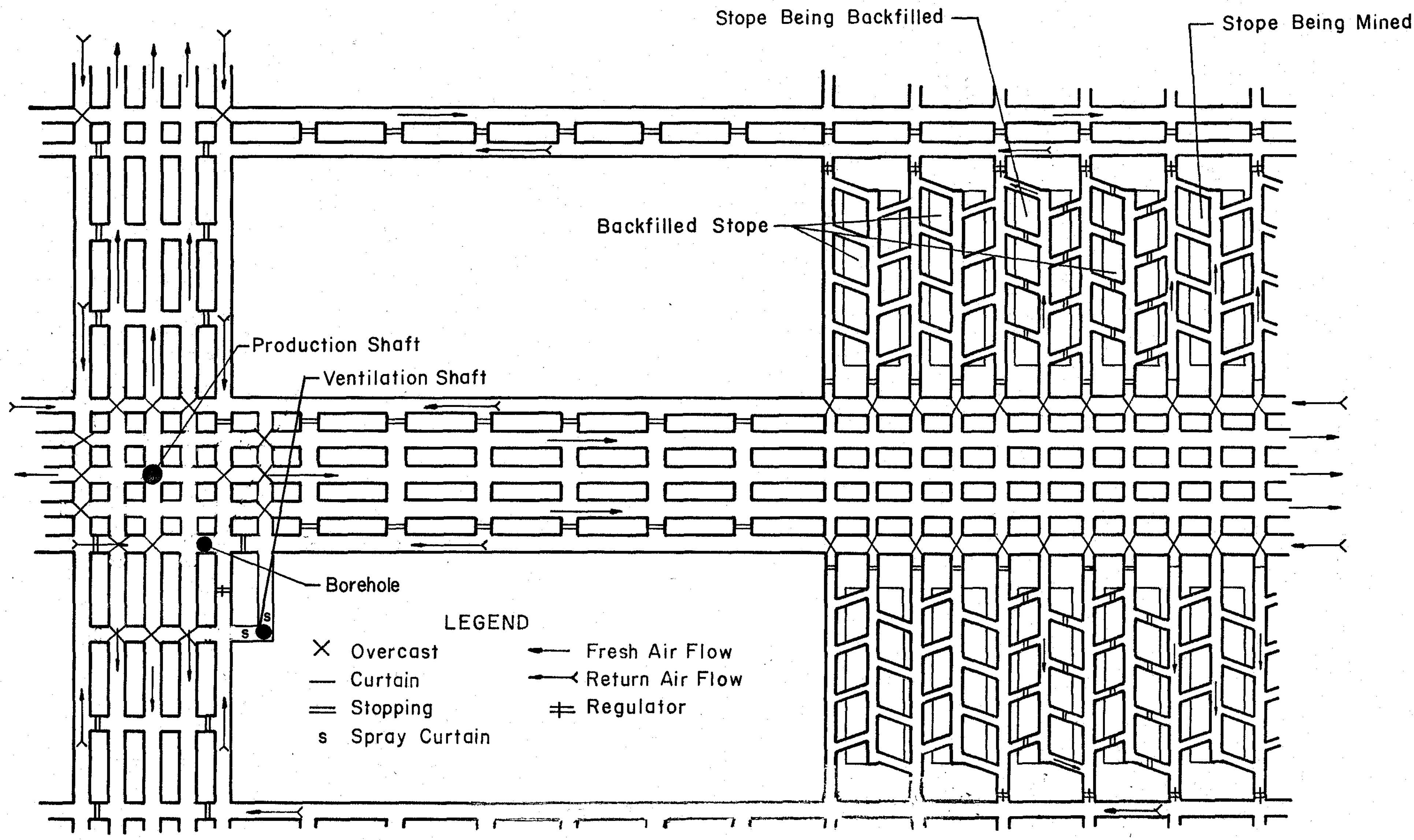
In addition to the potential presence of methane and hydrogen sulfide, other undesirable gases may be associated with an underground backfilling operation using retorted oil shale. Diesel equipment used will contribute varying amounts of CO, NO_x, nonmethane hydrocarbons, and particulate matter to the mine atmosphere. Retorted shale itself may contribute to the problem, especially if the ambient temperature of the stowed material is high.

Spontaneous ignition, with the accompanying combustion products, has been reported where hot (400°F) shale has not been adequately compacted (15). Due to adverse environmental, safety, and productivity effects associated with high temperatures, all retorted shale will



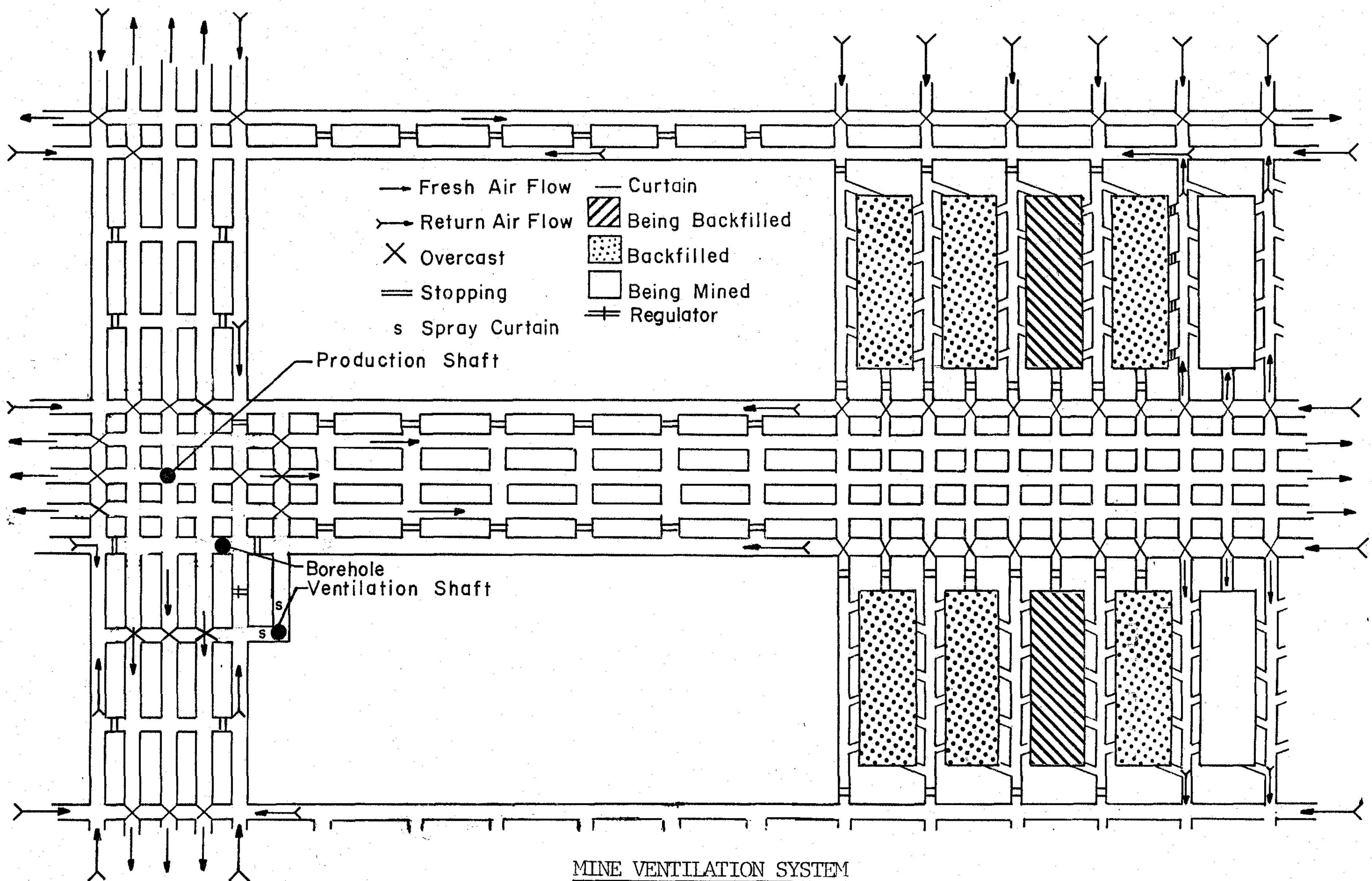
MINE VENTILATION SYSTEM
CHAMBER AND PILLAR MINING

FIGURE 3.5.2-1



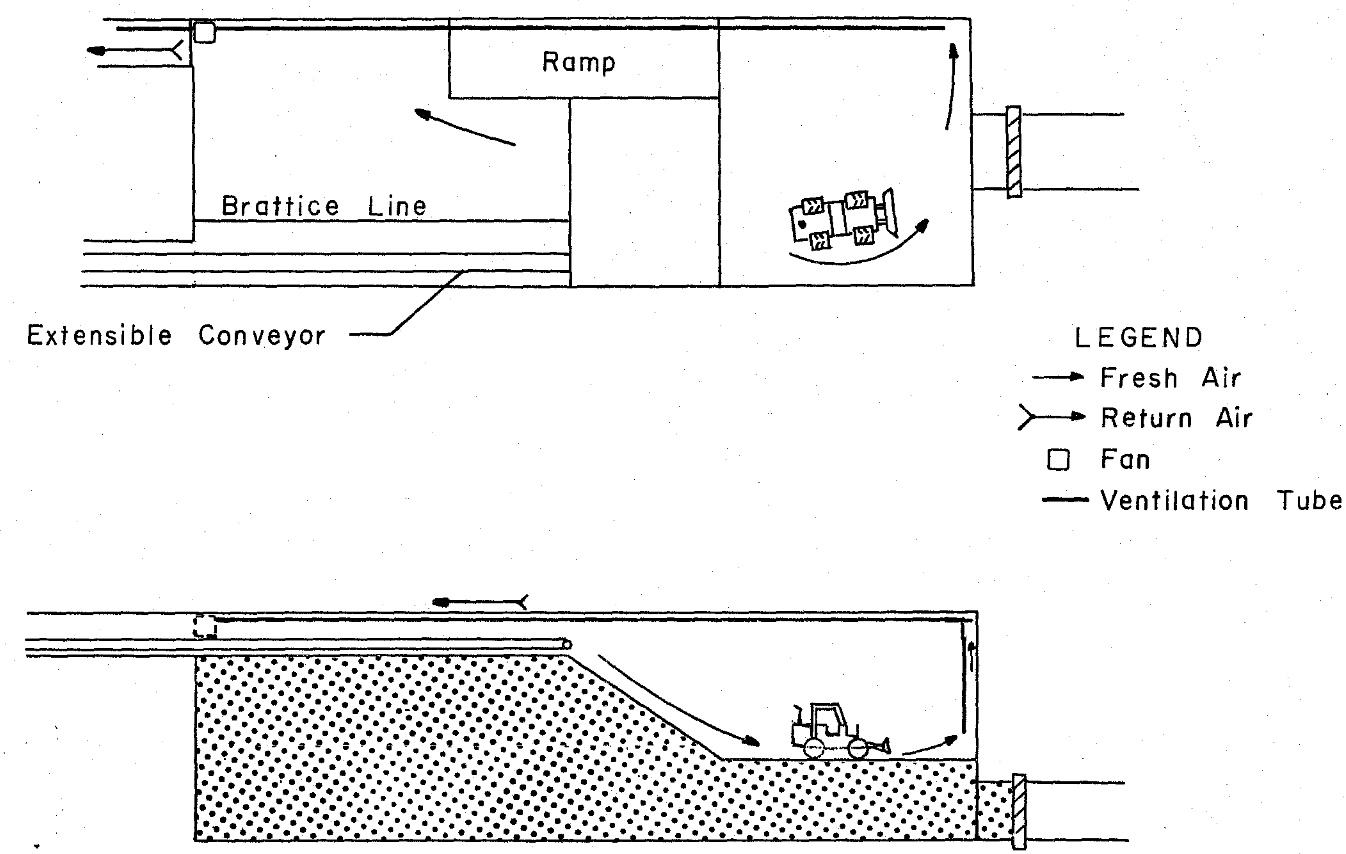
MINE VENTILATION SYSTEM
SUBLEVEL STOPING - BACKFILL LEVEL

FIGURE 3.5.2-2



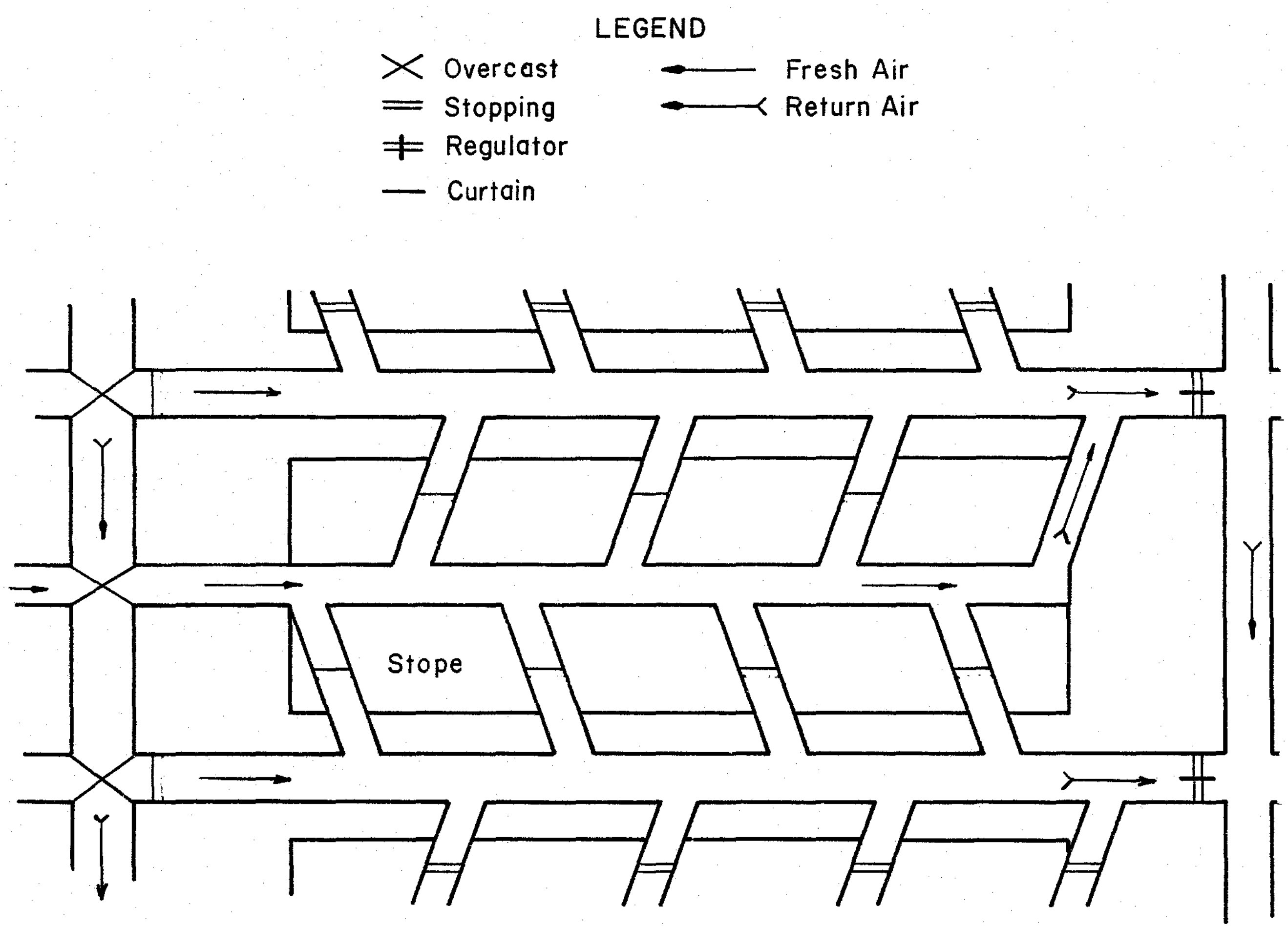
MINE VENTILATION SYSTEM
SUBLEVEL STOPING - PRODUCTION LEVEL

FIGURE 3.5.2-3



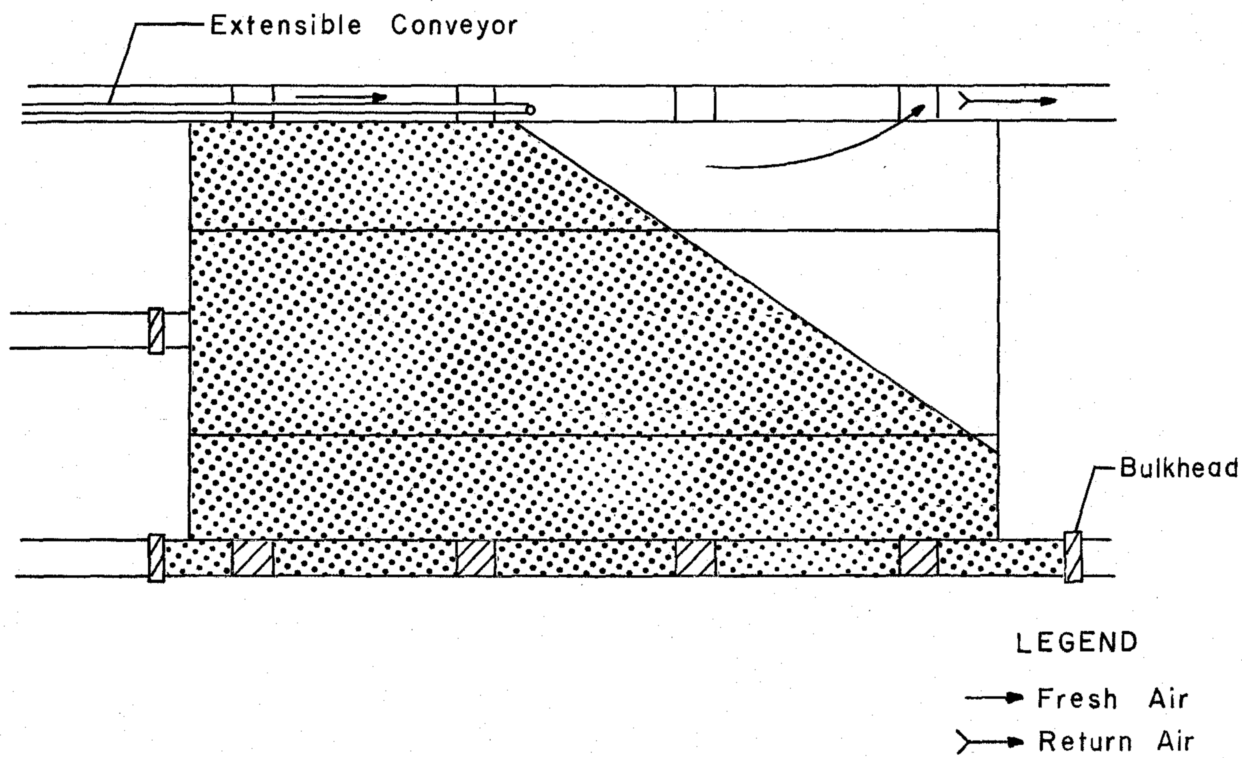
CHAMBER VENTILATION
MECHANICAL STOWING

FIGURE 3.5.2-4



SUBLEVEL STOPE VENTILATION
BACKFILLING LEVEL

FIGURE 3.5.2-5



SUBLEVEL STOPE VENTILATION
MECHANICAL STOWING

FIGURE 3.5.2-6

be cooled sufficiently, prior to stowage, to prevent spontaneous ignition.

A monitoring program, which is discussed in Section 3.5.4, has been developed to determine the presence and amount of noxious gases in a stope during and after backfilling. Specific gases to be monitored will depend on the stowing method used, but carbon monoxide, hydrogen sulfide, and methane will be monitored in all cases. The NO_x group may be monitored where mobile equipment is used. The polycyclic aromatic hydrocarbon group is another type of material that may be monitored. Many divergent opinions and findings have been presented on the occurrence of these materials in retorted oil shale and their carcinogenic hazard potential (6, 7, 20, 21). Information specifically relating to Paraho retorted shale's carcinogenic potential is not available. In all likelihood, some type of monitoring program will be required, at least until the extent of the hazard (if any) is determined. Personal hygiene will be stressed to eliminate potential hazards resulting from extended contact time with the material (6, 7).

3.5.2.2 Dust:

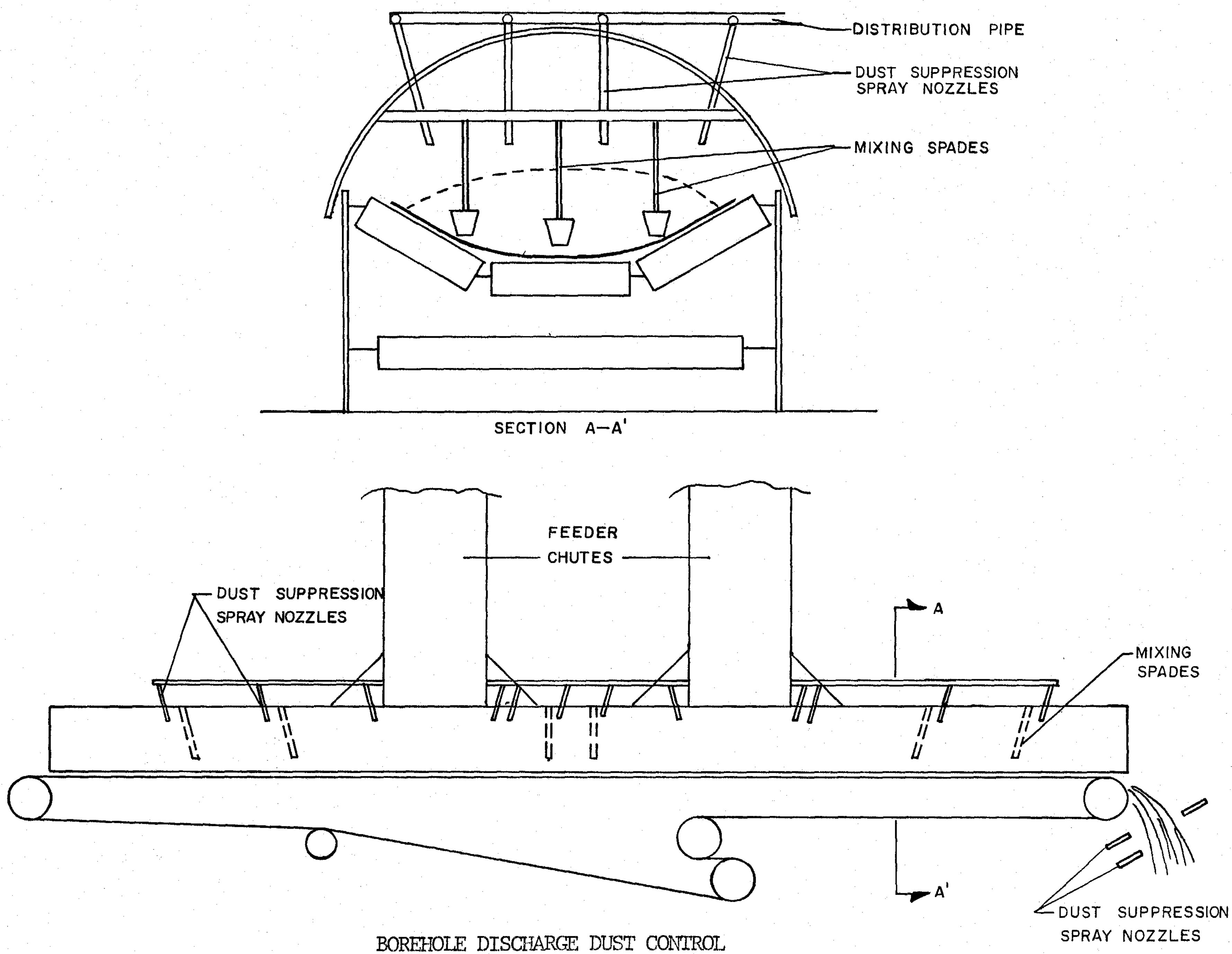
Retorted oil shale from the Paraho direct-heat-mode retort degrades easily when handled, so, unless dust abatement precautions are taken, dust will occur throughout the transporting and stowing operations. A combination of dust suppression and dust collection will be used to control the dust at its source.

3.5.2.2.1 Transfer Points:

To effectively eliminate the dust underground, the retorted oil shale must be treated with a chemical wetting solution as soon as it arrives at the backfill level. To accomplish this, the material will be sprayed with a wetting solution as it is fed out of the surge bin onto the feeder belt conveyor. Each feed point will be equipped with three distribution headers equipped with sufficient spray nozzles to ensure a uniform application rate across the entire stream of material (Figure 3.5.2.2.1-1). As the material flows along the feeder belt it will be wet down again, agitated, and mixed to provide for more complete wetting. A final application will be made as the shale is discharged onto the main conveyor belt. This will render the material dust free for a period of time.

Due to the expanding length of the transport conveyors as the project progresses, and the subsequent increased amount of time that the material will be on a conveyor, the material will begin to dry out and generate more dust. Thus, another dust suppressant application point at the first transfer station will be necessary to apply a carry-over amount which will compensate for the additional moisture loss, and will ensure a relatively dust-free material through the remainder of the transport system to the backfilling area.

The Johnson-March "Chem-Jet" system was used as the basis for describing the suppressant equipment. This is not intended as an endorsement by the U. S. Bureau of Mines for this system, but rather as an example of a feasible system. It is comprised of a proportioner,



BOREHOLE DISCHARGE DUST CONTROL
FIGURE 3.5.2.2.1-1

flow controllers at the application points, and spray system. It also includes a filter and flush assembly to filter the mine water before it enters the proportioner and is mixed with the wetting agent "Compound M-4 10/40." This wetting agent is mixed with water in a 1:3500 ratio and is applied to the retorted oil shale at a rate that will provide a moisture content of three percent which will render the material dust free. An additional one percent will be added as the carry-over amount to ensure dust free material throughout the transport system. The system will be completely automatic so that dust control treatment will occur only when the conveyors are transporting material.

A secondary dust collection system has been devised in the event some dust is generated at the transfer stations or a breakdown occurs in the dust suppressant system. Each conveyor transfer station will include a hood arrangement with a Vortex "Sepairator/Impactair" unit for the exhaust system which will capture 99 percent of all respirable dust which is generated, making the exhaust air sufficiently clean to be directed back into the haulageways. To provide adequate exhaust air volumes for the three transfer chutes at each transfer station, each station will require one 10,000 cfm rated Vortex unit (Figure 3.5.2.2.1-2). The Vortex unit consists of a turbine which pulls captured air through a flooded bed scrubber into the turbine blades which creates a vortex action which separates the dust-laden water from the cleaned air. The cleaned air is discharged into the mine atmosphere, and the slurry is discharged directly onto the downstream conveyor belt, thereby increasing the moisture content of the material somewhat and further acting as a dust suppressant. Each Vortex unit uses 13 gpm of unfiltered water.

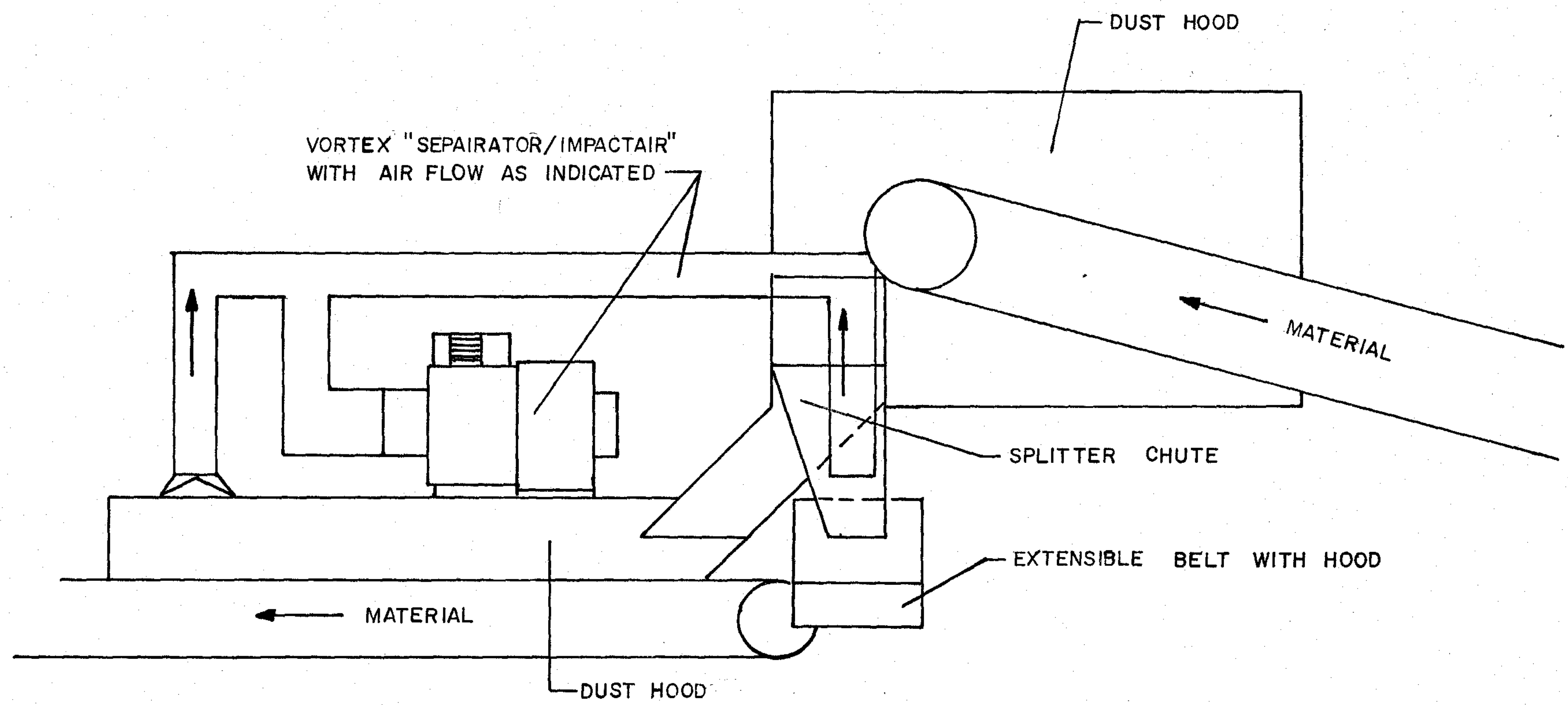
3.5.2.2.2 Haulageways:

Dust in the vehicular travelways will be controlled by spraying the roadways with a water-wetting agent solution as required. Ventilation air velocities will be kept low by the proper sizing of all entries. To further minimize dust potential, conveyor speeds will be no more than 500 feet per minute.

3.5.2.2.3 Stopes:

Dust control in the stopes during backfilling will vary with the stowing method and type of stope. Each stope will be on a separate split of air during the backfilling operation. With the exception of pneumatic stowing, the dust suppressant system described in Section 3.5.2.2.1 will make backfilling operations essentially dust free for either type of stope. This may enable the stowing and compaction to take place concurrently in the same stope.

Pneumatic stowing will generate more dust than any other method or other mine activity. Water will be injected into the pneumatic system near the discharge nozzle. This method of suppression will not be totally effective due to the extremely high particle velocity anticipated and the degradation on impact after discharge. Workers will not be allowed in the stope during stowing, and all air leaving the stope will be directed into the return air course.



TRANSFER POINT DUST COLLECTOR

FIGURE 3.5.2.2.1-2

Mechanical stowing presents the greatest challenge for dust control because men will be in the stope during placement. In all cases the moisture content of the retorted shale will be in the range of five to six percent prior to discharge into the stope. Since mechanical compaction is not possible in the sublevel stopes, intake air will be brought in through the stowing access entries and will be exhausted into the return air course at the inby end of the stope.

In chamber backfilling, retorted shale will be compacted mechanically. It may not be possible that stowing and compaction can be done concurrently in a single stope because of the amount of dust that will be created. Since compaction will be done at a lower elevation than stowing discharge, a positive ventilation technique will be needed during compaction. Figure 3.5.2-4 shows a proposed ventilation scheme for this phase of the operation. The compacting equipment will include pressurized, air-conditioned cabs for operator safety and comfort.

3.5.2.2.4 Return Air:

Eventually most of the airborne dust from all phases of backfilling will appear in the return air courses. Relatively low air velocities will cause some of the dust to settle, but a portion will reach the return air shafts. However, water or water-surfactant spray curtains will scrub the air prior to its entry into the upcast shafts. The resulting sludge from the spray curtains will be pumped to a stope for disposal.

3.5.2.2.5 Monitoring:

Dust will be continuously sampled and monitored at strategic locations in the transport and stowing areas and in the ventilation entries. The work force will be instructed in the importance of dust control and personal hygiene. Section 3.5.4 describes the dust monitoring system.

3.5.2.3 Temperature:

High underground temperatures will adversely affect an underground retorted oil shale disposal operation. Initially, it was felt that placement of hot retorted shale in the mine would result in reduced pillar stability due to accelerated creep and thermal stresses. These concerns are valid and are being investigated under other USBM contracts. However, temperatures high enough to affect physical rock characteristics will create far more severe environmental problems within the mine.

The ambient rock temperature at the depth planned for this mine is nearly 95^{o}F. This fact in itself creates a potential health and productivity problem for the work force. To avoid drastic aggravation of an existing problem, the retorted shale will be cooled prior to transport into the mine. Cooling before placement will also negate the possibility of creep and thermal stress in the pillars.

3.5.2.4 Summary:

Mine ventilation requirements will increase when an underground disposal system for retorted shale is introduced. The amount of additional air required will be a function of the transport and stowing methods used. Pneumatic methods will require large amounts of air which must be exhausted through the mine ventilation system.

Toxic and explosive gases may be present in minor amounts, and a monitoring system is planned for their detection and control (as described in Section 3.5.4). A sufficient volume of air will always be provided for dilution and removal of these gases.

Dust will be suppressed with water or water-surfactant sprays wherever possible and with dust collectors elsewhere. An ongoing program of dust sampling will be required, and all ambient dust standards will be met.

The adverse effects of high temperatures on men in the working areas, and on pillar stability, will be controlled by cooling the retorted shale to an acceptable level prior to stowing.

3.5.3 Hydrology:

Two major aquifers are present in Piceance Creek Basin and the effects of these aquifers must be considered in the design of backfilling systems. Tectonic activity in the basin has led to folding and fracturing of formations and to the development of fault structures (26). These faults and fractures produce variations in local hydrological conditions and have to be accounted for in the mine design. Since the mining methods incorporating backfilling are specifically designed for the deeper part of the Basin, the discussion below is based on hydrological conditions at the Horse Draw site of the U. S. Bureau of Mines.

The geohydrologic section at the Horse Draw site, from top to bottom, consists of an upper aquifer, a confining layer, a lower aquifer, and the unleached saline zone (8). The upper aquifer is about 750 feet thick and consists principally of sandstone (Uinta) and marlstone (Upper Parachute Creek Member of the Green River Formation). The confining layer is about 200 feet thick and consists of an oil shale section called the Mahogany Zone of the Parachute Creek Member. The lower aquifer is about 300 feet thick and consists of marlstone and oil shale that have been leached of soluble minerals (leached zone). The unleached saline zone, about 900 feet thick, consists of oil shale and saline minerals and is virtually impermeable. Results obtained during drilling of two core holes, and from two pump tests, have been augmented with experience in oil shale hydrology to prepare the geohydrological model shown in Figure 3.5.3.1-1 (10). The significant effects of backfilling on the aquifer systems are related to the physical and chemical characteristics of the backfill, permeability of overlying and underlying zones, and local geologic structures.

The two major hydrological aspects discussed here are the effects of backfilling on the ground-water system and the effects of ground water on the backfill. The placement method is affected by the quantity of ground water encountered in the mine, and the rate of mine water inflow is affected by the permeability of the formations in and around the mine, the mining method, and the rate of mine expansion. Saturation of the fill from unexpected large ground-water inflows could lead to potential liquefaction. Ground-water quality could be affected by saline leachate originating from permeable fill material.

3.5.3.1 Effects of Ground Water on Backfilling:

3.5.3.1.1 Placement Method:

Ground-water inflow can create wet conditions which will adversely affect the mechanical backfilling operation. Mechanical placement of the fill near the roof would be difficult under wet conditions. The rate of placement of the fill has an effect on its moisture content. If fill is placed at a slow rate (one foot per day), it could be progressively saturated by ground-water inflow. Areas of heavy water inflow will result in fill saturation and facilities for draining and collecting the water should be provided. Delays in fill placement will also require control of moisture content. The fill will attain its natural angle of repose, but additional moisture will lead to slope instability. However, severe water inflow problems are not anticipated (Section 3.5.3.1.2).

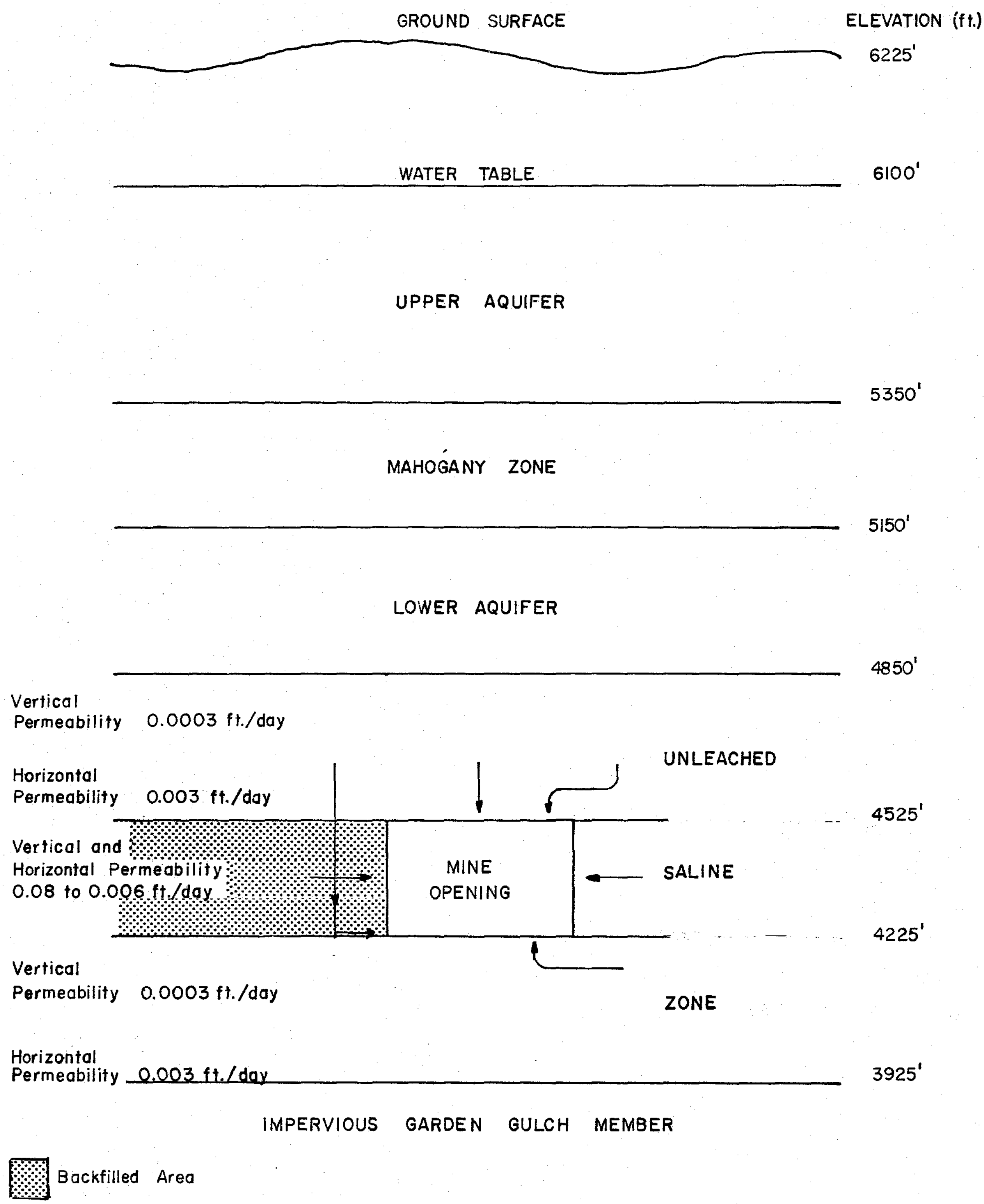
A major factor to be considered in selecting the placement method is the liquefaction potential of the fill. Liquefaction may be defined as the sudden decrease of shearing resistance of a soil from its normal value to almost zero, usually caused by a collapse of the soil structure. It is associated with soils having a low density and high moisture content, and is triggered by a sudden disturbance. The intensity of disturbance required to liquefy a soil differs for different soils. Blasts in the mine, especially in sublevel stopes, may be of sufficient magnitude to trigger liquefaction failures. Mechanically-placed fill has a relatively high density and low permeability. Thus, saturation and the resulting liquefaction of the fill material is not likely.

The effects of fill liquefaction can be twofold: disruption of activities to the extent that the entire operation could be endangered, and pillar failure resulting in unstable conditions underground. However, these effects could be significantly minimized by designing adequate bulkheads to completely retain the fill. In addition, regular monitoring of pore pressure and other hydrologic conditions behind the bulkheads could serve to indicate the development of potentially dangerous conditions.

3.5.3.1.2 Mine Inflows:

Hydrologic conditions at the Horse Draw site of the U. S. Bureau of Mines, Piceance Creek Basin, Colorado, are shown in Figure 3.5.3.1-1 (10). The discussion on mine inflows will be based on the assumption that mining will be conducted in the unleached saline zone Permeability of mechanically-placed fill varies from 0.08 to 0.006 feet per day for a low compaction effort (14).

The permeability of the unleached saline zone is very low; hence, the volume of flow has little impact on the hydrology of the relatively prolific rock mass above the mine roof. The vertical flow



HYDROLOGIC CONDITION AT HORSE DRAW

FIGURE 3.5.3.1-1

to the mine can be computed assuming a constant head gradient to the mine roof. Inflow analyses performed for a mining rate of 75,000 tons per day indicate that the inflow into the mine will vary from 440 gpm in year five to about 1,280 gpm in year 20 for the chamber and pillar method, and from 180 gpm in year five to 480 gpm in year 20 for the sublevel stoping method (10). The vertical permeability of the fill varies from 20 to about 270 times the permeability of the overlying and underlying formations. Hence, mine water will flow vertically through the fill. A schematic of the flow lines through the mining zone is presented in Figure 3.5.3.1-1.

As mentioned above, the presence of fill will cause vertical flow through the fill. Since the horizontal permeability of the formation below the mining zone is about 10 times greater than its vertical permeability, most of the water passing through the fill will flow beneath the floor back into the mine openings and can be handled there. Thus, fill placement may not reduce mine inflow, but will alter the pattern of flow.

3.5.3.2 Effects of Backfilling on the Ground Water System:

One of the main concerns associated with backfilling in the Piceance Creek Basin has been the potential effects on the ground water resources of the area. The two aquifers differ in their water quality. Water in the upper aquifer and the upper part of the lower aquifer has a concentration of dissolved solids of 960 mg/1, and the lower part of the lower aquifer has a concentration of dissolved solids of 9,400 mg/1 (10). Subsequent work by others indicates that the dissolved solids concentration for the upper part of the lower aquifer is 1,500 mg/1 and 4,000 mg/1 for the lower part of the lower aquifer (8). The principal dissolved constituents are chlorides, sulfates, and bicarbonates of sodium, magnesium, and calcium. The increase in dissolved solids concentration in the lower part of the lower aquifer is primarily due to increases in sodium chloride and sodium bicarbonate.

The quality of the leachate obtained from the fill is mainly dependent on the duration of contact between ground water and fill, the chemical nature of the fill constituents and the ground water, and the temperature of the fill at which chemical reactions take place. Test results indicate that the total dissolved solids in leachate from dry Paraho retorted shale can be as high as 11,500 mg/1 when the permeability rate approaches 0.001 feet per day (14). Other test results indicate that the total dissolved solids in the leachate is about 2,200 mg/1 (4). The reason for these widely varying results is not clear; it may be related to factors such as the particle size distribution, chemical nature of the retorted shale, contact time, and reaction rate.

During the mining phase most of the leachate will flow into mine openings and can be handled without causing any adverse impact on water quality in the project area. After cessation of mining and dewatering operations, the area will tend to attain a state of equilibrium and steady-state ground water flow will be established through the mining zone. The time period required for commencement of this flow may be several years (10). Since the permeability of the fill is greater than the surrounding host rock, ground water will flow through the fill and leach the salts until saturation of the ground water is achieved or the salts are exhausted. However, since the surrounding rock permeability is very low, the leachate may remain in the formation for a very long period without causing adverse impacts on the water quality in the aquifers. If any local geologic structures connect the unleached saline zone with the overlying aquifers, the leachate will lower the water quality in the aquifers. In addition, upward pressure gradients have been observed in the vicinity of the Horse Draw site (19). These pressure gradients may permit the leachate to flow into the overlying aquifers through fractures and other channels. Since the lower part of the Mahogany Zone has sparse fracturing and low vertical leakage, it is unlikely that the relatively good quality water of the upper aquifer will be adversely affected by the leachate.

Contaminants tend to move along with the ground water at ground water velocity, but are subjected to sorption, ion exchange, dispersions, and chemical reactions (9). The time scale for these effects is large, probably as much as several hundred years. The effect of these processes is to reduce the concentration of the contaminants and to retard their movement relative to that of ground water. However, the potential of the leachate spreading out over a wider area exists. These processes are complicated and an attempt to quantify them would require extensive laboratory and field tests. However, before the transport of contaminants in the groundwater system can be studied, an accurate assessment has to be made of the leaching characteristics of the fill and the effect of in situ ground-water quality on these characteristics.

Modification of fill characteristics could significantly alter leaching characteristics. Cemented backfill could have negligible permeability in addition to altered chemical characteristics. Leachate studies with cemented backfill are necessary to evaluate its effects. Assessment of the effects of varying types of backfill on the ground-water system can be made only if chemical and physical characteristics of the fill are well defined and the site specific geological conditions are adequately understood.

In summary, the above evaluation indicates that the post-mining impacts of backfilling mined-out voids will be minimal for mining in the unleached saline zone The magnitude of the impact is largely dependent on the presence of communication between the mining zone and the overlying aquifers. Areas for specific further investigation are the local geohydrological conditions at the project site and the leaching characteristics of the fill. Reduction of the permeability of the fill to prevent significant salt release should be studied. Preabandonment leaching of the backfill does not seem economically viable. It should be noted that the above evaluation is valid only for mines in the unleached saline zone which has very low permeability. If backfilling is practiced in more permeable zones, adverse impacts on regional ground-water quality could be considerable.

3.5.4 Monitoring:

Due to the potential presence of methane, carbon monoxide, hydrogen sulfide, and other nondesirable or toxic gases and dust in the mine during backfilling operations, monitoring systems have been devised to measure these substances in the working areas of the mine.

These gases which will be monitored are methane (CH_4), carbon monoxide (CO), hydrogen sulfide (H_2S), nitric oxide (NO), and nitrogen dioxide (NO_2) (Table 3.5.4-1). Dust concentrations will also be determined. Continuous monitoring will be utilized at semipermanent installations in the return air courses in conjunction with hand-held measuring instruments and equipment-mounted monitoring instruments where necessary in the working areas. All monitoring equipment will be instrinsically safe in accordance with MESA standards. Reference to specific manufacturers is for clarity and does not constitute an endorsement by the authors or the U. S. Bureau of Mines.

3.5.4.1 Methane:

Methane will be monitored at each split of return air and in the working chambers by use of a Mine Safety Appliance Company's (MSA) Series 510 Combustible Gas Detection System. In the monitoring system, a number of methane sensors will be installed in the return air splits or the working areas. The sensors utilize a catalytic combustion detector in conjunction with an air deflector to determine methane content in the 0-2 percent range. A set of four sensors is connected to a sensor power supply located in a fresh-air intake. The power supply provides intrinsically safe power for the sensors and also keeps the standby battery in the sensor assembly at full charge. The analog output assembly, located above ground, receives signals from each sensor, displays the reading on a meter, and provides output for the recorders. This assembly also has an auxiliary battery-powered standby power source. Specially modified mine telephone units provide a communication and calibration link for the sensors and readout/recorder units. The recorders furnish a continuous record of methane concentration at each sensor, and the analog output assembly is equipped with an annunciator panel which gives audio and visual warnings in the event of an increase of methane concentration above a predetermined level.

The MSA Methane Monitor Model VI, which is an equipment-mounted detector, will be utilized on all equipment operating in the working areas of the mine. When the methane level reaches a preset amount the unit automatically energizes a visual alarm and can shut down the equipment.

3.5.4.2 Other Gases:

Carbon monoxide (CO), hydrogen sulfide (H_2S), nitric oxide (NO), and nitrogen dioxide (NO_2) will be monitored at each split of return air and at each working chamber with the Series 3000 Ecolyser Analyser made by Energetics Science, Incorporated (ESI). A continuous monitoring system would be utilized providing monitoring of the air quality at any number of sampling points. Each sampling point will require a Series 3000 monitor which will transmit the data to a central annunciator panel on the surface. Both visual and audio alarms will be displayed on the annunciator panel.

3.5.4.3 Dust:

The dust level will be monitored with a Recording Respirable Dust Monitor, Model RDM-301, made by GCA/Technology Division. This unit provides for continuous and unattended monitoring of both the short-term

TABLE 3.5.4-1

CHARACTERISTICS OF GASES TO BE MONITORED

NAME	FORMULA	SPECIFIC GRAVITY (Air=1.0000)	EFFECT	EXPLOSIVE RANGE (%)	SOURCE	HARMFUL CONCENTRATION Ppm (1)
Methane	CH_4	0.5544	Explosive	5.0-15.0	Strata	500
Carbon Monoxide	CO	0.9671	Toxic, Explosive	12.5-74.0	Combustion	50
Hydrogen Sulfide	H_2S	1.1900	Toxic, Explosive	4.3-46.0	Strata	10
Nitric Oxide	NO	1.0366	Toxic	NONE	Combustion	25
Nitrogen Dioxide	NO_2	1.5890	Toxic	NONE	Combustion	5

(1) Colorado Mining Laws, Rules and Regulations, Bulletin 20.

and time-averaged respirable dust concentrations over a predetermined period of time which depends on the dust concentrations level and subsequently, the length of each monitoring period. It provides a direct digital printout of three variables for each measurement: the mass concentration of dust in milligrams per cubic meter for the preceding sample period, the accumulated mass of dust in milligrams from the initiation of the entire measurement cycle (i.e., the start of the shift), and the total elapsed effective sampling time. The unit is not equipped with a remote readout and, generally speaking, must be attended once per shift. The unit is equipped with an alarm circuit which will be mounted in the control building on the surface and will be activated when dust concentration rises above a set, predetermined level.

Portable, hand-held instruments to measure methane, CO, H_2S, NO, NO_2, and dust are readily available and will be used to supplement the monitoring data collected from the stationary, automated systems. MSA has shelf items available for measuring methane, carbon monoxide, and dust which will be used. Also, ESI manufactures portable monitoring instruments which will be used for measuring CO, H_2S, NO, and NO_2.

3.5.5 Energy and Water Requirements:

The water and energy requirements for each underground disposal method have been determined and are tabulated in Table 3.5.5-1. Retorted shale cooling accounts for a major portion of the water and electric power required.

TABLE 3.5.5-1

UNDERGROUND RETORTED SHALE DISPOSAL

ENERGY AND WATER REQUIREMENTS

	Water (Gal/Yr)	Power (KWH/Yr) Electric	Power (KWH/Yr) Diesel	Diesel Fuel (Gal/Yr)
	$x\ 10^6$	$x\ 10^6$	$x\ 10^6$	$x\ 10^3$
Chamber and Pillar				
Conveyor Only	838.58	175.16	20.63	633.80
Conveyor + Pneumatic	889.03	185.69	20.48	619.38
Sublevel Stoping				
Conveyor Only	757.11	157.47	21.44	642.47
Conveyor + Pneumatic	761.17	162.73	21.89	656.93

4.0 SURFACE DISPOSAL

Regardless of the underground backfilling method used, some retorted oil shale must be left on the surface. The swelling of the retorted shale, relative to undisturbed raw shale, results in an excess volume of waste material that must be stored on the surface. To achieve complete stowage underground would require a compacted-in-place density of 107.5 pcf. Using the data from Table 3.5.1.1-1 and the assumption that the density was 70 pcf before compaction, the theoretical compactive effort required to attain 107.5 pcf is approximately 2×10^6 foot pounds per cubic foot. The data was fitted to a power curve of the form $Y = ax^b$ by a least-squares curve fit. The values of the parameters are a = 69.62 and b = 0.03. The correlation coefficient for this curve fit is 0.9816. Thus, a degree of compaction that will allow complete underground disposal of retorted shale is neither technically feasible nor operationally possible.

The usual surface disposal method for retorted oil shale consists of piling the material in some convenient place, making allowances for the potential leaching and runoff water problems and providing some sort of a revegetative effort. This general approach will be followed for this study.

A surge facility will be provided near the surface disposal system truck loading area. This facility is needed to handle the surplus material during temporary slowdowns or stoppages in the underground disposal operation.

4.1 Stockpiling Method:

Since the Piceance Creek Basin is made up of a system of canyons and plateau-type highlands, a series of canyons are assumed to be available for dumping retorted oil shale. Each canyon has been assumed to drain an area of 600 acres lying above and including the dump site.

A containment basin will be constructed downstream from each dump and each will be designed to contain all of the runoff from a 100-year storm. The volume required is calculated using the equation Q = C i A where Q is peak flow in cubic feet per second, C is the runoff coefficient, i is the average rainfall intensity in inches per hour and A is the area drained in acres (Appendix B).

$$Q = 0.35 \times 4.1 \times 600 = 861 \text{ cfs}$$

Assuming a storm duration of 45 minutes, the capacity of the catchment basin is calculated as $V = 861 \times 45 \times 60/43{,}560 = 53.4$ acre-feet. An underdrain will be installed to bypass water from the upstream side of the dump area.

Dumps will be built up by hauling retorted shale in 100-ton bottom-dump haulers from surge bins near the retort facility. Face areas will be compacted using mechanical compactors while the main body of the dumps will be compacted only by haulers and dump maintenance equipment. Compaction will increase dump stability and restrict entry of surface water into the fill. The slope of the face may be as flat as 4:1 (5) to 2:1 (15). The 4:1 slope is less prone to erosion and it facilitates the establishment of vegetation on the dump surface. Lifts will be limited to approximately one foot to permit adequate compaction and a system of berms and roadways will be maintained for access and erosion control. See Figures 4.1-1 and 4.1-2 for typical surface disposal dumps.

The surface of the dump may be covered with a dressing of topsoil, if it is available. However, topsoil is not abundant in the Horse Draw area. Some riprapping may also be used to stabilize the face areas. Revegetation will feature a mixture of native shrubs and grasses with some enhancement from the development of salt-resistant plant strains. Fertilization and irrigation will be required for two years to establish plant growth, after which the revegetated areas will rely on natural precipitation. Much work is being conducted to revegetate retorted shale piles and development of improved methods or concepts is anticipated.

4.2 Partial Surface Disposal:

Regardless of the underground backfilling method used, some of the retorted shale must remain on the surface. Table 4.2-1 shows the percent of the total retorted shale left on the surface for the selected backfilling and mining methods, and the approximate acres required for the dumps and catchment basins.

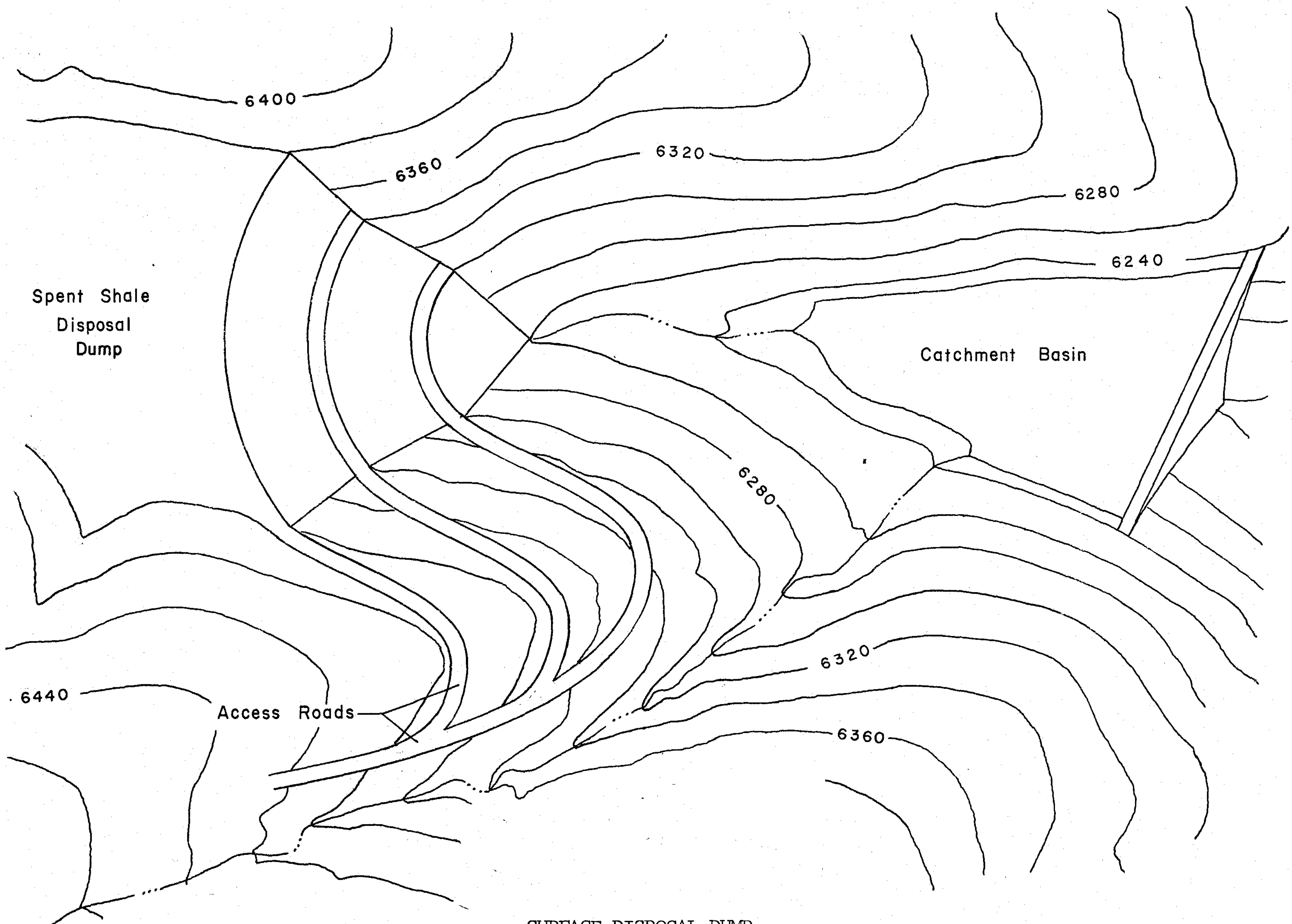
Costs and operating information for these partial surface disposal alternatives are found in Table 4.2-2.

4.3 Total Surface Disposal:

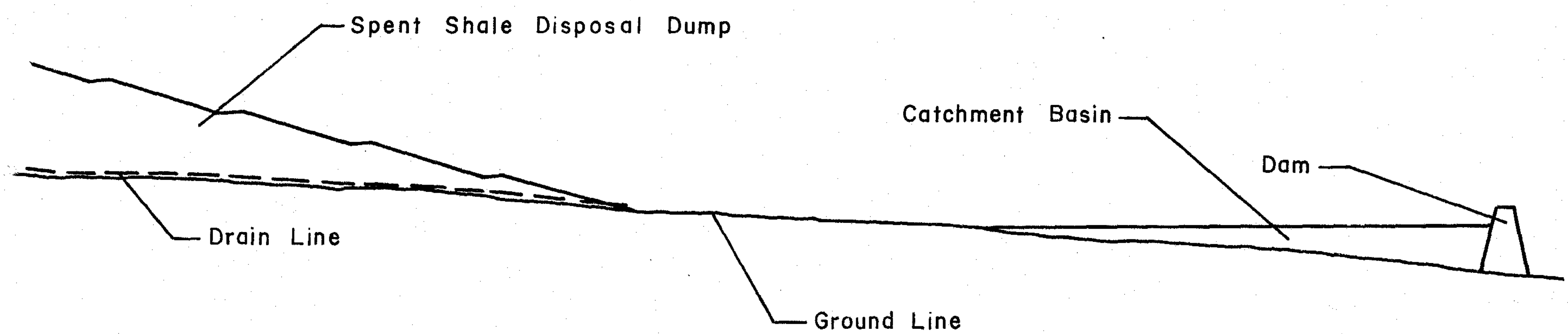
If the entire output of retorted shale were placed on the surface, the method described in Section 4.1 would be followed. Approximately 2,840 acres will be required for the dump and catchment basin. Capital and operating costs and data associated with total surface disposal for a 20-year project are shown in Table 4.3-1.

4.4 Summary:

Surface disposal is the lowest-cost method for disposing of retorted shale, but underground placement reduces environmental impact and allows greater resource recovery. However, all underground disposal systems require that some of the retorted material be stored on the surface. For the disposal methods selected, the amount left on the surface ranges from 15 to 27 percent of the total amount of retorted shale.



SURFACE DISPOSAL DUMP
FIGURE 4.1-1



SURFACE DISPOSAL DUMP - VERTICAL SECTION

FIGURE 4.1-2

TABLE 4.2-1

PARTIAL SURFACE DISPOSAL OF RETORTED SHALE

Mining Method	Backfilling Method	Percent To Surface Disposal	Acres Required For Dumps
Chamber & Pillar	Conveyor	15	480
	Conveyor & Pneumatic Topfill	17	535
Sublevel Stoping	Conveyor	24	740
	Conveyor & Pneumatic Topfill	27	820

TABLE 4.2-2

COST AND OPERATING SUMMARY OF THE

PARTIAL SURFACE DISPOSAL

	Chamber & Pillar		Sublevel Stoping	
	Conveyor	Conveyor & Pneumatic	Conveyor	Conveyor & Pneumatic
Capital ($)	11,515,000	11,933,000	14,634,000	15,087,000
Operating & Maintenance Costs ($/Ton Retorted Shale)				
Operating Labor	0.2784	0.2486	0.2219	0.2013
Operating Supplies	0.1525	0.1365	0.1241	0.1152
Maintenance Labor	0.1996	0.1779	0.1611	0.1475
Maintenance Supplies	0.2113	0.1893	0.1757	0.1636
TOTAL	0.8418	0.7523	0.6828	0.6272
Manpower - Total People	85	85	109	111
Water - Gal/Day x 10^3	49	57	80	90
Energy - KWH/Day x 10^3	31	31	43	48

TABLE 4.3-1

COST AND OPERATING SUMMARY

TOTAL RETORTED SHALE DISPOSAL ON SURFACE

Capital		$37,562,000
Operating and Maintenance Costs - $/Ton Retorted Shale		
Operating Labor	$	0.0871
Operating Supplies		0.0699
Maintenance Labor		0.0795
Maintenance Supplies		0.1082
TOTAL	$	0.3447
Manpower - Total Backfilling Payroll		195
Water - Gal/Day x 10^3		203
Energy - KWH/Day x 10^3		113
Diesel Fuel - Gal/Day		4,420

5.0 SURFACE ENVIRONMENTAL EFFECTS

5.1 Land Disturbance:

The most obvious surface environmental effects of retorted shale disposal are those associated with structures or activities which alter the appearance of the land area. Disposal dumps, roads, conveyors, and other material handling structures are the most common surface features that may be related to a retorted oil shale disposal operation.

5.1.1 Surface Dumps:

Since it is not possible to replace all of the retorted shale underground, some must be stored on the surface. Section 4.0 describes the surface disposal requirements and techniques. In most areas of the Piceance Creek Basin the canyon-plateau topography lends itself to canyon filling with retorted oil shale. Surface water diversion and containment structures and roads are part of the dump infrastructure. While the overall appearance of an area is altered by surface storage, contouring and revegetation will lessen the impact and provide a scene that is acceptable to most viewers.

5.1.2 Structures:

Underground retorted shale disposal operations will require surface facilities for cooling and transporting retorted shale from the retort to the mine. The largest facilities will be required for cooling the retorted shale prior to disposal in the mine. These structures will be in the mine-retort area and will be designed to blend into the local scene as much as possible. Upon completion of the project, all structures will be removed and the area will be regraded and revegetated; thus, the long-term effect of waste disposal should be negligible.

5.1.3 Ponds:

Surface ponds are required only for the containment of runoff from the surface waste dumps. Due to small drainage areas and low annual precipitation, permanent water impoundment structures will be small.

5.1.4 Subsidence:

Subsidence will be more severe unless mined areas are backfilled, or unless adequately-sized pillars are left in the mine. Resource recovery will be reduced if pillars are sized to support the entire overburden load.

5.2 Air Quality:

Air quality in the mine area will be affected primarily by fugitive dust from surface disposal activities and by underground stowing. Some gaseous emissions may occur as the result of surface and underground use of diesel engines in addition to potential, though unsubstantiated, emissions from the retorted shale.

5.2.1 Particulates:

Fugitive dust from surface disposal activities will be minimized by wetting the haul roads. Gravel or hard surfacing will be required in some areas, and revegetation efforts will closely follow the dumping operation. Dust from stowing and underground transport will be controlled primarily by water or water-surfactant suppression.

In addition to dust emissions, there will be some particulate emissions from diesel powered equipment. Table 5.2.1-1 shows the relative amounts of particulate emissions for the worst underground disposal case (truck transport with pneumatic stowing), and for total surface disposal (27).

5.2.2 Gaseous Emissions:

Principal gaseous emissions in an underground oil shale mine are methane and those associated with the use of diesel-powered equipment. Retorted shale, however, is not expected to be a source of gaseous emissions. Although it is possible for retorted shale to ignite spontaneously under certain conditions, it will not ignite when it has been cooled and properly compacted. Table 5.2.2-1 presents the estimated emissions for the worst underground case (truck transport and stowing in sublevel stopes), and for total surface disposal of retorted shale (27).

TABLE 5.2.1-1

ESTIMATED PARTICULATE EMISSIONS

	Underground Disposal (Lb./Day)	Total Surface Disposal (Lb./Day)
Fugitive Dust		
Underground Sources	1,060	-0-
Surface Sources	2,041	4,925
Combustion Particulates	139	120

TABLE 5.2.2-1

ESTIMATED GAS EMISSIONS

	Underground Disposal (Lb./Day)	Total Surface Disposal (Lb./Day)
CO	54	47
HC (nonmethane)	17	15
NO_x	2,735	2,380
SO_x	4	3
CH_4	Unknown	None

5.3 Water:

The effects of retorted shale disposal on surface water quality will vary with the disposal system used. Runoff flow, originating above the dumps in canyons where dumps are located, will be directed beneath the dumps and catchment basins. Catchment basins will be used to contain all saline leachate and runoff from the dumps.

5.4 Summary:

Underground disposal of retorted shale can reduce substantially the surface environmental and land disturbance effects of a mine-retort facility. Surface subsidence is likely to be minimal or negligible in areas that overlie backfilled stopes. Underground disposal is expected to have only minimal adverse effects on surface water.

Total surface disposal will disturb the land form and local drainage patterns in the immediate dump areas. However, except for an increase in fugitive dust, no appreciable differences in air quality are expected.

6.0 ECONOMICS

Detailed capital and operating costs were developed for the retorted shale disposal methods described in Sections 3.0 and 4.0 of this report. The preliminary economic considerations used in the selection of these disposal methods are discussed in Section 7.5. Capital costs are based on manufacturer's quoted prices, and labor rates are consistent with current levels for underground mining in Western Colorado. All costs were developed using fourth quarter 1977 dollars.

Capital costs include initial capital requirements, deferred and replacement capital expenditures for an assumed project life of 20 years with an additional four-year preproduction development period. All costs were escalated to their respective years' future worth at a rate of seven percent per year. These future worth values were then discounted to present worth based on discount rates of 10, 15, and 20 percent. These results can assist in the comparison of alternative investment opportunities which offer the same minimum rates of return mentioned above. Since it is out of the Scope of Work for this Contract, mining and processing costs and expected revenues were not developed for inclusion in a discounted cash flow analysis. The results reported may, however, be used by the reader for inclusion in a discounted cash flow analysis when the mining and processing costs and expected revenues are known.

6.1 Underground Disposal Costs:

Capital and operating costs for the cooling, surface conveying, vertical borehole transport, underground conveying and stowing, and partial surface disposal functions are presented in this section. Total project life is assumed to be 24 years, which includes a four-year preproduction development period and an operating life of 20 years. Operating costs for the selected underground disposal methods are higher than those initially reported in Phase I (Section 7.5) because of the more detailed analysis of each selected system. Whereas, in Phase I, a relative comparison on a limited-scale mine was used due to the Scope of Work and time constraints specified by the Contract. Table 6.1-1 shows the operating and capital costs for the various backfilling methods based on fourth quarter 1977 dollars.

6.1.1 Operating Costs:

The operating costs were developed using cost centers that will probably exist for a typical oil shale mining facility. Tables 6.1.1-1 through 6.1.1-4 show the development of the operating and maintenance costs for each of the underground disposal methods and its associated partial surface disposal costs. These Tables are based on fourth quarter 1977 dollars.

6.1.2 Capital Costs:

The capital and total costs for each backfilling method have been calculated and tabulated based on fourth quarter 1977 dollars, future worth at seven percent annual inflation, and present worth for the inflated costs with discount factors of 10, 15, and 20 percent. Table 6.1.2-1 shows the capital and total project costs for the various underground backfilling methods and the associated partial surface disposal requirements.

TABLE 6.1-1

UNDERGROUND RETORTED SHALE DISPOSAL

CAPITAL AND OPERATING COSTS
(1977 Dollars)

	Chamber and Pillar Mining		Sublevel Stoping	
	Conveyor Only	Conveyor & Pneumatic	Conveyor Only	Conveyor & Pneumatic
Capital (1)				
Total ($ x 10^3)	65,119	66,033	59,006	61,207
Retorted Shale ($/Ton)	0.1315	0.1334	0.1192	0.1236
Shale Oil ($/Barrel)	0.1784	0.1809	0.1617	0.1677
Operating (2)				
Total ($ x 10^3)	334,200	384,600	322,020	360,540
Retorted Shale ($/Ton)	0.6751	0.7769	0.6505	0.7283
Shale Oil ($/Barrel)	0.9156	1.0537	0.8822	0.9878
Combined Capital and Operating Costs				
Total ($ x 10^3)	399,319	450,633	381,026	421,747
Retorted Shale ($/Ton)	0.8066	0.9103	0.7697	0.8519
Shale Oil ($/Barrel)	1.0940	1.2346	1.0439	1.1555

(1) Based on 20-year operating life and a four-year preproduction period.

(2) Based on 20-year operating life.

TABLE 6.1.1-1

OPERATING COSTS - CHAMBER & PILLAR

CONVEYOR ONLY
($/Ton Retorted Shale*)

Cost Center	OPERATING Labor	OPERATING Supplies	MAINTENANCE Labor	MAINTENANCE Supplies	TOTAL
Surface Conveying	0.0157	0.0128	0.0100	0.0009	0.0394
Cooling	0.0219	0.1390	0.0393	0.0097	0.2099
Surface Disposal	0.0346	0.0221	0.0213	0.0315	0.1095
Borehole	0.0035	0.0014	0.0011	0.0022	0.0082
Underground Conveying	0.0361	0.0869	0.0099	0.0077	0.1406
Backfilling	0.0463	0.0243	0.0185	0.0136	0.1027
Supervision	0.0270	0.0021	0.0156	0.0014	0.0461
Technical, Administrative, and General	0.0166	0.0016	0.0003	0.0002	0.0187
TOTAL	0.2017	0.2902	0.1160	0.0672	0.6751

* 1977 Dollars

TABLE 6.1.1-2

OPERATING COSTS - CHAMBER & PILLAR

CONVEYOR & PNEUMATIC
($/Ton Retorted Shale*)

Cost Center	OPERATING Labor	OPERATING Supplies	MAINTENANCE Labor	MAINTENANCE Supplies	TOTAL
Surface Conveying	0.0157	0.0128	0.0100	0.0009	0.0394
Cooling	0.0219	0.1388	0.0393	0.0097	0.2097
Surface Disposal	0.0349	0.0224	0.0244	0.0320	0.1137
Borehole	0.0035	0.0014	0.0011	0.0021	0.0081
Underground Conveying	0.0361	0.0867	0.0099	0.0077	0.1404
Backfilling	0.0621	0.0489	0.0444	0.0454	0.2008
Supervision	0.0270	0.0021	0.0156	0.0014	0.0461
Technical, Administrative, and General	0.0166	0.0016	0.0003	0.0002	0.0187
TOTAL	0.2178	0.3147	0.1450	0.0994	0.7769

* 1977 Dollars

TABLE 6.1.1-3

OPERATING COSTS - SUBLEVEL STOPING

CONVEYOR ONLY
($/Ton Retorted Shale*)

Cost Center	OPERATING Labor	OPERATING Supplies	MAINTENANCE Labor	MAINTENANCE Supplies	TOTAL
Surface Conveying	0.0157	0.0119	0.0100	0.0006	0.0382
Cooling	0.0219	0.1213	0.0393	0.0087	0.1912
Surface Disposal	0.0436	0.0287	0.0312	0.0419	0.1454
Borehole	0.0035	0.0014	0.0011	0.0022	0.0082
Underground Conveying	0.0361	0.0790	0.0099	0.0069	0.1319
Backfilling	0.0463	0.0169	0.0052	0.0024	0.0708
Supervision	0.0270	0.0021	0.0156	0.0014	0.0461
Technical, Administrative, and General	0.0166	0.0016	0.0003	0.0002	0.0187
TOTAL	0.2107	0.2629	0.1126	0.0643	0.6505

* 1977 Dollars

TABLE 6.1.1-4

OPERATING COSTS - SUBLEVEL STOPING

CONVEYOR & PNEUMATIC
($/Ton Retorted Shale*)

Cost Center	OPERATING		MAINTENANCE		TOTAL
	Labor	Supplies	Labor	Supplies	
Surface Conveying	0.0157	0.0119	0.0100	0.0006	0.0382
Cooling	0.0219	0.1213	0.0393	0.0087	0.1912
Surface Disposal	0.0444	0.0299	0.0324	0.0439	0.1506
Borehole	0.0035	0.0014	0.0011	0.0022	0.0082
Underground Conveying	0.0361	0.0764	0.0099	0.0066	0.1290
Backfilling	0.0621	0.0254	0.0444	0.0144	0.1463
Supervision	0.0270	0.0021	0.0156	0.0014	0.0461
Technical, Administrative, and General	0.0166	0.0016	0.0003	0.0002	0.0187
TOTAL	0.2273	0.2700	0.1530	0.0780	0.7283

* 1977 Dollars

TABLE 6.1.2-1

UNDERGROUND RETORTED SHALE DISPOSAL

CAPITAL AND TOTAL PROJECT COSTS

	MINING AND BACKFILLING METHOD			
	Chamber and Pillar Mining		Sublevel Stoping	
	Conveyor Only $ x 10^3	Conveyor & Pneumatic $ x 10^3	Conveyor Only $ x 10^3	Conveyor & Pneumatic $ x 10^3
Current 1977 Dollars				
Capital	65,119	66,033	59,006	61,207
Total Project Costs(1)	399,319	450,633	381,026	421,747
Future Worth @7%				
Capital	118,839	119,361	101,457	105,654
Total Project Costs	1,079,634	1,225,053	1,027,237	1,142,174
Present Worth(2) @10%				
Capital	53,687	54,581	49,487	51,247
Total Project Costs	280,350	315,429	267,890	295,776
Present Worth @15%				
Capital	41,132	41,932	38,673	39,952
Total Project Costs	169,027	189,114	161,910	177,929
Present Worth @20%				
Capital	33,015	33,703	31,428	32,406
Total Project Costs	111,180	123,658	106,745	116,732

(1) Total Project Costs include capital plus operating costs.

(2) Present Worth is based on escalated future worth accumulated by year.

6.2 Economic Effects of Increased Resource Recovery:

Backfilling permits mining with relatively thin pillars which increases resource recovery. This increase is estimated to be approximately 16 percent for chamber and pillar mining and 15 percent for sublevel stoping. The relative economic effects of increased resource recovery were developed to meet contract requirements for this section and are based on the following assumptions which estimate the conditions likely to exist for an oil shale mining and retorting operation:

- A finite reserve capable of producing 50,000 barrels per day for 20 years with no backfilling.
- Mining, crushing, and retorting costs of $10.00 per barrel of shale oil produced.
- Shale oil selling price of $15.00 per barrel.
- No attempt to account for time value of money.

Based on the assumptions listed above, an analysis of the effects of increased resource recovery indicates that a probable cost benefit exists for the conveyor only method of backfilling for both mining methods but is of a marginal nature for conveyor plus pneumatic backfilling. Table 6.2-1 shows the relative effects of increased resource recovery on the economics of underground backfilling using 1977 dollars with no discounting.

6.3 Total Surface Disposal Costs:

Total surface disposal was considered as an alternative to underground retorted shale disposal. Costs were determined so that an objective comparison could be made of the capital and operating costs for the alternative disposal methods. Surface disposal costs were developed using the same guidelines that were used for underground disposal costs. Table 6.3-1 shows the surface disposal costs based on fourth quarter 1977 dollars.

6.3.1 Operating Costs:

The operating costs were divided into cost centers in the same manner as for underground disposal. The operating and maintenance costs, based on fourth quarter 1977 dollars, are detailed in Table 6.3.1-1.

6.3.2 Capital Costs:

The capital and total costs for total surface disposal were developed as described in Section 6.1.2 for underground disposal. The same escalation and discount factors were used in both total surface disposal and underground disposal analyses. The capital and total project costs for total surface disposal are tabulated in Table 6.3.2-1.

TABLE 6.2-1

RELATIVE EFFECTS OF INCREASED RESOURCE RECOVERY RESULTING FROM

UNDERGROUND BACKFILLING FOR A FINITE RESERVE(1)
(1977 Dollars)

	Chamber and Pillar Mining		Sublevel Stoping	
	Conveyor Only	Conveyor & Pneumatic	Conveyor Only	Conveyor & Pneumatic
	$ x 10^3	$ x 10^3	$ x 10^3	$ x 10^3
No Backfilling (1)				
Total Revenue (2)	5,475,000	5,475,000	5,475,000	5,475,000
Total Expense (3)	3,650,000	3,650,000	3,650,000	3,650,000
Cash Flow	1,825,000	1,825,000	1,825,000	1,825,000
With Backfilling (5)				
Total Revenue (2)	6,351,000	6,329,100	6,296,250	6,285,300
Total Expense (3)	4,482,139	4,517,455	4,421,718	4,460,040
Cash Flow (4)	1,868,861	1,811,645	1,874,532	1,825,260
Net Cash Flow Effect Related to Underground Waste Disposal and Increased Resource Recovery	43,861	-13,355	49,532	260

(1) Finite Reserve based on parcel with 20-year life with no backfilling and 50,000 barrels per day production.

(2) Based on $15.00 per barrel of shale oil produced.

(3) Based on $10.00 per barrel for mining, crushing, retorting, and retorted shale disposal. These costs are estimates and are not directly applicable to any specific operation.

(4) Does not include finance charges, depletion, depreciation, or taxes.

(5) Includes considerations for increased resource recovery resulting from smaller pillar requirements related to underground backfilling.

TABLE 6.3-1

TOTAL SURFACE RETORTED SHALE DISPOSAL

CAPITAL AND OPERATING COSTS
(1977 Dollars)

	Capital	Operating	Combined Capital and Operating Costs
Total ($ x 10^3)	37,563	170,640	208,203
Retorted Shale ($/Ton)	0.0759	0.3447	0.4206
Shale Oil ($/Barrel)	0.1029	0.4675	0.5704

TABLE 6.3.1-1

TOTAL SURFACE DISPOSAL

OPERATING COSTS
($/Ton Retorted Shale*)

Cost Center	OPERATING		MAINTENANCE		TOTAL
	Labor	Supplies	Labor	Supplies	
Hauling	.0279	.0484	.0459	.0820	.2042
Roads & Dump	.0262	.0116	.0179	.0196	.0753
Conveying	.0094	.0047	.0028	.0024	.0193
Revegetation	.0022	.0018	.0003	.0004	.0047
Supply & Service	.0045	.0018	.0029	.0034	.0126
Supervision	.0062	.0010	.0096	.0003	.0171
Technical, Administrative, and General	.0107	.0006	.0001	.0001	.0115
TOTAL	.0871	.0699	.0795	.1082	.3447

* 1977 Dollars

TABLE 6.3.2-1

TOTAL SURFACE DISPOSAL

CAPITAL AND TOTAL PROJECT COSTS

	$\$ \times 10^3$
Current 1977 Dollars	
Capital	37,563
Total Project Costs (1)	208,203
Future Worth @7%	
Capital	98,420
Total Project Costs	588,996
Present Worth @10% (2)	
Capital	26,643
Total Project Costs	142,377
Present Worth @15%	
Capital	16,291
Total Project Costs	81,594
Present Worth @20%	
Capital	10,839
Total Project Costs	50,751

(1) Total Project Costs include capital plus operating costs.

(2) Present Worth is based on escalated future worth accumulated by year.

6.4 Cost Comparison - Surface and Underground Disposal:

Underground disposal costs range from 2 to 2.5 times greater than surface disposal costs. Table 6.4-1 shows the relative magnitude of costs for each of the disposal and mining methods. The principal reasons for the difference in costs are as follows:

- ° No cooling facility requirements for surface disposal.
- ° Labor and capital efficiencies resulting from use of large haulers for surface disposal.
- ° Less equipment needed for surface disposal.
- ° Lower manpower requirements for surface disposal.

No attempt has been made to assign a dollar value to the adverse environmental effects on areas where surface disposal has taken place. A recent agreement between the Colorado Department of Wildlife and one of the Federal Prototype Oil Shale Lease holders, concerning some Colorado Department of Wildlife land within the lease boundaries, assigned a value of $376 per acre as compensation for lost wildlife habitat. If this figure should become a precedent, the overall costs would increase by approximately $200,000 for total surface disposal and by $85,000 for an underground disposal program with partial surface disposal.

TABLE 6.4-1

COST COMPARISON

UNDERGROUND VERSUS SURFACE DISPOSAL

	UNDERGROUND MINING AND BACKFILLING METHOD				
	Chamber and Pillar Mining		Sublevel Stoping		
	Conveyor Only	Conveyor & Pneumatic	Conveyor Only	Conveyor & Pneumatic	SURFACE
	$\$ \times 10^3$	$\$ \times 10^3$	$\$ \times 10^3$	$\$ \times 10^3$	$\$ \times 10^3$
Current 1977 Dollars					
Capital	65,119	66,033	59,006	61,207	37,563
Total Project Costs (1)	399,319	450,633	381,026	421,747	208,203
Future Worth @7%					
Capital	118,839	119,361	101,457	105,654	98,420
Total Project Costs	1,079,634	1,225,053	1,027,237	1,142,174	588,996
Present Worth @10% (2)					
Capital	53,687	54,581	49,487	51,247	26,643
Total Project Costs	280,350	315,429	267,890	295,776	142,377
Present Worth @15%					
Capital	41,132	41,932	38,673	39,952	16,291
Total Project Costs	169,027	189,114	161,910	177,929	81,549
Present Worth @20%					
Capital	33,015	33,703	31,428	32,406	10,839
Total Project Costs	111,180	123,658	106,745	116,732	50,751

(1) Total Project Costs include capital plus operating costs.

(2) Present Worth is based on escalated future worth.

7.0 PHASE I SUMMARY - SELECTION OF BACKFILLING METHODS

Phase I of Contract J0265052 included preliminary feasibility studies and selection of the most economically and technically feasible methods for the underground disposal of retorted oil shale. Three basic methods for transporting and stowing retorted oil shale were studied individually and in combinations of two or three in order to determine the most desirable system for the prescribed mining methods. Hydraulic, pneumatic, and mechanical methods (employing both belt conveyors and trucks) were studied. The two mining methods considered are chamber and pillar, and sublevel stoping. A hypothetical deep underground mine in the Piceance Creek Basin of northwestern Colorado was used as the site for this study.

As a result of the Phase I comparative study, conveyor transport and stowing has been selected as the most feasible system for underground disposal of retorted oil shale in the selected deep mine environment. This section describes the various methods studied and the ranking analysis which determined the most feasible methods.

Concepts and costs discussed or shown in this section may vary from other sections of this report because of refinements and additional data developed in the detailed engineering work performed during Phase II of the Contract.

7.1 Transport Methods:

The three general methods stipulated in the contract for transporting retorted oil shale to underground disposal areas are hydraulic, mechanical, and pneumatic. These methods were investigated in sufficient depth to determine the technical and operating feasibilities, relative costs, and environmental effects for ranking and selecting the most suitable method.

Due to the rugged terrain and a desire to minimize surface disruption, all methods investigated provide for vertical transport to the backfilling level from a point near the retort facilities. All lateral transport will be within the mine. The presence of two overlying aquifers and the expense of cementing casing in all boreholes to isolate these aquifers was an important point considered in selecting underground lateral transport of the retorted oil shale to the stopes.

7.1.1 Hydraulic Transport:

The hydraulic transport method envisioned here consists of preparing a slurry on the surface and transporting it through pipes to the backfill level. Two 18-inch-diameter pipes will transport the slurry and two 12-inch-diameter pipes will be used to return decant water to the surface.

Test work performed under USBM Contract No. H0166065 indicates that a high particle degradation rate is to be expected with Paraho retorted shale. Pumping a bulk sample containing 2.8 percent of minus 270 mesh for 1.5 hours produced 65.7 percent of the minus 270 mesh fraction. This reduction in particle size results in declining head losses, but significantly increases underground disposal problems. Particle degradation occurs during the slurrying and pumping operations and is less dependent on transport distance. Preliminary results obtained are summarized in Table 7.1.1-1. The degree of particle degradation reported is much greater than would be expected in an actual operation. During testing, the material passed through the pump many times with some additional particle degradation occurring each time. However, particle degradation will occur in the slurry mixing plant as well as during slurry pumping. A high degree of degradation is reported, but the order of magnitude is reasonable for the comparative aspects of this report. This work also shows that a slurry containing over 50 percent solids by weight cannot be transported hydraulically. Experience at mines where hydraulic backfilling is done indicates that dewatering of the fill and drainage from the fill are adversely affected by the presence of a large slime-size fraction. A maximum of five percent minus 400 mesh and 25 percent minus 325 mesh can be tolerated. Thus, over 35 percent of the slurried material will be removed by cyclones or centrifuges and pumped to slimes ponds on the surface.

Typical surface layout and cross section of a hydraulic transport system is shown in Figures 7.1.1-1 and 7.1.1-2. Cooled retorted shale will be transported by conveyor to the slurry mixing plant. A Marconajet system will be used in the slurry mixing plant to produce a slurry containing 40 percent solids by weight, although this is not to be construed as an endorsement of Marconajet systems. The slurry will then pass through a bank of cyclones or centrifuges to deslime it to a maximum of 25 percent minus 325 mesh. The overflow will be sent to the slimes pond and the underflow will flow through two 18-inch-diameter pipes to the backfill level. Automatic density control valves will maintain the slurry at 40 percent solids during pipe transport. It is assumed that the material will be transported vertically over a length of 2,000 feet, thus subjecting the pipe to high pressures. Wear-resistant fiberglass pipe will be used because of its lighter weight and resultant ease in handling.

The hydraulic head of 1,100 psi is sufficient to move the slurry through the 2,500 feet of horizontal pipe assumed for this study; however, slurry pumps will be provided in the backfill level for slurry handling if necessary. Box drainage tests indicate that 65-70 percent of the water can be drained from the fill (4). Drained water will be channeled to a settling sump and the clear effluent will be pumped to the surface (Figure 7.1.1-2). Recycled water will be utilized for slurry preparation since laboratory tests indicate that negligible quantities of chemical constitutents are leached out of the fill. Approximately 9.9 million gallons of water per day will be lost in the underground stopes and surface slimes ponds. See Appendix C for details.

The use of large-diameter pipe requires that highly mechanized pipe handling and maintenance techniques be used. Large openings will be needed to facilitate the use of mobile cranes and other pipe handling equipment. Pipe wear can be extreme in bends, so the radius of curvature of all bends will be as large as possible. Horizontal and vertical sections of straight pipe are expected to have an effective life of approximately two years or 2,500,000 tons per section.

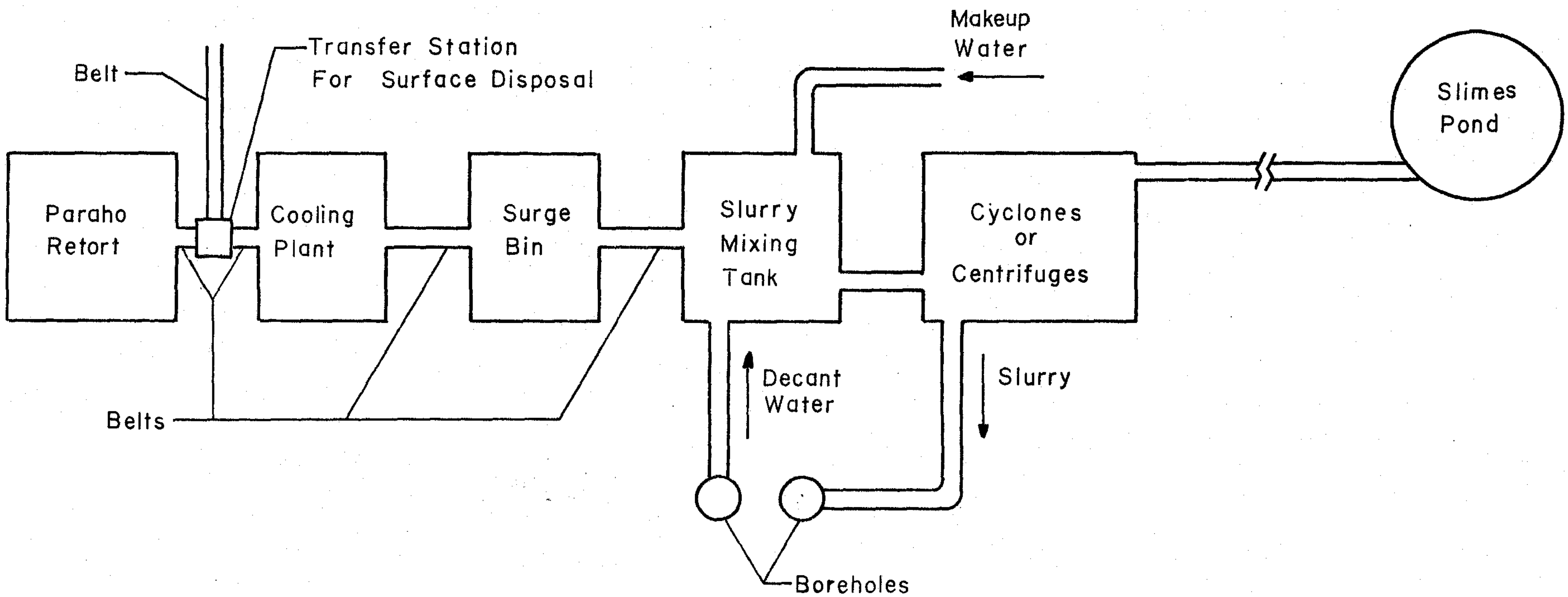
ADVANTAGES:

1. No dust problem

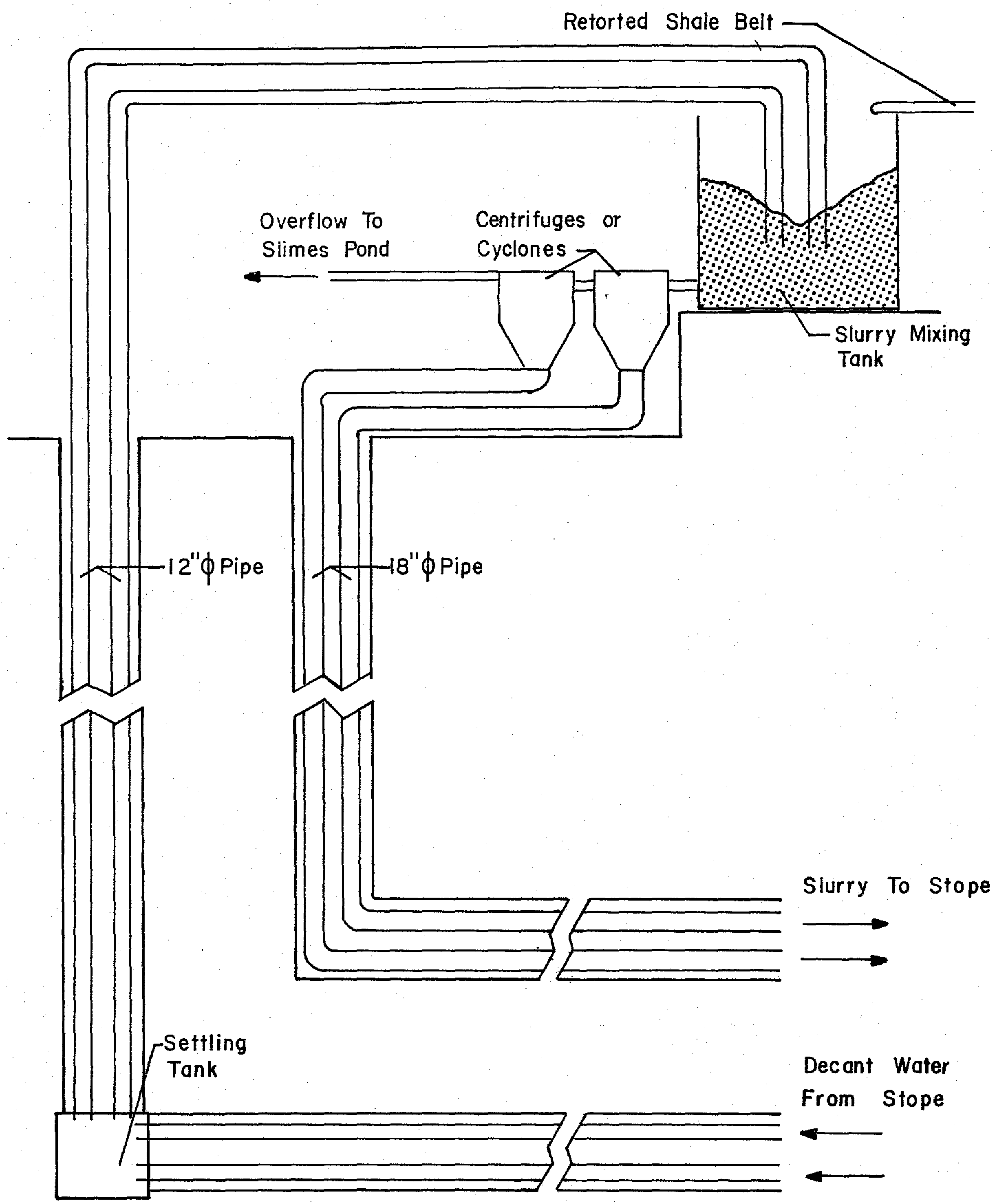
TABLE 7.1.1-1

HYDRAULIC TRANSPORT OF PARAHO SPENT SHALE THROUGH SIX-INCH PIPE

Horizontal Pipe					Vertical Pipe				
Test No.	Velocity fps	Head Loss Ft. H_2O/100'	Screen Analysis Wt. %	Solids Wt. %	Test No.	Velocity fps	Head Loss Ft. H_2O/100'	Screen Analysis Wt. %	Solids Wt. %
1	10.8	11.78	22.9 -325M 20.8 -1/4"+20M 23.7 -3/4"+1/4"	43.3	1-A	11.1	38.06		
2	2.5	3.10			1	10.0	35.52	26.4 -325M 44.1 -4M + 35M 10.7 -3/4"+1/4"	35.4
3	3.8	6.20			2	7.2	32.22		
4	3.2	4.40			3	8.7	31.72		
5	5.2	6.94			4	9.8	29.18		
6	7.5	7.07			5	11.6	30.45		
7	8.7	7.81			6	8.5	27.15		
8	10.0	8.98			7	7.3	26.14		
9	11.2	10.16	47.8 -325M 15.3 -1/4"+20M 12.9 -3/4"+1/4"	36.6	8	4.8	24.61		
10	13.1	11.91			9	9.75	27.15	53.3 -325M 22.1 -4M + 35M 5.1 -3/4"+1/4"	33.6



SURFACE LAYOUT
TYPICAL HYDRAULIC TRANSPORT SYSTEM
FIGURE 7.1.1-1



TYPICAL HYDRAULIC TRANSPORT SYSTEM

FIGURE 7.1.1-2

DISADVANTAGES:

1. Large water requirement

2. High capital and operating costs

3. Surface disruption due to slimes pond requirement

7.1.1.1 Summary:

Particle degradation that occurs during transport, imparting undesirable characteristics to the fill, is a severe disadvantage inherent to hydraulic transport of retorted shale. For this reason at least 35 percent of the material must be removed by cyclones or centrifuges and pumped to slimes ponds before the remainder can be transported to the fill level. This requirement will create more environmental disturbance than complete surface disposal. The water requirement is excessive because the material cannot be transported at a concentration much over 40 percent solids by weight, and more than 30 percent of the water is retained in the fill due to its low permeability.

7.1.2 Mechanical Transport:

Two mechanical transport methods have been evaluated for the underground transport of retorted oil shale. Conveyors and trucks were considered as the most likely methods for hauling the material to the stopes. In both cases, the retorted shale is moved to the backfill level in a large diameter, lined borehole (Figure 7.1.2-1). An eight-foot-diameter borehole will allow a mass flow rate in the range of 20 to 25 feet per minute. Dust control and suppression facilities have been included in the evaluations.

7.1.2.1 Conveyor Transport:

Apron feeders will be used to move retorted oil shale from the bottom of the borehole. A surge area will be provided for emptying the borehole in the event of conveyor system failure. All transfer points will use water sprays and dust collection hoods to control dust.

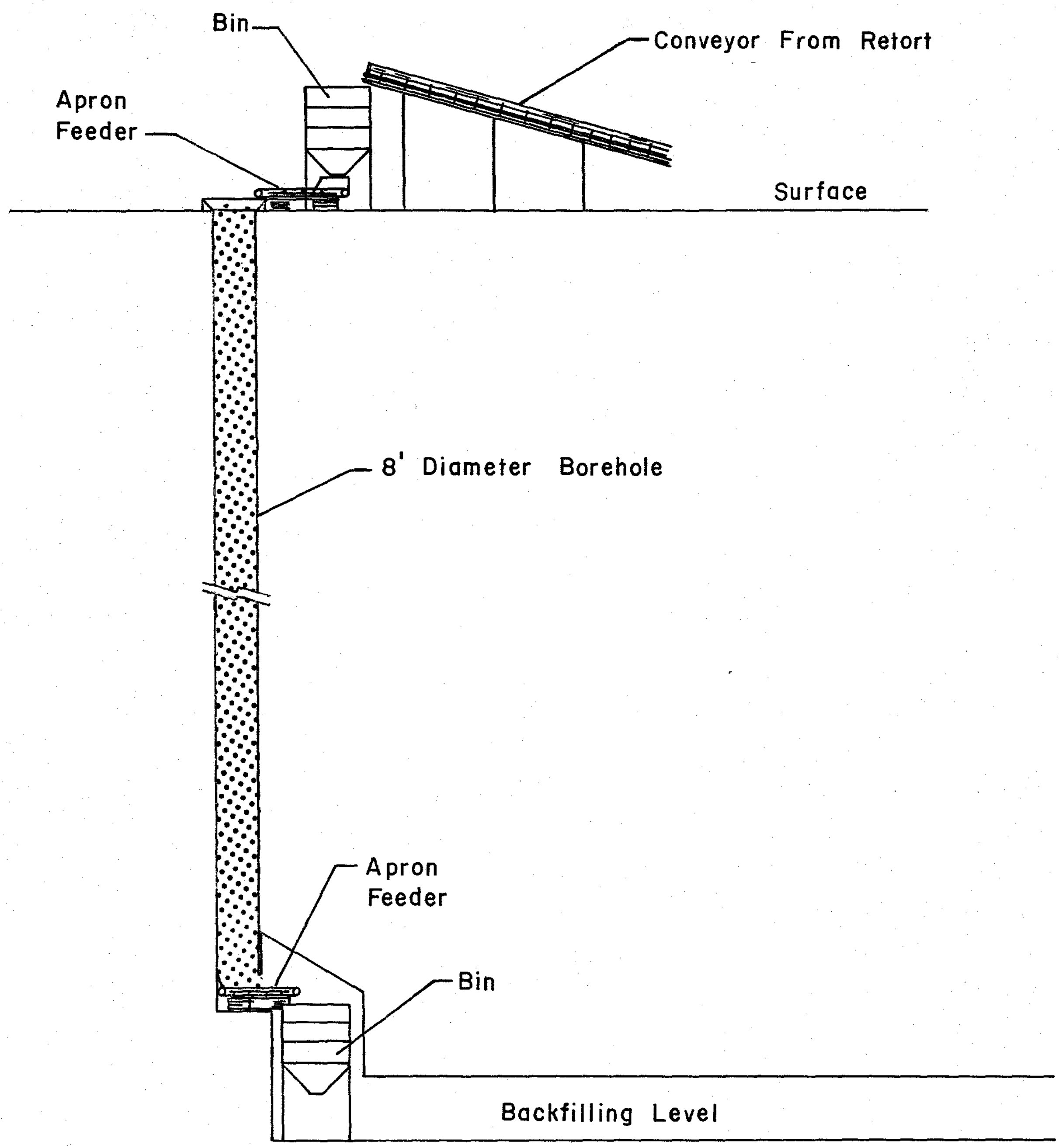
The main conveyor from the borehole transfer point to the stope area will be installed to meet all MESA standards. The 72-inch main conveyor belt will travel at 500 feet per minute and will move 2,900 tons per hour with surges to 4,000 tons per hour (11). Since two or more stopes will be backfilled simultaneously, trippers will be used to distribute the retorted oil shale to the proper stopes (Figure 7.1.2.1-1).

The backfill material will move from the tripper transfer points to the stopes by whatever stowing conveyance is used. The various stowing methods will be discussed in Section 7.2.

ADVANTAGES:

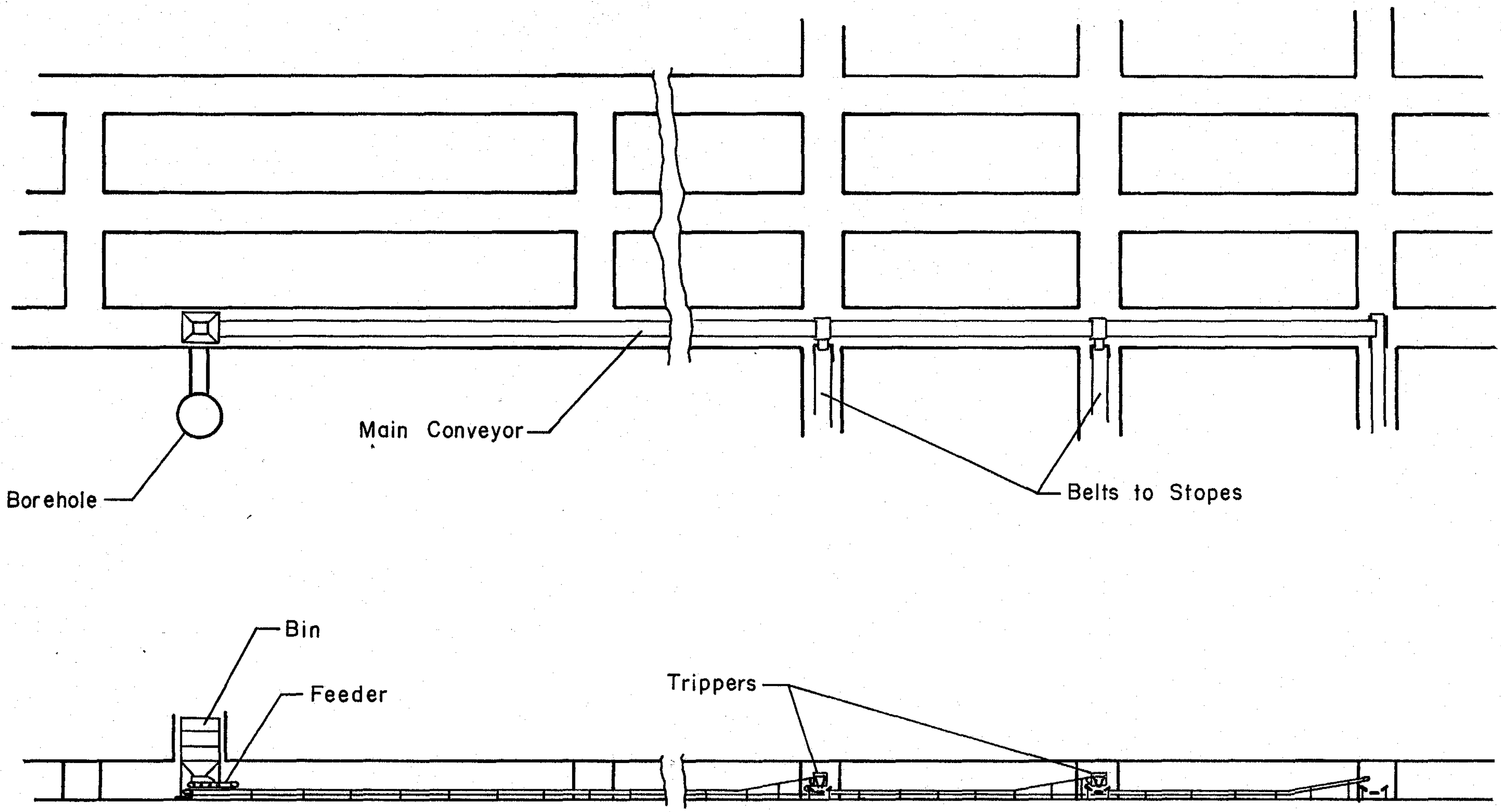
1. Low manpower requirements

2. Moderate ventilation requirements



BOREHOLE FOR MECHANICAL TRANSPORT

FIGURE 7.1.2-1



CONVEYOR TRANSPORT SYSTEM

FIGURE 7.1.2.1-1

3. Proved method

4. Low operating costs

DISADVANTAGES:

1. Inflexible system

2. Total transport dependence on one conveyor

7.1.2.2 Truck Transport:

Apron feeders will transport the material from the borehole to a surge bin for truck loading. An area will be provided to receive retorted shale from the borehole in the event that the backfilling operation is shut down unexpectedly. Water sprays and dust collectors will be used at all transfer points. Since all backfilling will be done from the upper access level, separate from the mining activity, all backfilling equipment will be used solely for retorted shale disposal.

Thirty-five-ton trucks were selected for hauling the retorted oil shale to the backfill areas because of their maneuverability and relatively small engines. A 400-horsepower engine is small enough to be tested for permissibility at existing facilities should the gassy mine conditions warrant meeting Schedule 31 rather than Schedule 24 standards. The number of trucks required is a function of the stowing method and distances traveled, and ranges from 11 to 15 operating at one time. For the comparative purposes of Phase I, an average haul distance of 2,500 feet was used with a 9.4 minute cycle time for hauling.

Trucks will haul the retorted oil shale from the surge bin at the borehole to the stope access entry. At that point the material may be disposed of in one of several methods which are discussed in Section 7.2 under Stowing Methods. Due to equipment density, a two-entry haulage system will be necessary (Figure 7.1.2.2-1). Roadways will be sprinkled to control dust and maintained to provide an efficient haulage system. Binding agents will be tested in an effort to improve dust control on the roadways.

ADVANTAGES:

1. Flexible system

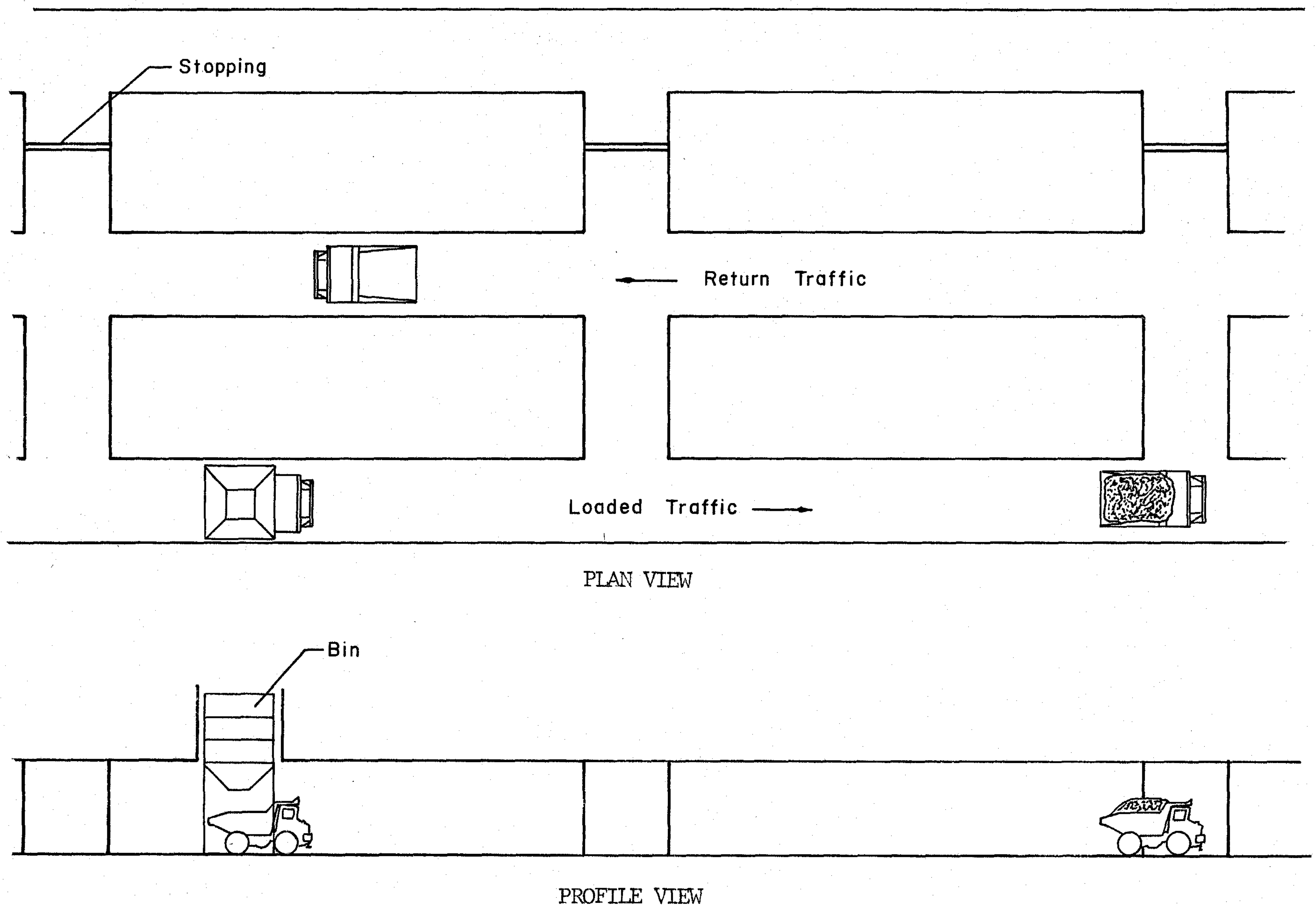
DISADVANTAGES:

1. High ventilation requirements

2. Large amount of mobile equipment

3. High energy consumption

4. High manpower requirements



TRUCK TRANSPORT SYSTEM

FIGURE 7.1.2.2-1

7.1.2.3 Summary:

The mechanical transport methods described are feasible for a retorted oil shale disposal operation. The conveyor system is preferable from the standpoint of low energy, manpower, and costs per unit conveyed. Truck haulage provides for more operational flexibility, but the environmental aspects outweigh this advantage. A tabulation of comparative factors is shown in Table 7.6.1.1.2-1 and 7.6.1.1.2-2.

7.1.3 Pneumatic Transport:

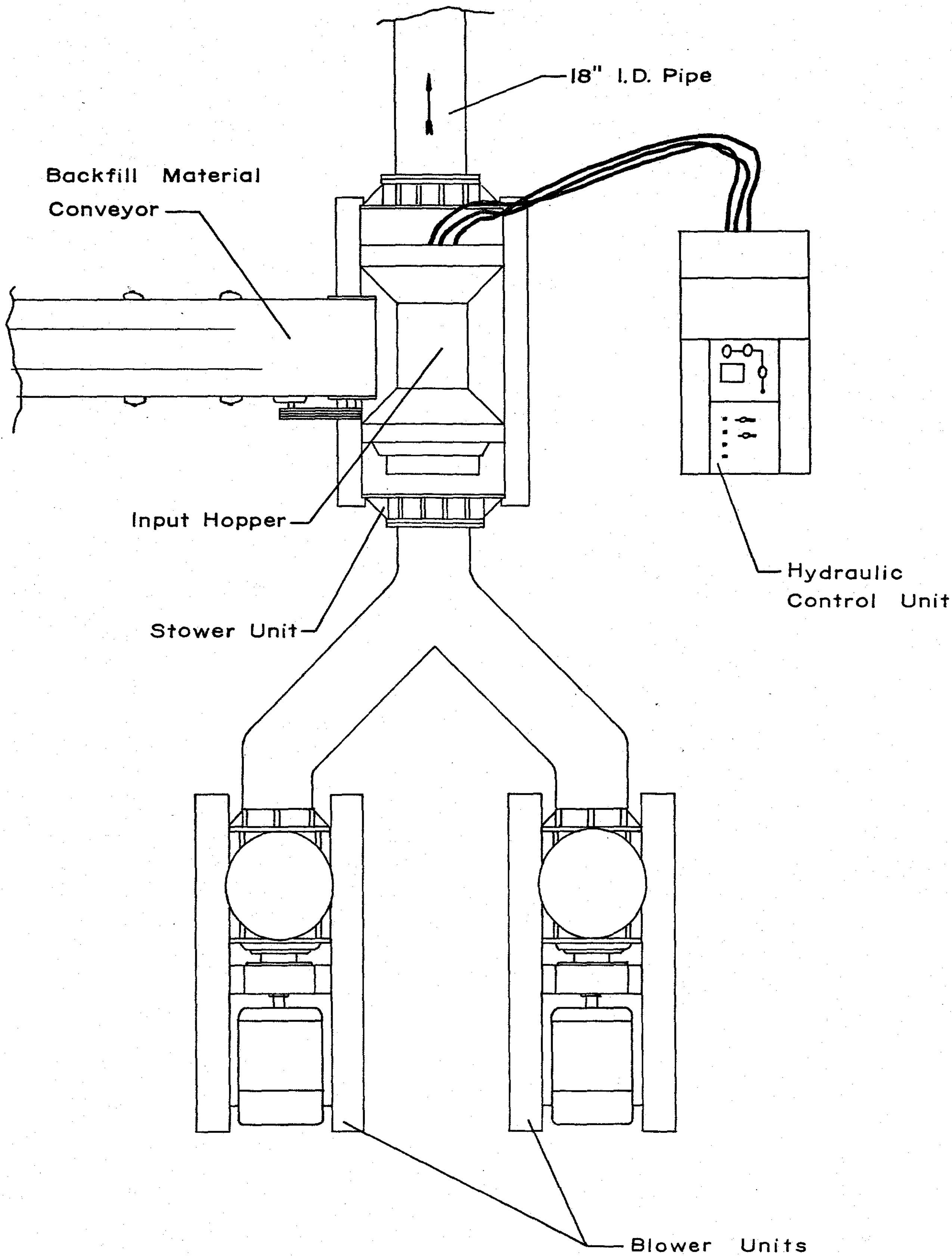
Two pneumatic transport systems will be considered in the ranking and selection of the most likely system for transporting retorted shale from the surface to the backfill area. The first case deals with pneumatic transport from the surface to the backfill area and the second case combines a gravity feed system to the backfill level with pneumatic transport from the borehole to the backfill area.

Pneumatic transport requires the material to be fed into the pipeline through an airtight feeder. The most common feeders are starwheel, screw conveyor, and airlock. The starwheel-type feeder is commercially available and provides continuous feed to the system. Air is the transporting medium and it is provided by high-volume, low-pressure blowers. Equipment and operating data were provided by the Radmark Division of Radar Canada Ltd. This is not to be construed as an endorsement of Radmark's products, but rather as an acknowledgment of assistance in investigating pneumatic transport and stowing methods.

A typical system would include a feeder system with adequate dust control devices, a blower and motor, and a pipeline to carry the material. Current technology limits a unit for the assumed mine conditions to one that is capable of conveying a nominal 200 tons per hour. Each unit includes a feeder, two 8,000-cfm blowers rated at 15 psig, driven by 700-horsepower motors and 18-inch-diameter transport pipe. Thirteen of these units are needed to meet the demands of a total pneumatic system to convey retorted shale. The number of units increases to 19 when considering the mechanical reliability of the system. Figure 7.1.3-1 is a schematic diagram of a typical feeder-blower layout.

Large-diameter pipe and the large number of units will require a highly mechanized method of pipe handling and maintenance. Large openings will facilitate the use of mobile cranes and other pipe-handling equipment. Wear is a severe problem when solids are transported pneumatically and any deviation from straight runs accelerates the rate of wear. It has been reported that pipe life for vertical sections is greater than five times the life for horizontal sections (18).

Energy requirements are extremely high for pneumatic conveying because of the unusually large amount needed for the blowers. Water requirements are lower than for other methods because of the enclosed transport system. Water injection into the pipeline (immediately prior to discharge into the stopes) and good ventilation practices are required. Even so, severe dust conditions are anticipated in the stopes during stowing.



TYPICAL FEEDER - BLOWER LAYOUT

FIGURE 7.1.3-1

No alternative stowing methods were considered in conjunction with pneumatic transport because of the extreme exit velocity and inherent dust problems. Therefore, the pneumatic transport system includes the equipment and labor for backfilling.

Backfilling will be done from the upper level access to the chambers or stopes. All lower level access entries will be bulkheaded and provided with drains should the stopes become saturated by ground water intrusion. Backfilling will progress from one end of the stope or chamber and the pipe and nozzle will be extended as the fill advances. Final filling to the roof will be accomplished by retreating as the fill is placed. An overall in-place density of 75 pounds per cubic foot has been assumed because impact packing will not occur except during placement of the final fill to the roof. Directional nozzles or end pieces can be used to control the filling of stopes.

7.1.3.1 Pneumatic Transport and Stowing:

Pneumatic transport infers that retorted shale will be transported from the surface to the stopes by pneumatic means. Since 19 pipes, 18 inches in diameter, are required, five boreholes, cased to 72-inch I.D. and divided into four compartments, are planned (Figures 7.1.3.1-1 and 7.1.3.1-2).

Severe wear is anticipated where the pipe bends but straight vertical pipe sections are expected to have an effective life of approximately two years or 2,500,000 tons per unit. Horizontal pipe sections will have an estimated life of 0.4 years or 500,000 tons. A special rig for handling vertical pipe changes and repair has been included in the costs for this method of transport.

ADVANTAGES:

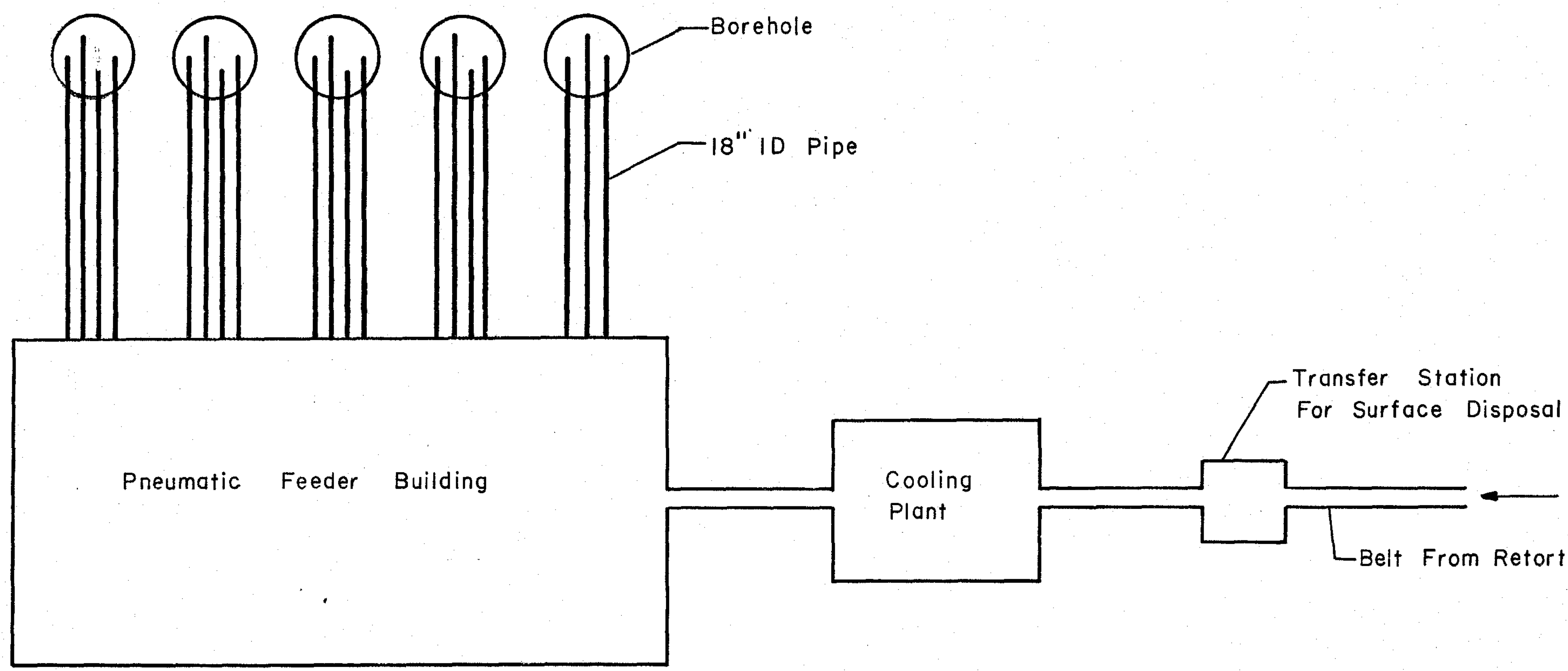
1. Low water requirements

DISADVANTAGES:

1. High energy requirements
2. Fill subject to saturation by ground water
3. High capital and operating costs
4. Large number of units required
5. Severe dust problems

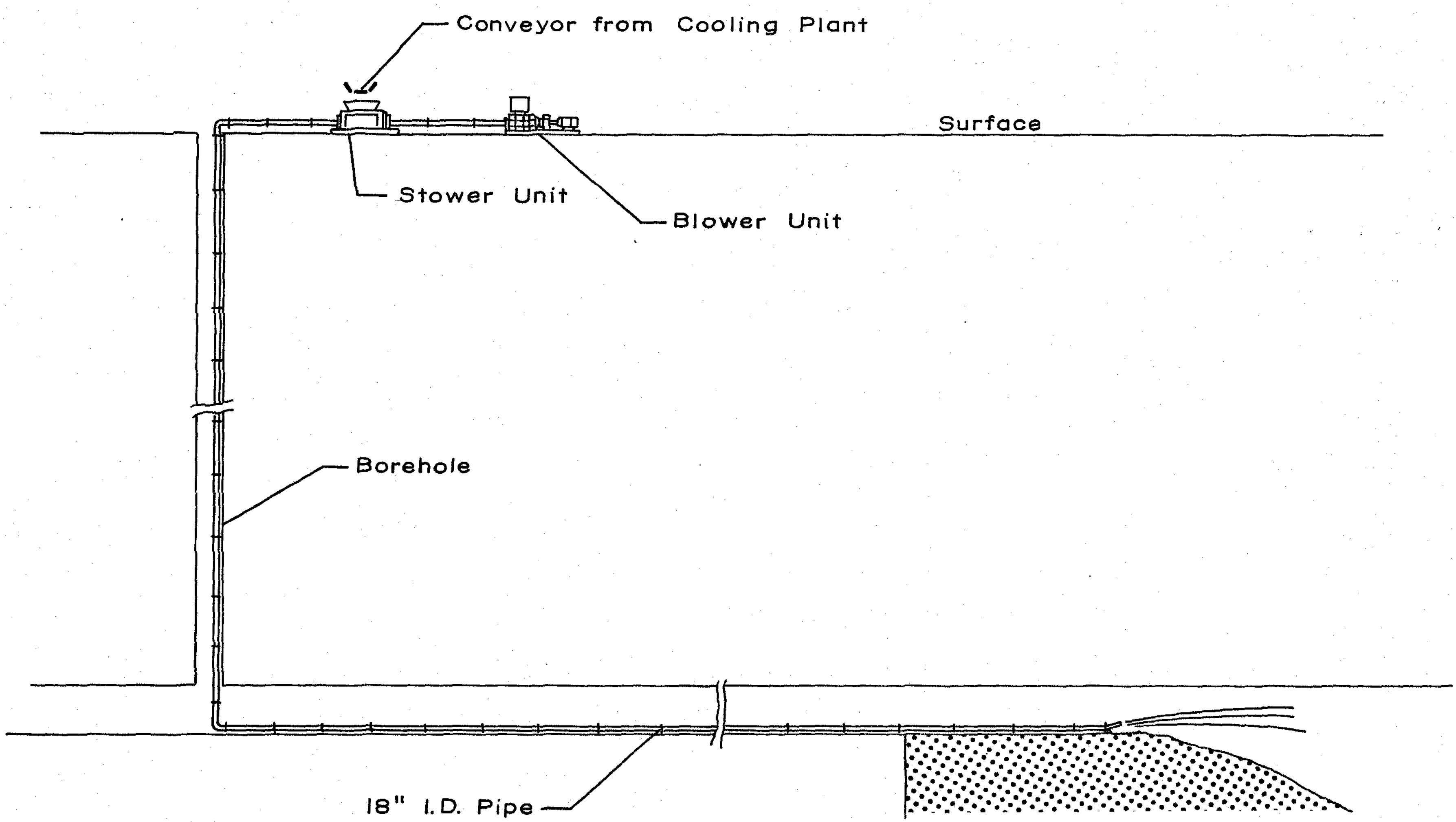
7.1.3.2 Pneumatic Transport and Stowing With Borehole Transport:

This variation of the pneumatic transport method utilizes a large diameter vertical borehole to convey the retorted oil shale to the backfilling level of the mine. An eight-foot I.D. borehole will permit a mass flow rate of approximately 20 feet per minute at the production



SURFACE LAYOUT
COMPLETE PNEUMATIC TRANSPORT SYSTEM

FIGURE 7.1.3.1-1



COMPLETE PNEUMATIC TRANSPORT AND STOWING

FIGURE 7.1.3.1-2

rate required for a 50,000-barrel-per-day operation. The material will be transferred from the borehole to a surge facility prior to being fed into the pneumatic transport system (Figure 7.1.3.2-1).

Once the retorted oil shale is in the pneumatic feeder, the system is identical to a pure pneumatic transport and stowing system. However, due to the addition of a gravity system of mass flow from surface to the backfilling level, the total number of feeder-blower units required is reduced from 19 to 15. A method for clearing the borehole, should it become plugged, will be included in the detailed analysis of Phase II.

The advantages and disadvantages of this system as compared to a pure pneumatic system are the same in most instances. However, the modified system with borehole transport does permit lower capital and operating costs.

7.1.3.3 Summary:

Pneumatic transport methods are possible, but the large amount of energy consumed and the large number of units required adversely affect the feasibility of the methods. The susceptibility of the fill to saturation by ground water is another potential problem source. Table 7.1.3.2-1 shows a comparison of the two pneumatic transport-stowing methods.

7.2 Stowing Methods:

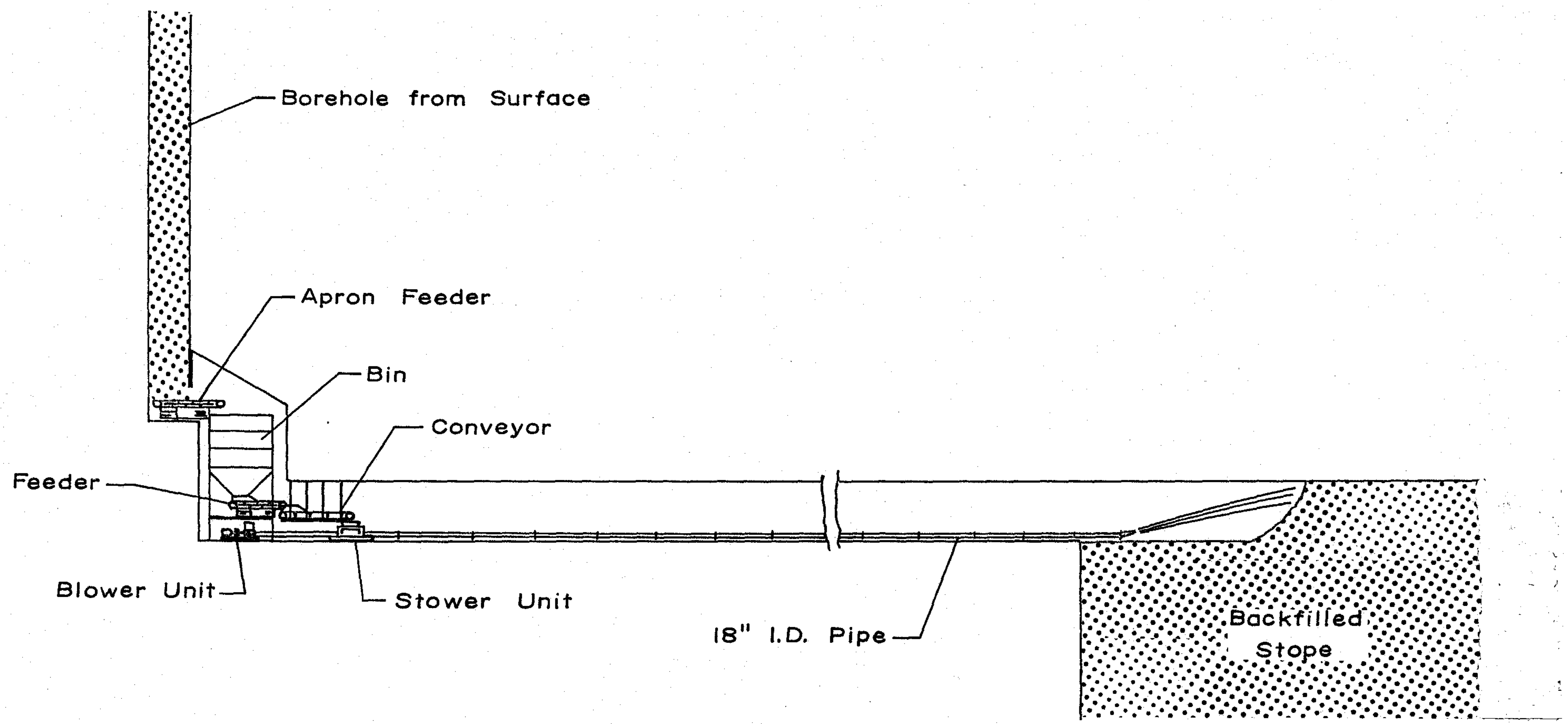
Three methods for stowing retorted oil shale in the mined-out underground areas of a mine were stipulated in the Scope of Work for this contract. They are hydraulic, mechanical, and pneumatic. The individual stowing methods were evaluated as an extension of the same transport method and as combinations of various transport and stowing methods.

The two mining methods that were stipulated (chamber and pillar, and sublevel stoping) are comparable for all three stowing methods, with one exception: mechanical compaction is not possible in the sublevel stopes.

7.2.1 Hydraulic Stowing:

Hydraulic stowing has been considered along with pneumatic and mechanical transport methods, as well as in conjunction with a complete hydraulic transport system. The hydraulic fill material will be placed in the chamber or stope from the upper level access drifts.

As explained in the hydraulic transport system, the retorted shale delivered to the stopes contains approximately 25 percent minus 325 mesh; hence, neither men nor equipment can work on the fill. Thus, the delivery pipe cannot be extended as filling progresses. As a result, coarser particles will drop out first near the end of the pipe, and the rest of the chamber or stope will be filled with the finer fractions. The finer fractions (minus 35 mesh) which constitute approximately 70 percent of the fill



PNEUMATIC TRANSPORT AND STOWING
WITH BOREHOLE TRANSPORT

FIGURE 7.1.3.2-1

TABLE 7.1.3.2-1

COMPARISON OF PNEUMATIC TRANSPORT METHODS

	Complete Pneumatic	Pneumatic & Borehole
Capital Costs ($\$ \times 10^3$)	\$ 60,833	\$ 43,295
Operating Costs (\$/Ton)	\$ 1.1855	\$ 1.0117
Energy Used (KWH/Day $\times 10^3$)	983	877
Water Used (Gal/Day $\times 10^3$)	874	874
Crew Size	380	353
Retorted Shale-Surface Disposal (%)	30	30
Fill Density (pcf)	75	75
Resource Recovery Increase (%)	13	13

material, reduce the permeability of the fill to as low as 0.001 feet per day. The percolation rate for hydraulic fill should be greater than 12 feet per day (3).

By means of laboratory experiments the hydraulic characteristics of Paraho retorted shale were investigated (4). Large scale drainage tests were conducted using plywood boxes measuring four feet by four feet by eight feet high. Vertical flow drainage and horizontal percolation were simulated in these tests. In addition, a column settling test was performed using a four-inch I.D. column. Test results indicated the following:

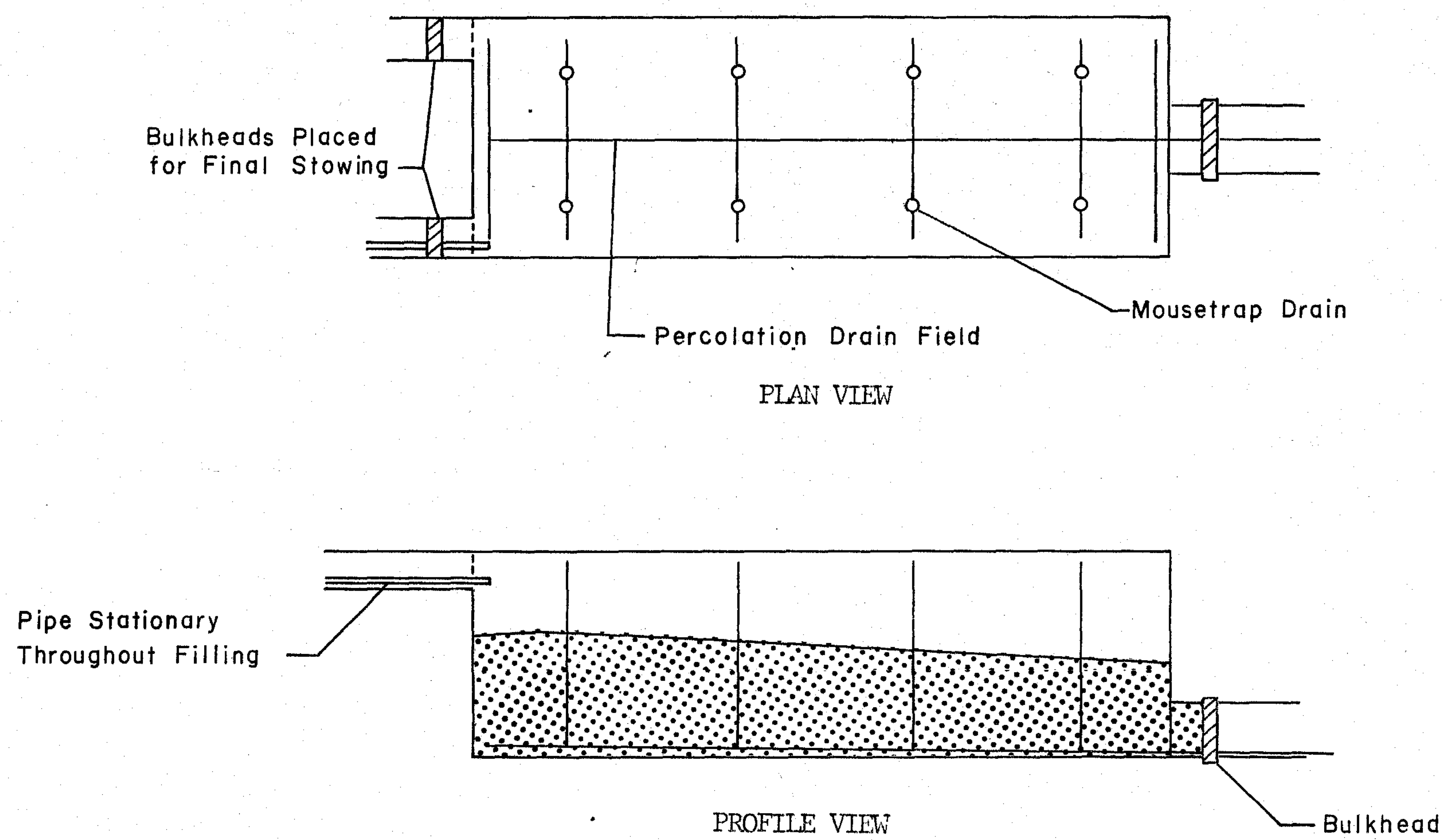
1. 65-70 percent of the water can be drained from the fill.
2. Bulk density of the drained material on a dry basis is 65 pounds per cubic foot.
3. Loss of chemical constituents due to water leaching is negligible.
4. Rate of settling declines rapidly and the percent solids in the column shows an irregular pattern.

Details of these tests and the results obtained are given in Appendix C.

Complete filling using hydraulic stowing is difficult to achieve. To minimize the effects of surface subsidence, a combined hydraulic stowing with penumatic topfilling system has been considered. Combinations of hydraulic stowing that are analyzed are as follows:

1. Hydraulic stowing combined with hydraulic transport
2. Hydraulic-pneumatic stowing combined with hydraulic transport
3. Hydraulic stowing combined with truck transport
4. Hydraulic-pneumatic stowing combined with truck transport
5. Hydraulic stowing combined with conveyor transport
6. Hydraulic-pneumatic stowing combined with conveyor transport

Hydraulic stowing requires that bulkheads be placed in all lower entries leading to the chambers and stopes. Bulkheads will be well constructed and will be capable of sustaining the hydrostatic pressures imposed by the fill. Graded mine-run material and concrete will be used to impound fill in the stopes. Fill drainage will be accomplished by decantation and percolation. For chamber stowing, mousetrap drains (which are perforated pipes covered with burlap) will be used in conjunction with a percolation drain field (Figure 7.2.1-1). Percolation drains will be spaced at intervals in the chambers to obtain maximum fill drainage.



HYDRAULIC STOWING
CHAMBER AND PILLAR MINING

FIGURE 7.2.1-1

Stopes will be dewatered mainly by decantation with a vertical drain pipe placed at each end of the stope (Figure 7.2.1-2). Monitoring piping and valves will be installed in each bulkhead so that subsequent hydrological conditions can be regularly checked. Water drained from the fill will be channeled to a settling sump and the clear effluent will be pumped to the surface. Laboratory tests indicate that a negligible amount of chemical constituents are leached from the fill; hence, treatment of drained water may not be necessary (4). However, a provision will be made to collect and test the drained water periodically.

7.2.1.1 Hydraulic Stowing Combined With Hydraulic Transport:

Level distribution lines will transport the fill from the main pipeline to the chambers and stopes. Ten-inch-diameter pipes, with a life of approximately 2,000,000 tons of fill, will be used for this purpose. The line will enter the chamber or stope at one end and will remain there until filling is completed. If the flow rate diminishes appreciably, the discharge end of the pipe will be promptly closed to prevent the loss of water from the hydraulic fill mixture in the pipe. The pipe can then be flushed in short lengths to clean out the plug.

Control of the filling operation will be accomplished with a reliable telephone system between fill operators. Cleanup of spills will be done mechanically with LHD's, or the spill material will be slurried and pumped. A limited amount of laboratory work performed on hydraulically pumped, retorted shale indicates that the addition of flocculants and cementing agents does not appreciably improve the strength of the fill (Appendix D). Hence, these agents will not be added to the fill before stowing.

ADVANTAGES:

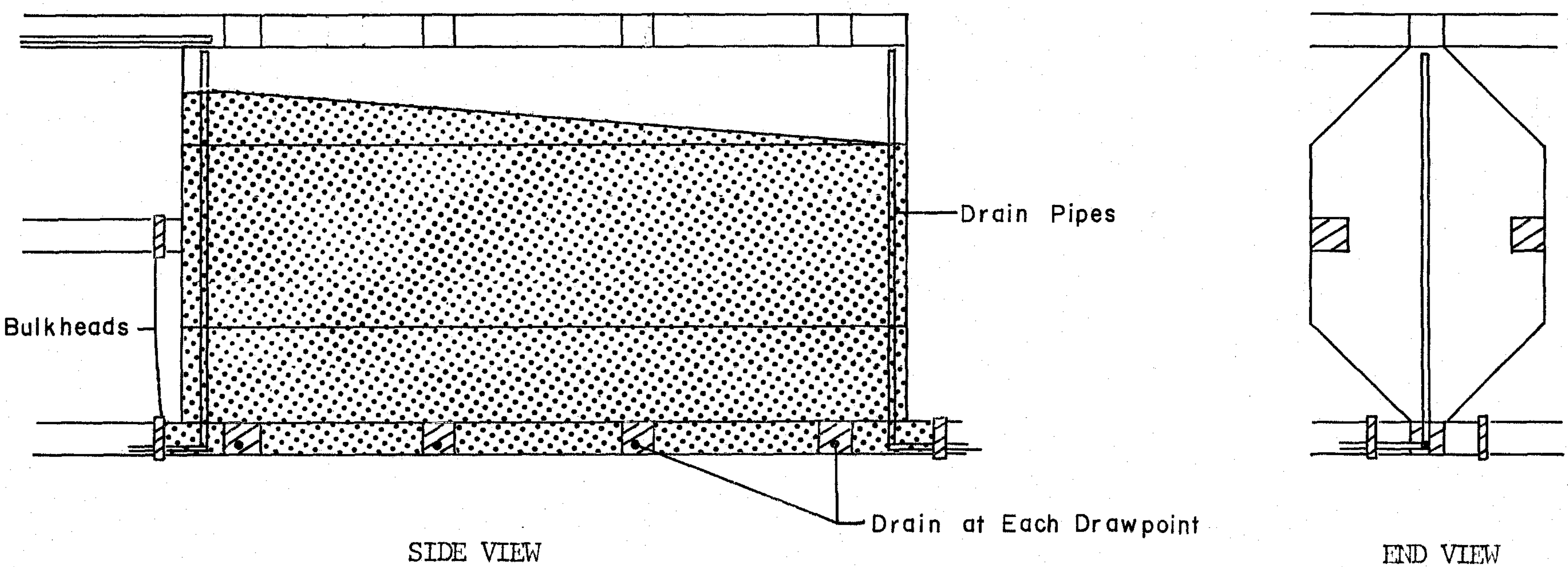
1. No dust potential

DISADVANTAGES:

1. Danger of liquefaction and mud flows
2. Increased humidity of mine air
3. Need for handling facilities for drained water
4. Unstable fill surface
5. Expanded surface disposal need due to low fill density
6. Slimes ponds requirement

7.2.1.2 Hydraulic-Pneumatic Stowing Combined With Hydraulic Transport:

In this combination method, final filling of the stope or chamber will be done pneumatically. Because the hydraulic fill is



HYDRAULIC STOWING
SUBLEVEL STOPING

FIGURE 7.2.1-2

unstable the pneumatic discharge nozzle must remain stationary at one end of the stope and spray the material over the hydraulic fill. This results in incomplete stope filling. However, the task of dewatering the hydraulically transported material prior to pneumatic stowing would require a large bank of filters. In addition, the adverse effect of moist feed from filters on energy consumption and productivity would be severe. Hence, this method is not feasible from operational and cost standpoints.

ADVANTAGES:

1. None

DISADVANTAGES:

1. High energy consumption
2. Large filter requirement
3. Potential for liquefaction and mud flows
4. Additional equipment need for pneumatic stowing
5. Handling facilities need for drained water
6. Increased ventilation requirements
7. Unstable fill surface
8. Expanded surface disposal need due to low fill density
9. Slimes ponds requirement

7.2.1.3 Hydraulic Stowing Combined With Truck Transport:

The problem of particle degradation discussed under hydraulic transport, Section 7.1.1, is no less severe in this instance since slurrying and pumping operations are principal sources of particle breakdown. Retorted oil shale is delivered by truck to a surge bin at the access entry to stope or chamber being backfilled. Water sprays and a dust collector system control the dust at the transfer points. The material is then slurried, cycloned, or centrifuged to remove excess slimes and transported to the stope or chamber for stowing. The stowing procedure is the same as in Section 7.2.1.1. The excess slimes are pumped to surface slimes ponds for disposal.

ADVANTAGES:

1. Minimal dust problem

DISADVANTAGES:

1. Potential for liquefaction and mud flows
2. Space requirement for transport-stowing interface
3. Unstable fill surface

4. Increased ventilation requirements

5. High operating costs

6. Increased manpower

7. Expanded surface disposal need due to low fill density

8. Slimes ponds requirement

7.2.1.4 Hydraulic-Pneumatic Stowing Combined With Truck Transport:

This combination method is comparable to Section 7.2.1.3 with the exception that final filling of the stope or chamber will be done pneumatically. Complete filling of the stope is not possible because the discharge line cannot be extended onto the hydraulic fill. Increased costs and operating problems are disproportionate to the results obtained.

ADVANTAGES:

1. None

DISADVANTAGES:

1. High energy consumption

2. High operating costs

3. Potential for liquefaction and mud flows

4. Increased ventilation requirements

5. Expanded surface disposal need due to low fill density

6. Slimes ponds requirement

7.2.1.5 Hydraulic Stowing Combined With Conveyor Transport:

This method is comparable to hydraulic stowing in conjunction with truck transport, Section 7.2.1.3, with the following exceptions:

(1) Lesser space requirements at the conveyor surge bin transfer point

(2) Lower manpower and energy requirements for conveyor transport

(3) Less expensive dust control at transfer points

ADVANTAGES:

1. Minimal dust

DISADVANTAGES:

1. Potential for liquefaction and mud flows
2. Space requirement for transport-stowing interface
3. Unstable fill surface
4. Increased ventilation requirements
5. Expanded surface disposal need due to low fill density
6. Slimes ponds requirement

7.2.1.6 Hydraulic-Pneumatic Stowing in Conjunction With Conveyor Transport:

This combination method is comparable to Section 7.2.1.5 with the exception that final filling of the stope or chamber will be done pneumatically. Complete stope filling is not possible because men and equipment cannot work on the hydraulic fill. Expanded costs and operating difficulties make this method unattractive.

ADVANTAGES:

1. None

DISADVANTAGES

1. Potential for liquefaction and mud flows
2. Space requirement for transport-stowing interface
3. Increased ventilation requirements
4. High energy consumption
5. High operating costs
6. Unstable fill surface
7. Expanded surface disposal need due to low fill density
8. Slimes ponds requirement

7.2.1.7 Summary:

Hydraulic stowing underground may be hazardous due to the potentially dangerous liquefaction characteristics of the stowed material. This circumstance stems from the undesirable dewatering characteristics of the fill resulting from the excessive amount of fine particles contained in the stowed shale. Men and equipment cannot work on the fill; thus, there is no control over the quality of fill. In addition, approximately 35 percent of the retorted shale has to be disposed in slimes ponds which cause greater environmental disturbance than complete surface disposal.

Most particle degradation occurs during slurrying and pumping and is less dependent on transport distance. Hence, hydraulic stowing in conjunction with other transport methods has the same drawbacks as hydraulic stowing in conjunction with hydraulic transport, apart from system interface problems. Retorted shale cannot be pumped at a concentration much over 40 percent solids by weight; hence, water requirements for this method are excessive. In addition, over 30 percent of the water used for transport is retained in the fill due to its poor dewatering qualities.

Pneumatic topfilling, combined with hydraulic stowing, provides incomplete stope filling. Also, the few benefits obtained do not justify the increased costs and operating difficulties.

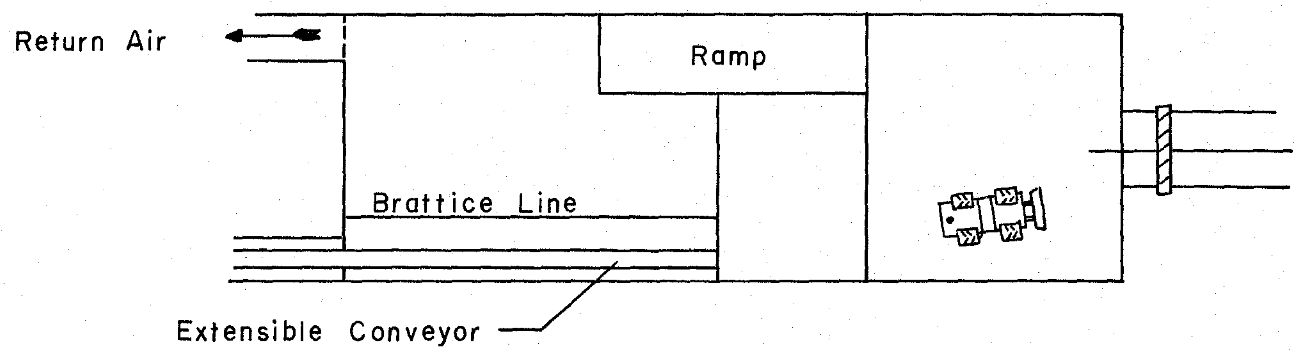
7.2.2 Mechanical Stowing:

The two mechanical methods considered for transport were also considered for stowing retorted oil shale in mined-out stopes. In either case, retorted shale will be deposited in stopes from the upper level access drifts. Mechanical compaction of the backfill is possible with chamber and pillar mining because of the completely bolted chamber roof, but no mechanical compaction of the backfill is possible in the unsupported open stopes of sublevel stoping operations. The stowing and compacting activities will alternate since dust problems during stowing may be too severe for concurrent operations within a single stope. Water will be added at the discharge point to bring moisture content up to five to six percent by weight. Water is used primarily for dust suppression since studies have shown that the density for a given compactive effort is not significantly affected by the addition of water (14). Compacted density will be about 90 pounds per cubic foot, and uncompacted density will be about 80 pounds per cubic foot.

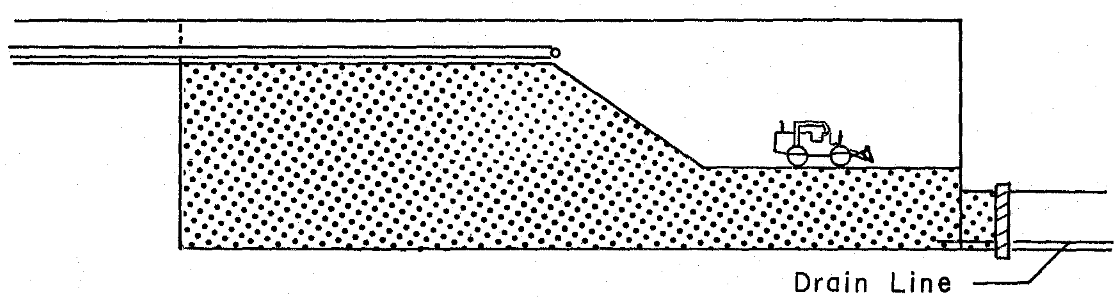
Complete filling will be difficult to achieve if a mechanical stowing method is used. Mechanical slingers have been used in practice, but they were not considered for this comparative study. Combined mechanical stowing with pneumatic topfilling was considered. Geotechnical studies have shown that the degree of subsidence does not increase appreciably between areas that are filled to the roof and areas where a gap is left between the backfill and the roof.

7.2.2.1 Conveyor Stowing:

For conveyor stowing, retorted shale will be picked up at a transfer point at the stope access entry, conveyed to the discharge points, discharged into the stope, and compacted whenever possible. The conveyor will be advanced into the stope as stowing progresses. To ensure continuity of the backfilling operation, several stopes will be backfilled concurrently. Conveyor stowing was studied in conjunction with conveyor and hydraulic transport methods, in addition to the combined conveyor-pneumatic stowing methods for the same transport methods. See Figures 7.2.2.1-1 and 7.2.2.1-2 for typical layouts for conveyor stowing and the mining methods being studied.



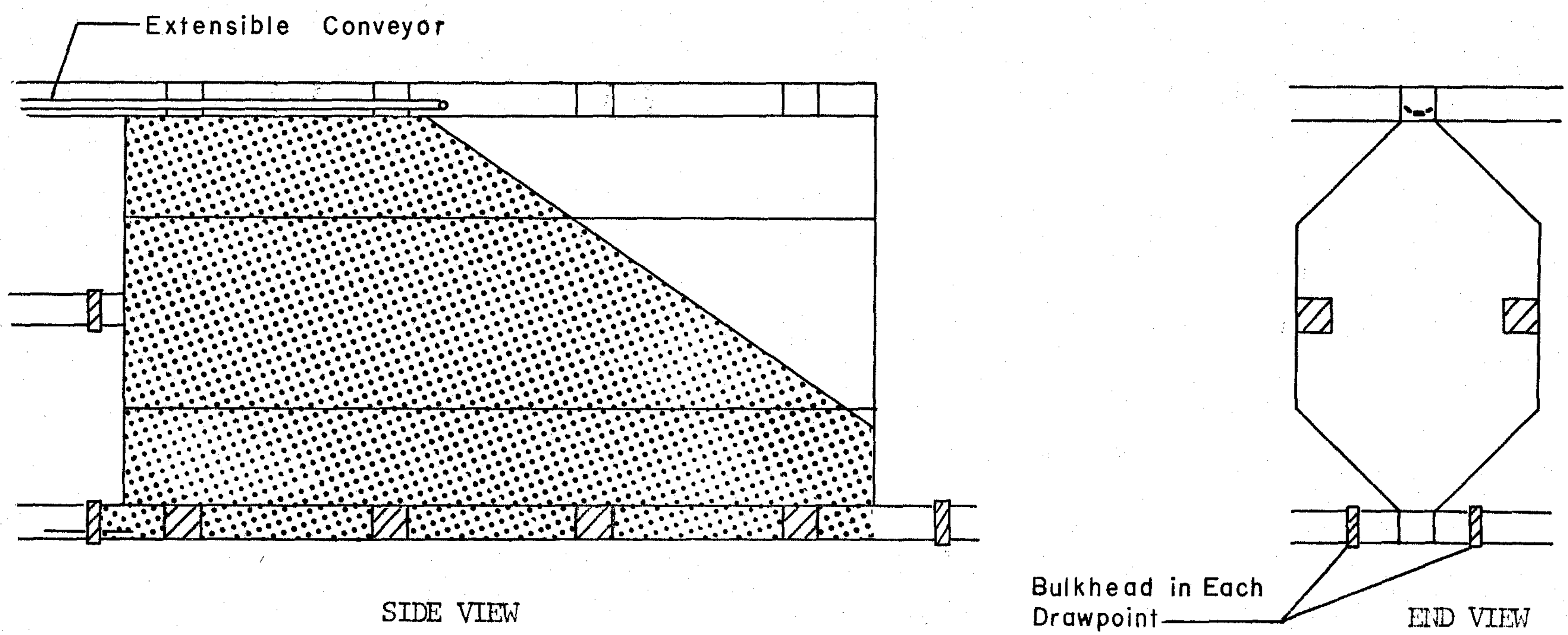
PLAN VIEW



PROFILE VIEW

<u>CONVEYOR STOWING</u>
<u>CHAMBER AND PILLAR MINING</u>

FIGURE 7.2.2.1-1



CONVEYOR STOWING
SUBLEVEL STOPING

FIGURE 7.2.2.1-2

7.2.2.1.1 Conveyor Stowing With Conveyor Transport:

In this system, retorted oil shale is taken from the main conveyor by a tripper and splitter that discharges the required quantity of material onto the stowing conveyor and returns the excess back onto the main conveyor. Water sprays and a dust hood will control dust emission at transfer points. In stopes, a compactor will be used to level and compact the stowed material. Stowing and compacting will alternate between stopes, as required, to effectively backfill each stope. Extensible belts will be used in stopes to facilitate conveyor advance and retreat during the stowing operation. By using modified dozers, backfill material will be packed as near to the roof as possible.

ADVANTAGES:

1. Good pillar support from high density fill
2. Low manpower requirement
3. High resource recovery
4. Favorable energy requirements
5. Moderate water requirements
6. Low costs
7. Well advanced technology

DISADVANTAGES:

1. Dust during stowing
2. Inability to pack against roof

7.2.2.1.2 Conveyor Stowing With Hydraulic Transport:

Retorted oil shale subsequent to initial cycloning or centrifuging degrades during hydraulic transport to the extent that approximately 25 percent of the material reaching the stopes will be minus 325 mesh. The task of dewatering the material prior to stowing will require large banks of filters which would create an untenable operating situation. Thus, the method is not feasible but was considered in order to satisfy contract stipulations that hydraulic transport be studied in combination with other stowing methods; however, this combination is not desirable under any condition.

ADVANTAGES:

1. Minor dust potential

DISADVANTAGES:

1. High water usage

2. Large filtering capacity requirement

3. Surface slimes ponds need

4. Incompatible methods

5. Need for complex pumping and water treatment facilities

6. High energy consumption

7. Excessive space requirements for equipment

7.2.2.1.3 Conveyor-Pneumatic Stowing With Conveyor Transport:

This combination stowing method consists of stowing by means of conveyors as described in Section 7.2.2.1.1 up to the floor level of the access drift and backfilling the remaining space pneumatically. The pneumatic discharge line will be extended to the inby end of the stope and stowing will follow a retreating pattern. Pneumatic stowing of the upper portion of the stope allows for complete filling and a compacted density of about 80 pounds per cubic foot. Reduction of surface subsidence is possible with complete filling of stopes and chambers. However, initial investigations indicate that the improvement in subsidence control will not warrant the increased costs and operating problems.

ADVANTAGES:

1. Complete stope filling with compacted material

2. Potentially good subsidence control

DISADVANTAGES:

1. Additional equipment requirement for pneumatic stowing

2. High energy consumption for pneumatic stowing

3. Increased dust problems

7.2.2.1.4 Conveyor-Pneumatic Stowing With Hydraulic Transport:

This combination stowing method has all the disadvantages described in Section 7.2.2.1.2 in addition to those associated with pneumatic stowing. The efficiency of the pneumatic system is low because of additional energy required to move high moisture material from the filter operation. The disadvantages of this method far outweigh any advantages that may exist.

ADVANTAGES:

1. Minimal dust problems

2. Complete stope filling

DISADVANTAGES:

1. High water usage
2. Large filtering capacity requirement
3. Surface slimes ponds need
4. Incompatible methods
5. High pumping and water treatment costs
6. High energy consumption
7. Excessive space requirements for equipment
8. Lowered efficiency of pneumatic system due to wet material

7.2.2.2 Truck Stowing:

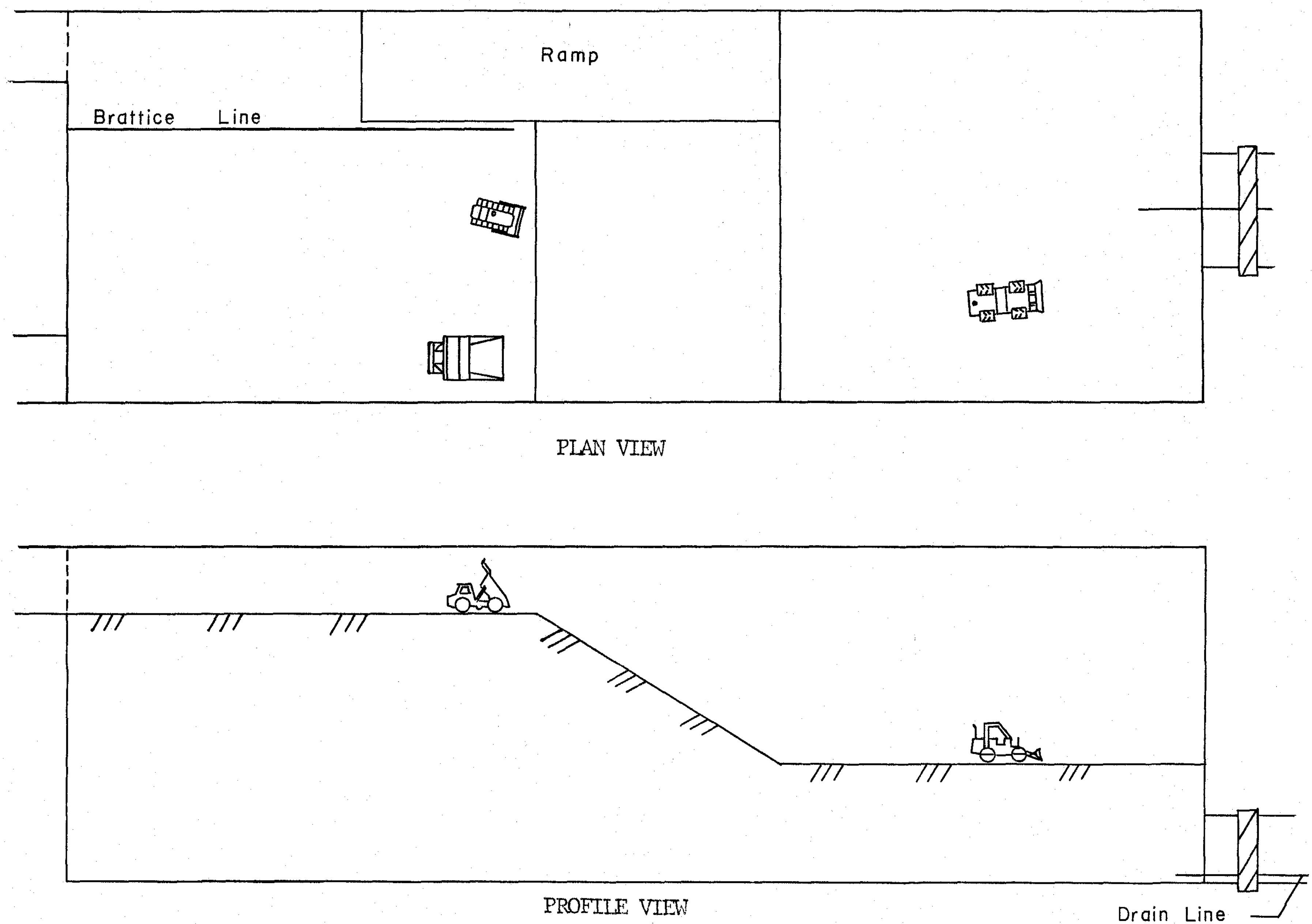
Trucks were considered for stowing in conjunction with truck transport. Conveyor transport with truck stowing was not investigated in depth because of excessive space required for surge bins and truck maneuvering. Hydraulic transport with truck stowing created even greater space and operating problems. The two cases involving truck stowing that were considered are truck stowing and a combination of truck stowing plus pneumatic topfilling. Figures 7.2.2.2-1 and 7.2.2.2-2 show the layouts for both mining methods being studied.

7.2.2.2.1 Truck Stowing With Truck Transport:

Trucks will be loaded at the borehole as described in Section 7.2.2.2 and will travel to the stope being filled. Retorted oil shale will be dumped on top of the fill and pushed over the edge by a dozer. Spreading and compacting probably will not be done while dumping is in progress. While one stope is being filled another will be undergoing compaction. These cycles will alternate until the stopes have been filled, but complete filling to the roof will be difficult to attain by this method. However, modified dozers will be used to pack the backfilling material as close to the roof as possible.

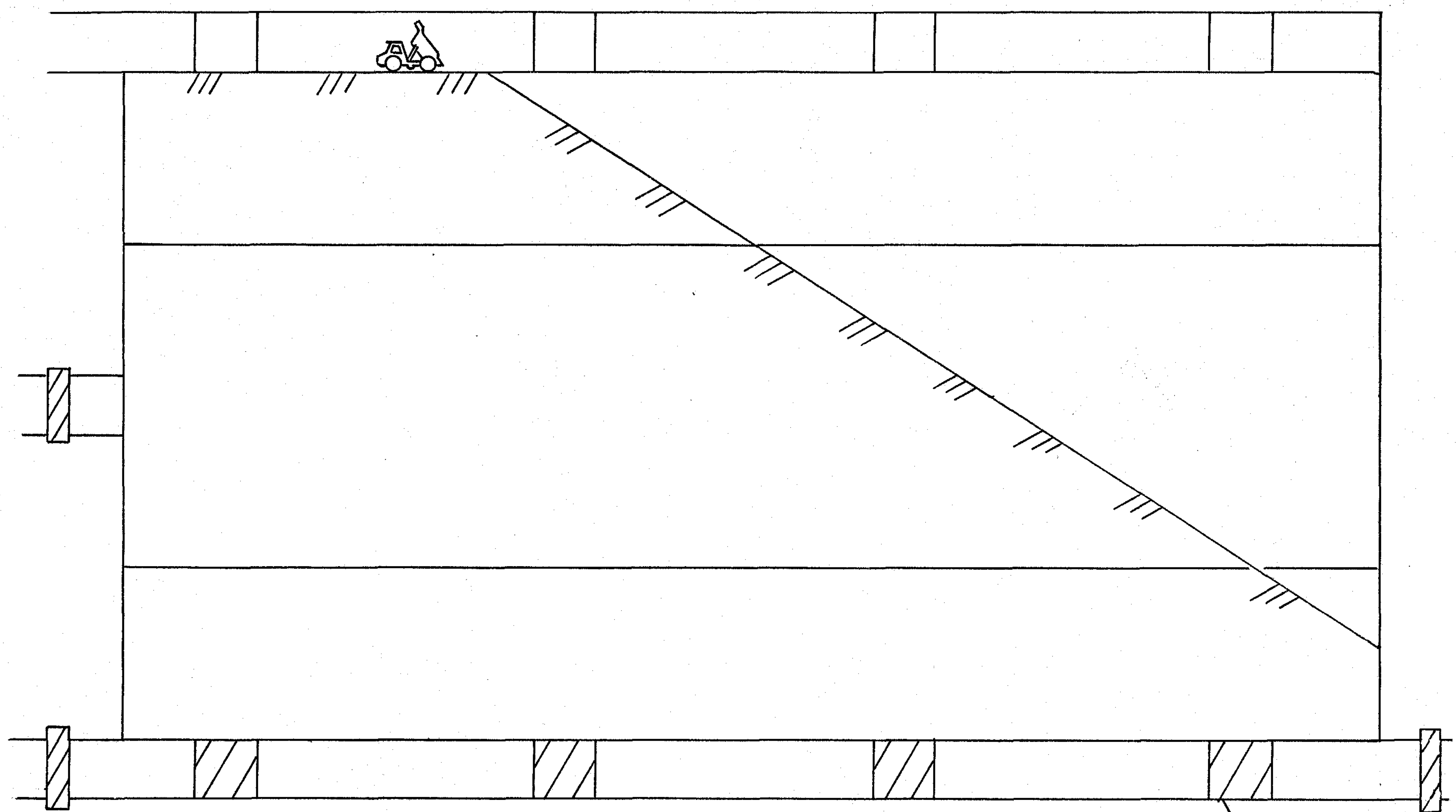
ADVANTAGES:

1. Continuous operation - no interface between transport and stowing
2. Good pillar support from high density fill
3. Increased resource recovery



TRUCK STOWING
CHAMBER AND PILLAR MINING

FIGURE 7.2.2.2-1



PROFILE VIEW

TRUCK STOWING
SUBLEVEL STOPING

FIGURE 7.2.2.2-2

DISADVANTAGES:

1. High energy consumption
2. Large manpower requirements
3. Increased ventilation requirements
4. Dust in stopes
5. Difficulty of packing against roof
6. Hazards of high dumps and fine material

7.2.2.2.2 Truck-Pneumatic Stowing With Truck Transport:

In this combination method, final filling of the stopes will be done pneumatically. Dumping and surge bin facilities are required in addition to the pneumatic stowing equipment. The stowing procedure is the same as that described in Section 7.2.2.1.3. This method increases the amount of equipment and men required for stowing with little improvement in overall subsidence control.

ADVANTAGES

1. Complete stope filling with compacted material
2. Some potential for increased subsidence control

DISADVANTAGES:

1. High energy consumption
2. High manpower requirements
3. Increased ventilation requirements
4. Dust suppression difficulties
5. Additional equipment requirement
6. Space requirement for transport-stowing interface activities
7. Hazards of high dumps and fine material

7.2.2.3 Summary:

The most desirable mechanical stowing methods minimize system interface problems. Conveyors generally have lower operating costs, lower manpower requirements, consume less energy, and are technically feasible for the job at hand. They are less flexible than other methods and some time is lost while extending them into the stopes, though the use of extensible belts reduces the seriousness of the problem.

Truck-related methods are high energy consumers and require a relatively large work force. As a result, systems involving the use of trucks are less desirable than other methods.

Pneumatic topfilling does provide complete stope filling, but preliminary findings are that subsidence control may not be improved significantly. The advantage of complete backfilling is offset by increased costs and greater dust problems.

7.2.3 Pneumatic Stowing:

Pneumatic stowing in combination with another transport or stowing method will require a feeder-blower installation at the transition point. This system will include a feeder, a 16-inch-diameter pipe, and a 17,000-cfm blower rated at 15 psig and powered by a 1,400-horsepower motor. A total of seven units, each rated at 400 tons per hour, will be required for a total pneumatic stowing system. Stowing will start at the outby end of the stope and the discharge line will be advanced as necessary. The in-place density of the backfill will be about 75 pcf. Upon completion of stowing to the level of the access entry, the system will retreat as final stowing to the roof takes place. The in-place density for topfilling will be about 80 pcf.

Pneumatic stowing will create a large amount of dust which will be partially alleviated by water injection near the discharge end of the pipeline. Although fugitive dust will be vented to the return air course, men will not be permitted in the stope or in the immediate return air course during stowing operations.

7.2.3.1 Pneumatic Stowing With Conveyor Transport:

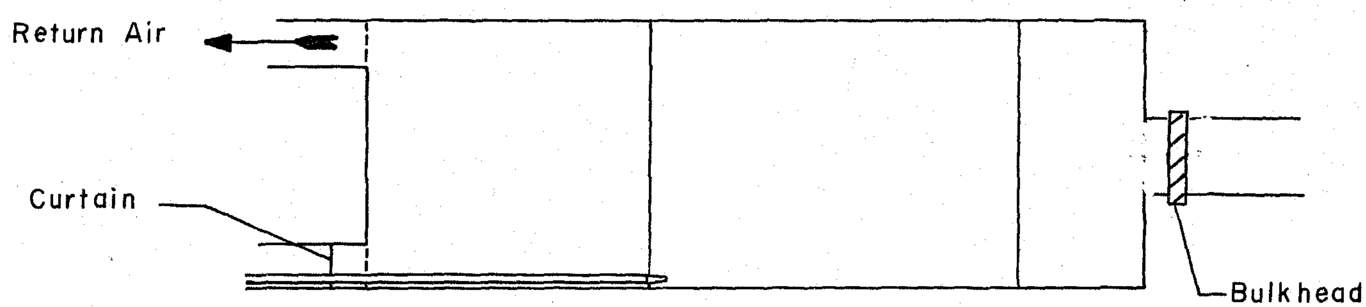
For this system, retorted oil shale will be delivered by a belt conveyor to a surge bin at the access entry to the stope being backfilled. Water sprays and a dust collector system will control dust at the transfer points where the material is fed into the pneumatic system and transported into the stope for stowing. Figures 7.2.3.1-1 and 7.2.3.1-2 show general views of the in-stope layout during pneumatic stowing.

ADVANTAGES:

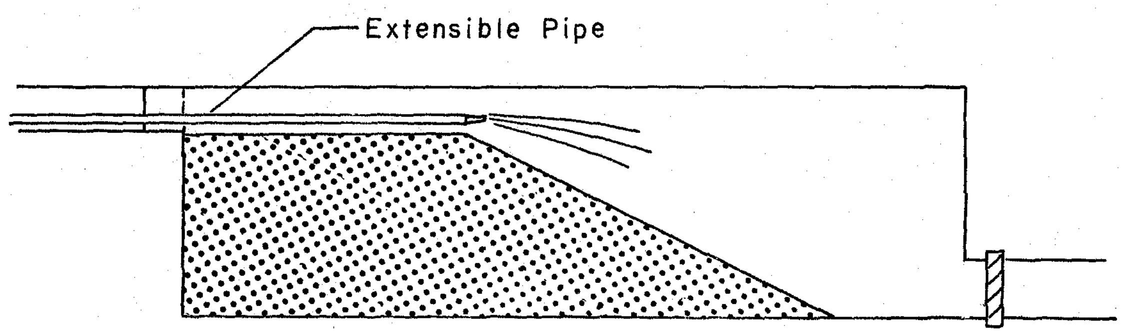
1. Complete stope filling

DISADVANTAGES:

1. Severe dust conditions
2. Potential for ground-water saturation of fill
3. High energy consumption for stowing activity
4. Need for large number of units
5. High operating costs



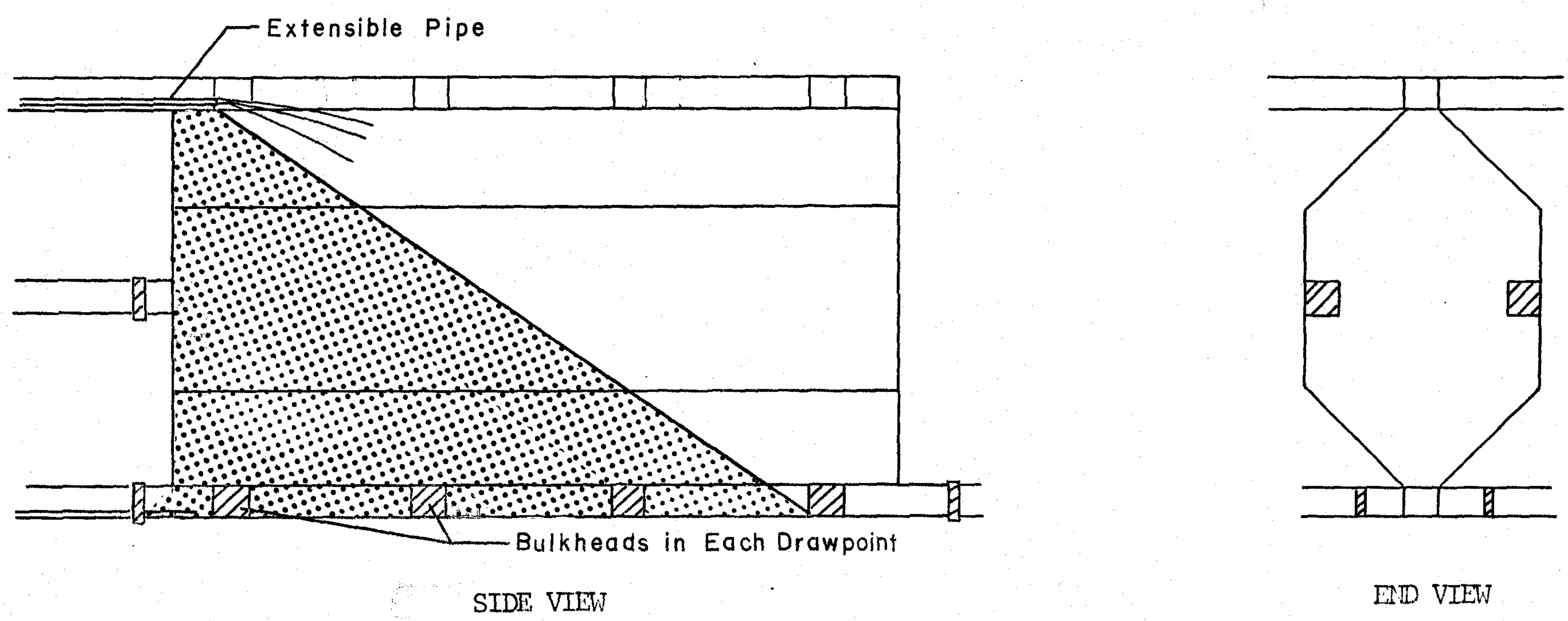
PLAN VIEW



PROFILE VIEW

<u>PNEUMATIC STOWING</u>
<u>CHAMBER AND PILLAR MINING</u>

FIGURE 7.2.3.1-1



PNEUMATIC STOWING
SUBLEVEL STOPING

FIGURE 7.2.3.1-2

6. Greater surface disposal area requirements compared to mechanical backfilling

7.2.3.2 Pneumatic Stowing With Truck Transport:

The combined system is comparable to pneumatic stowing with conveyor transport which was described in Section 7.2.3.1, with these exceptions:

a) More space will be required for truck dumping and surge bin facilities.

b) Overall manpower and energy consumption will be increased due to the requirements of a truck haulage system.

ADVANTAGES:

1. Complete stope filling

DISADVANTAGES:

1. Severe dust conditions
2. Potential for ground-water saturation of fill
3. High energy consumption
4. High manpower requirements
5. Need for large number of units
6. High operating costs
7. Greater surface disposal area requirement compared to mechanical backfilling

7.2.3.3 Pneumatic Stowing With Hydraulic Transport:

The operational difficulties associated with dewatering retorted oil shale from the hydraulic transport system are discussed in Section 7.2.2.1.2, and are equally severe in the instance of pneumatic stowing. In addition, adverse effects on energy consumption and productivity due to the moist condition of the feed from the filters are more severe. Thus, the method is not feasible from cost and operating points of view.

ADVANTAGES:

1. Complete stope filling
2. Minimal dust

DISADVANTAGES:

1. High water usage
2. High energy consumption
3. Large filtering facility requirement
4. Surface slimes ponds need
5. Extensive pumping and water treatment facilities requirement
6. Excessive space requirements for equipment
7. Greater surface disposal area requirements compared to mechanical backfilling

7.2.3.4 Summary:

Pneumatic stowing makes possible complete stope filling, but this principal advantage is offset by several disadvantages. Pneumatically placed fill has relatively low in-place density, is more susceptible to ground-water saturation, does not provide as much support for pillars, and requires that more of the material be placed on surface dumps. In addition, the pneumatic system generates large quantities of dust, it consumes a disproportionate amount of energy for the volume of material moved, and it requires a relatively large number of stowing units.

7.3 Surface Disposal:

Regardless of the backfilling method used, a portion of the retorted oil shale must be stored on the surface. This circumstance stems from expansion that occurs during mining and processing of raw shale, resulting in an excess volume of waste material that must be stored on the surface.

Retorted oil shale degrades when slurried and pumped, resulting in an excessive amount of slimes that will not settle out in the stopes. The slimes are separated by cyclones or centrifuges and pumped to the surface, thus, all hydraulic transport or stowing systems will require the use of surface slimes ponds. To handle the surplus material during temporary slowdowns or stoppages in the underground disposal operation, a small surge facility will be needed near the surface disposal system truck loading area.

7.3.1 Stockpiling Method:

Surface stockpiling of retorted shale is discussed in Section 4.

7.3.2 Slimes Ponds:

When retorted shale is transported hydraulically it degrades rapidly. By the time it reaches the stope as much as 55 to 60 percent of the material has degraded to a minus 325-mesh size. The slurry is then cycloned or centrifuged to remove more than half of the minus 325-mesh material prior to stowing. The rejected slimes will be pumped to a surface disposal site at a maximum rate of 10,850 gpm. At the production rates used in this study, and assuming that half of the water can be reclaimed, the volume of slurry deposited will amount to approximately 9,700 acre-feet per year. Test work has shown that flocculants are only marginally effective for slurry dewatering (Appendix D). Preliminary tests also indicate that retorted oil shale slurry is not amenable to electrokinetic densification and that chemical additives have not been effective for improving results in which the material has high pH and calcium content (22). The nature and extent of the slimes ponds are two of the primary reasons for rejecting the hydraulic system as a feasible method for transporting retorted shale.

7.4 Environmental Effects:

7.4.1 Land Disturbance:

The most obvious surface environmental effects are those associated with structures or activities which alter the appearance of the land area. Surface disposal dumps, roads, conveyors, other material handling structures, and slimes ponds are the most common surface features that may be related to a retorted oil shale disposal operation. Slimes ponds are associated only with hydraulic disposal systems. Surface dumps, structures, subsidence, and air quality are discussed in Section 5.0.

7.4.1.1 Ponds:

Surface ponds are required only if hydraulic transport and stowing methods are used. The Paraho retorted shale degrades when slurried and pumped to the extent that 55 to 60 percent of the material is minus 325 mesh. The slimes must be removed first if most of the water is to be separated from the material after stowing. The slimes are removed by cyclones or centrifuges and pumped to surface ponds for disposal. Section 7.5.2 states that approximately 9,700 acre-feet of slime storage capacity will be required each year. This material will never completely drain so that the ponds will be virtually impossible to reclaim. This severe long-range environmental effect is one of the principal objections to hydraulic transport and stowing methods.

7.4.2 Water:

The effects of a retorted shale operation on surface water quality will vary with the disposal system used. Some runoff flow from canyons that are used for dumps will be intercepted. Slimes ponds that are associated with hydraulic transport and stowing methods present the greatest threat for surface water contamination.

Catchment basins will be used to contain all saline leachate from dumps and slimes ponds. It will probably be necessary to line the slimes ponds with relatively impermeable material to prevent infiltration of low quality water into the ground-water system (4 and Appendix C).

Results from tests on decant water from drainage experiments using pumped, retorted shale slurry indicate total dissolved solids to be about 2,000 mg/1 (4).

The hydrologic conditions at the mine site and the possible effects of backfilling on the ground-water system are discussed in Section 3.5.3.

7.4.3 Summary:

Underground disposal of retorted shale can reduce substantially the surface environmental and land disturbance effects of a mine-retort facility. Surface subsidence is likely to be minimal or negligible when stopes are backfilled. The adverse effects of underground disposal on surface water are minimal, with nonhydraulic methods having the least effect. Hydraulic transport and placement methods result in the most severe environmental impacts because of the requirement for surface slimes ponds. With total surface disposal no appreciable differences in air quality are expected, except that more fugitive dust can be anticipated.

7.5 Economics:

Phase I capital and operating costs have been developed for various combinations of transport and stowing methods described in Sections 7.1 and 7.2. Costs developed are used to compare and rank the various methods. Actual quoted prices were used when they were available, but in some cases costs were factored to achieve relative cost relationships rather than absolute values. Productivity and usage rates are based on in-house experience and information supplied by vendors. Labor rates are consistent with current levels in Western Colorado. All costs are expressed in 1977 dollars. Due to the limited Scope of Work prescribed for Phase I, no effort is made in these comparisons to account for the time value of money or the return on investment. Capital costs include initial capital requirements and deferred capital expenditures over an assumed project life of twenty years.

7.5.1 Transport and Stowing Costs:

Capital and operating costs were developed for 17 backfilling systems incorporating combinations of the various transport and stowing methods. The systems have been studied only to the degree necessary for selecting the most promising ones for detailed analysis and engineering for Phase II. Costs for backfilling in conjunction with chamber and pillar, and sublevel stoping mining methods, are reported separately.

Tables 7.5.1-1 through 7.5.1-8 present capital and operating costs for the combinations of stowing methods related to each transport method for both mining methods being considered. Costs per ton are based on total tons of retorted shale. All manpower and equipment requirements are restricted to the backfilling activity with no overlap into the mining functions. Surface disposal costs for material not stowed in the mine have been included in the analysis of each system.

TABLE 7.5.1-1

PRELIMINARY CAPITAL AND OPERATING COSTS

CHAMBER AND PILLAR MINING

CONVEYOR TRANSPORT

	STOWING METHOD				
	Conveyor And Compact	Pneumatic	Conveyor With Pneumatic Topfill	Hydraulic	Hydraulic With Pneumatic Topfill
Capital Costs:					
$\$ \times 10^3$	19,670	34,186	23,015	100,106	91,883
\$/Ton	0.0397	0.0691	0.0465	0.2022	0.1856
Operating Costs:					
Operating:					
Labor (\$/Ton)	0.1200	0.1419	0.1495	0.2654	0.2512
Supplies (\$/Ton)	0.0537	0.5214	0.0813	0.4181	0.3555
Maintenance:					
Labor (\$/Ton)	0.0793	0.1340	0.1031	0.2589	0.2378
Supplies (\$/Ton)	0.0571	0.1659	0.0882	0.1361	0.1576
Total Operating Costs:					
\$/Ton	0.3101	0.9632	0.4221	1.0785	1.0021
$\$ \times 10^3$	153,512	476,823	208,956	533,901	496,080
Combined Costs:					
Total:					
$\$ \times 10^3$	173,182	511,009	231,971	634,007	587,963
\$/Ton	0.3498	1.0323	0.4686	1.2807	1.1877

TABLE 7.5.1-2

PRELIMINARY CAPITAL AND OPERATING COSTS

CHAMBER AND PILLAR MINING

TRUCK TRANSPORT

	STOWING METHOD				
	Truck And Compact	Pneumatic	Truck With Pneumatic Topfill	Hydraulic	Hydraulic With Pneumatic Topfill
Capital Costs:					
$\$ \times 10^3$	43,895	56,449	48,265	114,780	110,265
$/Ton	0.0887	0.1140	0.0975	0.2319	0.2227
Operating Costs:					
Operating:					
Labor ($/Ton)	0.1367	0.1819	0.1662	0.2703	0.2750
Supplies ($/Ton)	0.1613	0.6289	0.1889	0.5012	0.4532
Maintenance:					
Labor ($/Ton)	0.1337	0.1827	0.1574	0.2619	0.2618
Supplies ($/Ton)	0.1100	0.2121	0.1412	0.1590	0.1844
Total Operating Costs:					
$/Ton	0.5417	1.2056	0.6537	1.1924	1.1744
$\$ \times 10^3$	268,163	596,820	323,608	590,286	581,375
Combined Costs:					
Total:					
$\$ \times 10^3$	312,058	653,269	371,873	705,066	691,640
$/Ton	0.6304	1.3196	0.7512	1.4243	1.3971

TABLE 7.5.1-3

PRELIMINARY CAPITAL AND OPERATING COSTS

CHAMBER AND PILLAR MINING

PNEUMATIC TRANSPORT AND STOWING

	ALTERNATE TRANSPORT METHOD	
	Vertical Pipe	Borehole
Capital Costs:		
$\$ \times 10^3$	60,883	43,295
$/Ton	0.1230	0.0875
Operating Costs:		
Operating:		
Labor ($/Ton)	0.1233	0.1232
Supplies ($/Ton)	0.6969	0.5960
Maintenance:		
Labor ($/Ton)	0.1283	0.1255
Supplies ($/Ton)	0.2370	0.1670
Total Operating Costs:		
$/Ton	1.1855	1.0117
$\$ \times 10^3$	586,870	500,832
Combined Costs:		
Total:		
$\$ \times 10^3$	647,753	544,127
$/Ton	1.3085	1.0992

TABLE 7.5.1-4

PRELIMINARY CAPITAL AND OPERATING COSTS

CHAMBER AND PILLAR MINING

HYDRAULIC TRANSPORT

	STOWING METHOD				
	Hydraulic	Hydraulic With Pneumatic Topfill	Conveyor	Conveyor With Pneumatic Topfill	Pneumatic
Capital Costs:					
$\$ \times 10^3$	85,908	131,797	131,940	135,329	142,203
\$/Ton	0.1735	0.2662	0.2665	0.2734	0.2873
Operating Costs:					
Operating:					
Labor (\$/Ton)	0.1769	0.2551	0.2762	0.3058	0.2933
Supplies (\$/Ton)	0.3170	0.3684	0.5000	0.5276	0.9623
Maintenance:					
Labor (\$/Ton)	0.1748	0.2639	0.2547	0.2784	0.3054
Supplies (\$/Ton)	0.0953	0.1338	0.1439	0.1750	0.2472
Total Operating Costs:					
\$/Ton	0.7640	1.0212	1.1748	1.2868	1.8082
$\$ \times 10^3$	378,211	505,535	581,573	637,017	895,131
Combined Costs:					
Total:					
$\$ \times 10^3$	464,119	637,332	713,513	772,346	1,037,334
\$/Ton	0.9375	1.2874	1.4413	1.5602	2.0955

TABLE 7.5.1-5

PRELIMINARY CAPITAL AND OPERATING COSTS

SUBLEVEL STOPING
CONVEYOR TRANSPORT

	STOWING METHOD				
	Conveyor	Pneumatic	Conveyor With Pneumatic Topfill	Hydraulic	Hydraulic With Pneumatic Topfill
Capital Costs:					
$\$ \times 10^3$	22,328	34,186	25,557	100,106	91,883
\$/Ton	0.0451	0.0691	0.0516	0.2022	0.1856
Operating Costs					
Operating:					
Labor (\$/Ton)	0.1236	0.1419	0.1527	0.2654	0.2512
Supplies (\$/Ton)	0.0577	0.5214	0.0852	0.4181	0.3555
Maintenance:					
Labor (\$/Ton)	0.0738	0.1340	0.0975	0.2589	0.2378
Supplies (\$/Ton)	0.0611	0.1659	0.0922	0.1361	0.1576
Total Operating Costs:					
\$/Ton	0.3162	0.9632	0.4276	1.0785	1.0021
$\$ \times 10^3$	156,532	476,823	211,679	533,901	496,080
Combined Costs:					
Total:					
$\$ \times 10^3$	178,860	511,009	237,236	634,007	587,963
\$/Ton	0.3613	1.0323	0.4792	1.2807	1.1877

TABLE 7.5.1-6

PRELIMINARY CAPITAL AND OPERATING COSTS

SUBLEVEL STOPING

TRUCK TRANSPORT

	STOWING METHOD				
	Truck	Pneumatic	Truck With Pneumatic Topfill	Hydraulic	Hydraulic With Pneumatic Topfill
Capital Costs:					
$ x 10^3	45,665	56,449	44,575	114,780	110,265
$/Ton	0.0922	0.1140	0.0900	0.2319	0.2227
Operating Costs:					
Operating:					
Labor ($/Ton)	0.1318	0.1819	0.1608	0.2703	0.2750
Supplies	0.1612	0.6289	0.1881	0.5012	0.4532
Maintenance:					
Labor ($/Ton)	0.1315	0.1827	0.1547	0.2619	0.2618
Supplies ($/Ton)	0.1086	0.2121	0.1395	0.1590	0.1844
Total Operating Costs:					
$/Ton	0.5331	1.2056	0.6431	1.1924	1.1744
$ x 10^3	263,906	596,820	318,360	590,286	581,375
Combined Costs:					
Total:					
$ x 10^3	309,571	653,269	362,935	705,066	691,640
$/Ton	0.6253	1.3196	0.7331	1.4243	1.3971

TABLE 7.5.1-7

PRELIMINARY CAPITAL AND OPERATING COSTS

SUBLEVEL STOPING

PNEUMATIC TRANSPORT AND STOWING

	ALTERNATE TRANSPORT METHOD	
	Vertical Pipe	Borehole
Capital Costs:		
$\$ \times 10^3$	60,883	43,295
\$/Ton	0.1230	0.0875
Operating Costs:		
Operating:		
Labor (\$/Ton)	0.1233	0.1232
Supplies (\$/Ton)	0.6969	0.5960
Maintenance:		
Labor (\$/Ton)	0.1283	0.1255
Supplies (\$/Ton)	0.2370	0.1670
Total Operating Costs:		
\$/Ton	1.1855	1.0117
$\$ \times 10^3$	586,870	500,832
Combined Costs:		
Total:		
$\$ \times 10^3$	647,753	544,127
\$/Ton	1.3085	1.0992

TABLE 7.5.1-8

PRELIMINARY CAPITAL AND OPERATING COSTS

SUBLEVEL STOPING

HYDRAULIC TRANSPORT

	STOWING METHOD				
	Hydraulic	Hydraulic With Pneumatic Topfill	Conveyor	Conveyor With Pneumatic Topfill	Pneumatic
Capital Costs:					
$\$ \times 10^3$	85,908	131,797	131,940	135,329	142,203
\$/Ton	0.1735	0.2662	0.2665	0.2734	0.2873
Operating Costs:					
Operating:					
Labor (\$/Ton)	0.1769	0.2551	0.2762	0.3058	0.2933
Supplies (\$/Ton)	0.3170	0.3684	0.5000	0.5276	0.9623
Maintenance:					
Labor (\$/Ton)	0.1748	0.2639	0.2547	0.2784	0.3054
Supplies (\$/Ton)	0.0953	0.1338	0.1439	0.1750	0.2472
Total Operating Costs:					
\$/Ton	0.7640	1.0212	1.1748	1.2868	1.8082
$\$ \times 10^3$	378,211	505,535	581,573	637,017	895,131
Combined Costs:					
Total:					
$\$ \times 10^3$	464,119	637,332	713,513	772,346	1,037,334
\$/Ton	0.9375	1.2874	1.4413	1.5602	2.0955

7.5.2 Surface Disposal Costs:

All transport and stowing methods for underground disposal of retorted shale require that some material be left on the surface. Volumes to be left and costs of placement are a function of methods used. For comparative purposes, costs have been developed for disposing of all retorted shale on the surface.

7.5.2.1 Surface Disposal With Underground Backfilling:

Capital and operating costs for surface disposal in conjunction with underground disposal vary according to the disposal method being considered. The higher costs associated with hydraulic transport and stowing are primarily the result of slimes disposal pond construction and maintenance. The volume of dry retorted shale has a minor effect on costs. Table 7.5.2.1-1 presents relative costs for surface disposal with operating costs per ton based on total tons of retorted shale and as a function of surface tons only.

7.5.2.2 Total Surface Disposal:

When considering capital and operating costs, total surface disposal of retorted shale is the least costly method available. However, adverse environmental effects and future resource recovery considerations will strongly influence selection of the method of retorted shale disposal. Table 7.5.2.2-1 shows capital and operating costs for total surface disposal and revegetation activities.

7.5.3 Backfilling Costs At Other Room And Pillar Mines:

Two basic problems were encountered while trying to find representative room and pillar mining operations that are engaged in backfilling. First of all, no room and pillar mine in the United States produces at a rate comparable to a proposed oil shale facility. Secondly, most backfilling that is being done is on a sporadic basis and is usually quite localized. Many salt and limestone mines have rooms of comparable size to those envisioned for oil shale mines, but backfilling activities are minor and costs are not kept in sufficient detail to make a valid comparison.

Costs for underground disposal of coal mine wastes have been developed for active and abandoned mines and are shown in Table 7.5.3-1. (17). Parts of the estimate are based on actual experience in backfilling abandoned mines. Costs tend to be high but the nature of the projects was such that commercial-scale productivity was not required. In addition, the economics of large-scale operations is lacking.

7.5.4 Effects of Production Rate:

An analysis was made of the effects of production rates on the costs of underground disposal of retorted oil shale. Two production rates were selected that approximate the output of 10,000 barrels per day from one retort module and of 50,000 barrels per day which has been the traditional production parameter for a commercial-scale operation. Capital costs are based

TABLE 7.5.2.1-1

SURFACE DISPOSAL COSTS

Transport	Stowing	Chamber and Pillar Capital $\$ \times 10^3$	Chamber and Pillar Total Tons \$/Ton	Chamber and Pillar Surface Tons Only-\$/Ton	Sublevel Stoping Capital $\$ \times 10^3$	Sublevel Stoping Total \$/Ton	Sublevel Stoping Surface Tons Only-\$/Ton
Conveyor	Conveyor	5,222	0.0325	0.2169	7,880	0.0470	0.1959
	Pneumatic	8,828	0.0522	0.1740	8,828	0.0522	0.1740
	Conveyor and Pneumatic Topfill	5,603	0.0325	0.1914	7,880	0.0477	0.1935
	Hydraulic	63,076	0.2408	0.4424	63,076	0.2408	0.4424
	Hydraulic and Pneumatic Topfill	63,076	0.2408	0.4424	63,076	0.2408	0.4424
Truck	Truck	5,222	0.0325	0.2169	7,880	0.0470	0.1959
	Pneumatic	8,828	0.0522	0.1740	8,828	0.0522	0.1740
	Truck and Pneumatic Topfill	5,603	0.0325	0.1914	7,880	0.0477	0.1935
	Hydraulic	63,076	0.2408	0.4424	63,076	0.2408	0.4424
	Hydraulic and Pneumatic Topfill	63,076	0.2408	0.4424	63,076	0.2408	0.4424
Pneumatic	Vertical Pipe	8,828	0.0522	0.1740	8,828	0.0522	0.1740
	Borehole	8,828	0.0522	0.1740	8,828	0.0522	0.1740
Hydraulic	Hydraulic	63,076	0.2408	0.4424	63,076	0.2408	0.4424
	Hydraulic and Pneumatic Topfill	63,076	0.2408	0.4424	63,076	0.2408	0.4424
	Conveyor	63,076	0.2408	0.4424	63,076	0.2408	0.4424
	Conveyor and Pneumatic Topfill	63,076	0.2408	0.4424	63,076	0.2408	0.4424
	Pneumatic	63,076	0.2408	0.4424	63,076	0.2408	0.4424

TABLE 7.5.2.2-1

CAPITAL AND OPERATING COSTS

TOTAL SURFACE DISPOSAL

Capital ($ x 10^3)	$19,055
Operating and Maintenance ($/Ton Retorted Shale)	
Operating Labor	$0.0602
Operating Supplies	0.0668
Maintenance Labor	0.0484
Maintenance Supplies	0.0684
TOTAL	$0.2438

on a twenty-year production period and it is assumed that all labor and equipment needs are restricted to the disposal operation. All costs are stated in 1977 dollars with no attempt to discount for the time value of money. Conveyor transport and stowing have been assumed for this comparison and costs include those related to disposal on the surface of 16 percent of the retorted shale. Table 7.5.4-1 presents the costs and related parameters for the two cases.

TABLE 7.5.3-1

ESTIMATED COSTS

UNDERGROUND COAL MINE WASTE DISPOSAL

Type of Operation and Stowing Method	Average $/Ton of Waste
Abandoned Mines - U.S.A. - Hydraulic	$4.00
Operating Mines - U.S.A. - Pneumatic	4.25
Operating Mines - U.S.A. - Hydraulic	5.00
Operating Mines - Europe - Pneumatic	6.00

TABLE 7.5.4-1

UNDERGROUND DISPOSAL COSTS COMPARISON

EFFECTS OF PRODUCTION RATE

	10,000 BPD	50,000 BPD
Capital ($ x 10^3)	$14,005	$19,670
Operating and Maintenance ($/Ton)		
Operating Labor	$0.3811	$0.1200
Operating Supplies	0.1352	0.0537
Maintenance Labor	0.2672	0.0793
Maintenance Supplies	0.1495	0.0571
TOTAL	$0.9330	$0.3101
Energy (KW/Day x 10^3)	88	335
Water (Gal/Day x 10^3)	305	1,527
Manpower (Total Backfilling Payroll)	182	263

Capital costs are not proportional to the production rate because equipment and construction costs are not directly related to production requirements, and equipment utilization is higher for the larger operation. Labor requirements are virtually the same for both conveyor transport and for stowing, but supporting functions vary with the rate of production. Supervisory and administrative labor remains relatively constant for both production levels.

The relative effects of the production rate exist regardless of the disposal system selected, but the magnitude of the variation is dependent upon manpower and equipment requirements. For example, a more labor intensive method causes less effect on the relative costs of the two production rates. This assumes comparable labor productivity levels for both cases.

7.5.5 Effects of Increased Resource Recovery:

Resource recovery can be improved by backfilling mined-out stopes with retorted oil shale. By using the proper mining sequence, pillar dimensions can be reduced, resulting in an increase in available ore. Tables 7.5.5-1 and 7.5.5-2 show the percentage improvement for each transport-stowing combination for each mining method. The following hypothetical conditions are used to develop the effects of backfilling on resource recovery:

- Retort feed: 20,120,000 tons per year
- Raw shale grade: 28 gallons per ton
- Total minable reserve before backfilling: 582,400,000 tons (Excluding pillar tonnage)
- All tons have been adjusted for retort recovery
- Shale oil selling price: $15.00 per barrel
- No consideration is given to mining and retorting costs

The previous comparisons illustrate that increased revenues will partially offset the cost of backfilling. The extent to which this is true depends on the mining and retorting costs. Subject to the assumptions made for this comparison, additional revenue derived from increased resource recovery will pay the backfilling costs of most of the systems studied.

7.6 Evaluation and Selection:

Seventeen combinations of transport and stowing methods were evaluated in order to select the most promising systems for underground disposal of retorted oil shale. A separate analysis was made for both chamber and pillar mining and for sublevel stoping. The evaluations involved a technique which integrated the technical and economic aspects of the various systems and provided the basis for selecting the most promising systems for further detailed analysis and design.

TABLE 7.5.5-1

EFFECTS OF INCREASED RESOURCE RECOVERY

CHAMBER AND PILLAR MINING

Transport	Stowing	% Increase	Added Years Operation	Gross Added Revenue ($ x 10^3*)	Total Disposal Cost ($ x 10^3)
Conveyor	Conveyor	16.0	3.20	931,840	197,744
	Pneumatic	13.0	2.60	757,120	572,995
	Conveyor & Pneumatic Topfill	15.6	3.12	908,544	264,569
	Hydraulic	11.0	2.20	640,640	692,736
	Hydraulic & Pneumatic Topfill	11.3	2.26	658,112	644,020
Truck	Truck	16.0	3.20	931,840	354,964
	Pneumatic	13.0	2.60	757,120	730,856
	Truck & Pneumatic Topfill	15.6	3.12	908,544	422,355
	Hydraulic	11.0	2.20	640,640	769,997
	Hydraulic & Pneumatic Topfill	11.3	2.26	658,112	757,335
Pneumatic	Vertical Pipe	13.0	2.60	757,120	724,046
	Borehole	13.0	2.60	757,120	609,235
Hydraulic	Hydraulic	11.0	2.20	640,640	505,722
	Hydraulic & Pneumatic Topfill	11.3	2.26	658,112	694,457
	Conveyor	16.0	3.20	931,840	806,565
	Conveyor & Pneumatic Topfill	15.6	3.12	908,544	871,721
	Pneumatic	13.0	2.60	757,120	1,153,701

*1) Based on $15.00 per barrel without consideration of mining and retorting costs.

2) Assuming a finite reserve based on production of 50,000 barrels per day for 20 years with no backfilling.

TABLE 7.5.5-2

EFFECTS OF INCREASED RESOURCE RECOVERY

SUBLEVEL STOPING

Transport	Stowing	% Increase	Added Years Operation	Gross Added Revenue ($ x 10^3*)	Total Disposal Cost ($ x 10^3)
Conveyor	Conveyor	15.0	3.00	873,600	202,339
	Pneumatic	14.0	2.80	815,360	577,764
	Conveyor & Pneumatic Topfill	14.8	2.96	861,952	268,565
	Hydraulic	12.0	2.40	698,880	698,075
	Hydraulic & Pneumatic Topfill	12.4	2.48	722,176	649,476
Truck	Truck	15.0	3.00	873,600	349,157
	Pneumatic	14.0	2.80	815,360	736,824
	Truck & Pneumatic Topfill	14.8	2.96	861,952	410,053
	Hydraulic	12.0	2.40	698,880	775,900
	Hydraulic & Pneumatic Topfill	12.4	2.48	722,176	763,730
Pneumatic	Vertical Pipe	14.0	2.80	815,360	729,915
	Borehole	14.0	2.80	815,360	614,243
Hydraulic	Hydraulic	12.0	2.40	698,880	509,503
	Hydraulic & Pneumatic Topfill	12.4	2.48	722,176	700,018
	Conveyor	15.0	3.00	873,600	800,749
	Conveyor & Pneumatic Topfill	14.8	2.96	861,952	866,625
	Pneumatic	14.0	2.80	815,360	1,162,653

*1) Based on $15.00 per barrel without consideration of mining and retorting costs.

2) Assuming a finite reserve based on production of 50,000 barrels per day for 20 years without backfilling.

7.6.1 Ranking Analysis:

The method used to select the most promising combination of transport and stowing systems for underground disposal of retorted oil shale is called Least Total Divisor Ranking Analysis (24). Economic and technical evaluations were made for each system prior to the ranking analysis. This ranking analysis technique provides a means for integrating the results of the individual system evaluations and also provides a basis for selecting the most promising system.

7.6.1.1 Methodology:

The method weights each set of factors so that the lowest relative value indicates the most desirable system. The development of the relative values depends on the nature of the variable being evaluated. In a case where a low value, such as energy consumption, is preferred, the lowest value for the set is used as the divisor by which all factors are divided. Where a high value, such as resource recovery, is preferred, the highest value is used as the dividend into which all factors are divided. In either case, the results are a set of dimensionless numbers ranging from 1.0 for the most desirable factor upward to some maximum value for the least desirable factor. This relative ranking of the factors is the basis for the method, and the evaluated areas to be discussed in detail are listed below:

- Subjective Technical Evaluation
- Objective Technical Evaluation
- Capital Costs
- Operating Costs

7.6.1.1.1 Subjective Technical Evaluation:

A subjective technical evaluation based on experience and qualified opinions of the engineering staff was performed for twenty-five elements included in each of the transport-stowing combinations. The relative values assigned to the various elements followed the rationale that the most desirable element is assigned the lowest value. Each element was rated on a scale of 10 to 100, depending on the individual's evaluation. Each element was then rated for all transport-stowing combinations with the relative values on a scale of 1 to 10. The final element value for each transport-stowing combination was obtained by multiplying the two values together. Average values were calculated for the results from all those who participated in the evaluation and these averages were used to calculate the final results. The element factors for each transport-stowing combination were totaled with the sum being the input to the ranking calculation. Table 7.6.1.1.1-1 presents the results of the subjective technical evaluation. Tables 7.6.1.1.1-2 through 7.6.1.1.1-9 are compilations of the subjective technical analyses for the various transport-stowing and mining combinations.

TABLE 7.6.1.1.1-1

SUBJECTIVE TECHNICAL ANALYSIS

Transport	Stowing	Chamber and Pillar Mining Factor	Chamber and Pillar Mining Rank	Sublevel Stoping Factor	Sublevel Stoping Rank
Conveyor	Conveyor	4,103	1.00	4,319	1.00
	Pneumatic	5,921	1.44	5,858	1.36
	Conveyor & Pneumatic Topfill	4,987	1.22	5,156	1.19
	Hydraulic	6,235	1.52	6,199	1.44
	Hydraulic & Pneumatic Topfill	6,524	1.59	6,441	1.49
Truck	Truck	4,161	1.01	4,463	1.03
	Pneumatic	5,720	1.39	5,703	1.32
	Truck & Pneumatic Topfill	4,970	1.21	5,117	1.18
	Hydraulic	5,977	1.46	5,924	1.37
	Hydraulic & Pneumatic Topfill	6,277	1.53	6,227	1.44
Pneumatic	Vertical Pipe	7,333	1.79	7,312	1.69
	Borehole	6,802	1.66	6,726	1.56
Hydraulic	Hydraulic	7,186	1.75	7,046	1.63
	Hydraulic & Pneumatic Topfill	7,638	1.86	7,482	1.73
	Conveyor	7,206	1.76	7,046	1.63
	Conveyor & Pneumatic Topfill	7,585	1.85	7,369	1.71
	Pneumatic	7,785	1.90	7,636	1.77

TABLE 7.6.1.1.1-2
SUBJECTIVE ANALYSIS
CHAMBER AND PILLAR MINING METHOD

ELEMENTS	RELATIVE EFFECT FACTORS	TRANSPORT METHOD									
		CONVEYOR									
		STOWING METHOD									
		MECHANICAL		PNEUMATIC		MECH. & PNEU.		HYDRAULIC		HYDR. & PNEU.	
	ELEMENT	FACTOR	RESULT	FACTOR	RESULT	FACTOR	RESULT	FACTOR	RESULT	FACTOR	RESULT
Safety Aspects-Feed Preparation	39	3.28	128	4.15	162	4.56	178	4.00	156	5.00	195
Safety Aspects-Transport	40	4.28	171	5.28	211	4.85	194	4.15	166	4.56	183
Safety Aspects-Stowing	54	4.72	255	5.85	316	5.72	309	4.43	239	5.72	309
Transport Method-Installation	44	4.14	182	4.36	192	4.36	192	4.36	192	4.36	192
Transport Method-Maintenance	39	3.44	134	3.59	140	3.59	140	3.59	140	3.59	140
Transport Method-Operation	34	3.56	121	3.82	130	3.82	130	3.82	130	3.82	130
Transport Method-Technical Feasibility	36	2.42	87	2.67	96	2.67	96	2.67	96	2.67	96
Stowing Method-Operation	39	3.44	134	6.50	234	6.03	217	5.58	201	6.50	234
Stowing Method-Technical Feasibility	47	2.72	128	7.15	336	4.00	188	5.28	248	5.85	275
Stowing Method-Manpower Skills	38	3.87	147	5.29	201	4.58	174	4.42	168	5.00	190
Stowing Method-Fill Characteristics	49	3.00	147	6.00	294	3.29	161	6.86	336	6.43	315
Stowing Method-Ground Support	44	3.14	138	5.57	245	3.30	145	5.30	233	5.43	239
Underground Environment-Dust Control	51	5.43	277	9.14	466	7.43	379	2.00	102	4.71	240
Underground Environment-Temp.-Humidity	47	4.15	195	7.00	329	5.43	255	6.15	289	6.85	322
Underground Environment-Gas	39	4.00	156	5.72	223	4.44	173	2.85	111	3.85	150
Effects on Mine Op.-Water	39	2.44	95	2.28	89	2.56	100	7.85	306	6.72	262
Effects on Mine Op.-Sched. & Interfer.	50	4.00	200	4.86	243	5.58	279	4.72	236	5.72	286
Effects on Mine Op.-Disaster Potential	40	2.58	103	5.58	223	3.28	131	7.73	309	7.28	291
Ground Water-Effect on Fill	56	3.86	216	7.14	400	4.86	272	5.71	320	6.29	352
Ground Water-Decant Water Quality	51	3.71	189	5.43	277	4.29	219	8.00	408	7.29	372
Ground Water-Ground Water Qaulity	46	4.57	210	4.72	217	4.85	223	5.00	230	5.43	250
Ground Water-Mine Discharge Quality	54	4.00	216	4.85	262	4.85	262	8.00	432	7.28	393
Surface Envir. Effects-Disposal Area	53	3.15	167	4.28	227	3.28	174	8.28	439	7.57	401
Surface Envir. Effects-Water Treatment	56	3.29	184	4.00	224	4.00	224	8.43	472	7.57	424
Surface Envir. Effects-Surface Fac.	43	2.86	123	4.28	184	4.00	172	6.42	276	6.58	283
TOTALS			4,103		5,921		4,987		6,235		6,524

TABLE 7.6.1.1.1-3

SUBJECTIVE ANALYSIS

CHAMBER AND PILLAR MINING METHOD

ELEMENTS	RELATIVE EFFECT FACTORS - ELEMENT	TRANSPORT METHOD: TRUCK — STOWING METHOD: MECHANICAL FACTOR	MECHANICAL RESULT	PNEUMATIC FACTOR	PNEUMATIC RESULT	MECH. & PNEU. FACTOR	MECH. & PNEU. RESULT	HYDRAULIC FACTOR	HYDRAULIC RESULT	HYDR. & PNEU. FACTOR	HYDR. & PNEU. RESULT
Safety Aspects-Feed Preparation	36	3.28	118	4.14	149	4.72	170	3.58	129	4.86	175
Safety Aspects-Transport	43	4.86	209	5.00	215	5.14	221	3.86	166	4.14	178
Safety Aspects-Stowing	54	4.72	255	5.57	301	5.72	309	5.15	278	6.72	363
Transport Method-Installation	31	2.42	75	2.74	85	2.74	85	2.74	85	2.74	85
Transport Method-Maintenance	44	3.86	170	4.05	178	4.05	178	4.05	178	4.05	178
Transport Method-Operation	41	3.56	146	3.76	154	3.76	154	3.76	154	3.76	154
Transport Method-Technical Feasibility	32	2.16	69	2.31	74	2.31	74	2.31	74	2.31	74
Stowing Method-Operation	39	3.72	145	5.44	212	5.00	195	4.72	184	5.85	228
Stowing Method-Technical Feasibility	43	2.28	98	6.58	283	4.28	184	5.58	240	5.86	252
Stowing Method-Manpower Skills	44	4.00	176	5.14	226	4.30	189	4.43	195	5.14	226
Stowing Method-Fill Characteristics	45	2.58	116	5.58	251	3.29	148	6.13	276	6.29	283
Stowing Method-Ground Support	41	2.85	117	5.71	234	3.44	141	5.56	228	5.71	234
Underground Environment-Dust Control	51	5.86	299	9.00	459	7.86	401	3.00	153	4.43	226
Underground Environment-Temp.-Humidity	49	4.57	224	7.86	385	6.00	294	5.00	245	7.14	350
Underground Environment-Gas	39	4.85	189	5.72	223	4.72	184	3.85	150	4.00	156
Effects on Mine Op.-Water	40	2.58	103	2.58	103	3.00	120	6.43	257	6.15	246
Effects on Mine Op.-Sched. & Interfer.	55	5.71	314	4.71	259	6.15	338	5.85	322	5.85	322
Effects on Mine Op.-Disaster Potential	39	2.56	100	5.44	212	3.72	145	7.00	273	7.28	284
Ground Water - Effect on Fill	54	3.57	193	6.72	363	4.43	239	5.15	278	5.72	309
Ground Water - Decant Water Quality	49	3.43	168	5.00	245	3.57	175	7.86	385	6.86	336
Ground Water-Ground Water Quality	45	4.42	199	4.87	219	4.71	212	5.13	231	5.13	231
Ground Water-Mine Discharge Quality	50	3.86	193	4.72	236	4.42	221	7.42	371	7.00	350
Surface Envir. Effects-Disposal Area	55	3.44	189	4.56	251	3.56	196	8.44	464	7.56	416
Surface Envir. Effects-Water Treatment	54	3.15	170	3.85	208	3.85	208	8.28	447	7.43	401
Surface Envir. Effects-Surface Fac.	44	2.86	126	4.43	195	4.30	189	4.86	214	5.00	220
TOTALS			4,161		5,720		4,970		5,977		6,277

TABLE 7.6.1.1.1-4
SUBJECTIVE ANALYSIS
CHAMBER AND PILLAR MINING METHOD

ELEMENTS	RELATIVE EFFECT FACTORS - ELEMENT	TRANSPORT METHOD			
		PNEUMATIC			
		TRANSPORT METHOD FROM SURFACE			
		PIPE		BOREHOLE	
		FACTOR	RESULT	FACTOR	RESULT
Safety Aspects-Feed Preparation	46	4.28	197	3.85	177
Safety Aspects-Transport	31	4.71	146	3.71	115
Safety Aspects-Stowing	48	5.00	240	4.71	226
Transport Method-Installation	51	6.57	335	4.86	248
Transport Method-Maintenance	51	7.00	357	4.43	226
Transport Method-Operation	46	6.43	296	4.15	191
Transport Method-Technical Feasibility	58	6.86	398	4.29	249
Stowing Method-Operation	49	6.29	308	6.29	308
Stowing Method-Technical Feasibility	56	5.82	328	5.82	328
Stowing Method-Manpower Skills	41	5.29	217	5.29	217
Stowing Method-Fill Characteristics	63	6.43	405	6.43	405
Stowing Method-Ground Support	57	6.14	350	6.14	350
Underground Environment-Dust Control	72	8.57	617	8.57	617
Underground Environment-Temp.-Humidity	59	6.86	405	6.86	405
Underground Environment-Gas	44	5.30	233	5.36	236
Effects on Mine Op.-Water	40	3.28	131	3.43	137
Effects on Mine Op.-Sched. & Interfer.	48	5.00	240	5.21	250
Effects on Mine Op.-Disaster Potential	46	5.15	237	5.15	237
Ground Water-Effect on Fill	67	6.28	421	6.28	421
Ground Water-Decant Water Quality	54	4.72	255	4.72	255
Ground Water-Ground Water Quality	49	5.00	245	5.00	245
Ground Water-Mine Discharge Quality	56	4.86	272	4.86	272
Surface Envir. Effects-Disposal Area	59	4.29	253	4.29	253
Surface Envir. Effects-Water Treatment	57	4.42	252	4.42	252
Surface Envir. Effects-Surface Fac.	44	4.43	195	4.14	182
TOTAL			7,333		6,802

TABLE 7.6.1.1.1-5

SUBJECTIVE ANALYSIS

CHAMBER AND PILLAR MINING METHOD

ELEMENTS	RELATIVE EFFECT FACTORS - ELEMENT	TRANSPORT METHOD: HYDRAULIC; STOWING METHOD: HYDRAULIC		HYDR. & PNEU.		MECHANICAL		MECH. & PNEU.		PNEUMATIC	
		FACTOR	RESULT	FACTOR	RESULT	FACTOR	RESULT	FACTOR	RESULT	FACTOR	RESULT
Safety Aspects-Feed Preparation	47	3.72	175	4.72	222	4.85	228	5.28	248	5.28	248
Safety Aspects-Transport	44	4.43	195	5.00	220	5.43	239	6.00	264	6.00	264
Safety Aspects-Stowing	50	4.72	236	5.72	286	6.42	321	6.72	336	6.72	336
Transport Method-Installation	44	4.70	207	4.89	215	4.89	215	4.89	215	4.89	215
Transport Method-Maintenance	44	4.86	214	5.07	223	5.07	223	5.07	223	5.07	223
Transport Method-Operation	40	4.28	171	4.50	180	4.50	180	4.50	180	4.50	180
Transport Method-Technical Feasibility	46	4.28	197	4.33	199	4.33	199	4.33	199	4.33	199
Stowing Method-Operation	44	4.70	207	6.86	302	7.43	327	8.14	358	8.43	371
Stowing Method-Technical Feasibility	46	3.85	177	6.28	289	7.43	342	8.00	368	8.15	375
Stowing Method-Manpower Skills	36	3.86	139	4.58	165	5.58	201	6.28	226	5.28	190
Stowing Method-Fill Characteristics	57	5.58	318	6.00	342	5.42	309	5.86	334	6.28	358
Stowing Method-Ground Support	52	5.42	282	5.87	305	5.29	275	5.71	297	6.71	349
Underground Environment-Dust Control	29	1.86	54	3.28	95	3.00	87	4.28	124	5.72	166
Underground Environment-Temp.-Humidity	55	5.29	291	5.85	322	5.29	291	5.85	322	6.56	361
Underground Environment-Gas	35	3.00	105	4.14	145	3.71	130	4.43	155	5.00	175
Effects on Mine Op.-Water	63	7.57	477	7.00	441	5.57	351	5.57	351	5.43	342
Effects on Mine Op.-Sched. & Interfer.	46	4.28	197	5.28	243	5.43	250	5.72	263	5.28	243
Effects on Mine Op.-Disaster Potential	60	7.57	454	7.15	429	5.00	300	5.43	326	5.85	351
Ground Water - Effect on Fill	62	5.00	310	5.71	354	5.71	354	6.15	381	6.71	416
Ground Water-Decant Water Quality	67	7.43	498	6.85	459	6.00	402	5.57	373	6.28	421
Ground Water-Ground Water Quality	50	5.58	279	5.58	279	5.00	250	5.42	271	6.14	307
Ground Water-Mine Discharge Quality	73	7.86	574	7.14	521	6.00	438	6.29	459	6.42	469
Surface Envir. Effects-Disposal Area	70	7.71	540	7.29	510	7.14	500	6.71	470	6.43	450
Surface Envir. Effects-Water Treatment	73	8.14	594	7.86	574	7.14	521	7.29	532	7.00	511
Surface Envir. Effects-Surface Fac.	53	5.57	295	6.00	318	5.15	273	5.85	310	5.00	265
TOTALS			7,186		7,638		7,206		7,585		7,785

TABLE 7.6.1.1.1-6
SUBJECTIVE ANALYSIS
SUBLEVEL STOPING

ELEMENTS	RELATIVE EFFECT - ELEMENT	TRANSPORT METHOD: CONVEYOR; STOWING METHOD: MECHANICAL FACTOR	MECHANICAL PRODUCT	PNEUMATIC FACTOR	PNEUMATIC PRODUCT	MECH. & PNEU. FACTOR	MECH. & PNEU. PRODUCT	HYDRAULIC FACTOR	HYDRAULIC PRODUCT	HYDR. & PNEU. FACTOR	HYDR. & PNEU. PRODUCT
Safety Aspects-Feed Preparation	39	3.28	128	4.15	162	4.56	178	4.00	156	5.00	195
Safety Aspects-Transport	39	4.15	162	5.15	201	4.44	173	4.15	162	4.15	162
Safety Aspects-Stowing	54	4.00	216	5.57	301	4.85	262	4.00	216	5.00	270
Transport Method-Installation	43	4.00	172	4.23	182	4.23	182	4.23	182	4.23	182
Transport Method-Maintenance	39	3.56	139	3.72	145	3.72	145	3.72	145	3.72	145
Transport Method-Operation	33	3.27	108	3.55	117	3.55	117	3.55	117	3.55	117
Transport Method-Technical Feasibility	36	2.58	93	2.83	102	2.83	102	2.83	102	2.83	102
Stowing Method-Operation	36	3.14	113	5.42	195	4.86	175	4.28	154	5.28	190
Stowing Method-Technical Feasibility	43	3.00	129	6.72	289	4.42	190	4.86	209	5.58	240
Stowing Method-Manpower Skills	33	2.85	94	4.73	156	3.73	123	4.15	137	4.58	151
Stowing Method-Fill Characteristics	56	4.57	256	6.29	352	5.00	280	6.86	384	6.86	384
Stowing Method-Ground Support	53	4.00	212	6.00	318	4.72	250	5.43	288	5.57	295
Underground Environment-Dust Control	51	5.29	270	8.86	452	7.14	364	2.00	102	4.57	233
Underground Environment-Temp.-Humidity	47	4.43	208	7.00	329	5.15	242	6.15	289	6.72	316
Underground Environment-Gas	39	3.56	139	5.56	217	4.28	167	2.85	111	4.00	156
Effects on Mine Op.-Water	41	2.85	117	2.56	105	3.29	135	7.71	316	6.71	275
Effects on Mine Op.-Sched. & Interfer.	47	3.85	181	4.57	215	5.15	242	4.72	222	5.57	262
Effects on Mine Op.-Disaster Potential	37	3.43	127	5.57	206	4.30	159	8.14	301	7.57	280
Ground Water-Effect on Fill	55	4.29	236	7.00	385	5.00	275	5.71	314	6.29	346
Ground Water-Decant Water Quality	52	4.13	215	5.42	282	4.58	238	8.00	416	7.29	379
Ground Water-Ground Water Quality	49	4.86	238	4.86	238	5.00	245	5.29	259	5.43	266
Ground Water-Mine Discharge Quality	55	4.44	244	4.85	267	5.15	283	8.00	440	7.44	409
Surface Envir. Effects-Disposal Area	50	3.42	171	4.28	214	3.86	193	8.42	421	7.42	371
Surface Envir. Effects-Water Treatment	57	3.58	204	4.28	244	4.42	252	8.42	480	7.58	432
Surface Envir. Effects-Surface Fac.	43	3.42	147	4.28	184	4.28	184	6.42	276	6.58	283
TOTALS			4,319		5,858		5,156		6,199		6,441

TABLE 7.6.1.1.1-7
SUBJECTIVE ANALYSIS
SUBLEVEL STOPING

ELEMENTS	RELATIVE EFFECT FACTORS	TRANSPORT METHOD									
		TRUCK									
		STOWING METHOD									
	-	MECHANICAL		PNEUMATIC		MECH. & PNEU.		HYDRAULIC		HYDR. & PNEU.	
	ELEMENT	FACTOR	PRODUCT	FACTOR	PRODUCT	FACTOR	PRODUCT	FACTOR	PRODUCT	FACTOR	PRODUCT
Safety Aspects-Feed Preparation	36	3.28	118	4.14	149	4.72	170	3.72	134	5.00	180
Safety Aspects-Transport	42	4.57	192	4.57	192	5.00	210	4.00	168	4.00	168
Safety Aspects-Stowing	53	4.00	212	4.85	257	4.43	235	4.57	242	6.00	318
Transport Method-Installation	31	2.42	75	2.74	85	2.74	85	2.74	85	2.74	85
Transport Method-Maintenance	44	3.86	170	4.05	178	4.05	178	4.05	178	4.05	178
Transport Method-Operation	41	3.56	146	3.76	154	3.76	154	3.76	154	3.76	154
Transport Method-Technical Feasibility	32	2.16	69	2.31	74	2.31	74	2.31	74	2.31	74
Stowing Method-Operation	33	3.15	104	4.73	156	4.00	132	4.15	137	5.00	165
Stowing Method-Technical Feasibility	39	2.56	100	6.00	234	3.44	134	4.44	173	5.15	201
Stowing Method-Manpower Skills	39	3.00	117	4.72	184	3.56	139	3.85	150	4.15	162
Stowing Method-Fill Characteristics	54	4.15	224	6.00	324	4.72	2.55	6.57	355	6.57	355
Stowing Method-Ground Support	47	4.28	201	5.85	275	4.72	222	5.00	235	5.43	255
Underground Environment-Dust Control	53	6.43	341	8.85	469	7.85	416	2.28	121	4.43	235
Underground Environment-Temp.-Humidity	47	4.57	215	7.72	363	5.85	275	6.00	282	7.00	329
Underground Environment-Gas	38	4.87	185	5.58	212	4.58	174	3.13	119	4.00	152
Effects on Mine Op.-Water	40	2.85	114	2.73	109	3.00	120	7.28	291	6.15	246
Effects on Mine Op.-Sched. & Interfer.	49	5.00	245	4.57	224	5.86	287	4.86	238	5.71	280
Effects on Mine Op.-Disaster Potential	41	3.56	146	5.56	228	4.56	187	7.56	310	7.44	305
Ground Water-Effect on Fill	58	4.57	265	7.00	406	5.14	298	5.43	315	6.14	356
Ground Water-Decant Water Quality	49	4.00	196	5.14	252	4.12	202	7.71	378	7.00	343
Ground Water-Ground Water Quality	47	4.85	228	5.00	235	5.00	235	5.15	242	5.43	255
Ground Water-Mine Discharge Quality	52	4.58	238	4.87	253	5.00	260	7.87	409	7.29	379
Surface Envir. Effects-Disposal Area	55	4.15	228	4.85	267	4.29	236	8.44	464	7.56	416
Surface Envir. Effects-Water Treatment	55	3.56	196	4.15	228	4.44	244	8.29	456	7.56	416
Surface Envir. Effects-Surface Fac.	44	3.14	138	4.43	195	4.43	195	4.86	214	5.00	220
TOTALS			4,463		5,703		5,117		5,924		6,227

TABLE 7.6.1.1.1-8

SUBJECTIVE ANALYSIS

SUBLEVEL STOPING

ELEMENTS	RELATIVE EFFECT FACTORS - ELEMENT	TRANSPORT METHOD PNEUMATIC TRANSPORT METHOD FROM SURFACE PIPE FACTOR	PIPE PRODUCT	BOREHOLE FACTOR	BOREHOLE PRODUCT
Safety Aspects-Feed Preparation	46	4.28	197	3.85	177
Safety Aspects-Transport	39	4.72	184	3.72	145
Safety Aspects-Stowing	54	4.43	239	4.15	224
Transport Method-Installation	50	6.42	321	4.72	236
Transport Method-Maintenance	50	6.86	343	4.42	221
Transport Method-Operation	44	6.30	277	4.14	182
Transport Method-Technical Feasibility	56	6.86	384	4.29	240
Stowing Method-Operation	40	6.00	240	6.00	240
Stowing Method-Technical Feasibility	54	5.28	285	5.28	285
Stowing Method-Manpower Skills	37	4.70	174	4.70	174
Stowing Method-Fill Characteristics	66	6.71	443	6.71	443
Stowing Method-Ground Support	60	6.43	386	6.43	386
Underground Environment-Dust Control	70	8.29	580	8.29	580
Underground Environment-Temp.-Humidity	60	6.85	411	6.85	411
Underground Environment-Gas	42	5.14	216	5.14	216
Effects on Mine Op.-Water	41	3.56	146	3.44	141
Effects on Mine Op.-Sched. & Interfer.	47	4.85	228	3.85	181
Effects on Mine Op.-Disaster Potential	48	5.44	261	5.44	261
Ground Water-Effect on Fill	69	6.43	444	6.43	444
Ground Water-Decant Water Quality	55	5.00	275	5.00	275
Ground Water-Ground Water Quality	50	5.14	257	4.86	243
Ground Water-Mine Discharge Quality	58	5.00	290	5.00	290
Surface Envir. Effects-Disposal Area	61	4.57	279	4.57	279
Surface Envir. Effects-Water Treatment	59	4.58	270	4.58	270
Surface Envir. Effects-Surface Fac.	44	4.14	182	4.14	182
TOTALS			7,312		6,726

TABLE 7.6.1.1.1-9
SUBJECTIVE ANALYSIS
SUBLEVEL STOPING

ELEMENTS	RELATIVE EFFECT FACTORS - ELEMENT	TRANSPORT METHOD: HYDRAULIC — STOWING METHOD: HYDRAULIC		HYDR. & PNEU.		MECHANICAL		MECH. & PNEU.		PNEUMATIC	
		FACTOR	PRODUCT	FACTOR	PRODUCT	FACTOR	PRODUCT	FACTOR	PRODUCT	FACTOR	PRODUCT
Safety Aspects-Feed Preparation	47	3.72	175	4.57	215	4.72	222	5.15	242	5.15	242
Safety Aspects-Transport	44	4.43	195	4.70	207	5.30	233	5.86	258	5.86	258
Safety Aspects-Stowing	53	4.15	220	5.00	265	5.00	265	5.43	288	6.43	341
Transport Method-Installation	44	4.57	201	4.75	209	4.75	209	4.75	209	4.75	209
Transport Method-Maintenance	43	4.58	197	4.79	206	4.79	206	4.79	206	4.79	206
Transport Method-Operation	39	4.15	162	4.38	171	4.38	171	4.38	171	4.38	171
Transport Method-Technical Feasibility	44	4.00	176	4.05	178	4.05	178	4.05	178	4.05	178
Stowing Method-Operation	37	4.14	153	6.30	233	6.70	248	6.86	254	7.70	285
Stowing Method-Technical Feasibility	41	3.71	152	6.00	246	7.29	299	7.71	316	8.15	334
Stowing Method-Manpower Skills	32	3.28	105	4.00	128	4.72	151	5.56	178	4.84	155
Stowing Method-Fill Characteristics	57	5.72	326	6.14	350	6.28	358	6.58	375	6.72	383
Stowing Method-Ground Support	52	5.42	282	5.87	305	5.87	305	6.13	319	6.71	349
Underground Environment-Dust Control	29	1.86	54	3.59	104	3.14	91	4.28	124	5.41	157
Underground Environment-Temp.-Humidity	54	5.15	278	6.00	324	5.15	278	5.57	301	6.57	355
Underground Environment-Gas	35	3.00	105	4.14	145	3.43	120	4.29	150	4.86	170
Effects on Mine Op.-Water	61	7.57	462	7.15	436	5.72	349	5.72	349	5.57	340
Effects on Mine Op.-Sched. & Interfer.	44	4.14	182	5.30	233	5.30	233	5.70	251	5.30	233
Effects on Mine Op.-Disaster Potential	70	7.57	530	7.14	500	5.00	350	5.57	390	5.86	410
Ground Water-Effect on Fill	53	5.00	265	5.85	310	5.72	303	6.15	326	6.72	356
Ground Water-Decant Water Quality	70	7.43	520	6.86	480	6.14	430	5.71	400	6.29	440
Ground Water-Ground Water Quality	56	5.57	312	5.57	312	5.00	280	5.43	305	6.14	344
Ground Water-Mine Discharge Quality	71	7.86	558	7.28	517	6.14	436	6.28	446	6.42	456
Surface Envir. Effects-Disposal Area	69	7.71	532	7.29	503	7.14	493	6.71	463	6.57	453
Surface Envir. Effects-Water Treatment	74	8.15	603	7.85	581	7.15	529	7.28	539	7.00	518
Surface Envir. Effects-Surface Fac.	54	5.57	301	6.00	324	5.72	309	6.15	332	5.43	293
TOTALS			7,046		7,482		7,046		7,369		7,636

7.6.1.1.1.1 Elements For Subjective Technical Analysis:

Twenty-five elements were selected for the subjective technical analysis of the various transport-stowing combinations. These elements were divided into seven categories which are briefly described in this section.

1. Safety

   1.1 Feed Preparation:

   Preparation of the retorted shale for the particular transport or stowing method may include cooling, slurrying, or other changes in the physical state of the material.

   1.2 Transport:

   The three basic methods and combinations present varying conditions which could be potentially hazardous.

   1.3 Stowing:

   The three general methods for placing material in the various stopes present dissimilar working conditions and require different safety considerations. Stope dimensions and roof conditions must be considered.

2. Transport:

   2.1 Installation:

   The various methods present different problems with respect to materials and equipment required and to the frequency at which the system must be extended.

   2.2 Maintenance:

   The total maintenance effort and the difficulties associated with maintaining the systems will vary from method to method.

   2.3 Operation:

   The problems associated with the operation of the various systems and the equipment which is unique to each of them are to be considered.

   2.4 Technical Feasibility:

   The state of the art and anticipated problems have to be considered.

3. Stowing:

3.1 Operation:

Stope preparation and actual stowing activities are significantly different for the basic methods and combinations of methods being investigated.

3.2 Technical Feasibility:

The state of the art and anticipated problems have to be considered.

3.3 Manpower Skills Required:

Preparation and operational problems will dictate varying skill levels for the different methods.

3.4 Fill Characteristics:

Will the fill collect water, can it be strengthened with additives, and what special considerations must be made for controlling it?

3.5 Ground Support Achieved:

The in-place density, strength, and percent of void filled have to be considered.

4. Underground Environmental Effects:

4.1 Dust Control:

Severity of dust problems and extent of dust control methods require careful analysis. Potential respiratory and carcinogenic hazards should be included.

4.2 Temperature-Humidity:

The effects of high temperatures and humidity on the workers must be evaluated. High temperatures may also adversely affect pillar strength.

4.3 Toxic and/or Explosive Gas:

Safeguards against the accumulation of explosive gas in the stope areas, along with the possible emission of toxic agents from the fill, should be considered. Toxic agents would probably be more of a problem at higher temperatures than are presently envisioned for backfilling.

5. Effects on Mining Operation:

5.1 Water:

Additional pumping capacity and possible alternate disposal facilities are required for lower quality water resulting from the backfilling operation.

5.2 Mine Scheduling and Operational Interferences:

Mining plans and sequence of operations may be affected by the backfilling activity. Additional facilities and/or coordinated use of facilities may be required.

5.3 Disaster Potential:

Methods may present potential for disaster, such as flooding due to liquefaction, if water is trapped in the backfilled areas.

6. Ground Water:

6.1 Effects of Ground Water on Emplaced Fill:

Will the fill absorb and retain a large amount of water and perhaps lose some of its stability?

6.2 Quality of Decant Water and Its Effect on the Aquifers:

Consider the possibility of saline water from the fill areas entering the ground water system.

6.3 Quality of Ground Water:

Consider the effect of potentially low quality water on the fill and on any additives which might be used for strength enhancement.

6.4 Quality of Mine Discharge:

Will the effluent from the filled areas adversely affect the quality of the water to be discharged from the mine?

7. Surface Environmental Effects:

7.1 Surface Disposal Area:

What will be the effects of surface disposal on land water, vegetation, wildlife, and aesthetics of the area?

7.2 Water Retention and/or Treatment Facilities:

What will be the effect of water treatment facilities upon the surface area?

7.3 Surface Facilities:

The effect of size and extent of required surface facilities must be considered.

7.6.1.1.2 Objective Technical Evaluation:

Six quantifiable elements were selected for the objective technical evaluation of the various transport-stowing combinations. Relative values for every transport-stowing combination were calculated for each element. Tables 7.6.1.1.2-1 and 7.6.1.1.2-2 show the input values and the relative values for each element. The next step is to assign a relative importance factor value to each element and then multiply the relative element values calculated from Table 7.6.1.1.2-1 and 7.6.1.1.2-2 by the relative importance factor for each element. These results are then totaled for each transport-stowing combination to obtain the final objective technical ranking values. For a given mining method, the final objective technical ranking values are divided by the lowest value from the group to yield the ranking value for input into the final ranking analysis. Tables 7.6.1.1.2-3 and 7.6.1.1.2-4 show the development of the final objective technical ranking values.

The relative importance factors used in Tables 7.6.1.1.2-3 and 7.6.1.1.2-4 are weighting factors for establishing the priority of each element. These factors are site-specific in that each organization may place different significance to the various elements. The values used in this report reflect the conditions and experience of the authors.

The elements elected for objective technical analysis are listed below:

1. Energy Consumption:

   Energy consumed from all sources converted to equivalent kilowatt hours.

2. Water Usage:

   Dust control, cooling, and transport requirements.

3. Crew Size:

   Total manpower requirements.

4. Retorted Shale for Surface Disposal:

   Percent of total retorted shale which must be left on the surface.

5. Fill Density:

   High density resulting in increased volume to be placed underground and increased strength for pillar and roof support.

TABLE 7.6.1.1.2-1

OBJECTIVE TECHNICAL ANALYSIS
INPUT VALUES

CHAMBER AND PILLAR MINING

Transport	Stowing	Energy Consumption KW/D x 10^3	Water Usage Gal/D x 10^3	Crew Size	Surface Disposal %	Fill Density PCF	Resource Recovery Increase %
Conveyor	Conveyor	335	1,527	263	15	90	16.0
	Pneumatic	652	916	382	30	75	13.0
	Conveyor & Pneumatic Topfill	440	1,424	339	17	87	15.6
	Hydraulic	763	13,953	570	60	65	11.0
	Hydraulic & Pneumatic Topfill	734	10,262	553	55	67	11.3
Truck	Truck	777	1,563	381	15	90	16.0
	Pneumatic	1,255	952	520	30	75	13.0
	Truck & Pneumatic Topfill	816	1,461	485	17	87	15.6
	Hydraulic	1,205	13,989	599	60	65	11.0
	Hydraulic & Pneumatic Topfill	1,176	10,298	582	55	67	11.3
Pneumatic	Vertical Pipe	983	874	380	30	75	13.0
	Borehole	877	874	353	30	75	13.0
Hydraulic	Hydraulic	457	13,837	463	60	65	11.0
	Hydraulic & Pneumatic Topfill	1,208	10,237	711	55	67	11.3
	Conveyor	1,121	13,158	651	35	90	16.0
	Conveyor & Pneumatic Topfill	1,158	13,158	703	35	87	15.6
	Pneumatic	1,373	13,158	905	35	75	13.0

TABLE 7.6.1.1.2-2

OBJECTIVE TECHNICAL ANALYSIS
INPUT VALUES

SUBLEVEL STOPING

Transport	Stowing	Energy Consumption KW/D x 10^3	Water Usage Gal/D x 10^3	Crew Size	Surface Disposal %	Fill Density PCF	Resource Recovery Increase %
Conveyor	Conveyor	335	1,180	270	24	80	15.0
	Pneumatic	652	916	382	30	75	14.0
	Conveyor & Pneumatic Topfill	440	1,135	343	27	79	14.8
	Hydraulic	763	13,953	570	60	65	12.0
	Hydraulic & Pneumatic Topfill	734	10,262	553	55	67	12.4
Truck	Truck	777	1,216	376	24	80	15.0
	Pneumatic	1,255	952	520	30	75	14.0
	Truck & Pneumatic Topfill	816	1,172	460	27	79	14.8
	Hydraulic	1,205	13,989	599	60	65	12.0
	Hydraulic & Pneumatic Topfill	1,176	10,298	582	55	67	12.4
Pneumatic	Vertical Pipe	983	874	380	30	75	14.0
	Borehole	877	874	353	30	75	14.0
Hydraulic	Hydraulic	457	13,837	463	60	65	12.0
	Hydraulic & Pneumatic Topfill	1,208	10,237	711	55	67	12.4
	Conveyor	1,121	13,158	651	35	80	15.0
	Conveyor & Pneumatic Topfill	1,158	13,158	703	35	79	14.8
	Pneumatic	1,373	13,158	905	35	75	14.0

TABLE 7.6.1.1.2-3

OBJECTIVE TECHNICAL ANALYSIS
ELEMENT FACTOR VALUES

CHAMBER AND PILLAR MINING

Transport	Stowing	Energy Consumption RI=4	Water Usage RI=4	Crew Size RI=5	Surface Disposal RI=6	Fill Density RI=4	Resource Recovery RI=5	Total	Rank
Conveyor	Conveyor	4.00	6.99	5.00	6.00	4.00	5.00	30.99	1.00
	Pneumatic	7.79	4.19	7.26	12.00	4.80	6.15	42.19	1.36
	Conveyor & Pneumatic Topfill	5.25	6.52	6.44	6.80	4.14	5.13	34.28	1.11
	Hydraulic	9.11	63.86	10.84	24.00	5.54	7.27	120.62	3.89
	Hydraulic & Pneumatic Topfill	8.76	46.97	10.51	22.00	5.37	7.08	100.69	3.25
Truck	Truck	9.28	7.15	7.24	6.00	4.00	5.00	38.67	1.25
	Pneumatic	14.99	4.36	9.89	12.00	4.80	6.15	52.19	1.68
	Truck & Pneumatic Topfill	9.74	6.69	9.22	6.80	4.14	5.13	41.72	1.35
	Hydraulic	14.39	64.02	11.39	24.00	5.54	7.27	126.61	4.09
	Hydraulic & Pneumatic Topfill	14.04	47.13	11.06	22.00	5.37	7.08	106.68	3.44
Pneumatic	Vertical Pipe	11.74	4.00	7.22	12.00	4.80	6.15	45.91	1.48
	Borehole	10.47	4.00	6.71	12.00	4.80	6.15	44.13	1.42
Hydraulic	Hydraulic	5.46	63.33	8.80	24.00	5.54	7.27	114.40	3.69
	Hydraulic & Pneumatic Topfill	14.42	46.85	13.52	22.00	5.37	7.08	104.68	3.38
	Conveyor	13.39	60.22	12.38	14.00	4.00	5.00	108.99	3.52
	Conveyor & Pneumatic Topfill	13.83	60.22	13.37	14.00	4.14	5.13	110.69	3.57
	Pneumatic	16.39	60.22	17.21	14.00	4.80	6.15	118.77	3.83

RI = Relative Importance Factor

TABLE 7.6.1.1.2-4

OBJECTIVE TECHNICAL ANALYSIS
ELEMENT FACTOR VALUES

SUBLEVEL STOPING

Transport	Stowing	Energy Consumption RI=4	Water Usage RI=4	Crew Size RI=5	Surface Disposal RI=6	Fill Density RI=4	Resource Recovery RI=5	Total	Rank
Conveyor	Conveyor	4.00	5.40	5.00	6.00	4.00	5.00	29.40	1.00
	Pneumatic	7.79	4.19	7.07	7.50	4.27	5.36	36.18	1.23
	Conveyor & Pneumatic Topfill	5.25	5.19	6.35	6.75	4.05	5.07	32.66	1.11
	Hydraulic	9.11	63.86	10.56	15.00	4.92	6.25	109.70	3.73
	Hydraulic & Pneumatic Topfill	8.76	46.97	10.24	13.75	4.78	6.05	90.55	3.08
Truck	Truck	9.28	5.57	6.96	6.00	4.00	5.00	36.81	1.25
	Pneumatic	14.99	4.36	9.63	7.50	4.27	5.36	46.11	1.57
	Truck & Pneumatic Topfill	9.74	5.36	8.52	6.75	4.05	5.07	39.49	1.34
	Hydraulic	14.39	64.02	11.09	15.00	4.92	6.25	115.67	3.93
	Hydraulic & Pneumatic Topfill	14.04	47.13	10.78	13.75	4.78	6.05	96.53	3.28
Pneumatic	Vertical Pipe	11.74	4.00	7.04	7.50	4.27	5.36	39.91	1.36
	Borehole	10.47	4.00	6.54	7.50	4.27	5.36	38.14	1.30
Hydraulic	Hydraulic	5.46	63.33	8.57	15.00	4.92	6.25	103.53	3.52
	Hydraulic & Pneumatic Topfill	14.42	46.85	13.17	13.75	4.78	6.05	99.02	3.37
	Conveyor	13.39	60.22	12.06	8.75	4.00	5.00	103.42	3.52
	Conveyor & Pneumatic Topfill	13.83	60.22	13.02	8.75	4.05	5.07	104.94	3.57
	Pneumatic	16.39	60.22	16.76	8.75	4.27	5.36	111.75	3.80

RI = Relative Importance Factor

6. Resource Recovery Increase:

Backfilling which permits reduction of pillar size with resulting increase in resource recovery.

7.6.1.1.3 Economic Analysis:

The economic analysis of each transport-stowing system is made up of two elements, capital costs and operating costs. Each is important enough to be considered as a separate element in the final analysis. The calculations for the ranking factors are similar to the objective technical analysis except that no relative importance factors are needed at this point. Tables 7.7.1.1.3-1 and 7.7.1.1.3-2 show the ranking factor development for capital and operating costs for each transport-stowing system for the two mining methods.

7.6.2 Selection:

The final phase of the ranking analysis is the selection of the most desirable system of transport-stowing for each mining method. In this phase the ranking factor for every element of each transport-stowing system is multiplied by the relative importance factor assigned to each element. These products are then totaled and the system with the lowest sum is considered to be the most desirable system for that mining method. Tables 7.6.2-1 and 7.6.2-2 present the final phase of the ranking analysis.

Based on this analysis, conveyor transport and stowing is the system selected for the underground disposal of retorted oil shale for both mining methods studied.

7.7 Phase I Conclusions and Recommendations:

Completion of the Scope of Work for Phase I involved several activities which included a literature search, site visits, vendor contacts, laboratory work, and engineering analysis. The literature search and site visits provided input covering the state of the art and the operating problems of underground waste disposal. Equipment capabilities and costs were obtained through the vendor contacts and in-house sources. Limited laboratory work was conducted to determine the hydraulic characteristics of Paraho retorted shale and the effects of cementing additives on the strength of the material. Engineering analyses included mine and disposal system layout, preliminary system analysis and design, operating and capital cost determinations, effects of production rate on underground waste disposal economics, and the ranking analysis to select the most desirable system for each mining method.

7.7.1 Conclusions:

Conveyor transport and stowing is the most feasible system studied for underground disposal of retorted shale using either a chamber and pillar or sublevel stoping mining method. This analysis is based on a deep mine in the Piceance Creek Basin and the use of retorted shale from the Paraho direct-heating-mode of retorting. When compared to the other alternatives, this system has the following advantages:

TABLE 7.6.1.1.3-1

ECONOMIC ANALYSIS
RANKING FACTORS

CHAMBER AND PILLAR MINING

Transport	Stowing	Capital Costs		Operating Costs	
		$\$ \times 10^3$	Rank	$/Ton	Rank
Conveyor	Conveyor	19,670	1.00	0.3101	1.00
	Pneumatic	34,186	1.74	0.9632	3.11
	Conveyor & Pneumatic Topfill	23,015	1.17	0.4221	1.36
	Hydraulic	100,106	5.09	1.0785	3.48
	Hydraulic & Pneumatic Topfill	91,883	4.67	1.0021	3.23
Truck	Truck	43,895	2.23	0.5417	1.75
	Pneumatic	56,449	2.87	1.2056	3.89
	Truck & Pneumatic Topfill	48,265	2.45	0.6537	2.11
	Hydraulic	114,780	5.84	1.1924	3.85
	Hydraulic & Pneumatic Topfill	110,265	5.61	1.1744	3.79
Pneumatic	Vertical Pipe	60,883	3.10	1.1855	3.82
	Borehole	43,295	2.20	1.0117	3.26
Hydraulic	Hydraulic	85,908	4.37	0.7640	2.46
	Hydraulic & Pneumatic Topfill	131,797	6.70	1.0212	3.29
	Conveyor	131,940	6.71	1.1748	3.79
	Conveyor & Pneumatic Topfill	135,329	6.88	1.2868	4.15
	Pneumatic	142,203	7.23	1.8082	5.83

TABLE 7.6.1.1.3-2

ECONOMIC ANALYSIS
RANKING FACTORS

SUBLEVEL STOPING

Transport	Stowing	Capital Costs		Operating Costs	
		$\$ \times 10^3$	Rank	\$/Ton	Rank
Conveyor	Conveyor	22,328	1.00	0.3162	1.00
	Pneumatic	34,186	1.53	0.9632	3.05
	Conveyor & Pneumatic Topfill	25,557	1.14	0.4276	1.35
	Hydraulic	100,106	4.48	1.0785	3.41
	Hydraulic & Pneumatic Topfill	91,883	4.12	1.0021	3.17
Truck	Truck	45,665	2.05	0.5331	1.69
	Pneumatic	56,449	2.53	1.2056	3.81
	Truck & Pneumatic Topfill	44,575	2.00	0.6431	2.03
	Hydraulic	114,780	5.14	1.1924	3.77
	Hydraulic & Pneumatic Topfill	110,265	4.94	1.1744	3.71
Pneumatic	Vertical Pipe	60,883	2.73	1.1855	3.75
	Borehole	43,295	1.94	1.0117	3.20
Hydraulic	Hydraulic	85,908	3.85	0.7640	2.42
	Hydraulic & Pneumatic Topfill	131,797	5.90	1.0212	3.23
	Conveyor	131,940	5.91	1.1748	3.72
	Conveyor & Pneumatic Topfill	135,329	6.06	1.2868	4.07
	Pneumatic	142,203	6.37	1.8082	5.72

TABLE 7.6.2-1

RANKING ANALYSIS
FINAL SELECTION

CHAMBER AND PILLAR MINING

Transport	Stowing	Subjective Technical Analysis RI=6	Objective Technical Analysis RI=3	Capital Costs RI=3	Operating Costs RI=1	Total	Final Rank	Final Position
Conveyor	Conveyor	6.00	3.00	3.00	1.00	13.00	1.00	1
	Pneumatic	8.66	4.08	5.21	3.11	21.06	1.62	5
	Conveyor & Pneumatic Topfill	7.29	3.33	3.51	1.36	15.49	1.19	2
	Hydraulic	9.12	11.67	15.27	3.48	39.54	3.04	11
	Hydraulic & Pneumatic Topfill	9.54	9.75	14.01	3.23	36.53	2.81	9
Truck	Truck	6.08	3,75	6.69	1.75	18.27	1.41	3
	Pneumatic	8.36	5.04	8.61	3.89	25.90	1.99	7
	Truck & Pneumatic Topfill	7.27	4.05	7.36	2.11	20.79	1.60	4
	Hydraulic	8.74	12.27	17.51	3.85	42.37	3.26	13
	Hydraulic & Pneumatic Topfill	9.18	10.32	16.82	3.79	40.11	3.09	12
Pneumatic	Vertical Pipe	10.72	4.44	9.29	3.82	28.27	2.17	8
	Borehole	9.95	4.26	6.60	3.26	24.07	1.85	6
Hydraulic	Hydraulic	10.51	11.07	13.10	2.46	37.14	2.86	10
	Hydraulic & Pneumatic Topfill	11.17	10.14	20.10	3.29	44.70	3.44	14
	Conveyor	10.54	10.56	20.12	3.79	45.01	3.46	15
	Conveyor & Pneumatic Topfill	11.09	10.71	20.64	4.15	46.59	3.58	16
	Pneumatic	11.38	11.49	21.69	5.83	50.39	3.88	17

RI = Relative Importance Factor

TABLE 7.6.2-2

RANKING ANALYSIS
FINAL SELECTION

SUBLEVEL STOPING

Transport	Stowing	Subjective Technical Analysis RI=6	Objective Technical Analysis RI=3	Capital Costs RI=3	Operating Costs RI=1	Total	Final Rank	Final Position
Conveyor	Conveyor	6.00	3.00	3.00	1.00	13.00	1.00	1
	Pneumatic	8.14	3.60	4.59	3.05	19.47	1.50	5
	Conveyor & Pneumatic Topfill	7.16	3.33	3.43	1.35	15.27	1.17	2
	Hydraulic	8.61	11.19	13.45	3.41	36.66	2.82	11
	Hydraulic & Pneumatic Topfill	8.95	9.24	12.35	3.17	33.71	2.59	9
Truck	Truck	6.20	3.75	6.14	1.69	17.78	1.37	3
	Pneumatic	7.92	4.71	7.58	3.81	24.02	1.85	7
	Truck & Pneumatic Topfill	7.11	4.02	5.99	2.03	19.15	1.47	4
	Hydraulic	8.23	11.79	15.42	3.77	39.21	3.02	13
	Hydraulic & Pneumatic Topfill	8.65	9.84	14.82	3.71	37.02	2.85	12
Pneumatic	Vertical Pipe	10.16	4.08	8.18	3.75	26.17	2.01	8
	Borehole	9.34	3.90	5.82	3.20	22.26	1.71	6
Hydraulic	Hydraulic	9.79	10.56	11.54	2.42	34.31	2.64	10
	Hydraulic & Pneumatic Topfill	10.39	10.11	17.71	3.23	41.44	3.19	14
	Conveyor	9.79	10.56	17.73	3.72	41.80	3.22	15
	Conveyor & Pneumatic Topfill	10.24	10.71	18.18	4.07	43.20	3.32	16
	Pneumatic	10.61	11.40	19.11	5.72	46.84	3.60	17

RI = Relative Importance Factor

° Highest fill density

° Most retorted shale placed underground

° Highest pillar support potential

° Greatest increase in resource recovery potential

° Lowest manpower requirement

° Lowest energy requirement

° Lowest capital and operating costs

° Least potential for ground water contamination

° Least surface disruption

° Least environmental degradation

° Safest overall method

The relatively high fill density obtainable, especially in chamber and pillar mining where mechanical compaction will be used, is the retorted shale placed underground, and low potential for fill liquefaction. The conveyor system has relatively high initial capital requirements, but low operating and labor costs, making the system economically attractive. The system is technically feasible and is readily adaptable to alternative stowing techniques. Manpower requirements for conveyor systems are low and space for mechanization of the conveyor extensions and moves is available. Self-contained extensible belt systems are available for the stowing activity.

Hydraulic transport and stowing systems are not feasible for several reasons. Extreme degradation occurs when the retorted shale is slurried and pumped and the resulting slimes disposal problems are economically and environmentally prohibitive. Design and construction of reliable bulkheads to close off large openings and to confine large amounts of saturated backfill material would be extremely difficult and expensive. In addition, high energy and water requirements are inherent to a hydraulic system.

A pneumatic transport system has many drawbacks -- both economical and operational. Capital and operating costs are excessive. Energy consumption is high due to high horsepower per ton requirements. The system does not lend itself to large-scale commercial applications. Many blower-feeder units and pipeline networks would be required to attain the necessary transport and stowage rates. Under certain conditions, however, a pneumatic stowage system does have merit. It may be used as a final step to completely fill and pack the retorted shale against the stope roof.

7.7.2 Recommendations:

Phase II addresses the detailed design, engineering analyses, and economics of the selected systems. Based on this evaluation, it is recommended that an alternative stowing combination be considered for the chamber and pillar mining method. A combined conveyor stowing system with pneumatic topfilling, which permits complete stope filling, results in a greater potential for subsidence control than conveyor stowing alone. This combined method should be investigated as an alternative to conveyor stowing.

As a further aid to maximum resource recovery, additional work in the area of dry retorted shale strength augmentation with additives is justified. The self-cementing characteristics of Paraho retorted shale have not been adequately defined. Therefore, more work is recommended to relate retort operating temperature to the degree of self-cementation that may be possible in the retorted shale. In addition, leaching characteristics of dry retorted shale should be investigated more extensively.

8.0 MATERIAL TESTING

The physical and chemical characteristics of retorted shale from the Paraho direct-heated retort that affect underground backfilling were reviewed and analyzed. Physical properties that have been investigated include: size distribution, cooling characteristics, compactability, effects of slurry transport, dewatering potential, effects of cementing and flocculant additives, and amenability to mass flow. The chemical properties investigated were mainly in the areas of leaching and self-cementing potentials.

A significant difference in the size distribution of the retorted shale was observed from one sample group to another. This is due mainly to the varying operating conditions in the retort for the different sampling periods and a possible difference in the raw shale feed. Degradation after sampling probably varied for the individual samples.

The slurry pumping tests subjected the retorted shale to severe degradation as it was cycled repeatedly through the pump. Therefore, the results of the dewatering tests are not conclusive.

8.1 Size Distribution:

Retorted shale samples were taken at two different times during the Paraho Demonstration Program and once during the subsequent extended production program. The first sample was tested as a part of U. S. Bureau of Mines Contract No. J0255004 and the results are listed in Table 8.1-1. A second sample, consisting of approximately 40 tons, was placed in steel drums and saved for several testing programs. Results of tests from three different laboratories are shown in Table 8.1-1. The final sample was taken during the subsequent extended production run. The results of tests from this sample are also shown in Table 8.1-1. In addition, Figure 8.1-1 shows the size distribution curves for samples from three different retort operation periods.

8.2 Cooling:

Tests were conducted to determine the heat capacity and, ultimately, the ability to cool retorted oil shale from 400°F to 100°F with minimal water consumption. The heat capacity was determined by differential scanning colorimetry. Table 8.2-1 shows the results of this analysis.

Based on bench scale cooling tests, the following parameters were developed and used in the design of the rotary air swept cooling units to cool retorted shale from 400°F to 100°F.

Retention time:	8 minutes
Air velocity:	500 feet/minute
Heat capacity:	0.28 BTU/Lb/°F

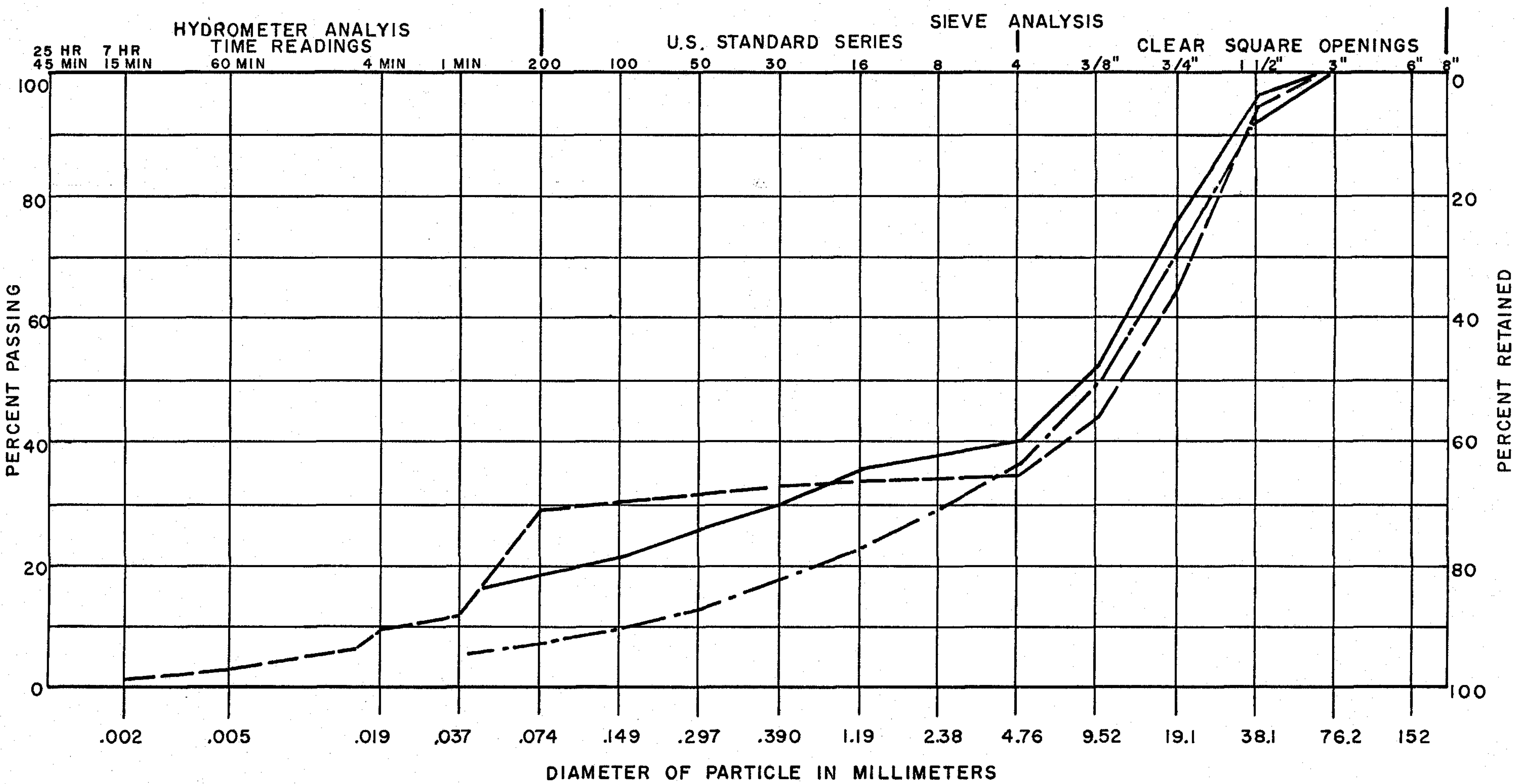
TABLE 8.1-1

SIEVE ANALYSIS

PARAHO SEMI-WORKS PLANT

Sieve Size	Demonstration Program				Extended Run
	Sample 1	Sample 2			
	Lab A (2)	Lab B (1)	Lab C (3)	Lab D (1)	Lab E (5)
	Range Percent Passing Each Sieve Size				
3"	100 - 100	100	100 - 100	100	100 - 100
1-1/2"	96 - 96	92	89 - 94	91	88 - 97
3/4"	76 - 77	72	65 - 78	70	53 - 76
3/8"	48 - 57	50	39 - 60	51	36 - 54
4M	34 - 45	37	28 - 46	37	27 - 44
8M	32 - 44	30	21 - 38	27	27 - 44
16M	29 - 43	23	17 - 30	20	26 - 44
30M	24 - 37	18	12 - 24	13	25 - 43
50M	18 - 33	14	7 - 20	9	23 - 41
100M	15 - 27	10	5 - 16	7	20 - 39
200M	13 - 22	8	3 - 13	5	18 - 36
325M	12 - 21	6	2 - 10	3	13 - 23

(2) Number of Individual Samples Tested



SIZE DISTRIBUTION - PARAHO RETORTED SHALE

FIGURE 8.1-1

TABLE 8.2-1

HEAT CAPACITY FOR PARAHO RETORTED OIL SHALE

Temperature (°F)	Capacity (BTU/Lb/°F)
100	0.269
150	0.280
200	0.286
250	0.290
300	0.289
350	0.287
400	0.289

8.3 Modification of Physical Properties:

Several methods of modifying the physical properties of retorted oil shale were investigated. The effects of compaction and chemical stabilization additives were studied. The results varied from substantial improvements to no change, and to deterioration of retorted oil shale with respect to its strength and load-carrying capacity.

8.3.1 Compaction:

Results of the compaction tests are found in Section 3.5.1.1.

8.3.2 Additives:

The effects of additives on both hydraulically pumped retorted shale and dry retorted shale were investigated by performing a series of laboratory tests. The primary purpose of the tests was to identify beneficial trends and not to obtain statistically significant results. Different percentages of flocculants were added to hydraulically pumped retorted shale to determine whether improvements in percolation and/or decantation rates would result. Two different ratios of cementing agents to retorted shale were investigated as to their effect on unconfined compressive strength. Details regarding test material, test equipment, experimental procedure, and results obtained are given in Appendix D.

8.3.2.1 Hydraulically Pumped Retorted Shale:

Paraho retorted shale was pumped at 48 percent solids-by-weight slurry for one hour at 10 feet per second to simulate a hydraulic backfilling operation. Results obtained from laboratory tests on these materials are as follows:

(1) The addition of flocculants can improve dewatering characteristics of the fill; however, due to the large slime size fraction in the material, a large amount of moisture will remain entrapped.

(2) Calcium chloride did not improve dewatering characteristics, whereas two pounds of Separan MGL per ton of solids produced an increase in water released from the specimen from 7.5 percent to 21.7 percent of the available water.

(3) Cementing agents did not have any beneficial effect on strength characteristics.

8.3.2.2 Dry Retorted Shale:

A mixture of 50 percent plus 4-mesh material and 50 percent minus 4-mesh material, with a maximum particle size of 0.5 inch, was used to simulate mechanical backfilling operations. Results obtained from laboratory tests on these materials are as follows:

(1) Increasing moisture content from 15 percent to 25 percent produced significant increases in compressive strength.

(2) With an eight-day cure, a 5 to 1 retorted shale-cement mixture produced a large increase in compressive strength, whereas a 30 to 1 mixture did not have any effect. A mixture containing equal amounts of cement, lime, and flyash produced noticeable improvements in compressive strength at 15 percent moisture, but little additional improvement at 25 percent moisture.

(3) Distribution of moisture, cementing agent, fines affects compressive strength. Curing period and curing environment are two other factors that exert a significant effect on compressive strength and deserve further study.

8.3.2.3 Other Tests:

The tests mentioned above were conducted to supplement the results obtained under USBM Contract No. J0255004. Under that contract, only dry retorted shale was tested and the major results obtained are as follows (14):

(1) Treatment with five percent hydrated lime promotes cementing, thus, increasing compressive strength and decreasing permeability rate.

(2) Accelerated curing times promote reactions of calcium and magnesium oxide present in Paraho retorted shale, increasing compressive strength from 17 psi to 125 psi with a 28-day curing period at 125°F.

8.3.3 Summary:

Compaction of stowed, retorted shale improves its strength and resistance to saturation by ground water, and increases the amount that can be stored underground. Some self-cementing is possible when moisture is added prior to compaction. However, more work is needed to determine the ultimate degree of self-cementation and the retorting conditions that will produce this and other favorable characteristics. Adequate compaction is possible using a compactor with attached blade to spread the material into one-foot lifts as it is compacted.

Flocculants and cementing agents do not produce significant improvements in physical properties of hydraulically pumped, retorted shale, and this can be attributed to the large slime-size fraction in the material. In contrast, the amounts of moisture and cementing agent that are added to dry retorted shale have a distinct effect on the strength characteristics of the fill material.

8.4 Mass Flow Characteristics:

Results of the lab tests performed in the investigation of the mass-flow characteristics of the Paraho retorted shale are discussed in Section 3.2.1.

8.5 Dewatering Characteristics and Chemical Analysis:

A series of tests were conducted to determine the dewatering characteristics of hydraulically placed, retorted oil shale. The leachate from these tests was analyzed to determine the extent of the chemical change in the retorted shale and the effluent. A column settling test was also done.

8.5.1 Dewatering Characteristics:

Large-scale drainage tests were conducted using two, four feet square by eight feet high, plywood boxes. One box had a vertical perforated pipe drain, while the other had a perforated bottom drain. Horizontal percolation and vertical flow drainage were simulated in these tests. Table 8.5.1-1 shows the results of these tests.

TABLE 8.5.1-1

BOX DRAINAGE TESTS

	Pipe Drain	Bottom Drain
Percent Solids by Weight - Slurry	37.6	34.7
Percent Solids by Weight - Drained Material	64.1	69.1
Bulk Density - Drained Material (Wet Basis)	104.9	93.7
Percent Water in Slurry - Drained	71	67

A column settling test was performed in a four-inch-diameter column that was 17.4 feet high. The solids level dropped to 16.6 feet in 35 hours, after which the settling was negligible. The retorted shale did not settle out appreciably and the rate of settlement decreased rapidly with time. Table 8.5.1-2 and Figure 8.5-1 show the results of the settling test.

TABLE 8.5.1-2

COLUMN SETTLING TEST

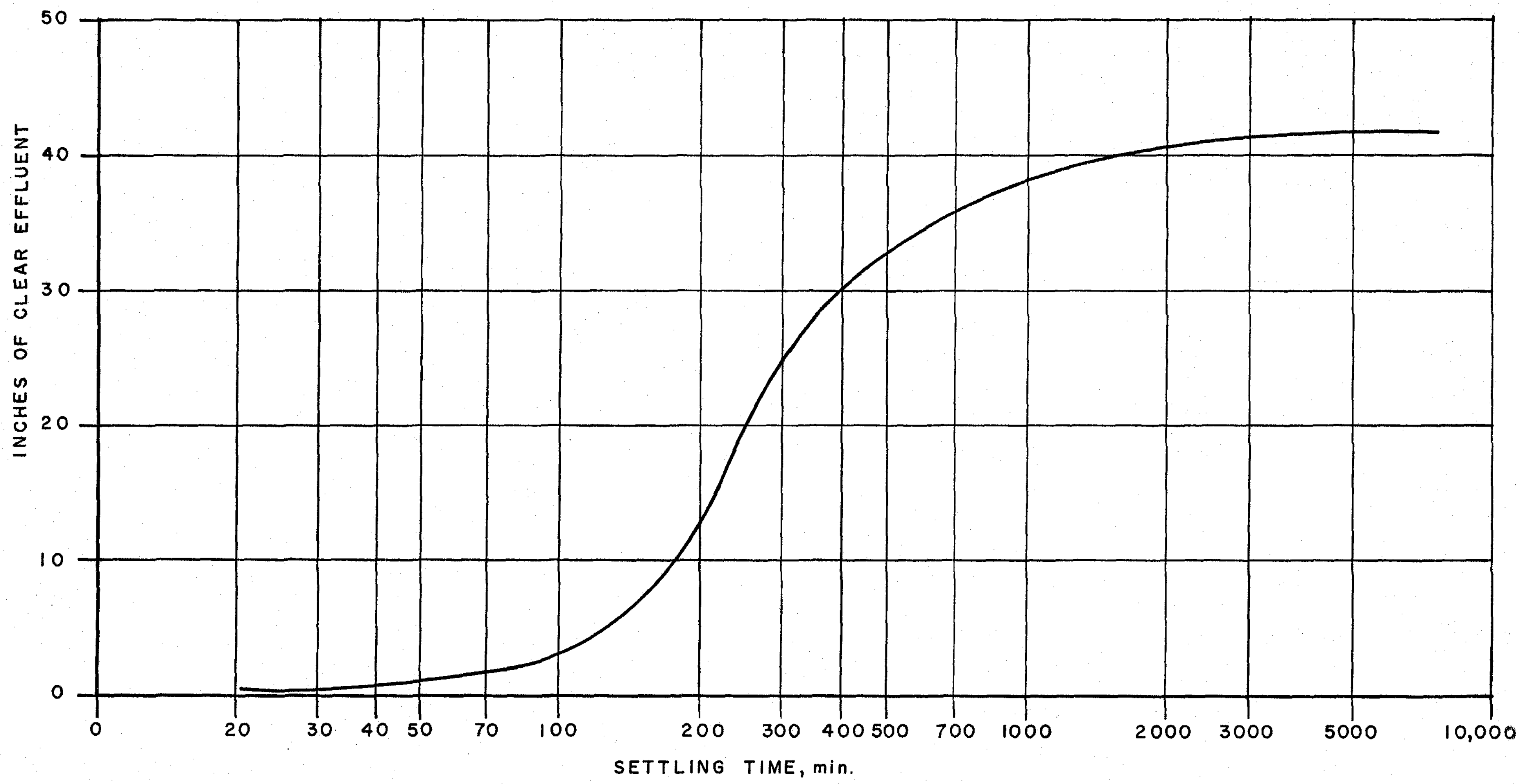
Sample Interval (Inches From Bottom of Column)	% Solids By Weight
Slurry Feed	53.2
167 - 120	64.7
120 - 96	60.5
96 - 72	58.1
72 - 48	58.7
48 - 24	59.3
24 - 0	63.5

8.5.2 Chemical Analysis:

The effects of hydraulic transporting and stowing on the chemical composition of Paraho retorted oil shale were investigated. In addition, leachate from drainage tests was analyzed although leaching was not as great as expected. Table 8.5.2-1 shows the results of this work.

8.6 Abrasive Properties:

The abrasive character of retorted oil shale was determined using the Bond Abrasion Index Method. The computed Bond Abrasion Index is 0.006 which indicates that the material is not significantly abrasive when compared to an index value of 0.016 for dolomite or 0.624 for taconite.



PARTICLE SETTLING RATES FOR HYDRAULICALLY STOWED PARAHO RETORTED SHALE

FIGURE 8.5-1

TABLE 8.5.2-1

CHEMICAL ANALYSIS OF LEACHATE - HYDRAULIC BACKFILL

	WEIGHT PERCENT				
		Pipe Drain Box		Filter Bottom Box	
Component	Unpumped Paraho Oil Shale	Drainage Water Composite	Calculated Residue Pulp Dry Basis	Drainage Water Composite	Calculated Residue Pulp Dry Basis
S	0.555	0.009	0.54	0.0085	0.54
CaO	16.1	0.0462	16.05	0.0259	16.08
MgO	7.05	Nil (3)	7.06	Nil (3)	7.06
Cl	Nil (1)	0.0029	Nil	0.0042	Nil
Br	Nil (2)	Nil (4)	Nil	Nil (4)	Nil
Na	1.815	0.0400	1.75	0.0490	1.73
Fe	2.285	Nil (5)	2.29	Nil (5)	2.29
K	0.61	0.0008	0.61	0.0010	0.61
Al	4.995	Nil (6)	5.00	Nil (6)	5.00
Moisture	0.45	--	--	--	--
LOI (1000°C)	19.4	--	--	--	--
Total Dissolved Solids	--	0.22	--	0.2	--

(1) Actual analysis - <0.05%
(2) Actual analysis - <0.001%
(3) Actual analysis - <0.2 ppm
(4) Actual analysis - <0.1 ppm
(5) Actual analysis - <0.5 ppm
(6) Actual analysis - <5 ppm

9.0 CONCLUSIONS AND RECOMMENDATIONS

The purpose of this contracted study was to investigate the possibilities for underground disposal of retorted oil shale from the Paraho retorting process. Two methods, chamber and pillar mining and sublevel stoping, were deemed most likely to be used for mining the deeper oil shale deposits of the Piceance Creek Basin in northwestern Colorado. Completion of the Scope of Work for the contract included an extensive literature search, site visits, manufacturer contacts, laboratory work, engineering analysis, and economic evaluation.

The literature search and site visits provided input covering the state of the art and the operating problems of underground waste disposal, and information on the physical and chemical properties of Paraho retorted shale. Equipment capabilities and costs were obtained through manufacturer contacts and in-house sources. Limited laboratory work was conducted to determine the hydraulic characteristics, the effects of cementing additives on the strength of the material, mass flow characteristics, and the cooling properties of Paraho retorted shale. Engineering analyses included mine and disposal system layout, systems analysis and design, effects of production rate on underground waste disposal economics, and the ranking analysis to select the most feasible disposal system for each mining method. The economic evaluation covered operating and capital cost determinations, present worth analysis of the costs for the selected systems, and the comparison of surface versus underground disposal of retorted shale.

9.1 Conclusions:

Conveyor transport and stowing is the most feasible system studied for underground disposal of Paraho retorted oil shale using either chamber and pillar mining or sublevel stoping. Based on fourth quarter 1977 dollars, underground disposal costs will be approximately $0.80 per ton of retorted shale or $1.10 per barrel of shale oil produced. Total surface disposal costs are about $0.40 per ton of retorted shale or $0.55 per barrel of shale oil. Underground disposal of retorted shale requires only 15 to 30 percent of the surface area that would be required for total surface disposal, which lessens the surface environmental impacts of the mining and retorting facility. The potential for surface subsidence is reduced significantly when the mined areas are backfilled. An increase in resource recovery of approximately 15 percent is possible since the stabilizing effect of backfilling permits the use of relatively thin pillars in the mining areas.

The alternative backfilling methods studied include mechanical, hydraulic, and pneumatic transport and stowing systems and several combinations of these basic systems. The conveyor transport and stowing systems, when compared to the alternatives studied, have the following advantages:

- Highest fill density
- Most retorted shale placement underground
- Highest pillar support potential

- Greatest potential for increase in resource recovery
- Lowest manpower requirement
- Lowest energy requirement
- Lowest capital and operating costs
- Least potential for ground-water contamination
- Least surface disruption
- Least environmental degradation
- Safest overall method

Thus, a conveyor transport and stowing system has many advantages. The relatively high fill density obtainable, especially in chamber and pillar mining where mechanical compaction can be used, is the principal reason for maximum support and resource recovery, large volume of retorted shale placed underground, and low potential for fill liquefaction. The conveyor system has relatively high initial capital requirements, but low operating and labor costs, making the system economically attractive overall. The system is technically feasible and is readily adaptable to alternative stowing techniques. Manpower requirements for conveyor systems are low and space is available for mechanization of conveyor extensions and relocation. Self-contained extensible belt systems are available for the stowing activity.

Hydraulic transport and stowing are not feasible for several reasons. Excessive degradation occurs when the retorted shale is slurried and pumped, and the resulting slimes disposal is economically and environmentally prohibitive. Design and construction of reliable bulkheads to close off large openings and to confine large amounts of saturated backfill material would be extremely difficult and expensive. In addition, a hydraulic system has inherently high energy and water requirements.

A pneumatic transport system has many drawbacks -- both economical and operational. Capital and operating costs are excessive. Energy consumption is high due to high horsepower per ton requirements. The system does not lend itself to large-scale commercial applications. Many blower-feeder units and pipeline networks would be required to attain the necessary transport and stowage rates. Under certain conditions, however, a pneumatic stowage system may be used as a final step to completely fill and pack the retorted shale against the stope roof.

9.2 Recommendations

Additional work is needed to better define the hydraulic characteristics of Paraho retorted shale. The initial work that was done subjected the retorted shale to repeated cycles in a closed loop system in which the pump caused recurrent and ever-increasing degradation to the shale.

As a further aid to maximum resource recovery, additional work directed at augmenting the strength of dry retorted shale by the use of additives is justified. The self-cementing characteristics of Paraho retorted shale have not been adequately defined. Therefore, more work is recommended to determine the relationships between retort operating temperature and the degree of self-cementation that is possible in the retorted shale. The Bureau of Mines has awarded Contract No. J0285001, "Natural Cementation of Retorted Oil Shale," for investigating this property.

Since the start of this contract, the trend in oil shale mining methods has shifted from the more conventional systems to modified in situ retorting. In situ retorting requires that approximately 20 to 25 percent of the oil shale from each retort be mined prior to rubblization of the retort and extraction of the contained shale oil. The mined portion, in most cases, will be retorted in surface retorts and will require some method for disposal of the retorted shale. Disposal in depleted in situ retorts would seem to be most advantageous from an environmental standpoint. A program to evaluate the environmental effects, technical feasibility, and costs for such an underground disposal system is recommended.

10.0 REFERENCES

The reference section is divided into two parts for convenience. The first section lists the sources of information that have been explicitly referred to in the report. The second section is a general bibliography that was compiled as a part of the literature search and problem definition activities.

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APPENDIX A

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APPENDIX A

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- Fairchild, Inc.
- Fisher and Porter
- FMC Corporation
- Frontier Industrial Controls Corp.
- Gardner-Denver Company
- GCA/Technology Division
- Getman Corporation
- Goodyear Tire & Rubber Company
- Grand Junction Machine & Plastics Co.
- Hecla Mining Company
- Hewitt-Robins
- Ingersoll-Rand Company
- Johnson-March Corporation
- Joy Manufacturing Company
- Kaman Bearing Company
- Kerr-McGee Nuclear Corp.
- Lafayette Engineering & Manufacturing, Inc.
- Lake Shore, Inc.
- Long-Airdox Company
- Marconaflo, Inc.
- Mine Safety Appliance Company
- National Iron Company
- Ohmart Corporation
- Portadrill, Inc.
- Radar Canada Ltd/Radmark Div.
- Split Mountain Pipe & Steel
- Stansteel Corporation
- Stephens-Adamson, Inc.
- Wagner Equipment Company
- Western Slope Iron & Supply
- Witco Chemical Company

APPENDIX B

SAMPLE CALCULATIONS

APPENDIX B

Sample Calculation - Surface Runoff

Reference: Handbook of Steel Drainage and Highway Construction Products

ASSUME:

1) Drainage area A = 600 acres
2) Weighted runoff factor C = 0.35
3) Duration 100 year storm = 45 minutes

From Figure 4-5 of Reference: Rainfall intensity i = 4.1 inches per hour

$$Q = C\ i\ A$$

$$Q = 0.35 \times 4.1 \times 600 = 861 \text{ cfs}$$

Catchment basin volume required:

$$V = 861\ \frac{\text{feet}^3}{\text{second}} \times 60\ \frac{\text{seconds}}{\text{minute}} \times 45 \text{ minutes} \times \frac{1 \text{ AF}}{43{,}560 \text{ feet}^3} = 53.4 \text{ AF}$$

APPENDIX B

Parameters used for comparison of methods for underground disposal of retorted shale from the Paraho retorting process.

- Oil production - 50,000 barrels per day
- Retort efficiency factor - .95 (REF)
- Raw shale grade - 28 gallons per ton
- Retorted shale weight loss factor - .82 (WLF)
- Moisture from dust and cooling - .05 (MF)
- Production schedule for retort - 365 days per year at rated capacity
- All surplus retorted shale will be disposed of on the surface

Calculation to determine daily retorted shale produced:

$$\text{Tons/Day} = \frac{50{,}000\ \text{bbl}}{\text{day}} \times \frac{42\ \text{gal}}{\text{bbl}} \times \frac{\text{ton}}{28\ \text{gal}} \times \frac{1}{0.95\ (\text{REF})} \times \underline{0.82\ (\text{WLF})}$$

$$\times\ 1.05\ (\text{MF}) = 67{,}974$$

Use 68,000 tons/day material to disposal system

APPENDIX B

Increase in extraction ratio with backfilling:

Assumptions and Symbols:

ASSUMPTIONS:

1. Rock strength parameters are from USBM holes 01-A and 02-A, Horse Draw, Piceance Basin, Colorado.
2. Dimensions used in the calculations are compatible with those that may be used in a commercial-scale mine.
3. Backfilling will provide pillar confinement and may eventually support the overburden load.
4. Pillars are designed to fail gradually, subsequent to backfilling, and create lateral stresses in the fill.

SYMBOLS:

L = Span

Sv = Vertical overburden stress

Co = Compressive strength of rock, determined in the laboratory on specimens having a height to diameter ratio of 2.0

Hp = Height of pillar

Wp = Width of pillar

Cp = Compressive strength of pillar

R = Extraction ratio

Sp = Vertical pillar stress

F = Factor of safety

APPENDIX B

SAMPLE CALCULATION:

Increase in extraction ratio with backfilling:

1) Chamber and Pillar Mining With Backfill:

Without backfill, the factor of safety for pillars should be between 2.0 and 2.5. The pillar size is calculated as follows:

L = Span = 70 feet

$S_v = \gamma H = 0.91 \quad 2{,}000 = 1{,}820$ psi

$C_o = 6{,}518$ psi

$H_p = 60$ feet, $W_p = 60$ feet

$$C_p = \frac{C_o}{.889}\{0.778 + 0.222 \quad (W_p/H_p)\} = 7{,}332 \text{ psi}$$

$R = L/(L + W_p) = 70/(70 + 60) = 0.54$

$S_p = S_v/(1 - R) = 1{,}820/0.46 = 3{,}956$ psi

$F = C_p/S_p = 7{,}332/3{,}956 = 1.85$

For $W_p = 75$ feet, $R = 0.48$

$F = 7{,}739/3{,}500 = 2.2$ (required for mining without backfill)

With backfilling, F can be 1.2 and thus $W_p = 40$ feet, $R = 0.64$. Thus, increase in panel extraction ratio due to backfill is $R = (64 - 48) = 16\%$.

For mechanical filling (90 pcf), increase in R = 16%

For pneumatic filling (75 pcf), increase in R = 13%

For hydraulic filling (65 pcf), increase in R = 11%

2) Sublevel Stoping With Backfill:

Without backfill, the factor of safety for pillars should be at least 2.0. The pillar size is calculated as follows:

L = 80 feet

Sv = γH = 0.91 x 2,000 = 1,820 psi

Co = 6,300 psi

Hp = 65 feet, Wp = 80 feet

R = 0.50

Cp = 7,450 psi

Sp = 1,820/0.5 = 3,640 psi

F = Cp/Sp = 7,450/3,640 = 2.04

With backfilling, F can be about 1.2, thus Wp = 40 feet and R = 0.67. Thus increase in panel extraction ratio due to backfill is R = (67 - 50) = 17%.

For mechanical backfilling (90 pcf), increase in R = 17%

For mechanical backfilling (80 pcf), increase in R = 15%

For pneumatic backfilling (75 pcf), increase in R = 14%

For hydraulic backfilling (65 pcf), increase in R = 12%

The extraction ratio may be increased further by robbing barrier pillars, if it is determined that support performance of backfill is such that it relieves the load on barrier pillars.

APPENDIX B

SAMPLE CALCULATIONS:

Effect of resource recovery:

$$80{,}000\ \frac{\text{tons}}{\text{day}} \times 7\ \frac{\text{days}}{\text{week}} \times 52\ \frac{\text{weeks}}{\text{year}} \times 20\ \frac{\text{years}}{\text{project}} = 582.4 \times 10^6 \text{ tons mined}$$

$$\frac{582.4 \times 10^6 \text{ tons}}{20 \text{ years}} = 29{,}120{,}000 \text{ tons per year}$$

Assume 16% increase in recovery -

$582.4 \times 10^6 \times 0.16 = 93{,}184{,}000$ tons additional reserves

Assume $15 per barrel -

$$93{,}184{,}000 \text{ tons} \times \frac{28 \text{ gal.}}{\text{tons}} \times \frac{1 \text{ bbl}}{42 \text{ gal.}} \times \$15 = \$931{,}840{,}000 \text{ additional revenue}$$

$$93{,}184{,}000 \text{ tons}/29{,}120{,}000\ \frac{\text{tons}}{\text{years}} = 3.2 \text{ years additional production}$$

APPENDIX B

SAMPLE CALCULATIONS:

Capital and Operating Costs:

Chamber and Pillar Mining - Conveyor Transport and Stowing (including surface disposal)

Initial Capital:

Borehole	$ 5,348,000
Substations and feeders	384,000
Underground structures and dust control	261,000
Conveyors - main	2,757,000
Conveyors - stope (including feeders)	816,000
Compactors	351,000
Loaders	267,000
Utility vehicles	226,000
Haul trucks	345,000
Dozer	138,000
Graders	106,000
Water trucks	60,000
Miscellaneous revegetation	80,000
Lube truck	40,000
Spare parts	870,000
Dam and road construction	2,054,000
Lighting	10,000
Cooling	20,797,000
Additional ventilation	336,000
Production supervision, engineering expense, and construction	1,273,000
TOTAL	$36,519,000

APPENDIX B

SAMPLE CALCULATIONS:

Capital and Operating Costs:

Chamber and Pillar Mining - Conveyor Transport and Stowing (including surface disposal)

Deferred Capital:

Conveyors	$13,448,000
Haul trucks	4,175,000
Loaders	2,010,000
Compactors	2,224,000
Dozer	1,584,000
Graders	1,516,000
Water trucks	396,000
Utility vehicles	1,841,000
Ventilation	185,000
Miscellaneous Equipment	554,000
Lube truck	402,000
Substation and feeder	265,000
TOTAL	$28,600,000

APPENDIX B

SAMPLE CALCULATIONS:

Capital and Operating Costs:

Chamber and Pillar Mining - Conveyor Transport and Stowing

BACKFILLING COST CENTER

Weekly Schedule: 476,000 tons per week.

Operating Labor:

Job	Men/ Shift	Shifts/ Week	Manshifts/ Week	$/ Manshift	Total $/Week
Conveyor Operator	4	21	84	54.48	$ 4,576
Leadman-Conveyor Moves	1	10	10	56.88	569
Conveyorman	3	10	30	54.48	1,634
Equipment Operator	4	21	84	57.84	4,859
Laborer	4	21	84	50.72	4,260
Subtotal					$15,898
Fringes and Absentee Allowance @36.4%					5,787
Shift Differential					353
TOTAL					$22,038

$22,038/476,000 = $0.0463 per ton

Operating Supplies:

Power:	900 HP x 1 $\frac{\text{KW}}{\text{HP}}$ x 133.4 $\frac{\text{Hrs.}}{\text{Week}}$ x $\frac{\$0.03}{\text{KWH}}$	=	$ 3,602
Water:	3,881,000 $\frac{\text{Gal}}{\text{Week}}$ x $0.00051/Gal	=	1,979
Idlers:	(Replace 10% per year) 1,600 Idlers x $\frac{\$210}{\text{Idler}}$ x .1/52	=	646
Belting:	(Replace every five years) 8 Conveyors x $\frac{\$21,420/5/52}{\text{Belt}}$	=	659
Ventilation:	8 Fans x 133.4 $\frac{\text{Hrs.}}{\text{Week}}$ x $\frac{\$0.04}{\text{Hr.}}$	=	43
Compactor:	3 x 133.4 $\frac{\text{Hrs.}}{\text{Week}}$ x $\frac{\$8.81}{\text{Hr.}}$	=	3,526
Loader:	2 x 66.7 $\frac{\text{Hrs.}}{\text{Week}}$ x $\frac{\$8.30}{\text{Hr.}}$	=	1,107
TOTAL			$11,562

$11,562/476,000 = $0.0243 per ton

APPENDIX B

SAMPLE CALCULATIONS:

Capital and Operating Costs:

Chamber and Pillar Mining - Conveyor Transport and Stowing

BACKFILLING COST CENTER (Weekly Schedule)

Maintenance Labor:

Loader:	$2 \times 66.7 \frac{\text{Hrs.}}{\text{Week}} \times \frac{\$6.56}{\text{Hr.}}$	=	\$ 875
Compactor:	$3 \times 133.4 \frac{\text{Hrs.}}{\text{Week}} \times \frac{\$15.87}{\text{Hr.}}$	=	6,351
Conveyor:	$404{,}600 \text{ Tons} \times \frac{\$0.0015}{\text{Ton}}$	=	607
Ventilation:	$8 \times 133.4 \frac{\text{Hrs.}}{\text{Week}} \times \frac{\$0.94}{\text{Hr.}}$	=	1,003
TOTAL			\$8,836

8,836/476,000 = \$0.0185 per ton

Maintenance Supplies:

Loader:	$2 \times 66.7 \frac{\text{Hrs.}}{\text{Week}} \times \frac{\$4.21}{\text{Hr.}}$	=	\$ 562
Compactor:	$3 \times 133.4 \frac{\text{Hrs.}}{\text{Week}} \times \frac{\$13.40}{\text{Hr.}}$	=	5,363
Conveyor:	$.05 \times 8 \times \$21{,}420/5/52$	=	33
Ventilation:	$8 \times 133.4 \frac{\text{Hrs.}}{\text{Week}} \times \frac{\$0.50}{\text{Hr.}}$	=	534
TOTAL			\$6,492

\$6,492/476,000 = \$0.0136 per ton

APPENDIX B

FUTURE WORTH CALCULATION:

Future worth is calculated using the equation $F=P(1+i)^{\eta}$ where F is the future worth at the end of time period η is present amount P and i is the period compound interest rate.

Example: $P = \$10,000$
$\eta = 20$ years
$i = .07$
$F = \$10,000\ (1 + .07)^{20} = \$38,697$

PRESENT WORTH CALCULATION:

Present worth is the single sum of money at time zero which, if invested at period compound interest rate i for η years, will yield a future amount F as shown by the equation $P = F/(1 + .07)^{\eta}$.

Example: $F = \$40,000$
$\eta = 20$ years
$i = .07$
$P = \$40,000/(1 + .07)^{20} = \$10,337$

APPENDIX C

HYDRAULIC CHARACTERISTICS REPORT
COLORADO SCHOOL OF MINES RESEARCH INSTITUTE

Colorado School of Mines Research Institute

P.O. BOX 112 • GOLDEN, COLORADO 80401
PHONE (303) 279-2581 CSMRI

December 9, 1976

CSMRI Project A60943

Mr. Paul McKie
Cleveland Cliffs Iron Company
P.O. Box 1211
Rifle, Colorado 81650

Dear Mr. McKie:

On September 15, 1976, Mr. Howard Earnest of Cleveland Cliffs Iron Company visited the Research Institute to discuss the hydrologic properties of spent oil shale. During that visit, Mr. Earnest requested a proposal for a research investigation concerning the pumping, drainage, compaction, and the cementation aspects of spent oil shale. The requested proposal was submitted in a letter dated September 21, 1976, to you from Mr. M. G. Pattengill. A revised form of that proposal was accepted by you during a meeting at the Research Institute on September 30, 1976. This letter reports the finding of that research investigation.

The objective of the investigation was to evaluate a sample of spent Paraho oil shale for: (1) attrition during pumping, (2) slurry drainage after pumping, (3) chemical alteration of the spent oil shale after pumping and drainage tests, (4) settling characteristics of pumped slurry, (5) abrasive character of slurry, (6) energy requirements for grinding, and (7) amenability of material to mixing with cementing agents.

The scope of the investigation was to include: (1) working with 29 barrels of spent oil shale on hand at CSMRI, (2) pumping of an approximately 50% solids slurry of the material in a 6-inch-diameter horizontal pipeline loop, (3) particle size determinations before and after pumping, (4) drainage tests on the pumped slurry using both horizontal percolation and vertical drainage techniques, (5) chemical analysis of the feed material and the drainage water, (6) one column settling test, (7) Bond and Miller abrasion tests, (8) Bond work index tests, and (9) consulting on cementation amenability.

SUMMARY

The following summary is based on a research investigation concerning the hydrologic properties of spent Paraho oil shale.

1. When pumped at 50% solids, the minus 65 mesh (Tyler) fraction in the spent oil shale increased from 8.2% to 63.2%, indicating its attrition is relatively high during pumping.

2. Drainage tests on the pumped oil shale slurry were performed in boxes measuring approximately 4 ft by 4 ft by 8 ft. One box had a perforated bottom and one had a solid bottom with a vertical perforated pipe to allow drainage. The resulting pulp in the box with the perforated bottom (after all drainage had ceased) contained 69.1% solids. The resultant pulp in the box containing the perforated vertical pipe contained 64.1% solids.

3. The column settling test showed that the material did not settle in a normal manner. The material settled from an initial column height of 17.4 ft to a height of 16.6 ft in 34.5 hours. At that point, the material essentially ceased to settle. The effluent above the settled pulp was clear. The percent solids in the column were determined for each 2-ft interval. These tests showed that the settled pulp averaged 61.25% solids over the length of the column of settled pulp, and that the percent solids in the 2-ft intervals ranged from 58.1% to 64.7% with no trend from top to bottom.

4. Chemical analyses of the head sample prior to pumping and the water drained from the boxes (No. 2 above) showed an insignificant loss of chemical constituents during drainage.

5. Bond and Miller abrasion tests on the original, unpumped material showed it to have low abrasion characteristics.

6. The results of the Bond work index test on the original, unpumped material indicate that the material is soft and could be easily ground.

7. The cementing aspect of the pumped spent shale are still under study. Dr. Rajaram of Cleveland Cliffs and M. G. Pattengill of CSMRI are involved in this work. M. G. Pattengill is acting on a consulting basis in this phase. Dr. Rajaram is performing test work and has in his possession the data collected thus far.

RECOMMENDATIONS

It is recommended that the cementing admixture development aspects of this project be continued in order to develop a pumpable slurry mixture with acceptable hardening characteristics.

RESULTS AND DISCUSSION

SAMPLE

The material used in this investigation involved 29 barrels of spent Paraho oil shale which was on hand at the Research Institute. The material is described in more detail in Exhibits 1, 2, and 4 of the Appendix.

PUMPING

Twenty-eight drums of the spent shale were slurried to approximately 50% solids and pumped through a 6-in.-diameter horizontal test pipeline loop at 10 to 13 fps for approximately 2 hours.

Exhibit 2 of the Appendix reports the screen size analyses of the shale before and after the pumping sequence. As will be noted, a considerable amount of particle attrition was produced by the pumping. Prior to pumping, only 8.2% by weight of the feed was minus 65 mesh (Tyler). After the pumping, the minus 65 mesh (Tyler) in the sample was 63.2%.

DRAINAGE TESTS

Large-scale drainage tests were conducted using plywood boxes measuring approximately 4 ft by 4 ft by 8 ft. More details concerning the procedure used in these tests can be found in Exhibit 3 of the Appendix. The first test involved a box containing a vertical center perforated drain pipe covered with filter media. This configuration was designed to simulate drainage by horizontal percolation. The second test involved a box fitted with filter media covering the entire bottom. This second box simulated vertical flow drainage. Following is a summary of the results of the two drainage tests.

	Pipe Drain Box	Filter Bottom Box
Solids in Incoming Slurry, %	37.6	34.7
Solids in Drained Pulp, %	64.1	69.1
Bulk Density of Drained Pulp, lb/cu ft	104.9	93.7
Bulk Density of Drained Pulp on a Dry Basis, lb/cu ft	67.3	64.8

More details of these tests are shown in Exhibit 3 of the Appendix.

The above data show that the bulk density of the drained pulp in the pipe drain was slightly higher than the bulk density of the drained pulp in the filter bottom box. The "bulk density on a dry solids basis" values were calculated from the bulk densities and the percent solids determined from the drained pulps.

Chemical analyses were made on the spent oil shale feed and on a composite sample of the liquid drained from each of the boxes. Based on these analyses, losses of chemical constituents due to water leaching was negligible in both tests (see Exhibit 4 of the Appendix for more details).

COLUMN SETTLING TEST

A column settling test was performed in a 4-in.-ID column. The height of the original slurry in this column was 17.4 ft. The percent solids in the slurry placed in the column was 53.15%. This test showed that the solids level in the column dropped to 16.6 ft in 34.5 hours. After this time period, the settling was negligible. All effluent above the solids level in the column was clear. Exhibit 5 of the Appendix details the results of the settling test. Figure 1 shows a graphical representation of the settling rate obtained in this column test.

Samples of the settled pulp were removed from the settling column at approximately 2-ft intervals, and the percent solids were determined on each. The results of these tests were as follows:

Sample Interval (in. from bottom of column)	Solids %
166.69-120	64.72
120-96	60.54
96-72	58.12
72-48	58.70
48-24	59.43
24-0	63.46

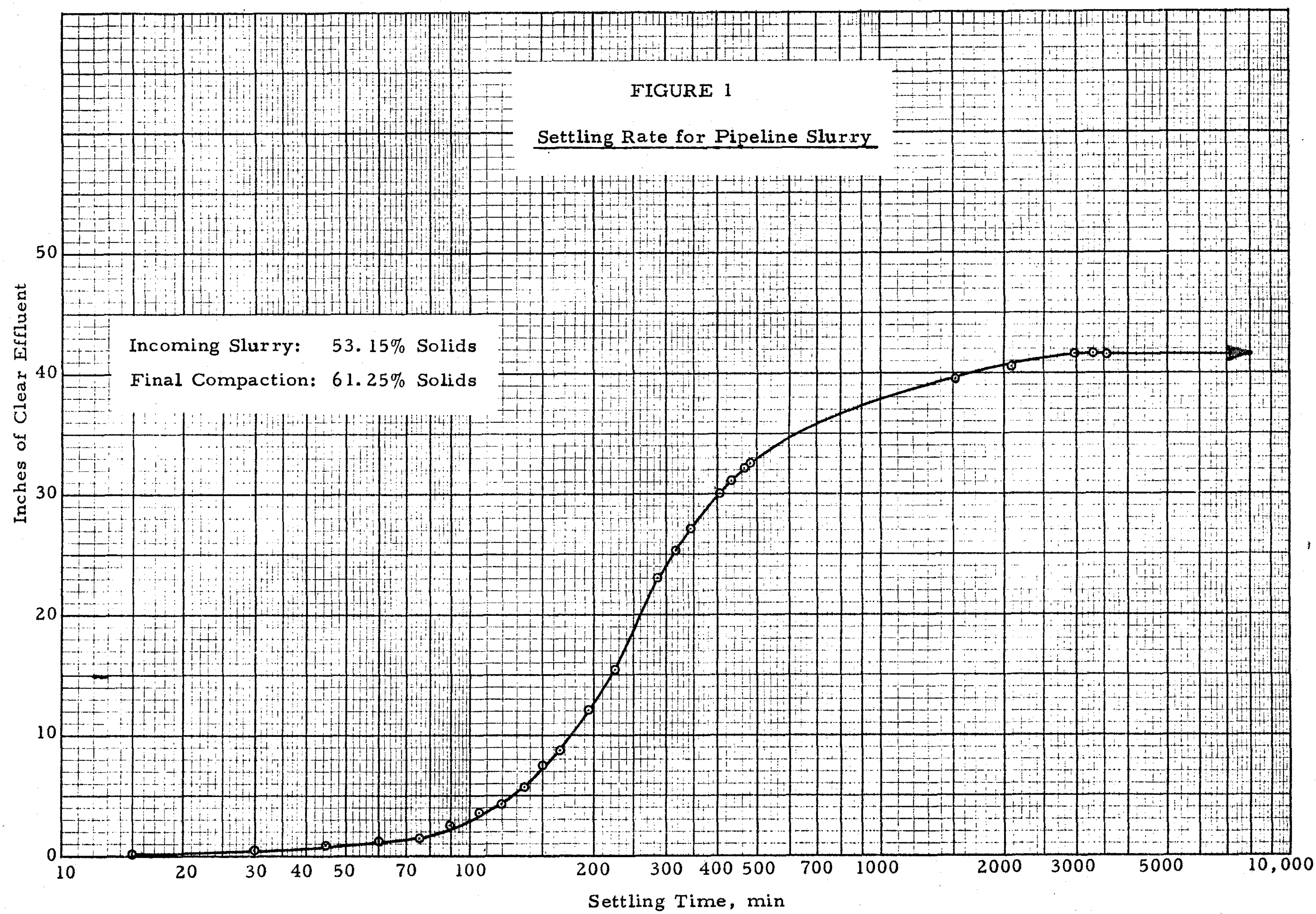
FIGURE 1
Settling Rate for Pipeline Slurry
Incoming Slurry: 53.15% Solids
Final Compaction: 61.25% Solids
Inches of Clear Effluent
0
10
20
30
40
50
10
20
30
40
50
70
100
200
300
400
500
700
1000
2000
3000
5000
10,000
Settling Time, min

These data show that the pattern normally expected in a settling test, i.e., an increase in percent solids from top to bottom, was not present. The above data shows an irregular percent solids pattern. The reason for the unusual settling behavior is not known at this time. It is interesting to note that the average percent solids determined for the settled pulp in this test, 61.25%, was similar to the percent solids determined for the drained pulp in the box drainage tests, i.e., 64.1% and 69.1%.

BOND WORK INDEX TEST

A sample of the spent oil shale feed prior to pumping was subjected to a Bond rod mill grindability test. The results of this test indicated a Bond work index of 6.2 (see Exhibit 6 of the Appendix for more details).

The Bond work index is the comminution parameter which expresses the resistance of the material to crushing and grinding. Numerically, the work index is the kilowatt-hour per short ton required to reduce the material from theoretically infinite feed size to 80% passing 100 microns, equivalent to about 67% passing a 200 mesh Tyler screen. A table of typical Bond work index values is shown in Exhibit 6 of the Appendix. This table shows that the grindability of the oil shale is in the same general range as clay.

BOND ABRASION TEST

A sample of unpumped spent oil shale was evaluated to determine its abrasive character on the Bond scale. As reported in Exhibit 7 of the Appendix, the computed Bond abrasion index was 0.0060. This value represents direct impact abrasion and is discussed in detail in an article by Fred C. Bond in E&MJ, June 1964, pages 169-175. Exhibit 7 of the Appendix contains a table of abrasion indexes for selected materials. This table shows that the spent oil shale has a lower abrasion index than dolomite.

MILLER ABRASION TEST

A sample of unpumped spent oil shale diluted to approximately 50% solids slurry was evaluated for abrasiveness on the Miller abrasion scale. As reported in Exhibit 8 of the Appendix, the resulting Miller number was 55+3. When a second sample was inhibited with sodium hydroxide to a pH of greater than 13, the Miller number dropped to 34-18.

The first figure in the Miller number is called the abrasivity and represents the rate of weight loss from a metallic wear block. The second value is called attrition and represents the effect of slurry particle breakdown as measured by a loss (or gain) of abrasivity as the test progresses. This second value is minus (-) if there is a loss of abrasivity and plus (+) if there is a gain in abrasivity.

Examples of Miller numbers for typical materials are as follows:

Sulfur-Water	25% solids	1-37
Lignite	25% solids	13+4
Limestone	N. D.	30+11
Magnetite	25% solids	64-1
70 Mesh Urn Sand	25% solids	104-14
Corundum 200 Mesh	25% solids	1040-12

The lower value of the NaOH inhibited sample shows that there is some corrosivity in the "raw" spent oil shale sample. (A polished mild steel coupon half-way submerged in both samples for 8 hours at room temperature showed evidence of corrosion in the "as received" sample).

However, in either case the abrasivity is in the region of the generally accepted value of 50, below which the slurry can be "easily" pumped. The rather small particle size certainly contributes to the rather low value of abrasivity and the "plus" value of attrition (+3) of the raw slurry indicates that the material resists mechanical breakdown or inherits the typical character of coal and limestone where the relatively soft material carries hard particles (silica, pyrite, etc.) that become exposed with breakdown or attrition as the test progresses.

CEMENTATION

Mr. M. G. Pattengill of CSMRI is acting in a consulting capacity on the cementation of the spent oil shale after pumping. He and Dr. Rajaram are experimenting with various admixtures of the shale to determine strength parameters, etc., with time. This work is still in progress.

We have appreciated the opportunity to be of service to you in this investigation, and we look forward to being of further assistance to you in the future.

If you have any questions concerning the results reported herein, please feel free to contact us.

Sincerely,

Maurice G. Pattengill
Projects Manager
Chemical Division

Hugh O. Van Male
Project Engineer
Chemical Division

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encl.

APPENDIX

EXHIBIT 1

SAMPLE DESCRIPTION AND PREPARATION

Sponsor's Designation of Sample:	Spent Paraho oil shale.
Sample Weight:	$\approx$8,000 lb.
Sample Container:	29 metal drums.
Sample Description:	Gray chunks, ranging in size from $\approx$2 in. to dust.
Preparation:	The material was used "as is."

EXHIBIT 2

SCREEN ANALYSES

Purpose: To determine the particle size distribution of the sample before and after pumping.

Sample: Spent Paraho oil shale, before and after pumping.

Procedure: Samples were screened at specified meshes.

Results:

Screen Size (Tyler) mesh		Weight % — Sample Prior to Pumping	Weight % — Sample of Shale After Pumping About 1 hr at 10 fps and 56% solids
	+2 in.	0	0
-2 in.	+1½ in.	3.9	0
-1½ in.	+1¼ in.	7.8	0.7
-1¼ in.	+1 in.	11.6	3.3
-1 in.	+½ in.	14.0	3.6
-½ in.	+¼ in.	15.4	2.8
-¼ in.	+4M	6.1	1.6
-4	+14	16.4	9.9
-14	+35	11.8	10.2
-35	+65	4.7	4.7
-65	+150	2.7	3.0
-150	+325	3.0	4.3
-325		2.6	55.9

Observations: These analyses show that the material after pumping is significantly finer than the material prior to pumping.

EXHIBIT 3

BOX DRAINAGE TEST RESULTS

Purpose: To evaluate the drainage characteristics of the slurried oil shale material.

Sample: Pumped pipeline slurry of Paraho oil shale.

Procedure: Two separate tests were made. One test involved a box with a perforated vertical pipe to facilitate drainage (pipe drain box). The other test involved a box with a perforated bottom (bottom drain box). Following are descriptions of these tests:

Pipe Drain Box

A box measuring 48 in. x 46.5 in. x 84 in. was fabricated out of $\frac{3}{4}$-in. plywood and fitted with a perforated 84-in. high plastic center pipe measuring $4\frac{5}{8}$ in. OD. The center pipe was covered with No. 33 cotton filter cloth (from National Filter Media Corporation, Salt Lake City, Utah). The box was filled with slurry to a depth of about 6 ft. The drainage solution was collected and its flow rate determined. A final composite sample of the drained pulp was obtained for chemical analysis after all drainage had ceased. The bulk density of the final drained pulp was calculated.

Filter Bottom Box

The procedure was basically the same as for the pipe drain test with the exception that the entire bottom of the box was perforated and fitted with No. 33 cotton filter cloth. No vertical pipe was used in this test. The filter media was covered with $\approx\frac{3}{4}$ in. of $-\frac{5}{8}$ in. $+\frac{1}{4}$ in. washed pea gravel and $\approx\frac{3}{4}$ in. of washed sand.

Results: Following are the results of these drainage tests.

EXHIBIT 3

Box Drainage Test Results -- continued

Box Drainage Tests Data Summary

	Pipe Drain Box	Filter Bottom Box
Slurry, gal	714.0	667.0
Residue Pulp After Drainage, gal	326.0	300.0
Residue Pulp After Drainage, cu ft	43.6	40.0
Solids in Incoming Slurry, %	37.6	34.7
Water in Initial Slurry, lb	4,862.8	4,874.3
Water in Initial Slurry, gal	583.1	584.4
Solids in Initial Slurry (Dry Basis), lb	2,933.7	2,590.7
Water Drained Out, gal	414.7	393.4
Water Drained Out, lb	3,458.6	3,281.0
Water Remaining in Drained Pulp, gal	168.4	191.0
Water Remaining in Drained Pulp, lb	1,404.2	1,593.3
Total Pulp, lb	4,337.9	4,184.0
Solids in Drained Pulp, %	67.63 (64.12)(1)	61.92 (69.09)(1)
Bulk Density of Drained Pulp, lb/cu ft	99.5 (104.9)(1)	104.6 (93.7)(1)
Bulk Density of Remaining Solids (Dry Basis), lb/cu ft	67.3	64.8

(1) Numbers in parentheses are based on percent solids in final pulp as determined from pipe samples. These values are considered the most accurate, as they were based on actual samples procured from the boxes.

EXHIBIT 4

CHEMICAL ANALYSES OF SAMPLES

Purpose: To determine the extent of change in the chemical composition of the test material brought about by the slurry pumping procedures.

Sample: Spent Paraho oil shale prior to pumping and a composite drainage water sample from each box drainage test.

Procedure: The samples were analyzed for specified constituents using standard wet chemical and instrumental analytical procedures. The chemical composition of the residue pulp, on a dry basis, was calculated.

Results:

	Analyses, Weight %				
		Pipe Drain Box		Filter Bottom Box	
Component	Unpumped Paraho Oil Shale	Drainage Water Composite	Calculated Residue Pulp Dry Basis	Drainage Water Composite	Calculated Residue Pulp Dry Basis
S	0.555	0.009	0.54	0.0085	0.54
CaO	16.1	0.0462	16.05	0.0259	16.08
MgO	7.05	Nil(3)	7.06	Nil(3)	7.06
Cl	Nil(1)	0.0029	Nil	0.0042	Nil
Br	Nil(2)	Nil(4)	Nil	Nil(4)	Nil
Na	1.815	0.0400	1.75	0.0490	1.73
Fe	2.285	Nil(5)	2.29	Nil(5)	2.29
K	0.61	0.0008	0.61	0.0010	0.61
Al	4.995	Nil(6)	5.00	Nil(6)	5.00
Moisture	0.45	--	--	--	--
LOI (1000°C)	19.4	--	--	--	--
Total Dissolved Solids	--	0.22	--	0.2	--

(1) Actual analysis -- <0.05%.
(2) Actual analysis -- <0.001%.
(3) Actual analysis -- <0.2 ppm.
(4) Actual analysis -- <0.1 ppm.
(5) Actual analysis -- <0.5 ppm.
(6) Actual analysis -- <5 ppm.

Observations: The comparison of the analyses of the original unpumped head sample and the calculated analyses of the pulps from the drainage box tests show no significant variations.

EXHIBIT 5

COLUMN SETTLING TEST

Purpose: To determine settling characteristics of pumped slurry.

Sample: Slurry of spent Paraho oil shale which had been pumped ≈$1\frac{1}{2}$ hr at 10-13 fps.

Procedure: The slurry sample was agitated and poured into a 4 in. ID x 240 in. high glass column. The quantity of clear effluent above the solids was measured at periodic intervals. No flocculant was used.

Results:

Solids in Incoming Slurry, %:	53.15
Total Column Height at Start of Test, in.:	208.5
Weight of Solids, lb:	74.77
Compaction, % solids	
At 19 hr (based on % solids in and volume of clear liquid):	60.75
At $49\frac{1}{4}$ hr (based on % solids in and volume of clear liquid):	61.41
At $49\frac{1}{4}$ hr (based on weighted average of samples taken from column at end of test):	61.25
Volume of Settled Pulp, cu ft:	1.21
Total Weight of Settled Pulp, lb:	122.07
Bulk Density (Settled Pulp), lb/cu ft (based on weighted average of samples taken from column):	100.70
Dry Basis:	61.68

EXHIBIT 5

Column Settling Test -- continued

Settling Column Observations

Settling Time min	Inches of Clear Effluent	Settling Time min	Inches of Clear Effluent
0	0	285	23.188
15	0.25	315	25.375
30	0.50	345	27.250
45	1.0	405	30.063
60	1.375	435	31.125
75	1.625	465	32.125
90	2.75	480	32.625
105	3.625	1140(1)	(38.813)
120	4.375	1515	39.625
135	5.938	2070	40.625
150	7.623	2955	41.813
165	8.938	3285	41.813
195	12.188	3525	41.813
225	15.500		

(1) Estimated 19 hr.

Percent Solids in 2-Foot Interval Samples Taken From the Column at End of the Settling Test

Sample Interval (in. from bottom of column)	Solids %
166.69-120	64.72
120-96	60.54
96-72	58.12
72-48	58.70
48-24	59.43
24-0	63.46

EXHIBIT 6

BOND WORK INDEX TEST

Purpose: To determine the rod mill grindability of the test sample in terms of a Bond work index number.

Sample: Spent Paraho oil shale, prior to pumping.

Procedure: The equipment and procedure duplicate the Bond method for determining rod mill work indices.

Test Conditions: Mesh of grind: 10.
Weight of undersize product for 100% circulating load: 749.5 g.
Weight % of undersize material in rod mill feed: 33.6.

Results:

Stage No.	New Feed g	Undersize In Feed g	Undersize To Be Ground g	Revolutions	Undersize in Product g	Undersize Produced Total g	Undersize Produced Per Mill Revolution g
1	1,499.0	503.7	245.8	51	1,410.7	907.0	17.784
2	1,410.7	474.0	275.5	16	997.8	523.8	32.738
3	997.8	335.2	414.3	13	841.3	506.1	38.931
4	841.3	282.7	466.8	12	786.3	503.6	41.967
5	786.3	264.2	485.3	12	744.0	479.8	39.983
6	744.0	250.0	499.5	12	740.0	490.0	40.833
7	740.0	248.6	500.9	12	727.0	478.4	39.867
8	727.0	244.3	505.2	13	781.3	537.0	41.308

Average last three = 40.669

EXHIBIT 6

Bond Work Index Test -- continued

Rod Mill Work Index Computations

$$Wi = \frac{62}{P_1^{0.23} \times Grp^{0.625} \times \left(\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}}\right)}$$

Wherein:	P_1	=	100% Passing Size of Product	=	1,680μ
	Grp	=	Grams per Revolution	=	40.669
	P	=	80% Passing Size of Product	=	1,200μ
	F	=	80% Passing Size of Feed	=	8,250μ

Wi = 6.2

EXHIBIT 6

Bond Work Index Test -- continued

Average Work Indexes of Typical Materials(1)

TABLE IIIA—Average Work Indexes by Types of Materials

Caution should be used in applying the average work index values listed here to specific installations, since the individual variations between materials in any classification may be quite large.

Material	No. of Tests	Average: Specific Gravity *Sg*	Average: Work Index *Wi*
All materials tested	2088	—	13.81
Andesite	6	2.84	22.13
Barite	11	4.28	6.24
Basalt	10	2.89	20.41
Bauxite	11	2.38	9.45
Cement clinker	60	3.09	13.49
Cement raw material	87	2.67	10.57
Chrome ore	4	4.06	9.60
Clay	9	2.23	7.10
Clay, calcined	7	2.32	1.43
Coal	10	1.63	11.37
Coke	12	1.51	20.70
Coke, fluid petroleum	2	1.63	38.60
Coke, petroleum	2	1.78	73.80
Copper ore	308	3.02	13.13
Coral	5	2.70	10.16
Diorite	6	2.78	19.40
Dolomite	18	2.82	11.31
Emery	4	3.48	58.18
Feldspar	8	2.59	11.67
Ferro-chrome	18	6.75	8.87
Ferro-manganese	10	5.91	7.77
Ferro-silicon	15	4.91	12.83
Flint	5	2.65	26.16
Fluorspar	8	2.98	9.76
Gabbro	4	2.83	18.45
Galena	7	5.39	10.19
Garnet	3	3.30	12.37
Glass	5	2.58	3.08
Gneiss	3	2.71	20.13
Gold ore	209	2.86	14.83
Granite	74	2.68	14.39
Graphite	6	1.75	45.03
Gravel	42	2.70	25.17
Gypsum rock	5	2.69	8.16
Ilmenite	7	4.27	13.11
Iron ore	8	3.96	15.44
Hematite	79	3.76	12.68
Hematite—specular	74	3.29	15.40
Oolitic	6	3.32	11.33
Limanite	2	2.53	8.45
Magnetite	83	3.88	10.21
Taconite	66	3.52	14.87
Kyanite	4	3.23	18.87
Lead ore	22	3.44	11.40
Lead-zinc ore	27	3.37	11.35
Limestone	119	2.69	11.61
Limestone for cement	62	2.68	10.18
Manganese ore	15	3.74	12.46
Magnesite, dead burned	1	5.22	16.80
Mica	2	2.89	134.50
Molybdenum	6	2.70	12.97
Nickel ore	11	3.32	11.88
Oil shale	9	1.76	18.10
Phosphate fertilizer	3	2.65	13.03
Phosphate rock	27	2.66	10.13
Potash ore	8	2.37	8.88
Potash salt	3	2.18	8.23
Pumice	4	1.96	11.93
Pyrite ore	4	3.48	8.90
Pyrrhotite ore	3	4.04	9.57
Quartzite	16	2.71	12.18
Quartz	17	2.64	12.77
Rutile ore	5	2.84	12.12
Sandstone	8	2.68	11.53
Shale	13	2.58	16.40
Silica	7	2.71	13.53
Silica sand	17	2.65	16.46
Silicon carbide	7	2.73	26.17
Silver ore	6	2.72	17.30
Sinter	9	3.00	8.77
Slag	12	2.93	15.76
Slag, iron blast furnace	6	2.39	12.16
Slate	5	2.48	13.83
Sodium silicate	3	2.10	13.00
Spodumene ore	7	2.75	13.70
Syenite	3	2.73	14.90
Tile	3	2.59	15.53
Tin ore	9	3.94	10.81
Titanium ore	16	4.23	11.88
Trap rock	49	2.86	21.10
Uranium ore	20	2.70	17.93
Zinc ore	10	3.68	12.42

(1) Crushing and grinding calculations, Fred C. Bond, Allis-Chalmers, Revised January 2, 1961, p. 14.

EXHIBIT 7

BOND ABRASION TEST

Purpose: To determine the Bond abrasion index of the test sample.

Sample: 1600 g, -¾ in. +½ in. fraction of spent Paraho oil shale, prior to pumping.

Procedure: The equipment and procedure duplicate the Bond method for determining an abrasion index.

Results:

Original Coupon Weight, g:	95.1068 (OW)
Final Coupon Weight, g:	95.1008 (FW)

Computed Abrasion Index, g, A_i = 0.0060 (OW-FW)

Note: Arbitrary lower limit has been established to be 0.021 for A_i.

Average Bond Abrasion Index Values[1]

No.	Material	A_i
1	Dolomite	0.0160
2	Shale	0.0209
3	L. S. for Cement	0.0238
4	Limestone	0.0320
5	Cement Clinker	0.0713
6	Magnesite	0.0783
7	Heavy Sulfides	0.1284
8	Copper Ore	0.1472
9	Hematite	0.1647
10	Magnetite	0.2217
11	Gravel	0.2879
12	Trap Rock	0.3640
13	Granite	0.3880
14	Taconite	0.6237
15	Quartzite	0.7751
16	Alumina	0.8911

(1) From Bond, Fred C., E&MJ, June 1964, p. 171.

EXHIBIT 8

MILLER ABRASION TEST

Purpose: To determine the Miller abrasion number of the test sample.

Sample: Spent Paraho oil shale, prior to pumping, diluted to ≈50% solids.

Procedure: The equipment and procedure duplicate the Miller method for determining an abrasion index. Twenty-seven percent chrome iron wear blocks used in all test.

Results:

	≈50% Solids Slurry		NaOH Inhibited pH 13+	
Block No.	1	2	1	2
Total Weight Loss, mg	53.8	45.0	15.5	27.2
Lap Wear	trace		trace	
Miller No.	55.4 + 2.9		33.8-18.1	

Colorado School of Mines Research Institute

P.O. BOX 112 • GOLDEN, COLORADO 80401
PHONE (303) 279-2581 CSMRI

December 14, 1976

CSMRI Project A60943

DEC 1976
RECEIVED
RIFLE, COLORADO

Dr. Rajaram
Cleveland Cliffs Iron Company
P. O. Box 1211
Rifle, Colorado 81650

Dear Dr. Rajaram:

In accordance with your telephone request of December 6, 1976, we have prepared the following photographs showing the equipment used in the box drainage test phase of our spent oil shale disposal study. The results of this study were reported in our letter of December 9, 1976.

Following are descriptions of the photographs:

Figure 1 - Exterior view of pipe drain box. Drain pipe is located on opposite side.
Figure 2 - Pipe drain box, exit drain pipe and collecting bucket.
Figure 3 - Interior of pipe drain box after drainage test showing location of center drain pipe. Pipe is caked with residue pulp.
Figure 4 - Exterior view of filter bottom box with drain trough at base.
Figure 5 - Close-up of drain trough on filter bottom box.
Figure 6 - Interior of filter bottom box after tests showing residue solids.

If you have any questions concerning these pictures, please feel free to contact us.

Sincerely,

M. G. Pattengill
Projects Manager
Chemical Division

Hugh O. Van Male
Project Engineer
Chemical Division

/cjm
Enc.

Figure 1

Figure 2

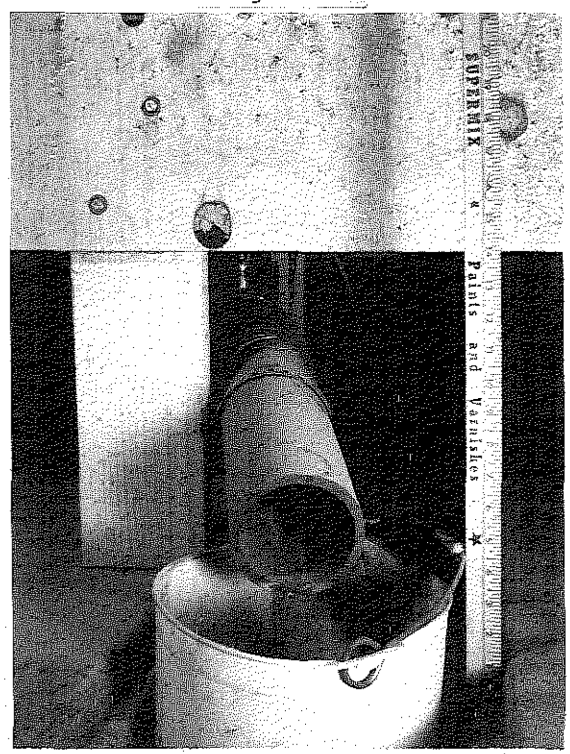

Figure 3

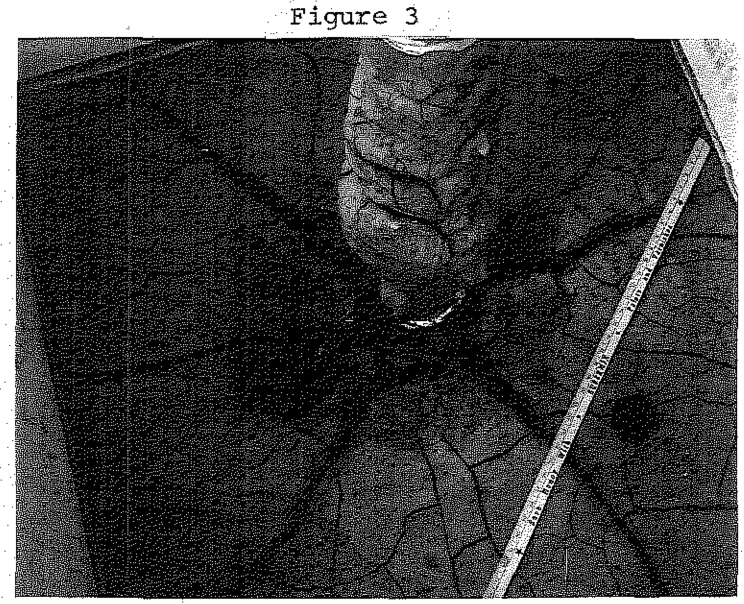

Figure 4

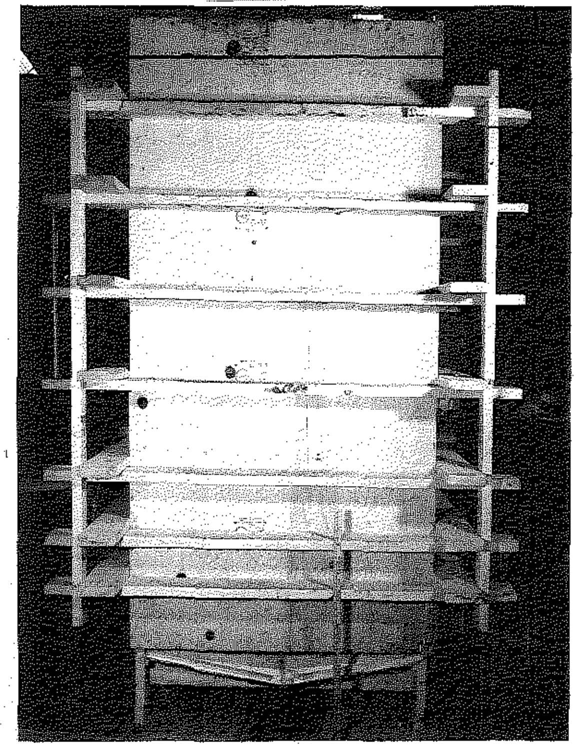

Figure 5

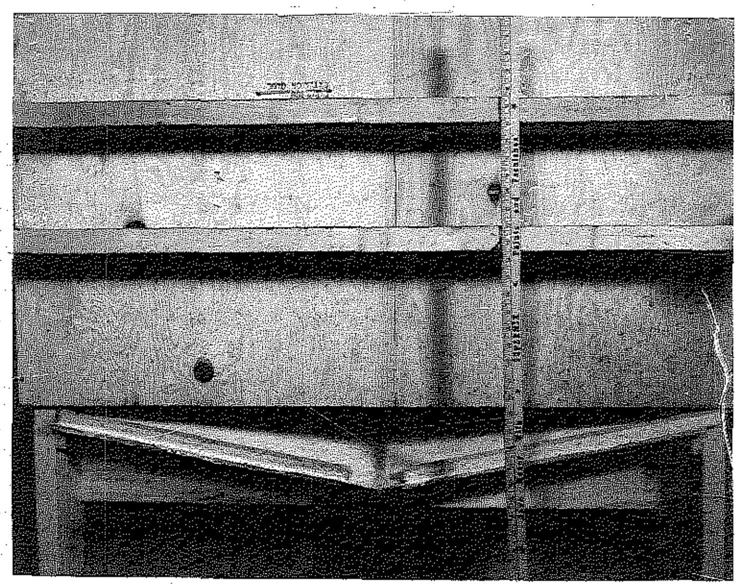

Figure 6

Colorado School of Mines Research Institute

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PHONE (303) 279-2581 CSMRI

December 21, 1976

CSMRI Project A60943

DEC 1976
RECEIVED
RIFLE, COLORADO

Mr. Paul McKie
Cleveland Cliffs Iron Co.
P. O. Box 1211
Rifle CO 81650

Dear Mr. McKie:

At the request of Dr. Rajaram, following is a summary of the calculations performed at CSMRI relating to the cementing phase of the spent Paraho oil shale. The major purpose of this phase was to determine if various proportions of fly ash, burned lime, or Portland cement would act as a cementing media for spent Paraho oil shale after reemplacement in worked-out areas. I was to perform a series of calculations to determine, on a chemical basis, what proportions of the three above-mentioned components appeared logical as a cementing media. Dr. Rajaram was to perform the actual physical testing in his laboratory.

Chemical Analyses of Spent Paraho Oil Shale

Following is the chemical analyses of the spent Paraho oil shale prior to pumping (dry basis):

	%
SiO_2	(39.54)[1]
Al_2O_3	9.49
Fe_2O_3	3.28
TiO_2	ND
P_2O_5	ND
CaO	16.07
MgO	7.13
Na_2O	2.46
K_2O	1.48
$Na_2O + 0.658\ K_2O$ (alkalies)	3.43
SO_3	0.56
Cl	<0.5
BrO_3	<0.001
LOI	19.49
Total	100.00

1/ Determined by difference.

Mr. Paul McKie
Cleveland Cliffs Iron Co.
December 21, 1976
Page 2

Chemical Analyses of Paraho Oil Shale After Pumping and Drainage Tests

As may be noted in the report of December 9, 1976, the chemical analysis of the Paraho oil shale after pumping and after subjection to the drainage box test was significantly similar to the unpumped shale. Owing to this, the chemical analysis of the unpumped Paraho oil shale was used for all subsequent calculations.

Synthesized Cementing Media

In an attempt to produce a synthesized cementing media from fly ash and burned lime, similar in nature to the chemistry of Portland cement, the chemistries of the two components were assumed as follows:

	%	
	Fly Ash(1)	Burned Lime
SiO_2	41.6	---
Al_2O_3	26.6	---
Fe_2O_3	5.4	---
TiO_2	1.4	---
P_2O_5	1.0	---
CaO	8.7	100.0(3)
MgO	1.8	---
Na_2O	1.6	---
K_2O	0.4	---
Na_2O + 0.658 K_2O (Alkalies)	1.9	---
SO_3	10.0	---
Difference(2)	1.5	---
Total	100.0	100.0

1/ The mathematic average of two published fly ashes. One ash was from Moffett County and one was from Routt County.
2/ This value is the mathematical difference between the sum of the above-listed oxides and 100% (includes LOI).
3/ The burned lime was assumed to contain 100% CaO.

Mr. Paul McKie
Cleveland Cliffs Iron Co.
December 21, 1976
Page 3

The ASTM specification for Type II Portland cement compared to a mixture of 50% Fly Ash and 50% Burned Lime Cementing Mix No. 1 is as follows:

	%	
	50% Fly Ash 50% Burned Lime (Cementing Mix No. 1)	Type II Portland Cement
SiO_2	20.8	21.0 (minimum)
Al_2O_3	13.3	6.0 (maximum)
Fe_2O_3	2.7	6.0 (maximum)
TiO_2	0.7	----
P_2O_5	0.5	----
CaO	54.4	55.4 (calculated)
MgO	0.9	5.0 (maximum)
Na_2O	0.8	----
K_2O	0.2	----
$Na_2O + 0.628 K_2O$ (Alkalies)	0.93	0.6 (maximum)
SO_3	5.0	3.0 (maximum)
Difference	(0.7)(1)	(3.0 LOI maximum)
Total	100.0	100.0

1/ This value is the mathematical difference between the sum of the above-listed oxides and 100% (includes LOI).

Since 75% of Portland cements are comprised of calcium silicates and since the above mix closely approximates the CaO and SiO_2 levels of the Type II Portland cement, it was chosen as a candidate for a cementing material.

Another cementing mix (Cementing Mix No. 2) was designed that incorporated 50% of the Cementing Mix No. 1 and 50% Type II Portland cement. Following is a comparison of chemistry of this mix and the chemistry of a Type II Portland cement (P.C.):

Mr. Paul McKie
Cleveland Cliffs Iron Co.
December 21, 1976
Page 4

	50% Cementing Mix No. 1 and 50% Type II P.C. (Cementing Mix No. 2)	Type II P.C.
SiO_2	20.9	21.0 (minimum)
Al_2O_3	9.6	6.0 (maximum)
Fe_2O_3	4.5	6.0 (maximum)
TiO_2	0.4	----
P_2O_5	0.2	----
CaO	54.8	55.4 (calculated)
MgO	2.9	5.0 (maximum)
$Na_2O + 0.658\ K_2O$ (Alkalies)	0.8	0.6 (maximum)
SO_3	4.0	3.0 (maximum)
LOI	1.5	3.0 (maximum)
Difference	0.4 (includes LOI)	----

As with Cementing Mix No. 1, this cementing mix also looks chemically promising as a cementing media.

Initial Shale-Cementing Media Mixes

The following mixes were suggested for initial study based on the above calculations:

Mr. Paul McKie
Cleveland Cliffs Iron Co.
December 21, 1976
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Mix A
(95% Spent Shale and 5% Cementing Mix No. 1

	%		
	95% Spent Shale(1)	5% Cementing Mix No. 1	Final Mix
SiO_2	37.56	1.04	38.60
Al_2O_3	9.02	0.66	9.68
Fe_2O_3	3.12	0.14	3.26
TiO_2	ND(2)	0.04	0.04
P_2O_5	ND	0.02	0.02
CaO	15.27	2.72	17.99
MgO	6.77	0.04	6.81
$Na_2O + 0.658 K_2O$	3.26	0.05	3.31
SO_3	0.53	0.25	0.78
Cl	<0.5	ND	<0.5
BrO_3	<0.001	ND	<0.001
LOI	18.52	0.04	18.29

1/ Moisture free basis.
2/ ND = not determined.

Mix B
(95% Spent Shale and 5% Cementing Mix No. 2)

	%		
	95% Spent Shale(1)	5% Cementing Mix No. 2	Final Mix
SiO_2	37.56	1.04	38.60
Al_2O_3	9.02	0.48	9.50
Fe_2O_3	3.12	0.22	3.34
TiO_2	ND(2)	0.02	0.02
P_2O_5	ND	0.01	0.01
CaO	15.27	2.74	18.01
MgO	6.77	0.14	6.91
$Na_2O + 0.658 K_2O$	3.26	0.04	3.30
SO_3	0.53	0.20	0.73
Cl	<0.5	----	<0.5
BrO_3	<0.001	----	<0.001
LOI	18.52	0.08	18.60

1/ Moisture free basis.
2/ ND = not determined.

Mr. Paul McKie
Cleveland Cliffs Iron Co.
December 21, 1976
Page 6

Mix C
(70% Spent Shale and 30% Cementing Mix No. 1)

	% 70% Spent Shale	% 30% Cementing Mix No. 1	% Sum
SiO_2	27.55	9.68	37.25
Al_2O_3	6.12	2.94	9.06
Fe_2O_3	2.28	0.71	2.99
TiO_2	------	0.04	0.04
P_2O_5	------	0.02	0.02
CaO	11.20	16.24	27.44
MgO	4.97	0.24	5.21
$Na_2O + 0.658\ K_2O$ (Alkalies)	2.40	0.16	2.56
SO_3	0.39	1.50	1.89
Cl	0.35	-----	0.35
BrO_3	0.001	-----	0.001
LOI	13.58	0.21	13.79

Mix D
(70% Spent Shale and 30% Cementing Mix No. 2)

	% 70% Spent Shale	% 30% Cementing Mix No. 2	% Sum
SiO_2	27.55	6.27	33.82
Al_2O_3	6.12	2.88	9.00
Fe_2O_3	2.28	1.35	3.63
TiO_2	------	0.12	0.12
P_2O_5	------	0.06	0.06
CaO	11.20	16.44	27.64
MgO	4.97	0.87	5.84
$Na_2O + 0.658\ K_2O$ (Alkalies)	2.40	0.24	2.64
SO_3	0.39	1.20	1.59
Cl	0.35	-----	0.35
BrO_3	0.001	-----	0.001
LOI	13.58	0.45	14.03

If you have any questions concerning the above, please feel free to contact me.

Sincerely,

M. G. Pattengill

M. G. Pattengill
Projects Manager
Chemical Division

/cjm

Colorado School of Mines Research Institute

P.O. BOX 112 • GOLDEN, COLORADO 80401
PHONE (303) 279-2581 CSMRI

January 10, 1977

CSMRI Project A60943

Mr. Paul McKie
Cleveland Cliffs Iron Company
P. O. Box 1211
Rifle CO 81650

Dear Mr. McKie:

This letter reports the water flow rate data for the two box drainage tests recently completed on spent Paraho oil shale at the Research Institute. This information was inadvertently omitted from our report to you dated December 9, 1976.

Liquid Flow Rates from Box Drainage Tests

PIPE DRAIN BOX

Time Interval In Test (hr)	Average Flow Rate (gpm)	Volume Flowing (gal) Per Time Interval	Volume Flowing (gal) Cumulative Total
0 - 0.67	0.1544	6.21	6.21
0.67- 5.42	0.1348	38.42	44.63
5.42- 20.67	0.1169	106.96	151.59
20.67- 29.67	0.0873	47.14	198.73
29.67- 44.17	0.0612	53.24	251.97
44.17- 72.42	0.0479	81.19	333.19
72.42- 77.42	0.0355	10.65	343.81
77.42- 96.92	0.0188	22.00	365.81
96.92-124.67	0.0106	17.65	383.46
124.67-147.67	0.0043	5.93	389.39
147.67-166.17	0.0056	6.22	395.61
166.17-243.84	0.0041	19.11	414.72

Mr. Paul McKie
Cleveland Cliffs Iron Company
January 10, 1977
Page 2

FILTER BOTTOM BOX

Time Interval In Test (hr)	Average Flow Rate (gpm)	Volume Flowing (gal) Per Time Interval	Volume Flowing (gal) Cumulative Total
0 - 1.00	0.1045	6.27	6.27
1.00- 2.08	0.0780	5.05	11.32
2.08- 3.42	0.0665	5.35	16.67
3.42- 7.50	0.0557	13.64	30.31
7.50- 10.00	0.0466	6.99	37.30
10.00- 19.50	0.0378	21.55	58.85
19.50- 24.00	0.0313	8.45	67.30
24.00- 28.75	0.0311	8.86	76.16
28.75- 44.00	0.0314	28.73	104.89
44.00- 53.00	0.0308	16.63	121.52
53.00- 67.50	0.0303	26.36	147.88
67.50- 95.75	0.0291	49.32	197.20
95.75-100.75	0.0280	8.40	205.60
100.75-120.25	0.0274	32.06	237.66
120.25-148.00	0.0254	42.29	279.95
148.00-171.00	0.0230	31.74	311.69
171.00-189.50	0.0214	23.75	335.44
189.50-267.17	0.0120	55.92	391.36
267.17-290.00	0.0015	2.05	393.41

We apologize for any inconvenience this omission may have caused. If you have any questions concerning this data, please feel free to contact us.

Sincerely,

Hugh O. Van Male

Hugh O. Van Male
Project Engineer
Chemical Division

/cjm

APPENDIX D

STRENGTH OF PARAHO RETORTED SHALE MODIFIED BY ADDITIVES

THE CLEVELAND-CLIFFS IRON COMPANY

THE CLEVELAND-CLIFFS IRON COMPANY — WESTERN DIVISION, RIFLE, COLORADO 81650

APPENDIX D
STRENGTH OF PARAHO RETORTED SHALE MODIFIED BY ADDITIVES

ABSTRACT

Various flocculants and cementing agents were added to Paraho retorted shale in an effort to improve its strength characteristics. Tests were conducted on both dry retorted shale and hydraulically pumped retorted shale. Two different ratios of cementing agents to retorted shale were investigated. Different percentages of flocculants were added to hydraulically pumped retorted shale to determine any improvements in percolation and/or decantation rates. Percent of moisture added to dry retorted shale was also varied. Results of unconfined compression tests and the effect of flocculants are discussed. The amount of water released from the hydraulic fill was a maximum when Separan MGL was added at the rate of two pounds per ton of solids. Maximum strength of dry fill was achieved with a five to one retorted shale cement mixture.

1.0 Introduction:

The strength characteristics of a backfill material can be increased either by (1) compaction alone or (2) compaction aided with additives. As part of the study to determine the technical and economic feasibility of underground disposal of Paraho retorted shale, tests were performed to investigate economical and efficient methods of increasing the support potential of backfill. The primary purpose of the tests were to identify beneficial trends with the use of flocculants and cementing agents and not to obtain statistically significant results.

The characteristics of Paraho retorted shale are totally different from that of other materials used for backfilling in North American mines. Research directed toward studying the effects of flocculants and cementing agents on Paraho retorted shale is extremely limited. However, several studies have been conducted on the effects of flocculants on cemented sandfill (1,2,3,4). Studies have determined the effect of lime on the strength characteristics of Paraho retorted shale (5)*. The results of this study, as related to the effect of cementing agents, are as follows:

1. Treatment with 5% hydrated calcium lime promotes cementing, thus, increasing compressive strength and decreasing permeability.
2. Accelerated curing times promote reactions of calcium and magnesium oxides present in the Paraho retorted shale and thus increase compressive strength.
3. The amount of fines and their distribution could have a significant effect on compressive strength.

2.0 Test Material:

Two different types of test material to simulate hydraulic backfill and mechanically placed backfill were used during this testing program. Gradation test results on the as received Paraho retorted shale, as obtained by the Colorado School of Mines Research Institute (CSMRI), are shown in Figure 1. The as received material was pumped at a 48%

* References listed at back of Appendix D

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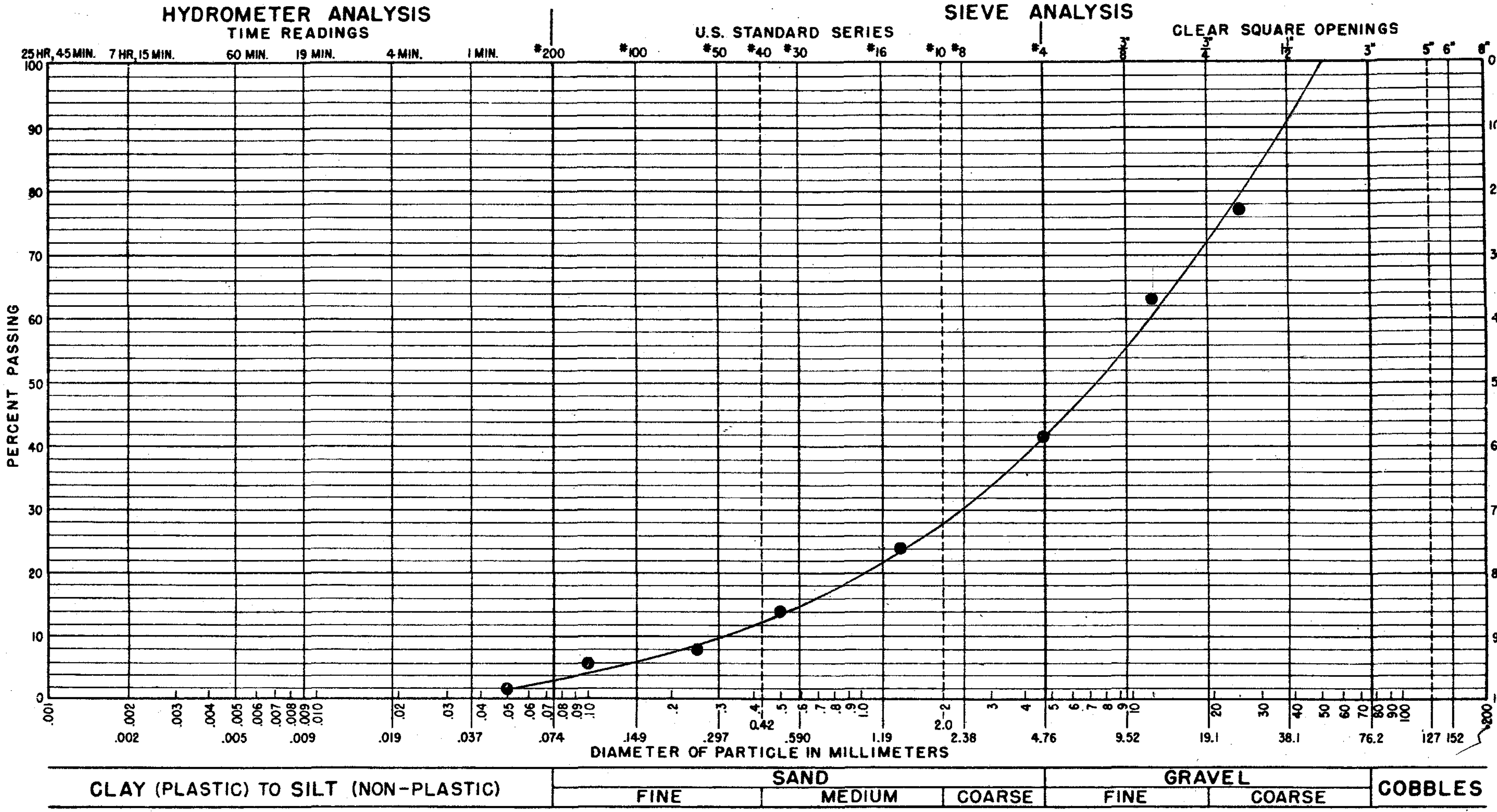

NOTES:

GRADATION TEST

TEST NO. I — Paraho Retorted Shale, As Received Sample

Figure I Grain-Size Distribution of As Received Paraho Retorted Shale

LABORATORY SAMPLE No. ________ FIELD DESIGNATION ________ EXCAVATION No. ________ DEPTH ________ FT.

Nov. 1976

FIGURE NO.

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solids slurry for one hour at 10 feet per second at CSMRI to simulate hydraulic backfilling operations. Grain-size distribution after pumping, as obtained by CSMRI, is shown in Figure 2. As shown in Figure 2, the material used for preparing hydraulic backfill samples contains almost 90% of minus 4-mesh size of which slime sizes (-325 mesh) make up 56%.

To simulate the attrition and abrasion that would occur to Paraho retorted shale during borehole transport and mechanical stowing, the as received material was crushed and a sample consisting of 50% plus 4-mesh material and 50% minus 4-mesh material was prepared. The maximum particle size is 0.5 inch and the grain size distribution of the minus 4-mesh material is shown in Figure 3.

3.0 Equipment:

The equipment used for the testing program included the following:

1. Cardboard cylinders, waxed inside, 6 inches in diameter by 12 inches high
2. Balance and weights
3. Compression testing machine - Soil Test CT-650 (Figure 4)
4. General laboratory equipment

4.0 Experimental Procedure:

4.1 Sample Preparation:

Hydraulically pumped retorted shale was mixed thoroughly in the shipping drum and a sufficient quantity (five quarts) was placed in each cardboard cylinder. The base plate in some cardboard cylinders was perforated and covered by two layers of burlap (Figure 5). Various

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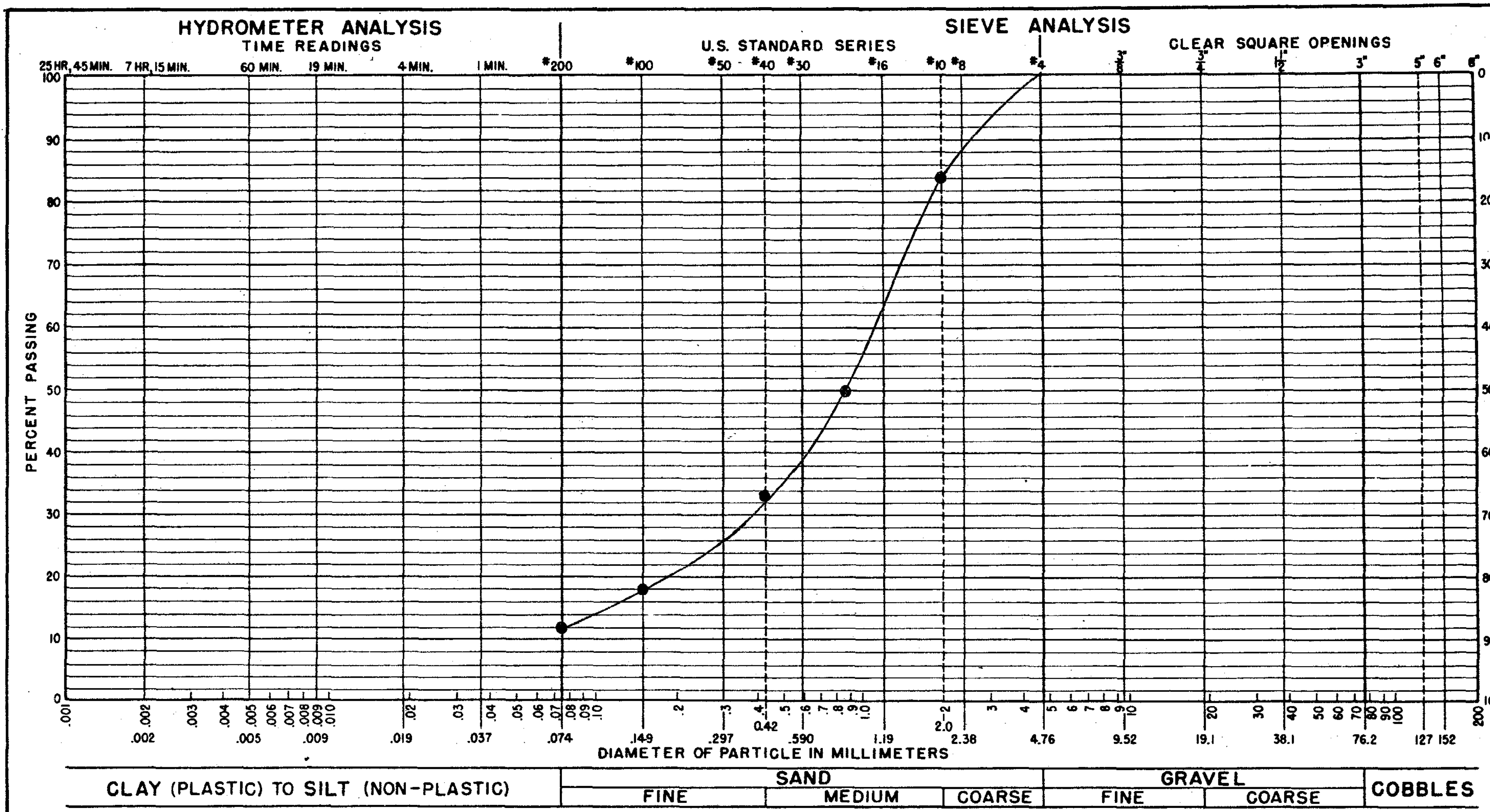

NOTES:

GRADATION TEST

TEST NO. 3 Paraho Retorted Shale- Crushed and screened to minus 4 mesh

Figure 3 Grain-Size Distribution of Minus 4 Mesh Paraho Retorted Shale

LABORATORY SAMPLE No. ______ FIELD DESIGNATION ______ EXCAVATION No. ______ DEPTH ______ FT.

Nov. 1976

FIGURE NO.

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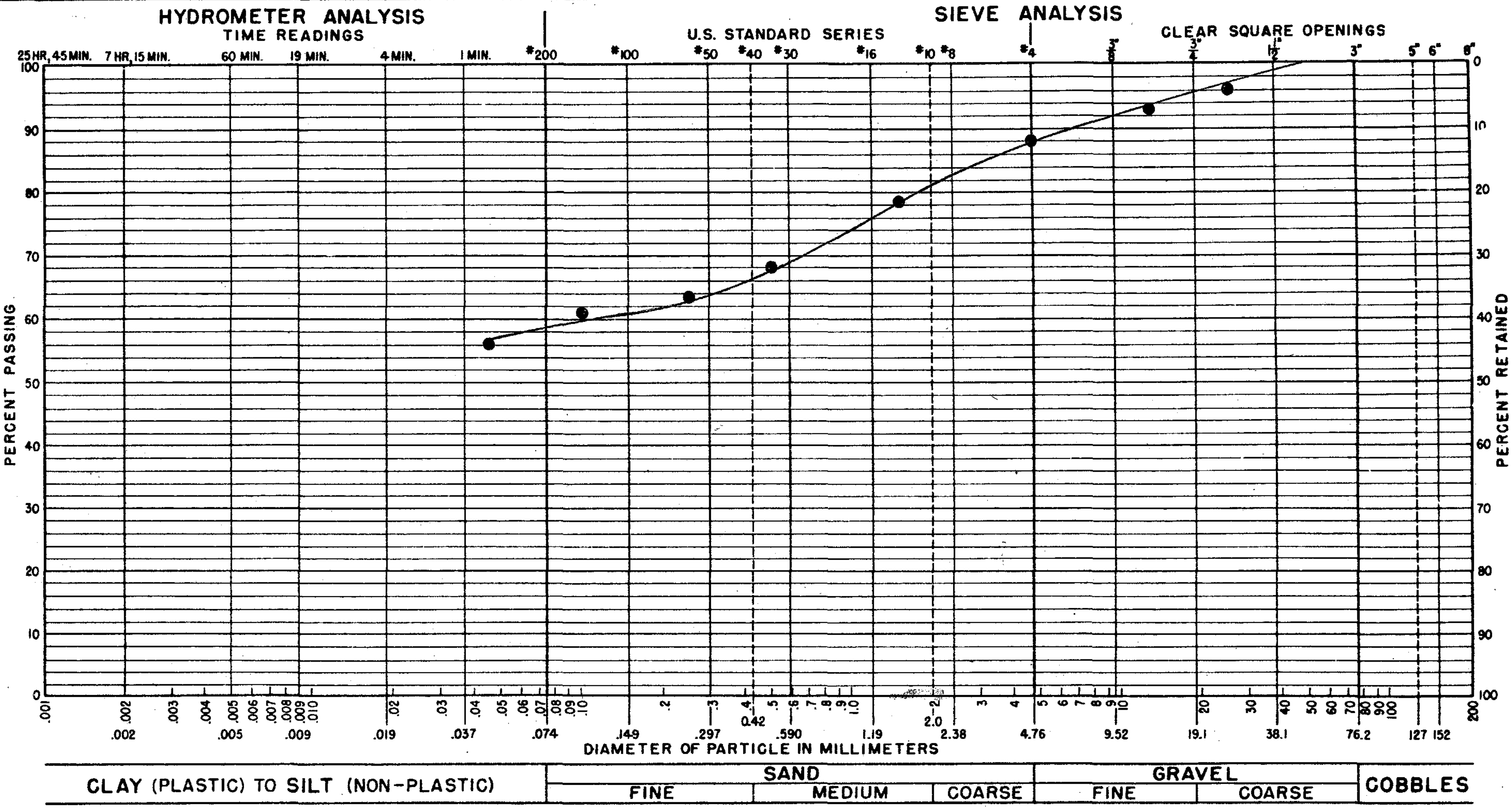

NOTES:

GRADATION TEST

TEST NO. 2 Paraho Retorted Shale - Pumped for 1.0 hr at 10 fps

Fig 2 Grain - Size Distribution of Hydraulically Pumped Paraho Retorted Shale

LABORATORY SAMPLE No. ______ FIELD DESIGNATION ______ EXCAVATION No. ______ DEPTH ______ FT.

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FIGURE NO.

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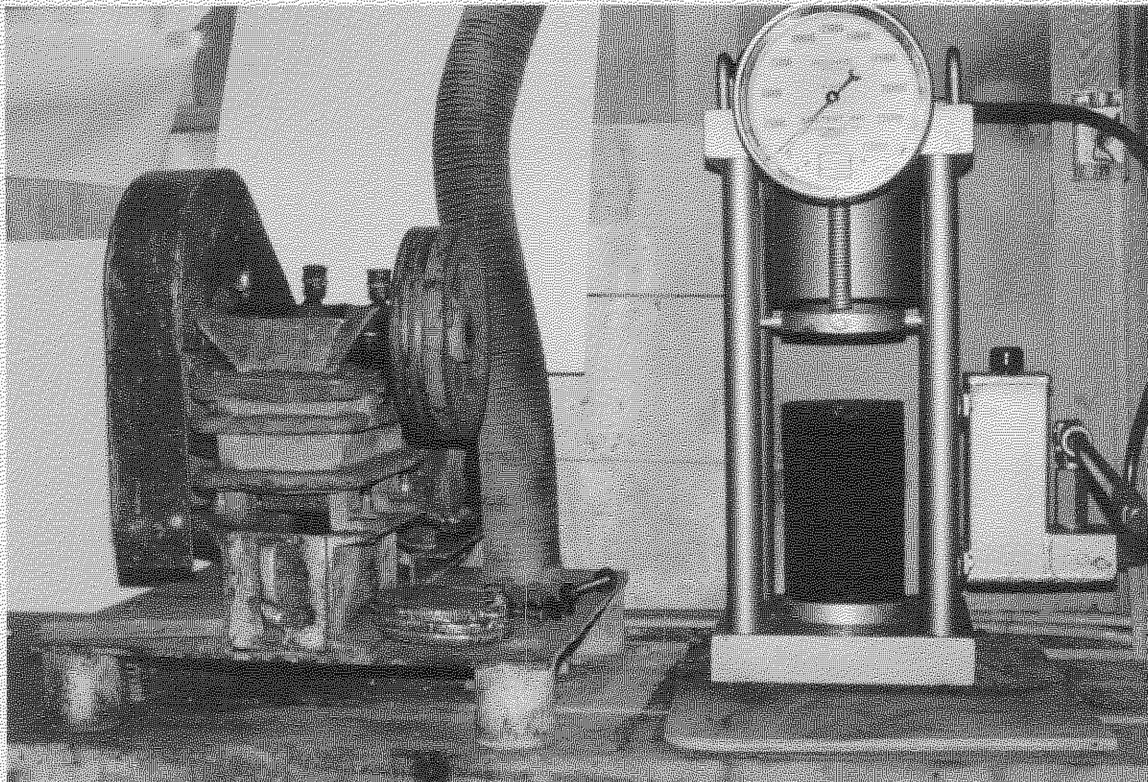
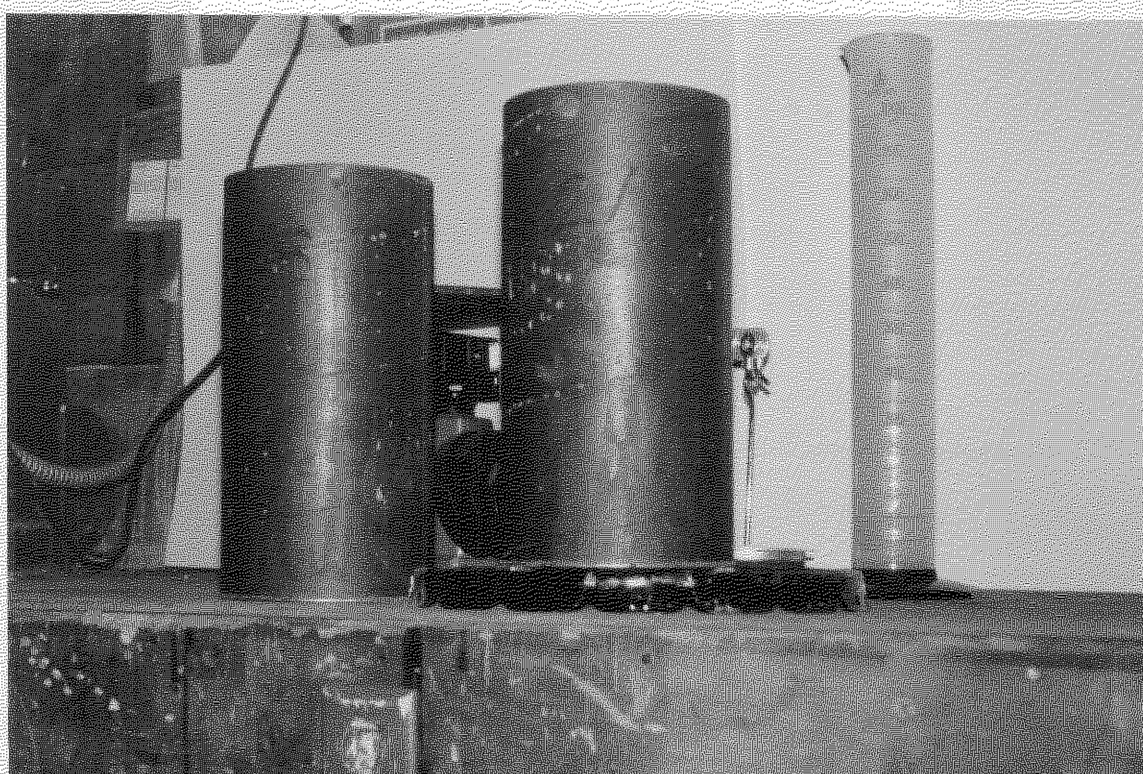


FIGURE 4: JAW CRUSHER AND COMPRESSION TESTING MACHINE

FIGURE 5: EXPERIMENTAL SETUP FOR DECANTATION AND PERCOLATION TESTS



combinations of flocculants and cementing agents were mixed and tested.

The two flocculants used in this study were calcium chloride and polyacrylamide. Four different variations of polyacrylamide, Separan MGL, Separan MG 200, Separan MG 500, and Separan MG 700 (trademark of the Dow Chemical Company), were used in two proportions: 1.0 and 2.0 pounds per ton of solids. The proportions of calcium chloride used were 1, 5 and 10 pounds per ton of solids. The two flocculants were handled differently because of their varying characteristics. Calcium chloride was added in its dry state, while polyacrylamide was dissolved in water prior to its addition to the hydraulically pumped retorted shale.

Three types of cementing agents were used in various proportions. Portland cement, hydrated lime, and flyash (obtained from the Hayden Power Plant, Colorado) were mixed as follows:

100% Portland cement

50% each of Portland cement and flyash

33.3% each of Portland cement, flyash and lime

50% each of lime and flyash

The ratios of retorted shale-cementing agent used were 5 to 1 and 30 to 1. The cementing agent was mixed thoroughly with the retorted shale and allowed to cure for a period of 10 days.

Dry retorted shale samples were prepared by thoroughly mixing 50% minus 4 mesh material and 50% plus 4-mesh material. Every specimen was placed in two lifts per cylinder, with each lift being compacted with 10 drops of a 10-pound steel weight falling through an 18-inch controlled drop. Cementing agents were mixed thoroughly as each lift was placed. Samples were mixed at two different water contents. Based on optimum moisture of about 22% for Paraho retorted shale, moisture contents of 15% and 25% were used in this study. Curing period for this series of tests was eight days.

4.2 Testing Procedure:

Hydraulically pumped retorted shale specimens were observed periodically and the amount of water percolated and/or decanted from the specimen was measured (Figure 7). After the specified curing period, specimens were removed from the cardboard mold and tested in the compression testing machine (Figure 6). The unconfined compressive strength at failure and specimen shortening was determined for each specimen.

5.0 Analysis of Data:

Unconfined compression tests were performed on both hydraulically pumped retorted shale specimens and dry retorted shale specimens. In addition, the effect of flocculants on dewatering characteristics of hydraulically pumped retorted shale was also determined.

It was found that specimens containing 1, 5, and 10 pounds of calcium choloride per ton did not show any improvement in dewatering characteristics (Table 1). From this Table, it can be seen that the four different types of polyacrylamide flocculant had beneficial effects,

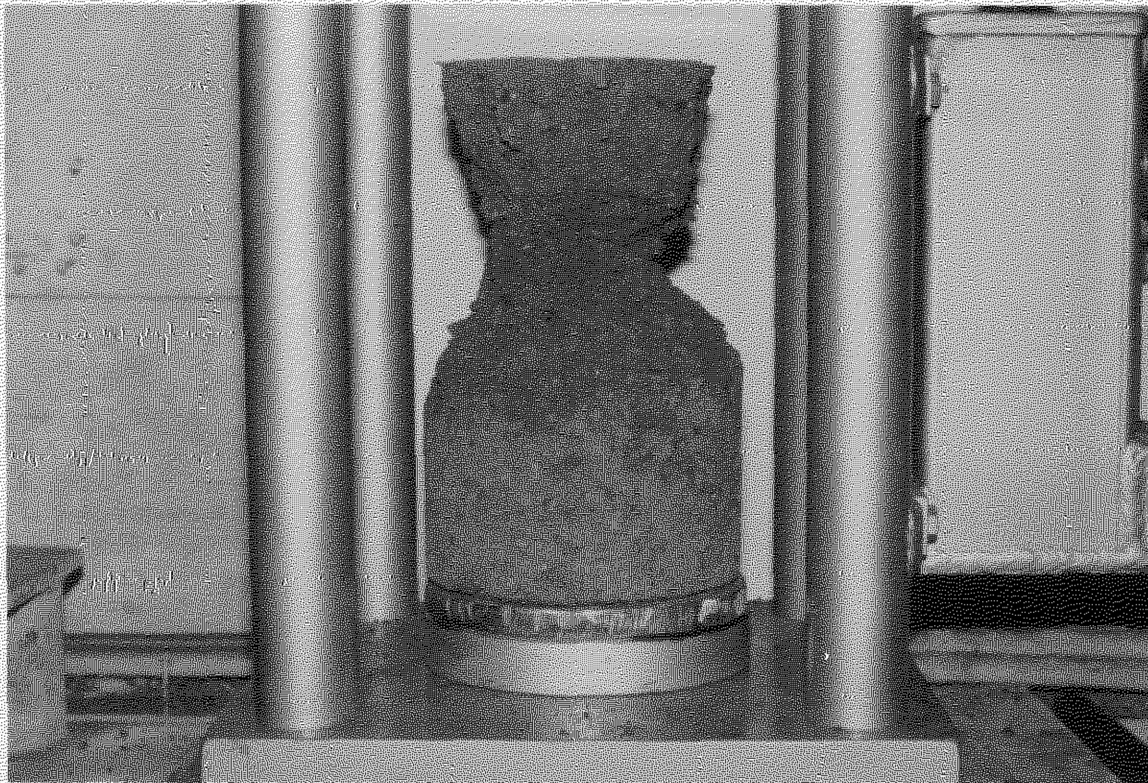


FIGURE 6: HOUR GLASS SHAPE OF FAILED DRY RETORTED SHALE SPECIMEN

FIGURE 7: DECANTED WATER IN A HYDRAULICALLY PUMPED RETORTED SHALE SPECIMEN

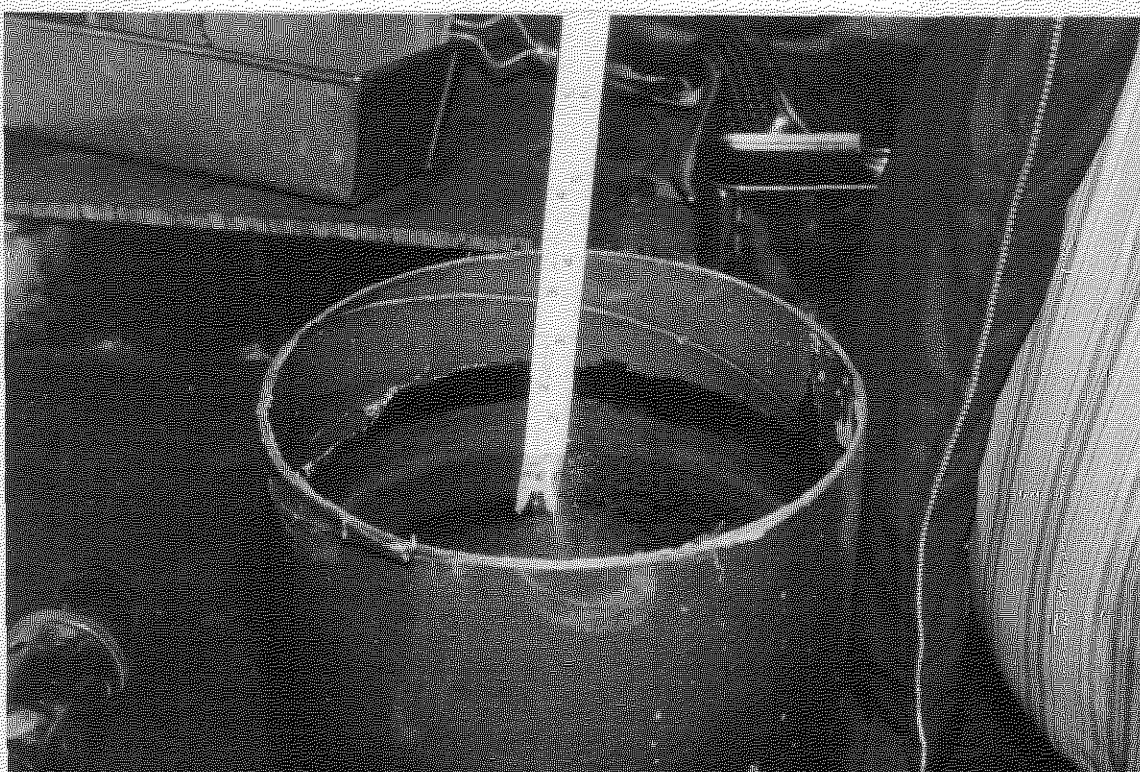


TABLE 1

Effect of Flocculants on Dewatering Characteristics of Hydraulically Pumped Paraho Retorted Shale.

Type of Flocculant	Amount Added Lbs./Ton	Water Released From Specimen %
None	-	7.5
Calcium Chloride	1	5.3
Calcium Chloride	5	6.4
Calcium Chloride	10	5.6
Separan MGL	1	11.6
Separan MGL	2	21.7
Separan MG 200	1	13.2
Separan MG 200	2	14.7
Separan MG 500	1	14.1
Separan MG 500	2	13.7
Separan MG 700	1	11.3
Separan MG 700	2	17.8

the best result being obtained with two pounds of Separan MGL per ton. It was noted that specimens placed in the perforated bottom molds dewatered better than specimens placed in the solid bottom molds. Cementing agents did not improve strength characteristics mainly because of the large amount of water retained in the slime size fraction, which forms 56% of the material.

Results of unconfined compression tests performed on dry retorted shale specimens are shown in Table 2. Moisture content had a significant effect on specimen strength, an increase of moisture content from 15% to 25% producing significant increases in strength. A retorted shale cementing agent ratio of 5 to 1 produced significant increases in strength. At 15 percent moisture and a 5 to 1 retorted shale-cementing agent mixture, specimens containing equal amounts of cement, flyash, and lime resulted in a higher ultimate strength than those containing cement alone. At 25 percent moisture and a 5 to 1 retorted shale-cementing agent mixture, the strength of specimens containing cement alone was 4.5 times the strength of those containing a cement, flyash, and lime mixture. With an 8-day cure the 30 to 1 retorted shale cementing agent mixture did not have any beneficial effect on strength.

6.0 Discussion:

The variables studied in these tests were type and amount of flocculant and cementing agent, and moisture content. Other variables

TABLE 2

Unconfined Compressive Strengths of Dry Retorted Shale Specimens At 8 Day Cure

Cementing Agents	Retorted Shale Cementing Agent Ratio	Compressive Strength PSI 15% Moisture	Compressive Strength PSI 25% Moisture
-	-	N.D.	17.7
100% Cement	5:1	26.5	389.2
100% Cement	30:1	N.D.	17.7
50% Each Cement & Flyash	5:1	N.D.	63.7
50% Each Cement & Flyash	30:1	N.D.	N.D.
50% Each Lime & Flyash	5:1	-	N.D.
50% Each Lime & Flyash	30:1	-	N.D.
33.3% Each Cement, Flyash & Lime	5:1	60.1	70.8
33.3% Each Cement, Flyash & Lime	30:1	N.D.	N.D.

N.D. - Not determined because specimen fell apart when removed from mold.

that have a significant effect on strength are curing period and curing environment; however, these were not studied because of the limited time available.

Calcium chloride and polyacrylamide had significantly different mixing and reaction characteristics. Calcium chloride could be dissolved with ease whereas polyacrylamide formed a sticky gel, giving the batch mixture a consistency similar to that of bread dough. Thus polyacrylamide cannot be used in hydraulic transport lines but will have to be mixed just before stowing. Calcium chloride did not seem to react with the chemicals in retorted shale and did not improve dewatering characteristics whereas Separan MGL produced a noticeable improvement. The rate of water release from the specimen was at a maximum in the first 24 hours and later reduced to a negligible value. The best dewatering results were obtained when both the processes of percolation and decantation were acting simultaneously.

The method of sample preparation affected the strength characteristics of dry retorted shale. Even distribution of moisture and cementing agent produced the best results. Curing period and curing environment affect the reaction rate between the cementing agent, water, and retorted shale, and it is possible that better strength characteristics would have resulted if the curing period had been increased and a constant temperature of 125°F had been maintained. Previous studies (5) have shown that the optimum moisture for Paraho retorted shale is approximately 22% and the results of this study indicate that a moisture content close to the optimum promotes cementing processes. The amount and distribution of fines has a considerable effect on the cementing process; thus, the particle breakdown in any mechanical backfilling method will affect strength characteristics. A retorted shale cementing agent ratio of 30 to 1 would be an economical mixture but it was found that no improvement in strength characteristics resulted from this mixture.

7.0 Conclusions and Recommendations:

7.1 Conclusions:

The addition of flocculants can improve the dewatering characteristics of hydraulically pumped retorted shale; however, due to the large slime-size fraction in the material, a large amount of moisture is still entrapped. Two pounds of Separan MGL per ton produced the best dewatering results. Moisture content close to the optimum promotes cementing processes in dry retorted shale. Maximum strength of dry fill was achieved with a 5 to 1 retorted shale cement mixture.

7.2 Recommendations:

It is recommended that further testing be performed on strength characteristics of dry retorted shale as modified by cementing agents. Curing period, curing environment, and proportions of the cementing agents should be varied in this testing program.

Further study of the effect of flocculants and cementing agents on hydraulically pumped retorted shale is not warranted because of the negative results obtained in this study.

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APPENDIX E

BOREHOLE TRANSPORT OF RETORTED SHALE REPORT
JENIKE & JOHANSON, INC.

APPENDIX E

2000 Foot-Deep Borehole
for the Transport of Retorted Oil Shale

The Cleveland-Cliffs Iron Company

77 - 42

Jenike & Johanson, Inc.
Storage and Flow of Solids
No. Billerica, Massachusetts

August 1977

Contents

29 July 1977

2000 FOOT-DEEP BOREHOLE FOR THE TRANSPORT OF RETORTED OIL SHALE

Background

The Cleveland-Cliffs Iron Company under contract with the Bureau of Mines is developing the concept of borehole transport of retorted oil shale for backfilling.

The Company has subcontracted to Jenike & Johanson, Inc. the analysis of the flow of shale in the borehole and from the borehole onto a belt feeder.

Introduction

Retorted oil shale is to flow down a 2000 foot deep borehole and onto a belt feeder at a rate of up to 3000 tph. In order for the flow onto the belt to be uniform and controlled it is necessary that:

(a) The flow of shale in the regions of the hopper above the outlets be steady. This means that the hopper must be mass flow, i.e. all the shale must be in motion whenever any of it is withdrawn.

(b) The pressure of air in the pores of the shale discharging onto the belt feeder be close to the ambient air pressure. If pore pressure is too high, shale will flush uncontrollably and flood the belt; if pore pressure is too low, flow will be intermittent with arching followed by flushing and flooding of the belt.

(c) The area of the outlets be sufficiently large to assure unobstructed flow at the specified rate.

(d) The chute and skirt design at the outlet ensure fully live outlets.

In addition to assuring controlled flow under continuous flow conditions, it is also necessary to provide for start-up during the filling of the borehole and for restarting after a stoppage of flow.

Particulate solids tend to flow in pulsating motion. Coarse, permeable solids pulsate more, fine impermeable solids pulsate less. Pulsation is particularly pronounced in tall vertical channels of constant cross section, like the borehole under consideration. The magnitude of likely pulsation in the borehole cannot be predicted for the lack of an appropriate theory. It is therefore necessary to provide a disengaging region between the borehole and the hopper outlets so that borehole pulsations do not affect the feed on the belt. Disengagement is obtained by providing a space in the hopper where the solid can form a free fluctuating surface. Through that surface air can also be introduced or evacuated, as needed, to maintain pore air pressure at the outlets close to ambient air pressure.

Material properties

Pertinent material properties refer to:

(1) The flow of solid neglecting the gaseous phase. Here belong the flow functions and the wall frictional and adhesive properties. Mass flow hopper configurations and minimum outlet sizes for flow without arching are determined from these properties.

Tests of this type have been performed for some twenty years [1,2,3]* and are carried out on the Flowfactor Tester and Consolidating Bench shown in Figs. 1 and 2, respectively. The tests run on shale have defined a minimum outlet diameter of 1.3 feet and hopper wall slope angles of 19° from the vertical for a circular cone, and 30° for a wedge, for a hopper made of carbon steel.

(2) The effect of the gaseous phase, in this case air, on the flow of the solid. Air is entrained with the shale down the borehole. As the solid compacts under the increasing solids pressure, the pore size is reduced and air pressure increases. At the outlets onto the belt feeder, shale expands and air pressure drops. How much air is entrained and what the air pressure is at the exit depends on the surface density, the compressibility and the permeability of the solid as well as on the ambient pressure and viscosity of air.

Compressibility is measured on the instrument shown in Fig. 3. A sample of solid in a shallow cylinder is compressed by a loaded piston and the bulk density is determined as a function of the effective head of solid. The function

$$\gamma = f(h) \tag{1}$$

is derived and plotted in Fig. 3.

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*Numbers in square brackets designate references at end of paper.

Permeability is measured on the instrument shown in Fig. 4. A sample of solid is placed in a cylinder and the air flow rate is measured for a given air pressure drop as a function of the bulk density of the solid. A combination of the results of these tests with the tests of compressibility leads to Darcy's relation in the following form (see below for notations)

$$u = .00437\ \left(\frac{\gamma}{68.8}\right)^{-7.88} \times \left(\frac{1}{\gamma}\frac{dp}{dz}\right) \tag{2}$$

Surface density is determined on the instrument sketched in Fig. 5. A known weight W of solid is placed in a tall cylinder of cross sectional area A with a porous bottom. An air pressure

$$p = \frac{W}{A} \tag{3}$$

is continuously applied at the bottom. Then, for a short period of time, a large volume of air is blown through the bottom to lift and fluidize the solid in the cylinder. As that air is turned off, material settles down. The lowest density at which the solid forms a distinct surface is taken as the surface density. This density was measured at $\gamma_s = 60$ pcf

Particle density was measured at $\Gamma = 165$ pcf.

Solid-gas steady-state flow relations in one dimension

Velocities

Call

- v - true absolute solid velocity
- w - true absolute gas velocity
- u - superficial slip gas velocity (relative to solid)

and the voids ratio

$$\nu = \left(\frac{1}{\gamma} - \frac{1}{\Gamma}\right)\gamma \tag{4}$$

where

- γ - solids bulk density
- Γ - particle density

The true slip gas velocity is u/ν, and the true velocities are related by

$$w = v + \frac{u}{\nu} \tag{5}$$

or, using (4),

$$\nu\, w = (\frac{1}{\gamma} - \frac{1}{\Gamma})\gamma\, v + u \tag{6}$$

Darcy

Darcy's relation for the gas superficial slip velocity is in the following form

$$u = u_o(\frac{\gamma}{\gamma_o})^n\, (\frac{1}{\gamma}\frac{dp}{dz}) \tag{7}$$

where dp/dz is the gas pressure gradient, while u_o, γ_o and n are experimentally determined constants.

Substitute this in (6)

$$\nu\, w = (\frac{1}{\gamma} - \frac{1}{\Gamma})\gamma\, v - u_o(\frac{\gamma}{\gamma_o})^n\, (\frac{1}{\gamma}\frac{dp}{dz}) \tag{8}$$

This relation applies at any cross section of a one dimensional channel.

Flow rates

Define the flow rates as follows:

$$Q = A\,\gamma\,v\,[\text{lb/sec}] \qquad \text{for the solid} \tag{9}$$

$$P = A\,p\,\nu\,w\,[\text{psia} \times \frac{ft^3}{sec}] \qquad \text{for the gas} \tag{10}$$

Substitute (8) in (10), then for $A\,\gamma\,v$ from (9)

$$P = \left[(\frac{1}{\gamma} - \frac{1}{\Gamma})Q - A\,u_o(\frac{\gamma}{\gamma_o})^n\,(\frac{1}{\gamma}\frac{dp}{dz})\right]p \tag{11}$$

Continuity

1. Channel without source

Consider two sections of a channel: 1 and 2.

Evidently

$$Q_2 = Q_1 = Q \tag{12}$$

and

$$P_2 = P_1 = P \tag{13}$$

In particular, the ratio of gas pressures is

$$\frac{p_2}{p_1} = \frac{(\frac{1}{\gamma_1} - \frac{1}{\Gamma})\ Q - A_1\ u_o\ (\frac{\gamma_1}{\gamma_o})^n\ (\frac{1}{\gamma}\frac{dp}{dz})_1}{(\frac{1}{\gamma_2} - \frac{1}{\Gamma})\ Q - A_2\ u_o\ (\frac{\gamma_2}{\gamma_o})^n\ (\frac{1}{\gamma}\frac{dp}{dz})_2} \tag{14}$$

2. Branch

For the branch shown in Fig. 6

$$Q_3 = Q_1 + Q_2 \tag{15}$$

and

$$P_3 = P_1 + P_2 \tag{16}$$

In particular, if $Q_2 = 0$, i.e. there is only gas in- or out-flow at section 2, then $Q_1 = Q_3 = Q$, and

$$- A_2\ u_o\ (\frac{\gamma_2}{\gamma_o})^n\ (\frac{1}{\gamma}\frac{dp}{dz})_2\ p_2 = \left[(\frac{1}{\gamma_3} - \frac{1}{\Gamma})\ Q - A_3\ u_o (\frac{\gamma_3}{\gamma_o})^n\ (\frac{1}{\gamma}\frac{dp}{dz})_3\right] p_3 +$$

$$- \left[(\frac{1}{\gamma_1} - \frac{1}{\Gamma})\ Q - A_1\ u_o\ (\frac{\gamma_1}{\gamma_o})^n\ (\frac{1}{\gamma}\frac{dp}{dz})_1\right] p_1 \tag{17}$$

Equilibrium in one dimension

$$\frac{d\sigma_z}{dz} + \frac{dp}{dz} + \frac{4\ K\ \tan\phi'}{D}\ \sigma_z = \gamma \tag{18}$$

in this equation of equilibrium for a vertical channel [4], σ_z is the solids pressure in the vertical direction, K is the Janssen ratio between the horizontal and vertical solids pressure and ϕ' is the friction angle between the solid and the borehole casing.

At the top surface: $z = 0$, $\sigma_z = 0$, $p = p_{ambient}$. In the limit as $z \to \infty$

$$\lim_{z\to\infty} \frac{d\sigma_z}{dz} = 0$$

$$\lim_{z\to\infty} \frac{dp}{dz} = 0 \tag{19}$$

the effective consolidating head of solid for this converged condition then is

$$\lim_{z\to\infty} \frac{\sigma_z}{\gamma} = \frac{D}{4\ K\ \tan\phi'} \tag{20}$$

The relation between γ and σ_z is taken in the form

$$\gamma = \gamma_o \left(\frac{\sigma_z}{\sigma_o}\right)^{\beta} \tag{21}$$

where σ_o, γ_o and β are determined from the curve shown in Fig. 3. This relation applies for $\sigma_z \geq \sigma_s$, where σ_s corresponds to the surface density γ_s; below σ_s, the density is taken constant, $\gamma = \gamma_s$.

Borehole and hopper

The layout of the borehole and hopper is shown in Fig. 7; the hopper is shown in greater detail in Figs. 8 and 9. The top 100 or 200 feet of the borehole are of 10 ft diameter, the remainder of 8 ft diameter. Aeration rings are placed at 500 ft intervals of the borehole, closer in permeable rock, to maintain or restore borehole air pressure during and after shutdown, thus allowing easy restart. The borehole discharges into a hopper. The shale flows down the mass flow hopper into two

3 ft diameter outlets which feed onto a 72 inch, 35° trough-angle belt. Since the belt needs to be reversible, the chutes have pivoted skirts to permit each outlet to discharge approximately the same layer of material on the belt.

Calculations indicate that an excess of air is likely to be entrained into the borehole. That excess will be evacuated at the free surface of the hopper by maintaining an air pressure at the top of the hopper lower than the pressure in the pores of the shale issuing from the borehole. The flow pattern of shale expanding from the borehole into the hopper cannot lead to uniform deaeration. The proposed hopper design aims at the prevention of gross nonuniformities. In addition, hopper ring-expansions are indicated. These rings provide a passage through which air pressure equalization can take place across a hopper prior to discharge. This will help prevent flushing of solid through the side of the outlet with an excess of air, while the other side flows sluggishly because of air deficiency.

Flow calculations will be done with respect to the three ~~five~~ Sections, indicated in Figs. 7 and 8.

At the surface of the borehole - Section 1

Ambient air pressure is taken at $p_1 = 22''$ Hg = 10.8 psia, shale bulk density $\gamma_1 = \gamma_s = 60$ pcf, cross sectional area of borehole $A_1 = \frac{\pi}{4}\, 10^2\ \text{ft}^2$, the surface layer of solid is assumed in a state close to fluidization $\left(\frac{1}{\gamma}\frac{dp}{dz}\right)_1 \approx 1$, particle density $\Gamma = 165$ pcf.

The air flow rate is then computed from (11) as follows:

$$p_1 = \left[\left(\frac{1}{60} - \frac{1}{165}\right)\frac{3000}{1.8} - \frac{\pi}{4}\, 10^2 \times .00437 \left(\frac{60}{68.8}\right)^{-7.88}\right] 10.8$$

$$= 180.0 \text{ psia} \times \frac{\text{ft}^3}{\text{sec}} \tag{22}$$

In the borehole - Sections 10 and 8

Equations (11), (18) and (21) permit to compute the pressures in the borehole. The converged values are computed first. The effective consolidating head of the solid is computed from (20) with K = .4 and $\phi' = 22°$, the latter as measured on instruments, Figs. 1 and 2. For the 10 foot borehole, the head is

$$h_{10} = \frac{10}{4 \times .4 \times \tan 22°} = 15.5 \text{ ft} \tag{23}$$

and for the 8 foot borehole

$$h_8 = \frac{8}{4 \times .4 \times \tan 22°} = 12.4 \text{ ft} \tag{24}$$

The corresponding bulk densities are read from Fig. 3 at: $\gamma_{10} = 78.2$ pcf and $\gamma_8 = 77.8$ pcf

The solids pressures are

$$(\sigma_z)_{10} = h_{10} \times \gamma_{10} = 15.5 \times 78.2 = 1210 \text{ psf} \tag{25}$$

$$(\sigma_z)_8 = h_8 \times \gamma_8 = 12.4 \times 77.8 = 963 \text{ psf} \tag{26}$$

The relation of Fig. 3 determines the constants in formula (21) at

$$\gamma = 60.0 \left(\frac{\sigma_z}{.114}\right)^{.0289} \tag{27}$$

for $\sigma_z \geq .114$. For smaller values of σ_z, $\gamma = \gamma_s = 60$ pcf.

Since $P = P_1$ in accordance with (13), the corresponding gas pressures are found from (11), with $(\frac{1}{\gamma}\frac{dp}{dz}) = 0$, as follows

$$p_{10} = \frac{180.0}{(\frac{1}{78.2} - \frac{1}{165})\frac{3000}{1.8}} = 16.1 \text{ psia} \tag{28}$$

$$p_8 = \frac{180.0}{(\frac{1}{77.8} - \frac{1}{165})\frac{3000}{1.8}} = 15.9 \text{ psia} \tag{29}$$

The calculation then proceeds from: $z = 0$, the converged value of p, a value of σ_z either slightly lower or slightly higher than the converged value, and $\Delta z < 0$. If a lower than the converged value of σ_z is taken, then σ_z decreases monotonically to zero at a surface; if a higher value is taken, then σ_z increases monotonically. The borehole under study involves both cases, as shown in Fig. 10.

The pressure distribution is almost independent of the solids flow rate Q for Q between 500 and 4,000 tph.

At the hopper outlets - Section 3

The effective head at the outlet is

$$h_3 = \frac{\sigma_3}{\gamma_3} = ff\ \frac{B}{H} = 1.3\ \frac{3.0}{2.3} = 1.7\ \text{ft} \tag{30}$$

B = 3 ft is the diameter of the outlet, while ff and H are hopper and material parameters taken from reference [3]. The corresponding bulk density is read from Fig. 3 at $\gamma_3 = 73.2$ pcf.

The ambient air pressure at a depth of 2000 feet is

$$p_3 \approx p_1 + \frac{.074}{144} \times \frac{p_1 + p_3}{2 \times 14.7} \times 2000 = 11.6\ \text{psia} \tag{31}$$

where $p_1 = 10.8$ psia.

For controlled flow from the hopper, it is necessary to have $(\frac{1}{\gamma}\frac{dp}{dz})_3 \approx 0$. Hence the required air flow rate at the hopper outlets is, in accordance with (11),

$$P_3 = (\frac{1}{73.2} - \frac{1}{165})\ \frac{3000}{1.8} \times 11.6 = 146.9\ \text{psia} \times \frac{ft^3}{sec} \tag{32}$$

Gravity flow rate. This is computed from the formula derived in reference [5]

$$Q_g = 2 \times 1.8\ \gamma\ \frac{\pi B^2}{4}\ (\frac{B\ g}{4\ \tan\theta_c})^{\frac{1}{2}}$$

$$= 2 \times 1.8 \times 73.2\ \frac{\pi \times 3^2}{4}\ (\frac{3 \times 32}{4\ \tan 19°})^{\frac{1}{2}} = 15{,}551\ \text{tph} \tag{33}$$

This allows a uniform flow rate by providing an excess rate factor of

$$\frac{15{,}551}{3{,}000} = 5.2 \tag{34}$$

Belt feeder. Use a 72 inch belt with a 35° trough angle. Then for a conservative surcharge angle of 5°, the cross sectional bed area is 3.229 ft^2 (Jeffrey-Dresser catalog) and a capacity of 19,380 ft^3/hr at 100 fpm. For 3,000 tph, the belt speed should be

$$V_b = \frac{3000 \times 2000}{73.2 \times 19{,}380} \times 100 = 423 \text{ fpm} \tag{35}$$

Shut-off gates. Use gates at the outlets of the hoppers only for shut-off during initial filling of the borehole, for emergency and for maintenance.

At the free surface at top of hopper - Section 2

Since the desirable air flow rate at the hopper outlets P_3 is less than the air entrained in the borehole P_1, the excess air in the amount of

$$P_2 = P_3 - P_1 = 146.9 - 180.0 = -33.1 \text{ psia} \times \frac{\text{ft}^3}{\text{sec}} \tag{36}$$

must be evacuated through the surface at the top of the hopper. At the top of the hopper, solids pressure $\sigma_2 = 0$ and material flowing from the borehole to the top surface is free to expand so that pore air pressure at the surface equalizes with the air pressure above the solid. Assume that pressure $p_2 = 12.0$ psia, somewhat higher than the ambient pressure in the mine. p_2 is lower by 3.9 psi than the air pressure in the borehole. The surface level of material in the hopper will, therefore, rise to balance that difference. The surface density γ_2 is computed from (14) on the assumption that rapid expansion with negligible gas flow occurs as shale issues from the borehole to the hopper, i.e. for $(dp/dz)_2 = (dp/dz)_8 = 0$

$$\gamma_2 = \frac{1}{\frac{1}{165} + \left(\frac{1}{77.8} - \frac{1}{165}\right)\frac{15.9}{12.0}} = 66.4 \text{ pcf}$$

The surface area A_3' needed to remove air at the rate P_2 is estimated from (11) with $Q_2 = 0$ at

$$A_2' = \frac{\frac{33.1}{12.0}}{.00437\left(\frac{66.4}{68.8}\right) - 7.88} = 477 \text{ ft}^2$$

An area $A_2 = 1728 \text{ ft}^2$ is recommended in view of the uncertainties involved in the flow at the surface (F.S. = [illegible]).

Operation

Controls. In order to assure smooth start-up, restart after a stoppage and steady state operation, it will be necessary to monitor and control the following:

(a) Level of solids in the borehole. While the borehole will provide a substantial surge capacity, the level of solids should be monitored and the in- and out-flow rates controlled to maintain the shale level within specified high-low limits.

(b) Air pressure at the top of the hopper. The amount of air entrained into the borehole depends on the bulk density of the shale at the surface of the borehole. That density cannot be accurately predicted from a one-dimensional analysis and experiment because, in fact, the problem is two-dimensional with, to date, no available solutions. Similarly, the amount of air expelled through the top surface in the hopper is uncertain through the lack of a two-dimensional solution of the flow from the borehole to the hopper. However, since the rate P_2 of air evacuation is directly proportional to the air pressure p_2 at the top of the hopper, that pressure can be used effectively to control the uniformity of flow on the belt. Provision should be made to hold pressure p_2 at any required level between 10 and 12 psia.

(c) Feed rate from hoppers. The height of the chutes above the belt feeder can provide gross feed rate control. Fine adjustment will be best achieved with a variable speed drive. The slide gates at the hopper outlets should not be used for rate control because, in a partly open position, they would prevent mass flow from developing and would lead to nonuniform flow with likely flushing.

Initial filling of borehole

Initial filling should proceed at a low charging rate in order to minimize impact loads on the hopper. After the initial 500 tons have been dropped, the gates should be opened and the belt started. The withdrawal should proceed through both outlets at a rate, say, one fourth of the rate of charge into the borehole. The top of the hopper should be vented at the ambient pressure in the mine. When the operating level of solids in the borehole has been reached, all the rates can be increased to normal operating values.

Emergency cessation of flow

Air pressure at the aeration rings should be monitored and, if that pressure drops to 13 psia during a cessation of flow of shale from the hopper, air at that pressure should be supplied to the borehole. This will permit rapid return to full-rate borehole flow when the emergency ends.

Emergency outlet from hopper

The position of a side chute is shown in Fig. 8. This chute can be installed, if required, for use when the main belt feeder is down.

Higher Level Withdrawal

Backfilling will start at the lowest level of the mine and, as that level is filled, a new hopper will be constructed at a higher level. In order not to lose backfilling time as the operation is shifted from one level to the next, the construction of the new hopper should proceed without interrupting the backfilling at the lower level.

If possible, the new hopper should be built around the borehole. The two pantleg hoppers can be completed on the sides of the operating borehole. Fig. 11 shows this Stage 1 of construction. The operation would then have to stop while, in Stage 2, the work shown with dashed lines is carried out to install the reversible conveyor at the bottom and connect the borehole with the hoppers at the top. This method would be the most satisfactory from the solids flow standpoint.

The alternative method would be to build the complete hopper in a location offset from the borehole by whatever distance is required. The top of the hopper would then be connected, in Stage 2, to the borehole by means of a steel-lined 8-foot diameter shaft inclined at an angle of, say, 15° to the vertical, Fig. 12.

Further Fundamental Work Needed

The solids and gas flow patterns at an expansion, like the one of the falling stream on the top of the borehole as well as the one from the borehole to the hopper, are problems in two dimensions; the former in axial symmetry, the latter in plane strain. Solutions to these problems are needed but unavailable. The one-dimensional analysis provides only a rough approximation.

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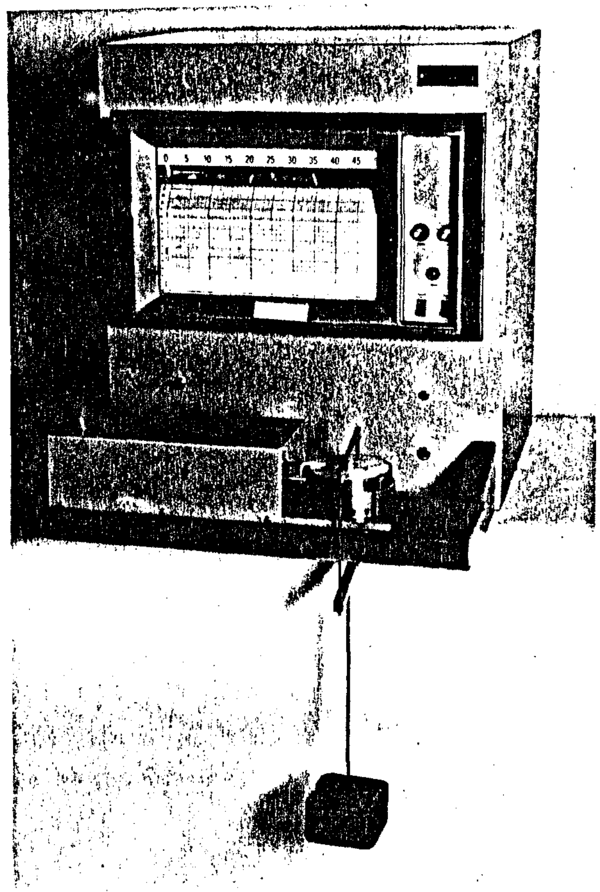
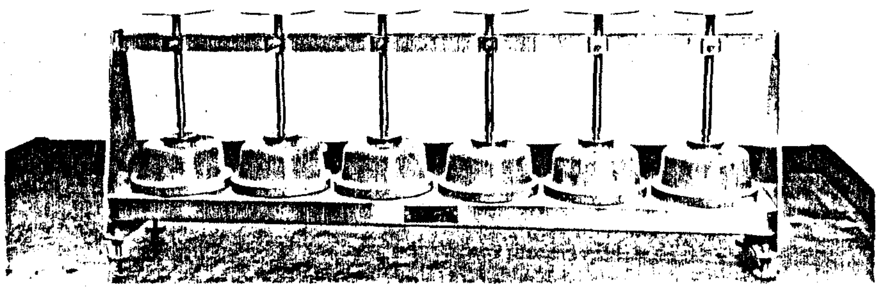


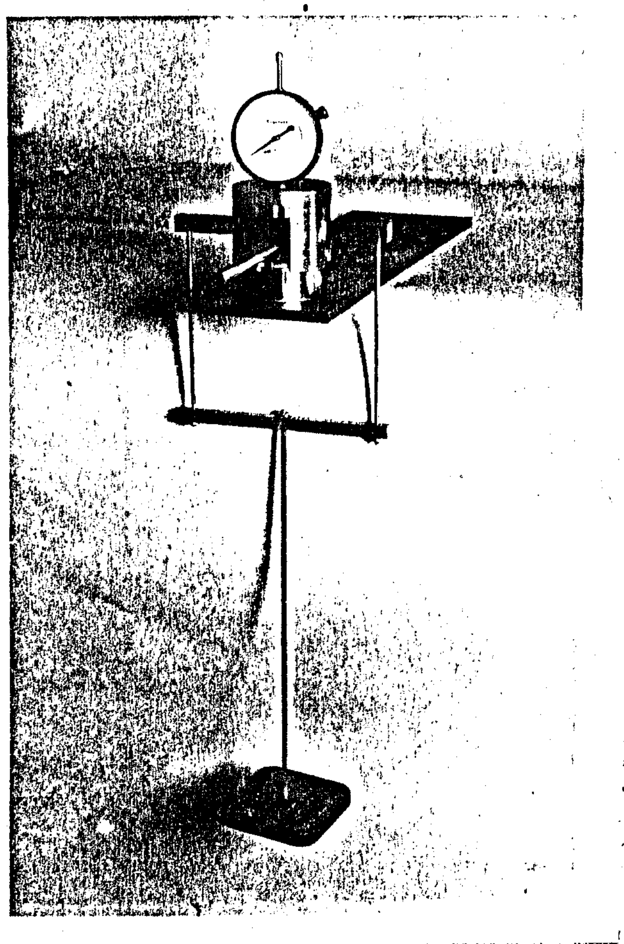
Flowfactor Tester (Jenike & Johanson, Inc.)

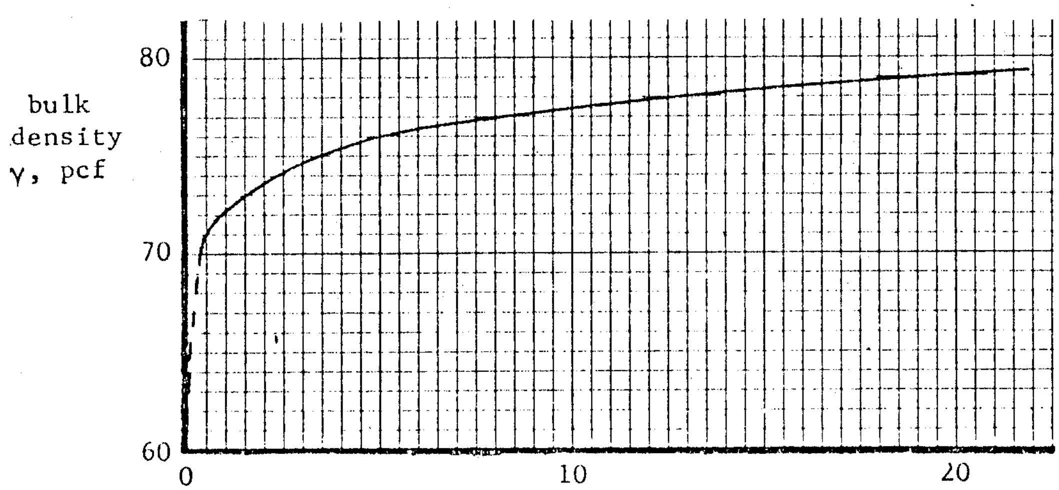
Fig. 1



Consolidating Bench (Jenike & Johanson, Inc.)

Fig. 2

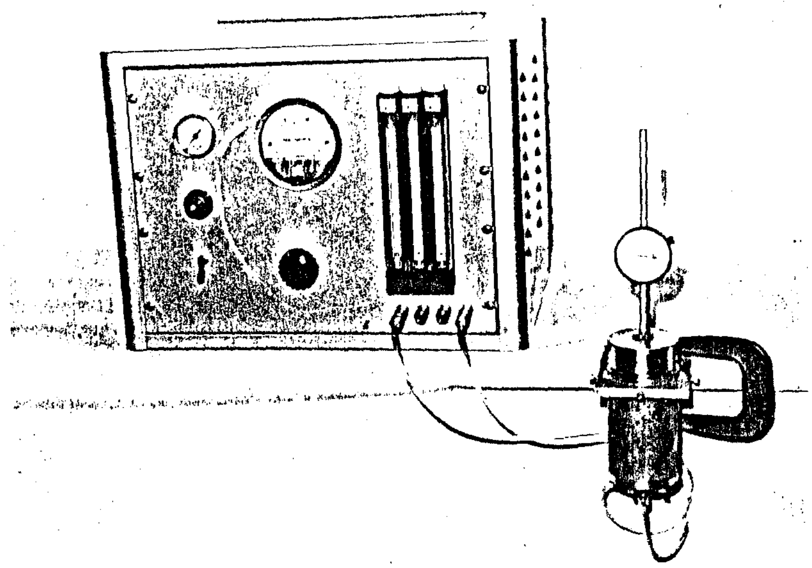




Compressibility Tester (Jenike & Johanson, Inc.)

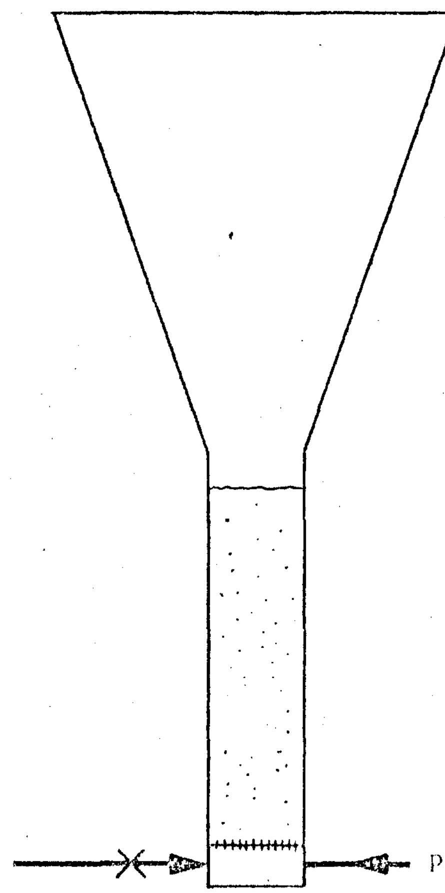
and function $\gamma = f(h)$

Fig. 3



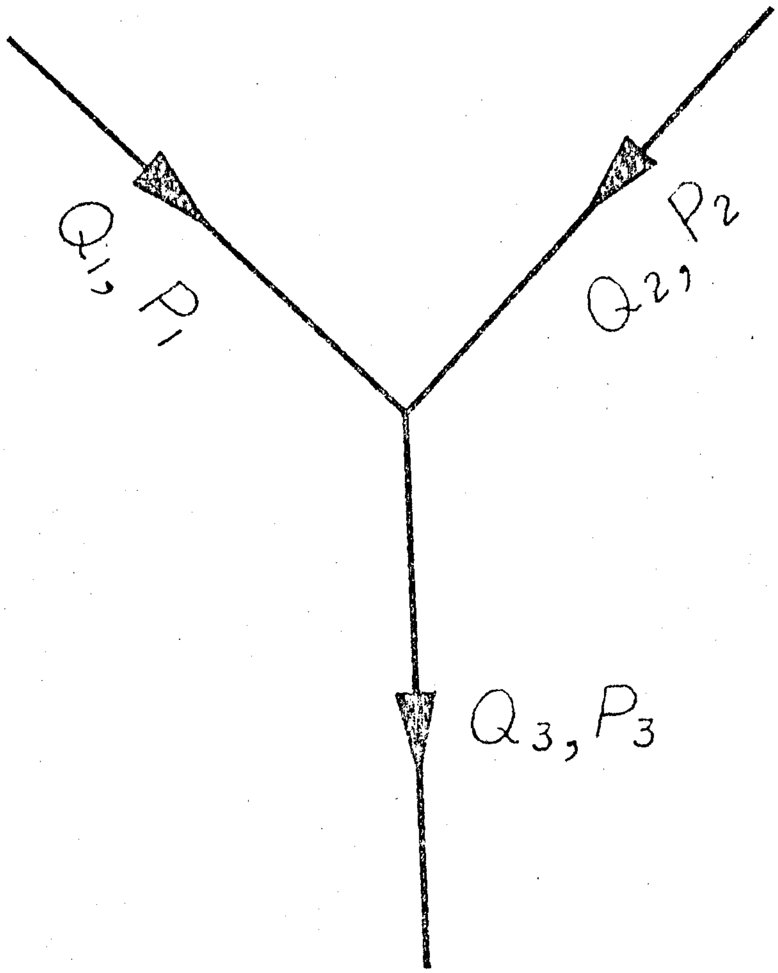
Permeability Tester (Jenike & Johanson, Inc.)

Fig. 4



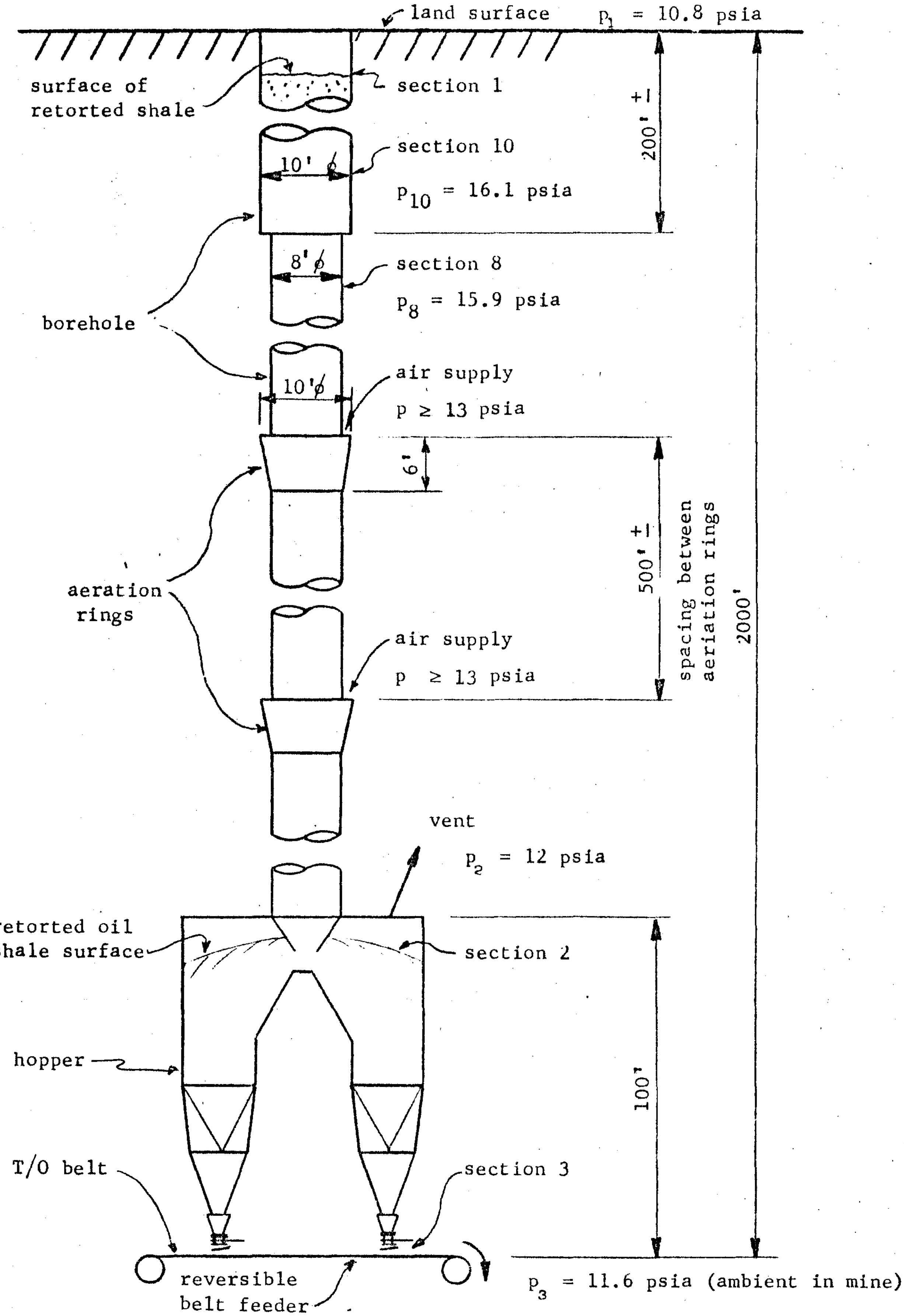
Surface Density Tester

Fig. 5



Branch

Fig. 6



Borehole and Hopper Layout

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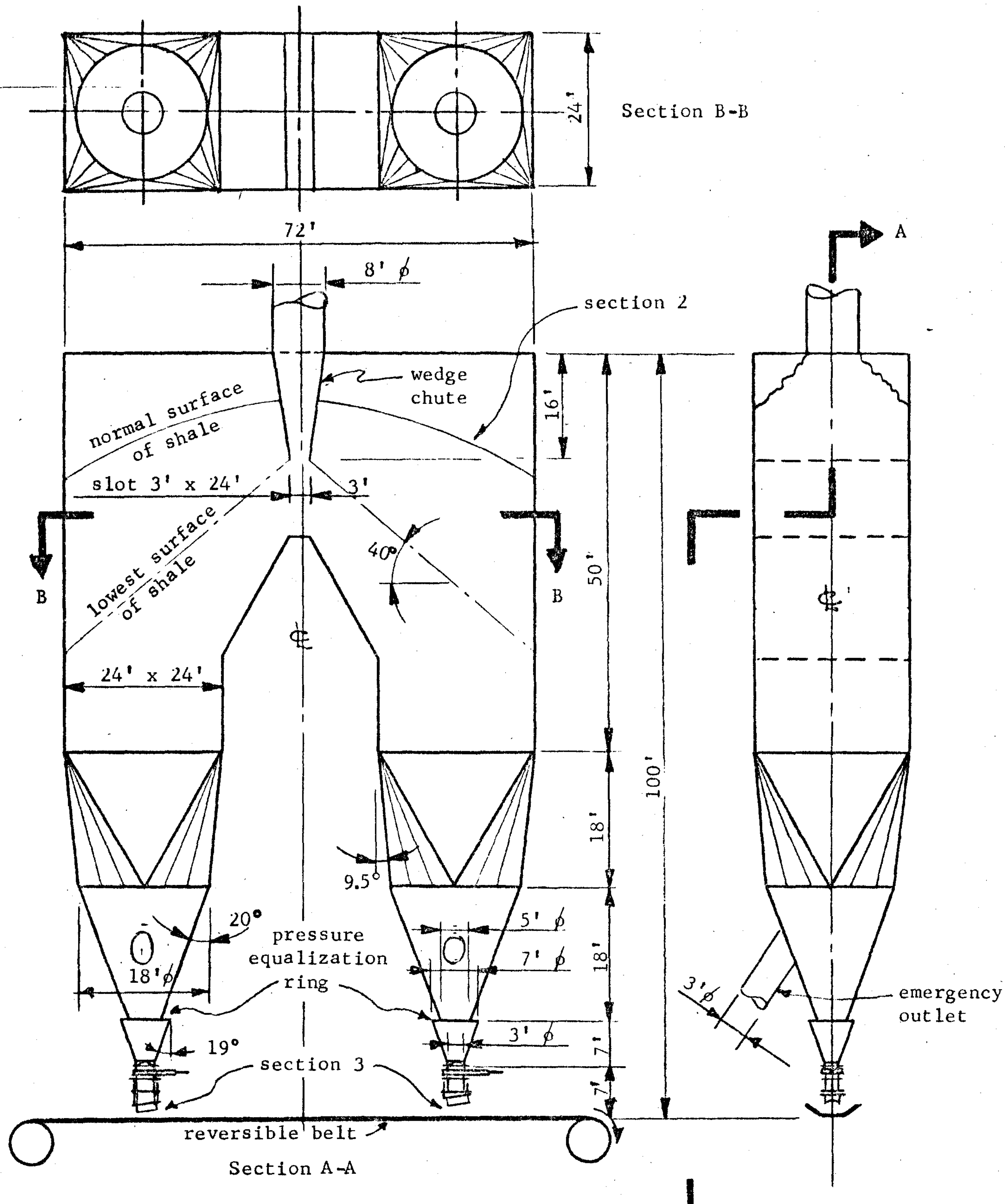
Fig. 7



Note: The lower 45 feet of hopper made of carbon steel

- Dust control closure of belt not shown

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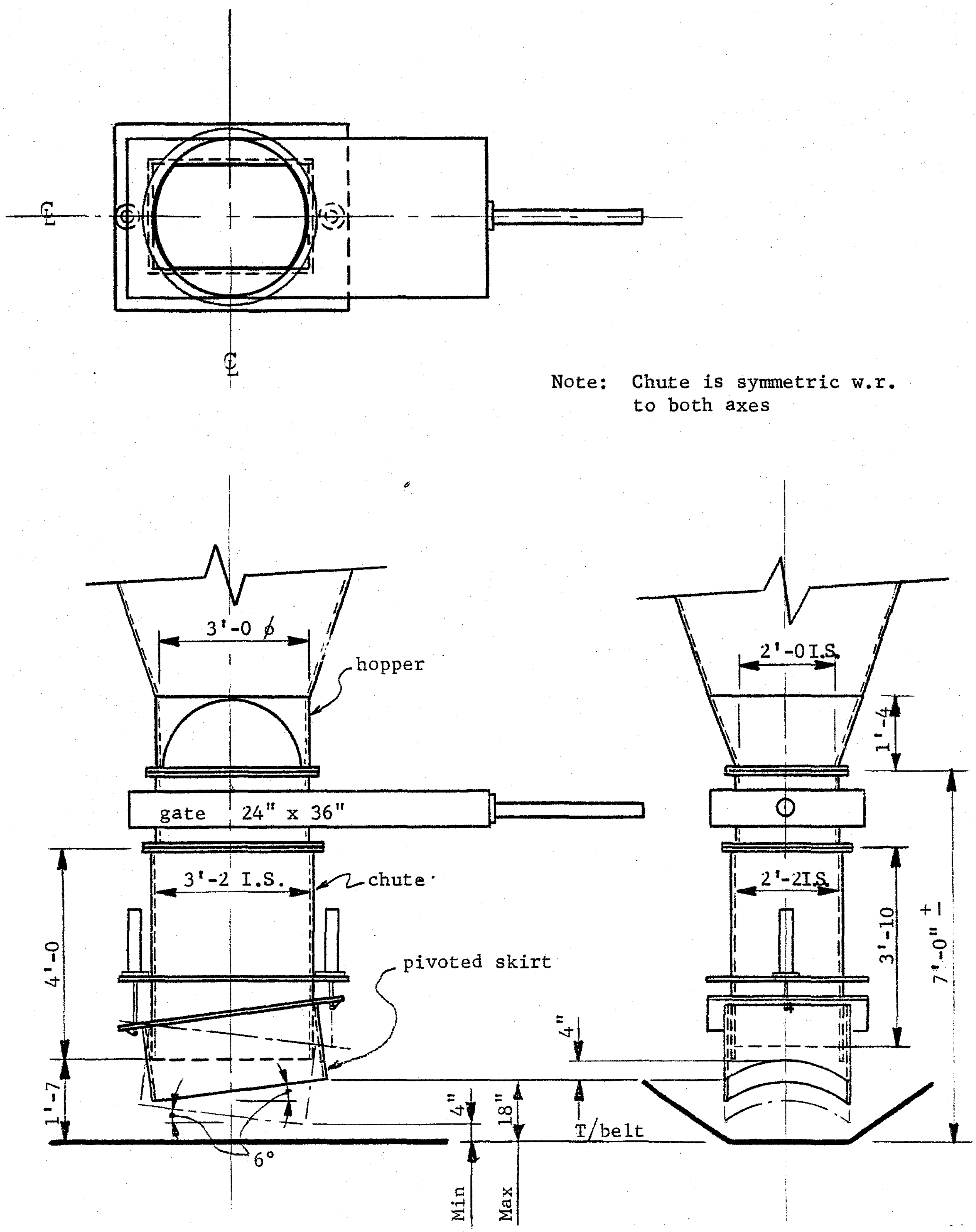
Hopper Layout

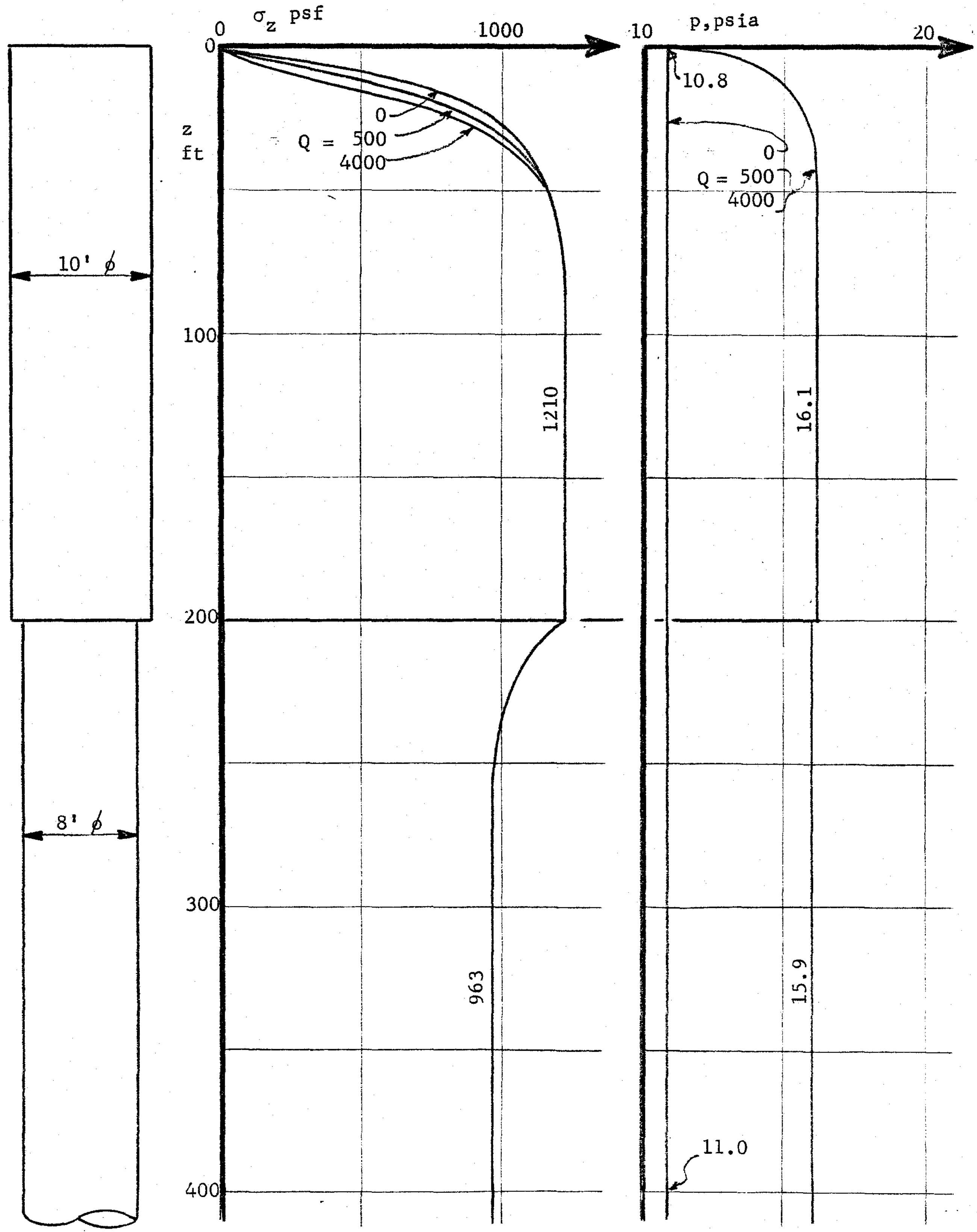
Fig. 8



Note: Chute is symmetric w.r. to both axes

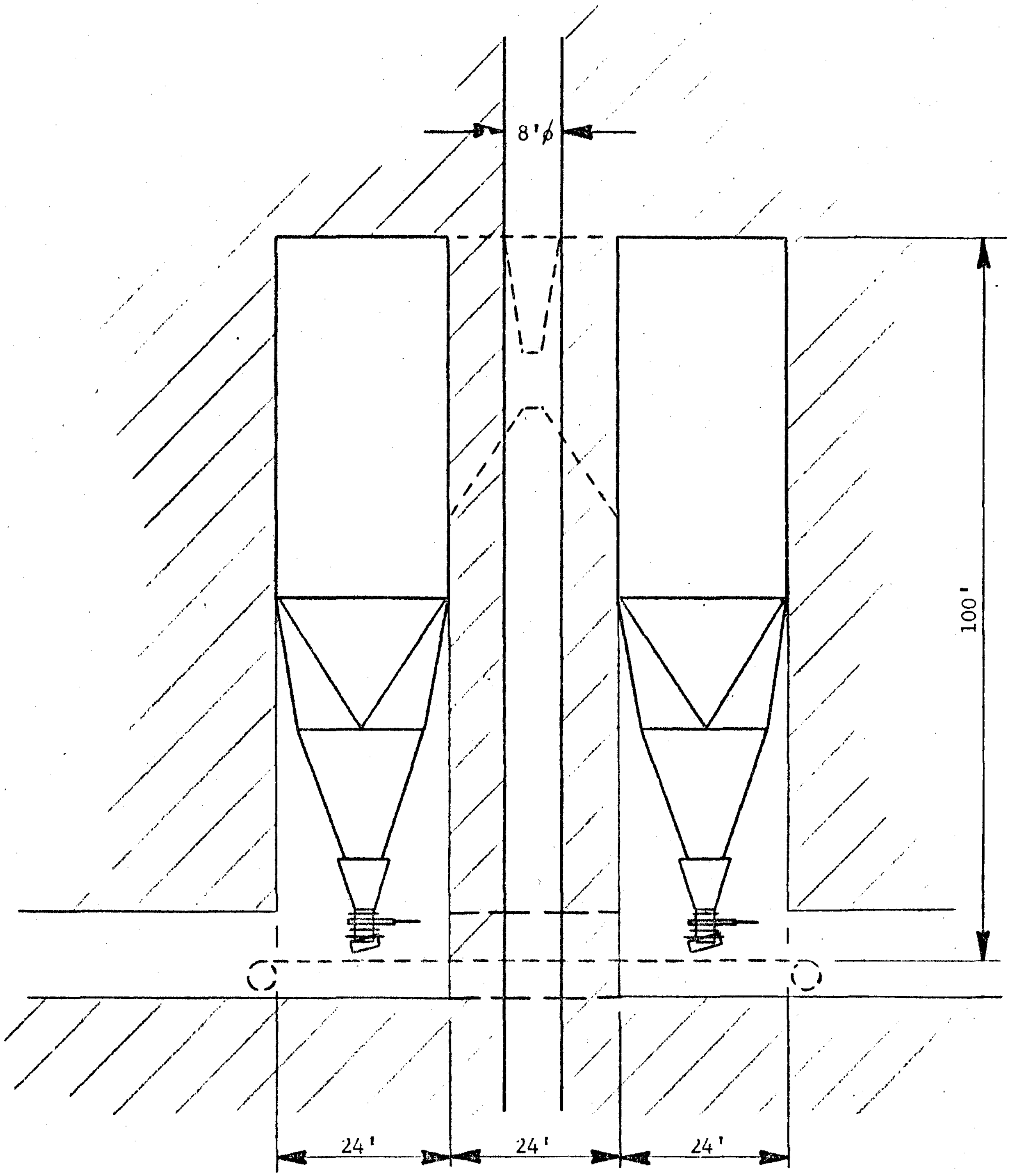
Hopper and Chute Detail

Fig. 9



Pressures σ_z and p in borehole

Fig. 10



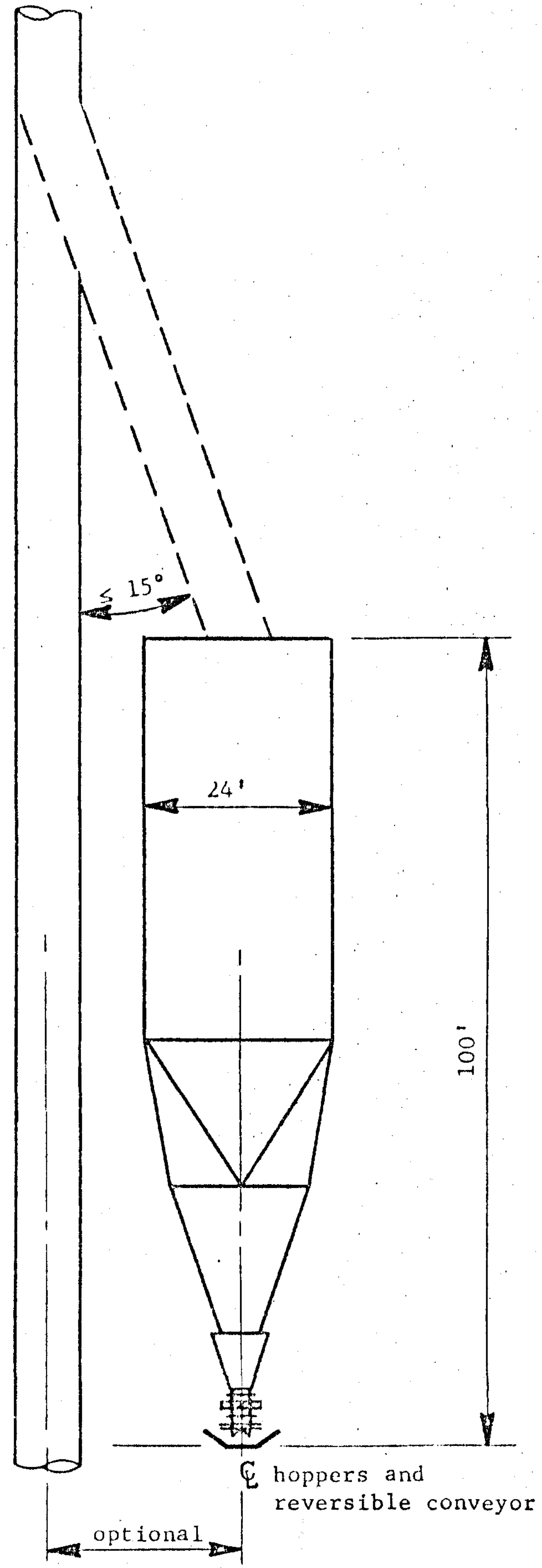
Stage 1 - continuous lines

Stage 2 - dashed lines

Construction of Higher Level Withdrawal

Alternative #1

Fig. 11



Construction of Higher Level Withdrawal

Alternative #2

Fig. 12

new 8.22.77

STANDARD TEST REPORT

Retorted Oil Shale

Cleveland Cliffs

77 - 42

Jenike & Johanson, Inc.
Storage and Flow of Solids
No. Billerica, Massachusetts

June 1977

CONTENTS

SELECTION OF BIN TYPE AND FEEDER SIZE

Types of bins

In the process of selection of a bin [1,2]*, the first step is to decide on the type of bin. From the standpoint of flow, there are three types of bins: mass-flow, funnel-flow and expanded-flow.

Mass-flow bins are particularly suitable for powders, cohesive solids, solids which degrade with time, and when segregation needs to be minimized. Mass-flow bins give a uniform feed density often permitting the use of volumetric feed control in place of gravimetric control. Low level indicators work reliably in mass-flow bins.

Examples of mass-flow bins are shown in Figures 1a and 1b. In mass-flow bins, the hopper, i.e. the converging part of a bin, is sufficiently steep and smooth to cause flow of all the solid without stagnant regions whenever any solid is withdrawn. This produces uniform flow and a feed density which is practically independent of the head of solid in the bin. Segregation is minimized because, while a solid may segregate at the point of charge into the bin, continuity of flow enforces remixing of the fractions within the hopper [1,3]. Mass-flow bins have a first-in first-out flow sequence. This is useful in the storage of solids which degrade with time and of powders which thus deaerate with a minimum of residence time. Powders do not flush as long as a sufficient head of solid is maintained and the gravity flow rate is not exceeded; air locks are not needed [4,5,6].

Fig. 1a defines the outlet width for oval outlets and the hopper slope angles for transition hoppers.

Fig. 1b defines the outlet diameter and the hopper slope angle for circular, conical hoppers.

The minimum outlet dimensions are given in "Results of Flow Tests," Sections I-A and I-B. The length of an oval outlet should be at least

* Numbers in brackets designate references listed on page 5.

three times its width. The maximum slope angles (measured from the vertical) are given in Section III for the specified wall materials and conditions of storage. For some solids and wall materials, the slope angles vary with the size of the outlet.

Valleys are not permitted. Ledges and protrusions into a mass-flow channel are not allowed. If a hopper is equipped with a shut-off gate, the gate must be fully open for mass flow to be possible.

Mass-flow bins of special design can be used for in-bin blending by circulation of solid [7].

Funnel-flow bins are suitable for coarse, free-flowing or slightly cohesive, non-degrading solids when segregation is unimportant. Examples of funnel-flow bins are shown in Figures 2a, 2b and 2c. Funnel flow occurs when the hopper is not sufficiently steep and smooth to allow material to slide along the walls or when the outlet of a mass-flow bin is not fully effective (see "Feeders" below). In these bins, solid flows toward the outlet through a channel that forms within stagnant solid. The diameter of that channel approximates the largest dimension of the effective outlet. When the outlet is fully effective, this dimension is the diameter of a circular outlet, or the diagonal of a square or rectangular outlet. As the solid within the channel flows out, layers slough off the top of the stagnant mass into the channel. This spasmodic behavior is particularly pronounced with cohesive solids and with powders. A powder, sloughing off the top, aerates as it falls into the channel and may flush out if the level of solid in the bin is low. With a cohesive solid, a channel may empty out completely (rathole) and powder charged into the bin then flushes through. A rotary valve is often used under these conditions to contain the material, but a uniform flow rate cannot be ensured with this arrangement because flow into the valve is erratic. These bins are more prone to cause arching of cohesive solids than are mass-flow bins and, hence, frequently require larger outlets for dependable flow.

Such a large outlet may preclude the use of a rotary valve. These bins also cause segregation of solids [1,3] and are unsuitable for solids which degrade with time because of the stagnant regions.

Figures 2a and 2b define the outlet dimensions for rectangular outlets. Fig. 2c defines the outlet dimension for circular outlets.

The minimum dimensions are listed in Sections I-A and I-B. The length of a slot should be either not less than D_f, Fig. 2a, or equal to the full diameter of the bin, Fig. 2b. Usually D_f is much larger than B_f and a long rectangular (slot) outlet is more advantageous than a circular or square one.

Clean out of a funnel-flow bin is often uncertain because solid in the stagnant regions may pack hard and cake. Mass-flow bins should be specified when clean out is mandatory.

Expanded-flow bins are recommended for the storage of large quantities of non-degrading solids. This design is also useful as a modification of existing funnel-flow bins to correct erratic flow caused by arching, rat-holing or flushing. The concept can be used with single or multiple outlets.

Examples of expanded-flow bins are shown in Figures 3a and 3b. The lower part of such a bin is designed for mass-flow. The mass-flow outlet usually requires a smaller feeder than would be the case for a funnel-flow bin. The mass-flow hopper expands the flow channel to a diagonal or diameter equal to or greater than D_f, thus eliminating the likelihood of rat-holing. In the case of powders, the volume of the mass-flow hopper should ensure sufficient residence time for deaeration.

Feeders

The specified outlet must be fully effective. If flow from the bin is controlled by means of a feeder, the feeder must be so designed as to draw uniformly through the entire cross-section of the outlet [1,2].

Vibrating equipment

Vibration has two effects: on the one hand it tends to break up the arches that obstruct flow, on the other it packs so stagnant solid gains greater strength. In order to allow for this packing, the outlet dimensions for flow with vibration are often larger than those for gravity flow without vibration.

Many fine and wet materials tend to pack severely when vibrated, and vibrating equipment is not recommended for them. The applicability of vibrating equipment can be judged from Section I-B of the "Results of Flow Tests" which lists the minimum outlet dimensions required for flow with vibrating equipment. These dimensions refer to the outlet dimensions of a static bin that discharges into vibrating equipment.

External vibrators are particularly suitable for materials which are free-flowing under conditions of continuous flow, but set up and gain strength when stored at rest for hours or days. Hoppers for these materials should always be equipped with pads for the mounting of external vibrators.

Flow rate of powders

The rate at which a coarse solid discharges through an outlet [8] is usually sufficient to satisfy the requirements of the process. The situation is different with powders. Density changes which occur within a flowing mass, combined with the low permeability of a powder, cause air pressure gradients within the mass and these gradients critically affect the rate of powder discharge from a bin. Depending on the fineness of the powder, its volumetric rate of gravity discharge may be one hundredth or one thousandth that of a coarse solid.

If the flow rate tests have been run, conservative estimates of the maximum rate for mass-flow hoppers are shown in Section IV of the "Results of Flow Tests." They are based on material discharging onto a belt feeder.

These flow rates can be increased significantly using an air permeation technique (patent pending). Examples of the application of air permeation are given in references [4,5,6].

Calculation of effective consolidating head h_e

The critical rathole diameter D_f (Section I), bulk density (Section II), wall friction angle φ' (Section III), and fine powder flow rate (Section IV) are functions of the consolidation pressure of the solid in the pertinent regions of the bin. To calculate the critical rathole diameter and fine powder flow rate, this pressure is specified by the effective head of solid in the bin h_e as follows:

$$h_e = \frac{R}{\mu K}\left[1 - e^{-\mu K \frac{h}{R}}\right] \tag{1}$$

or $\qquad h_e = 2R \qquad$ whichever is larger

where

R = hydraulic radius of the cylinder (cross-sectional area/circumference)

$R = D/4$ for a circular cylinder of diameter D

$R = W/2$ for a long rectangular cylinder of width W

μ = coefficient of friction on the cylinder walls

$\mu = \tan \phi'$

K = ratio of horizontal to vertical pressures = 0.4

h = height of vertical bin section

To calculate the bulk density and wall friction angle at the outlet of a mass flow bin,

$$h_e = B_p \text{ for transition hopper}$$
$$= B_c/2 \text{ for conical hopper}$$

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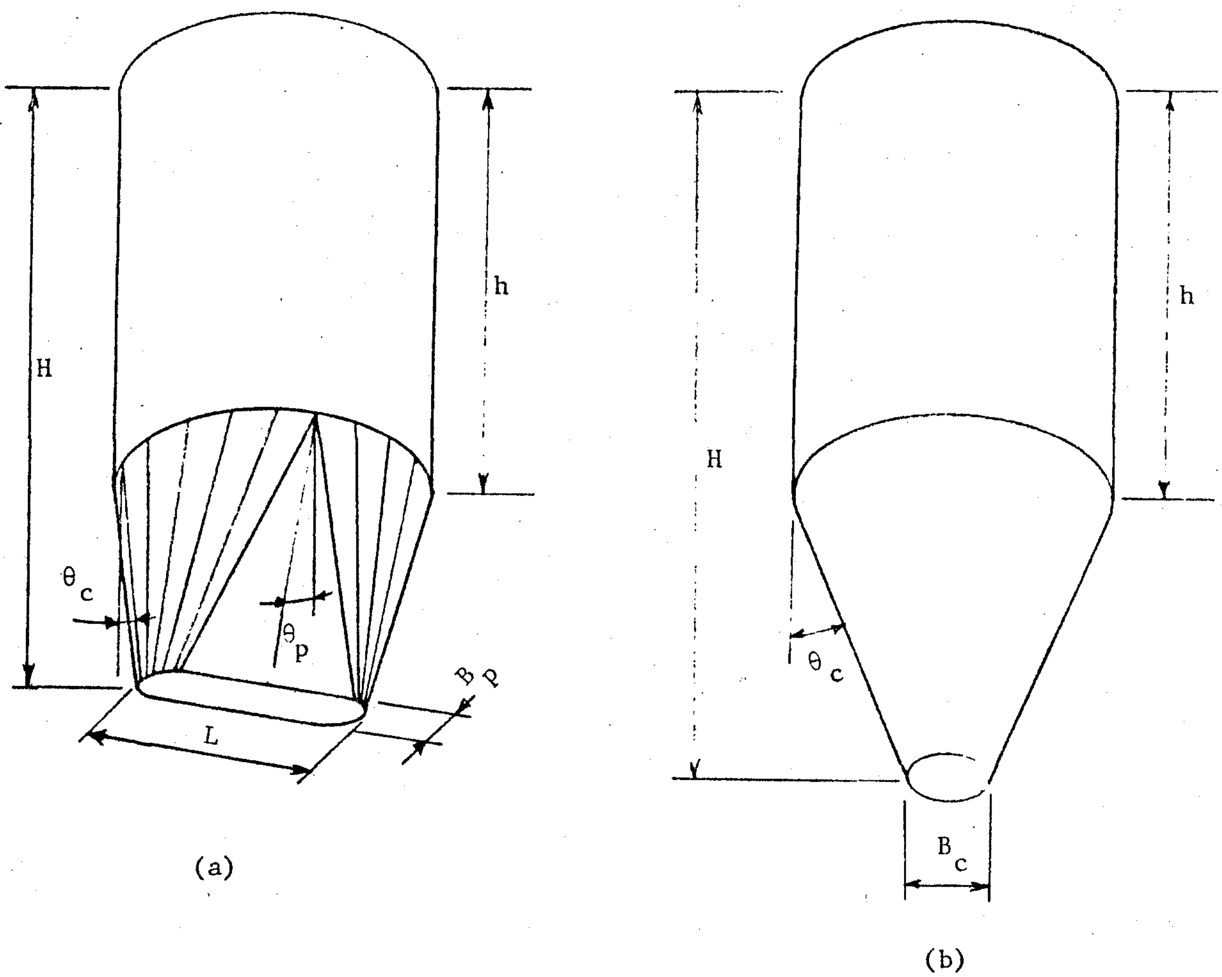
Glossary of terms and symbols

Arching	- a no-flow condition in which material forms a stable dome (or arch) across the bin
Bin	- container for bulk solids with outlets for withdrawal of solids either by gravity alone or by gravity assisted by flow-promoting devices
Cylinder	- vertical part of a bin
Expanded flow	- flow pattern which is a combination of mass flow and funnel flow
Feeder	- device for controlling the rate of withdrawal of bulk solid from a bin
Flow channel	- space in bin through which a bulk solid is actually flowing during withdrawal
Funnel flow	- flow pattern in which solid flows in a channel formed within stagnant material
Hopper	- converging part of a bin
Mass flow	- flow pattern in which all solid in a bin is in motion whenever any of it is withdrawn
Piping	- a no-flow condition in which material forms a stable vertical hole within the bin
Ratholing	- same as piping
B_c	- minimum diameter of a circular outlet in a mass-flow bin to prevent arching, ft
B_f	- minimum width of a rectangular outlet in a funnel-flow bin to prevent arching, ft
B_p	- minimum width of an oval outlet in a mass-flow bin to prevent arching, ft
D_f	- critical piping or ratholing dimension, ft
F	- unconfined compressive force, lb; equal to 1/13 times the unconfined compressive strength, psf
H	- total height of bin, ft
h	- height of cylinder, ft
h_e	- effective consolidating head, ft
L	- length of hopper outlet, ft

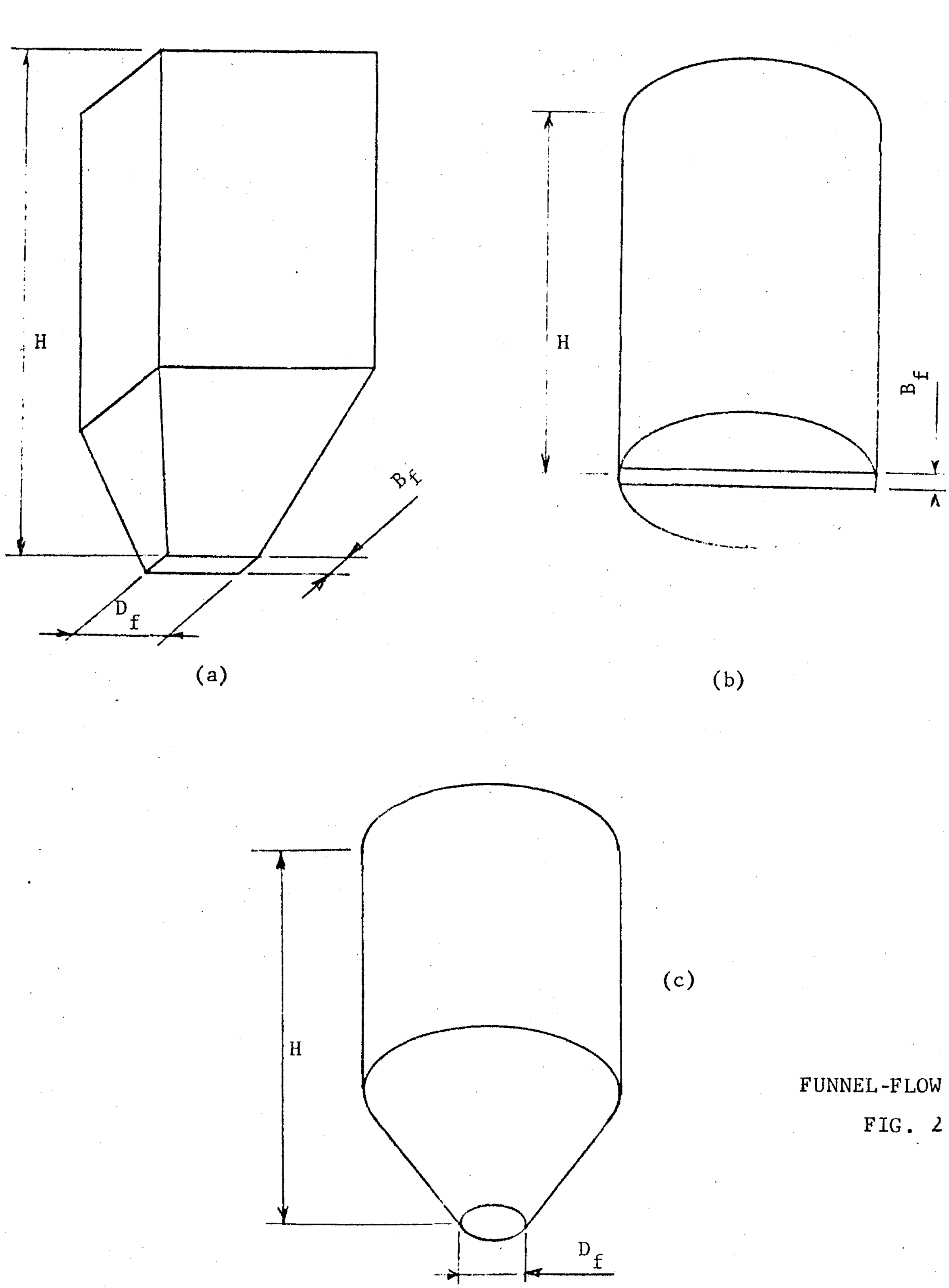
Glossary (cont.)

S	-	shearing force applied to a shear cell, lb
V_1	-	major consolidating force, lb; equal to 1/13 times the major consolidating pressure, psf
V	-	normal force applied to a shear cell, lb
δ	-	effective angle of internal friction of a solid during flow, degree
θ_c	-	maximum recommended angle (from vertical) of conical hoppers and end walls of transition hoppers for mass flow, degree
θ_p	-	maximum recommended angle (from vertical) of side walls of transition hoppers for mass flow, degree
ϕ'	-	kinematic angle of friction between a solid and a wall, degree
ϕ_t	-	angle of internal friction of a solid for incipient flow, degree



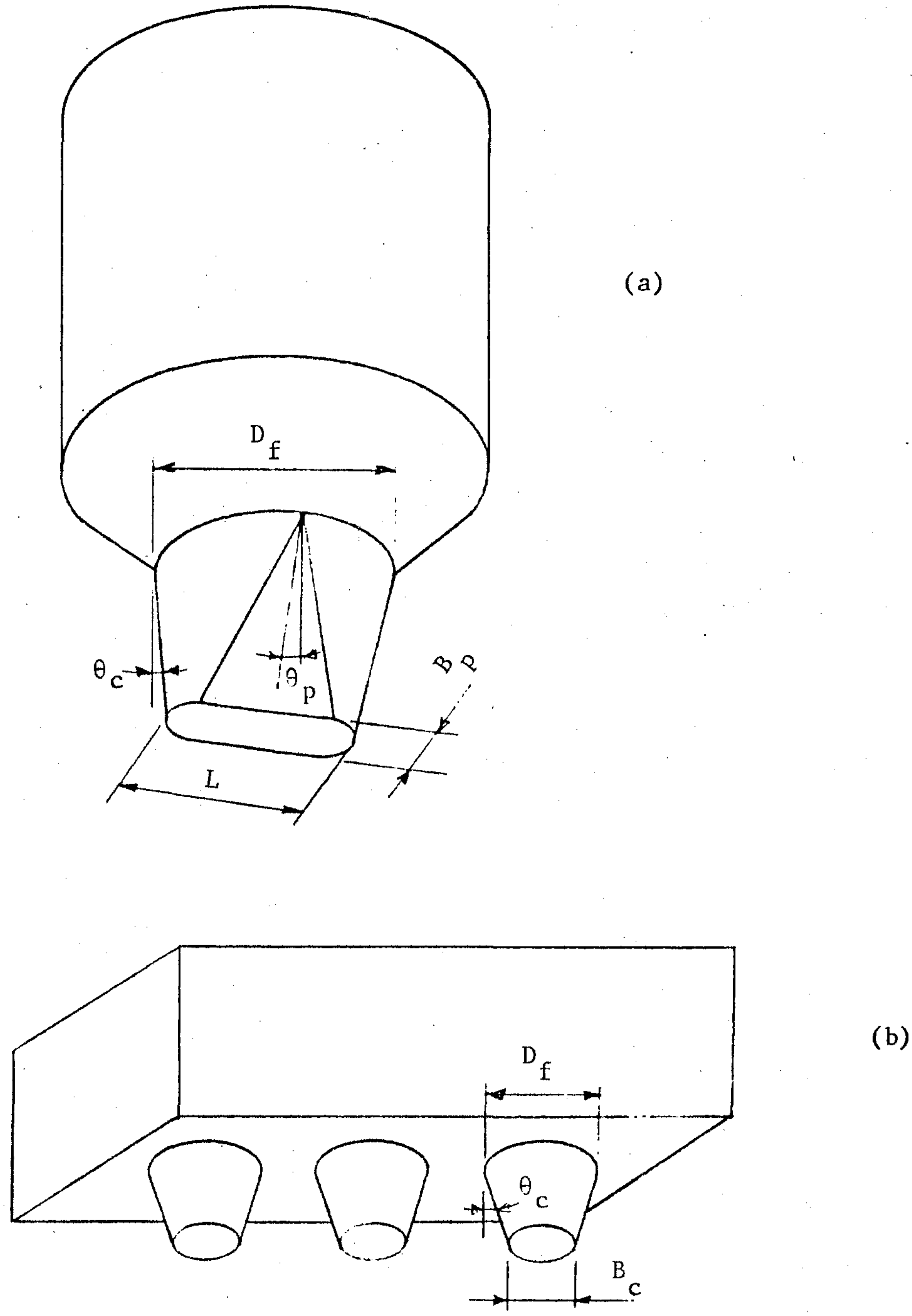
MASS-FLOW BINS

FIG. 1



FUNNEL-FLOW BINS

FIG. 2



EXPANDED-FLOW BINS

FIG. 3

RESULTS OF FLOW TESTS

Material(s) tested: Retorted Oil Shale

For each material, results are presented in the following order:

Section I	Minimum outlet dimensions for dependable flow
	A. Gravity flow
	B. With vibrating equipment
Section II	Bulk density as a function of head of solid
Section III	Maximum allowable hopper slope angles
Section IV	Maximum solids flow rate (if tested)

Material Retorted Oil Shale

Section I. Minimum outlet dimensions for dependable flow

A. Gravity flow

Storage time at rest	Water content, % wet weight	Temperature, °F	Mass-flow: Circular outlet diameter B_c, ft	Mass-flow: Oval outlet width B_p, ft	Funnel-flow: Rectangular outlet width B_f, ft	Funnel-flow: Critical rathole diameter h_e	D_f	Ref. Fig.
Continuous flow	0.4	100	1.3	0.6	0.6	5′ 10 20 40	5 11 24 49	4 - 6
4 Days		100 to Amb.	1.3	0.6	0.6	5′ 10 20 40	6 12 25 50	

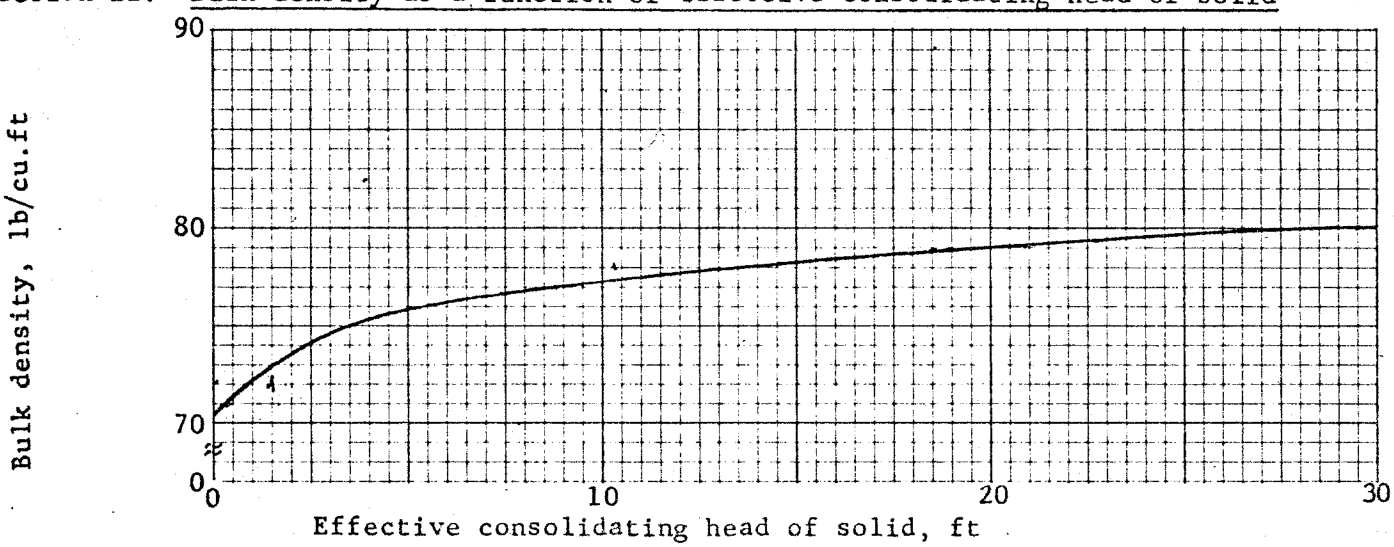
Material Retorted Oil Shale

B. With vibrating equipment

☐ Material not suited to vibration

Water content, % wet weight	Temperature, °F	Mass-flow: Circular outlet diameter B_c, ft	Mass-flow: Oval outlet width B_p, ft	Funnel-flow: Rectangular outlet width B_f, ft	Funnel-flow: Critical rathole diameter h_e	Funnel-flow: Critical rathole diameter D_f	Ref. Fig.
0.4	100	1.3	0.6	0.7	5	6	4 - 6
					10	12	
					20	25	
					40	50	

Section II. Bulk density as a function of effective consolidating head of solid



Material Retorted Oil Shale

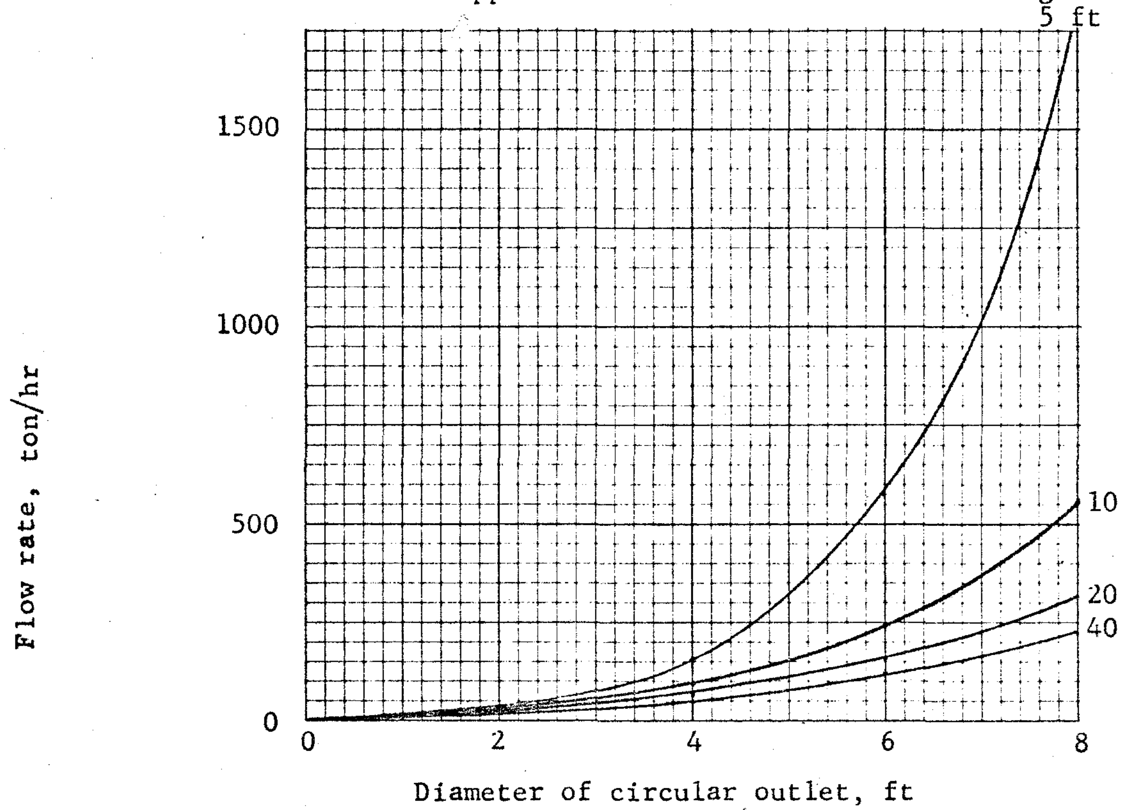
Section III Friction test results for various wall materials and maximum recommended hopper angles (measured from the vertical) for mass flow

Effective consolidating head, ft		0.25	0.5	1.0	2.0	4.0	5.0
Width B_p of oval outlet, ft		0.25	0.5	1.0	2.0	4.0	5.0
Dia. B_c of Circular outlet, ft		0.5	1.0	2.0	4.0	8.0	10.0
Wall material	Angle deg.						
Carbon steel	ϕ'	31	26	24	23	23	22
	θ_c	8	15	18	19	19	20
	θ_p	18	25	28	29	30	30
Carbon steel 4 day time test	ϕ'	31	26	24	23	23	22
	θ_c	8	15	18	19	19	20
	θ_p	18	25	28	29	30	30
304 - 2B finish stainless steel	ϕ'	18	18	18	18	18	18
	θ_c	25	25	25	25	25	25
	θ_p	36	36	36	36	36	36
304 - 2B finish stainless steel 4 day time test	ϕ'	18	18	18	18	18	18
	θ_c	25	25	25	25	25	25
	θ_p	36	36	36	36	36	36
	ϕ'						
	θ_c						
	θ_p						

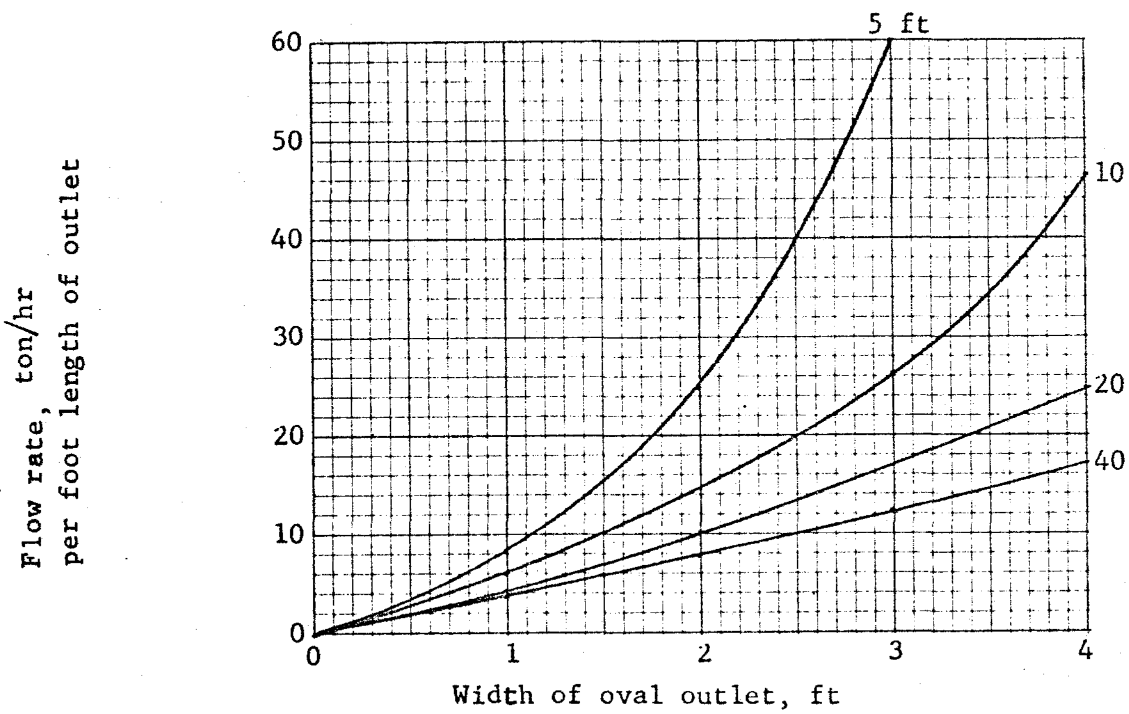
Material Retorted Oil Shale

Section IV. Maximum solids flow rate onto a belt feeder without air permeation

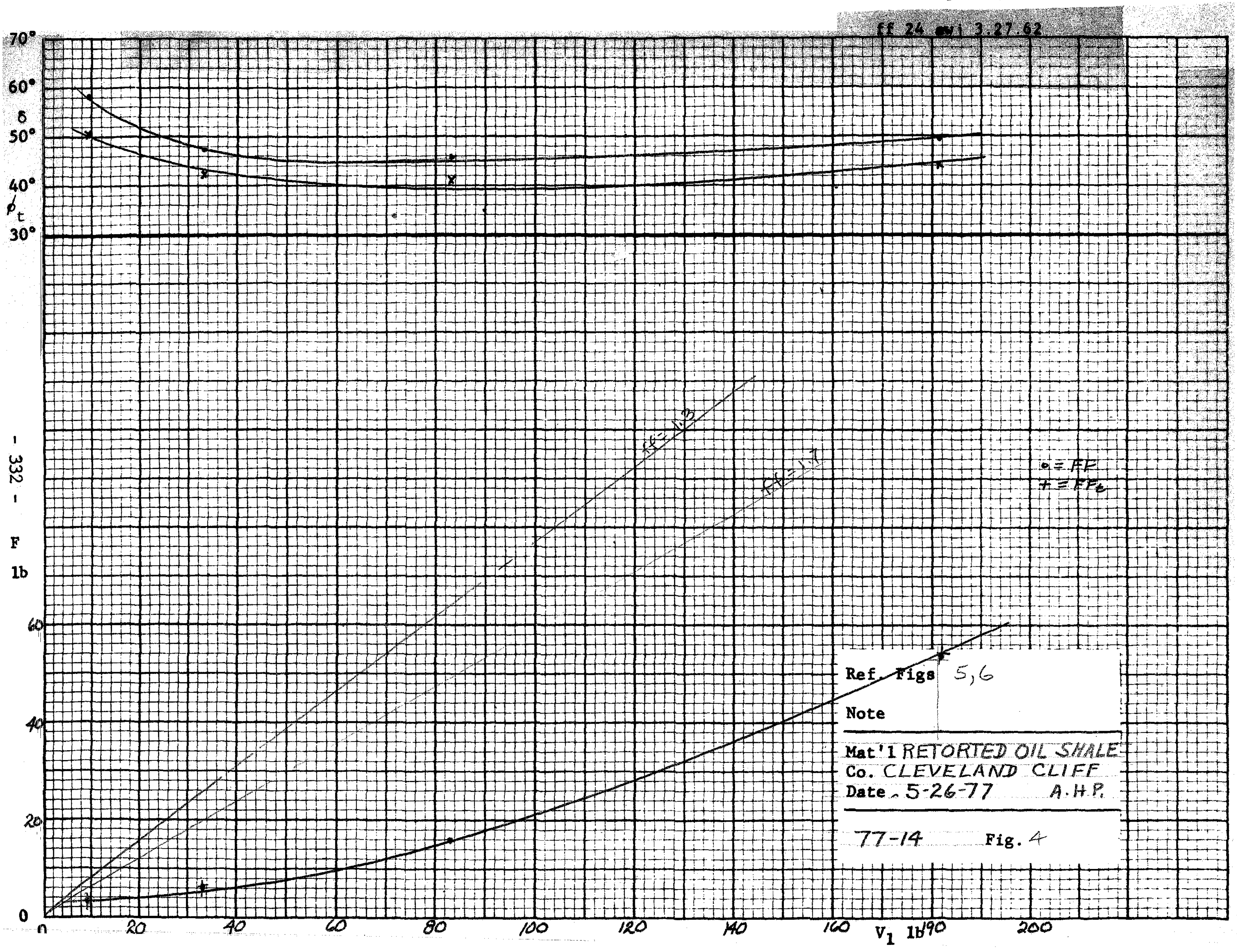
A. From a conical mass-flow hopper with an effective consolidating head h_e =



B. From a transition mass-flow hopper with an effective consolidating head h_e

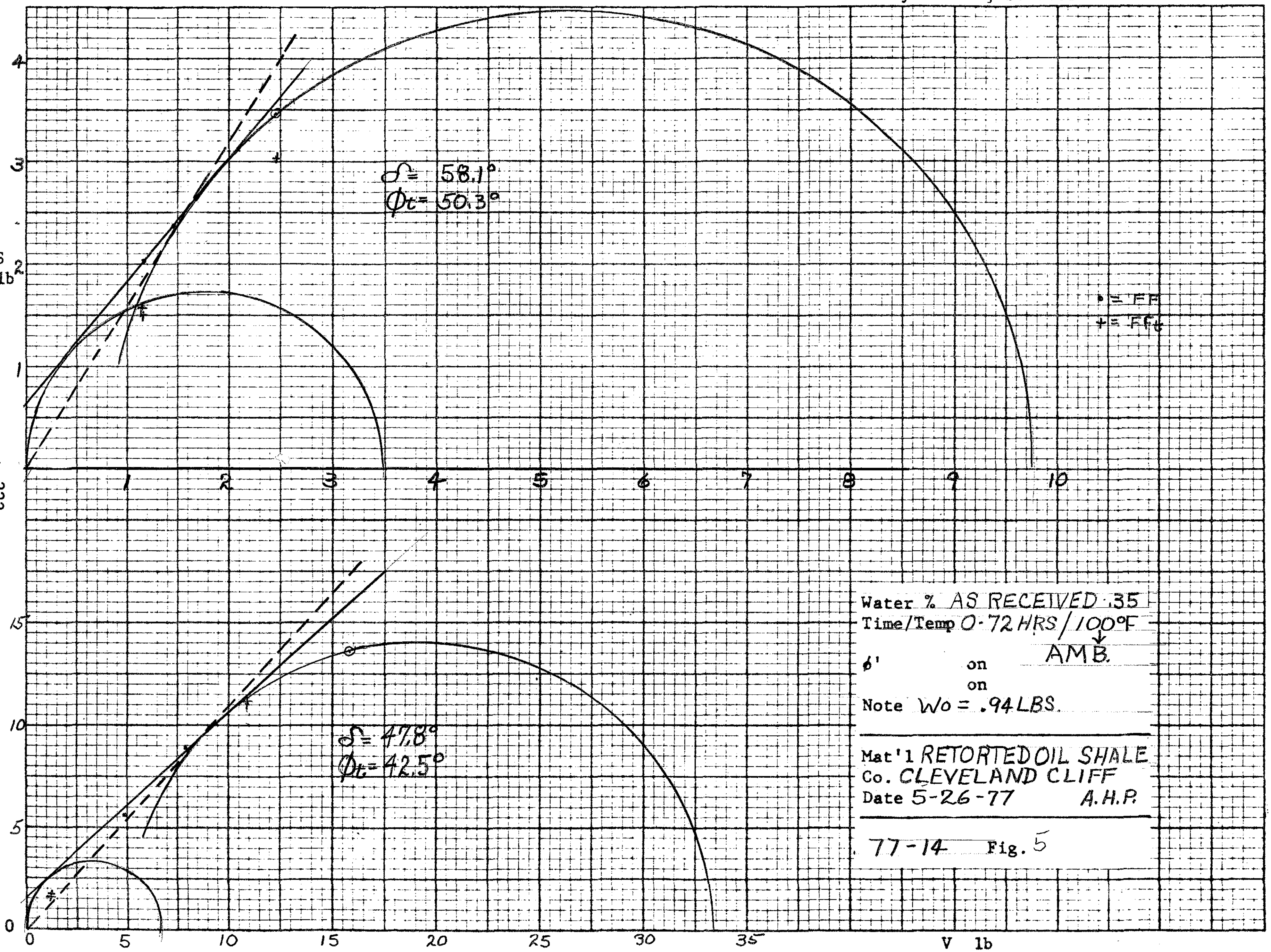


RECORD OF TESTS


ff 24 gwi 3.27.62
70°
60°
δ
50°
40°
ϕ_t
30°
F
lb
60
40
20
0
0
20
40
60
80
100
120
140
160
V_1 lb
190
200
ff = 1.3
ff = 1.7
• = FF
+ = FFb
Ref. Figs 5,6
Note
Mat'l RETORTED OIL SHALE
Co. CLEVELAND CLIFF
Date 5-26-77 A.H.P.
77-14 Fig. 4

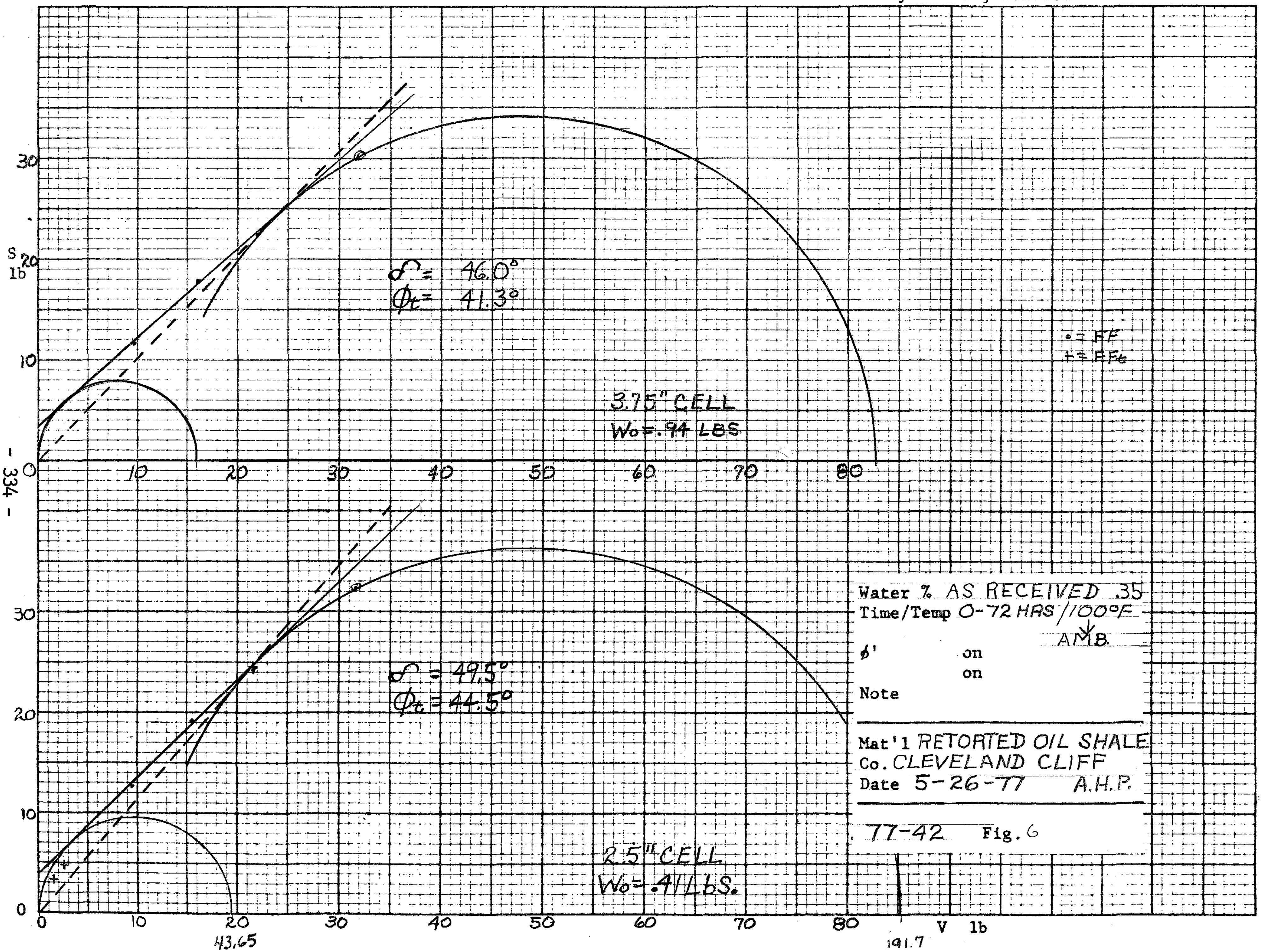
12 X 12 TO THE INCH

y1 48 awj 3.27.62



12 X 12 TO THE INCH

y1 48 awj 3.27.62



APPENDIX F

COOLING TESTS REPORT
STANSTEEL CORPORATION

ALLIS-CHALMERS

STANSTEEL CORPORATION

5001 SOUTH BOYLE AVENUE • LOS ANGELES, CALIFORNIA 90058 / 213-585-1231

June 27, 1977

JUN 1977
RECEIVED
CCI
RIFLE, COLORADO

The Cleveland Cliffs Iron Company
P. O. Box 1211
Rifle, CO 81650

Attention: Mr. R. A. Heisler

Reference: Your P. O. WD 77149
Our Order 1813

Dear Mr. Heisler:

Enclosed are three copies of our laboratory test report covering the testing of your samples of oil shale.

I believe you will find the enclosures to be self-explanatory, but please let me know if further data are required.

Very truly yours,

Robert R. Johnson, Manager
Contract Administration
Heat Transfer Equipment

RRJ:ae
Enclosures

APPENDIX F

RETORTED
OIL SHALE

INVESTIGATION OF COOLING PARAMETERS

THE CLEVELAND - CLIFFS IRON COMPANY
WESTERN DIVISION
RIFLE, COLORADO 81650

STANSTEEL JOB NO. 1813

STANSTEEL CORPORATION
SUBSIDIARY OF ALLIS-CHALMER CORPORATION
PROCESS ENGINEERING DEPARTMENT
5001 SOUTH BOYLE AVENUE
LOS ANGELES, CALIFORNIA 90058

DATE OF LABORATORY TEST
MAY 31 TO JUNE 3, 1977

TESTS PERFORMED BY J. RODRIGUEZ AND THE WRITER,
GREGORY L. BRIGGS
TESTS WITNESSED BY ROBERT A. HEISLER OF CLEVELAND-CLIFFS IRON COMPANY

ABSTRACT

Laboratory tests conducted in Stansteel's pilot rotary cooler showed that the retorted oil shale supplied to us could be cooled from 400°F to 100°F, given a retention time of about 8 minutes. The required air velocity at cooler exit is about 500 ft/min. Acquired information will be used to provide a budgetary proposal for plant scale units to cool 3000 tons per hour.

TABLE OF CONTENTS

\- - - - - - - -

INTRODUCTION

Cleveland-Cliffs is involved in the extraction of oil from Colorado oil shale. In the process of extraction, raw oil shale passes through a mill prior to the retort. The retort is a vertical column in which the shale flows downward, countercurrent to the upward flowing hot gases. In this operation the oil shale is heated to 1200-1500°F as it passes downward through rotating horizontal grates. The oil and gas extracted are then sent to a condenser.

The retorted oil shale leaves the retort at 400°F and needs to be cooled to 100°F prior to being dumped back into the mine through an 8' Ø x 2000' deep borehole. The retorted oil shale is 100% minus 4 inches particle size, reportedly containing 15% minus 325 mesh fines due to the grinding action of the rotating grates.

It is important to minimize degradation of the material as it passes through the cooler because coarser material will flow more easily down the borehole; fine material has greater tendency to bridge and adversely affect the operation of the borehole.

The use of water to facilitate cooling has been investigated by others but this is said to require too much water in an area that is short of water. Furthermore, if the product contains water, it cannot be placed down the borehole because of flow problems.

PLAN OF TEST

The variables which seemed most important were:

1. The maximum practical air velocity inside dryer without excessive dust carry-over to the flue handling system.

2. The minimum retention time needed for cooling the material.

3. Attrition of the oil shale in a rotary unit.

SUMMARY & CONCLUSIONS

1. Seven tests were run in our 15" Ø pilot plant rotary cooler. All tests used countercurrent flow.

2. Feed is dark gray, contains rocks as large as 4" and fines less than 10 microns (see graph 1). Most of these rocks are flat and seem to be formed of numerous horizontal plates; some are porous, while others are very dense and hard (like slate). Bulk density of feed is 55-65 #/ft^3.

3. To be able to run the cooler, the oil shale had to be heated in our laboratory. During initial testing of material it was found that severe attrition occurred when the shale was heated using 650°F inlet gas temperature in our 21" Ø heater. To avoid this attrition we had to use our 8" Ø indirect fired heater, see graph 2 for particle size.

4. In test 1 the feed rate was 1 #/min and the gas velocity was 655 ft/min. Dust carryover to cyclone was 59% of product. Fillage was low.

SUMMARY & CONCLUSIONS (continued)

5. In test 2 the feed rate was increased to 3#/min to increase fillage. The gas velocity was decreased to 359 ft/min to decrease dust carryover to cyclone. Fillage was adequate. Dust carryover was 26% of product.

6. At the end of test 2, the feed hood was inspected. Over 90# of material had collected there from the two tests, or a total feed of 228#. The bulk of the material was fine particles which had settled in the enlarged feed hood. There was also large pieces of feed material which had backspilled over the feed end plate. This indicated that the pitch of the cooler was not high enough for this feed rate.

7. In test 3, the cooler pitch was increased from 1/16 to 1/4 in/ft, which eliminated the backspill problem at the feed hood. Attendant to this was a decrease in the retention time to 7 minutes and hence lower fillage, 5.3%. Gas velocity was 331 ft/min. and dust carryover was 7.2%.

8. In test 4, gas velocity was decreased to 246 ft/min to observe the effect on dust carryover. Retention time increased to 9 minutes. Dusty carryover remained at 7.2%, while fillage was 6.1%.

9. Normally, a decrease in gas velocity will result in a decrease in dust carryover, if the particle size distribution is normal and does not change. Since this material is easily comminuted, the retention time could be a factor affecting dust carryover. As retention time increases, milling between particles increases and hence the amount of dust and carryover also increase. (see graphs 5 and 6).

10. In test 5, the feed rate was increased to 5#/min to increase fillage. This test was aborted when it became clear that the feed chute to the cooler was too small to accept this load.

11. In test 6, the feed rate was decreased to 4#/min which eliminated the plugging of the cooler feed chute. Gas velocity was 347 ft/min. Dust carryover was 6.8% of product. Fillage was adequate - 9.1%.

12. In test 7, we duplicated the 4#/min. feed rate but the gas velocity was increased to 535 ft/min to observe the effect on dust carryover. Dust carryover increased to 12.4% of product. Fillage was adequate - 9.7%.

13. For our heat balance calculations and at the request of Cleveland-Cliffs' Mr. Robert A. Heisler, we had a sample of the feed material analyzed to find its heat capacity between 400°F and 100°F. Heat capacity ranged from .269 to .290 BTU/#°F. (See Exhibit A).

EXPERIMENTAL DETAILS

All tests in our laboratory were performed in our 15" Ø unit. The data is summarized in table 1.

All tests were run with countercurrent flow.

The cooler was fitted with a cyclone, scrubber and an I. D. fan. (See Figure 1).

EXPERIMENTAL DETAILS (continued)

Material was heated in our 8" Ø indirect fired rotary heater.

Material was hand fed to the heater to prevent degradation of size in a screw conveyor.

Material was gravity fed to the cooler from the discharge hood of the heater.

The temperature of heated material fed to the cooler was measured by a thermocouple immersed in the bed at the discharge end of the heater.

Screen analysis product was done on Tyler Standard Screens using a mechanical shaker, timed for 2 minutes.

Sample of the product and cyclone dust from each run were given to Mr. Robert A. Heisler of Cleveland-Cliffs Iron Company.

RECOMMENDATIONS

The following parameters should be used in the design of rotary airswept units to cool retorted oil shale from 400°F to 100°F.

Retention time:	about 8 minutes
Air velocity at cooler exit:	about 500 ft/min.
Heat capacity:	.28 BTU/# °F

An enlarged feed hood should be used as a settling chamber to collect most of the fines.

Test No.		1	2	3	4	5	6	7	
Feed Rate	#/min	1	3	3	3	5	4	4	
Slope	in/ft	1/16	1/16	1/4	1/4	1/4	1/4	1/4	
RPM		10	10	10	10	10	10	10	
Air Temp. in/out	°F	77/96	75/113	77/114	86/116	70/85	71/115	77/101	
Mat'l Temp in/out	°F	520/77	550/75	414/82	441/87	270/70	450/71	439/78	
Retention Time	min.	25	18	7	9		8	8	
Air Velocity at Cooler Exit	ft/min	655	359	331	246	345	347	535	
Dust Carryover % of Product	%	59	26	7	7		7	12	
Fillage Volume %	%	4.2	9.3	5.3	6.1		9.1	9.7	

TABLE I
SUMMARY OF LAB DATA

STANSTEEL JOB 1813
6/14/77, GLB

What % E dust collected came from heater ?

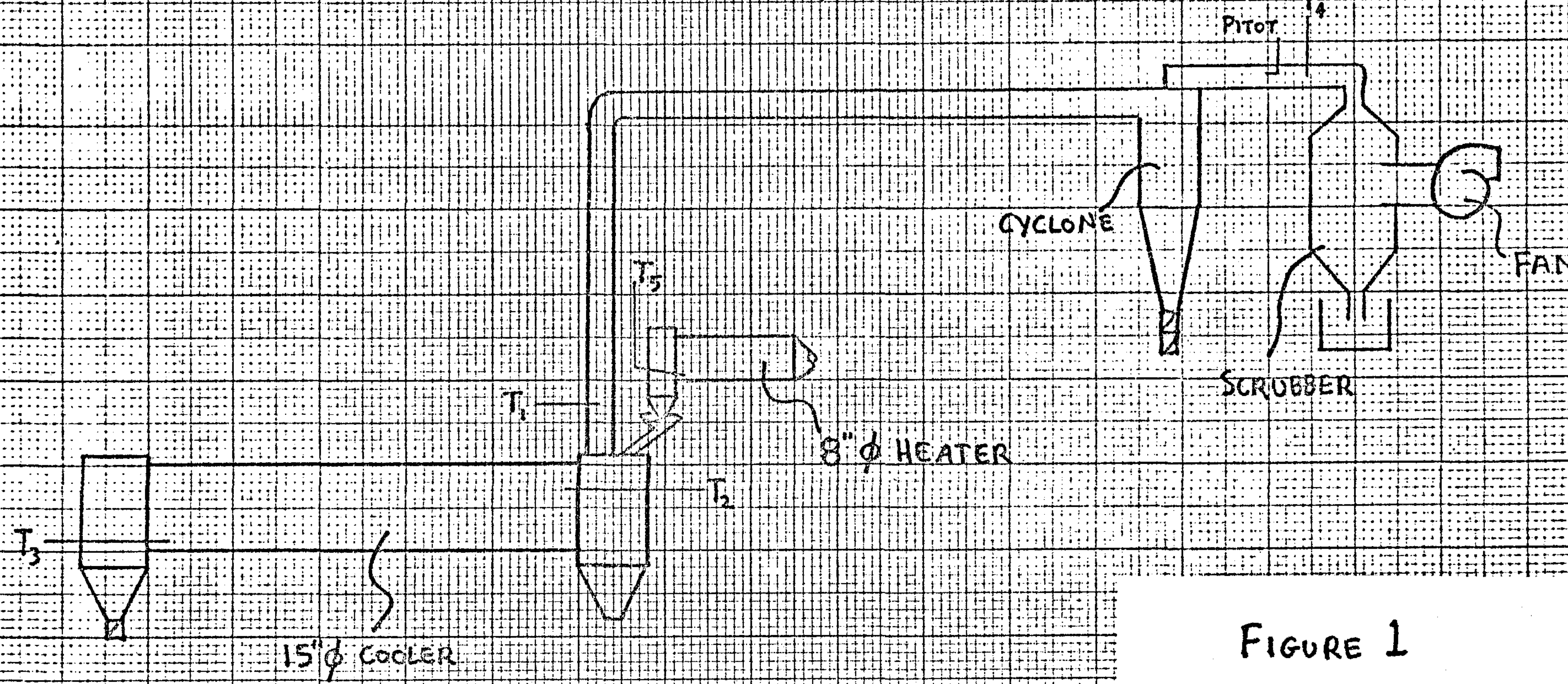
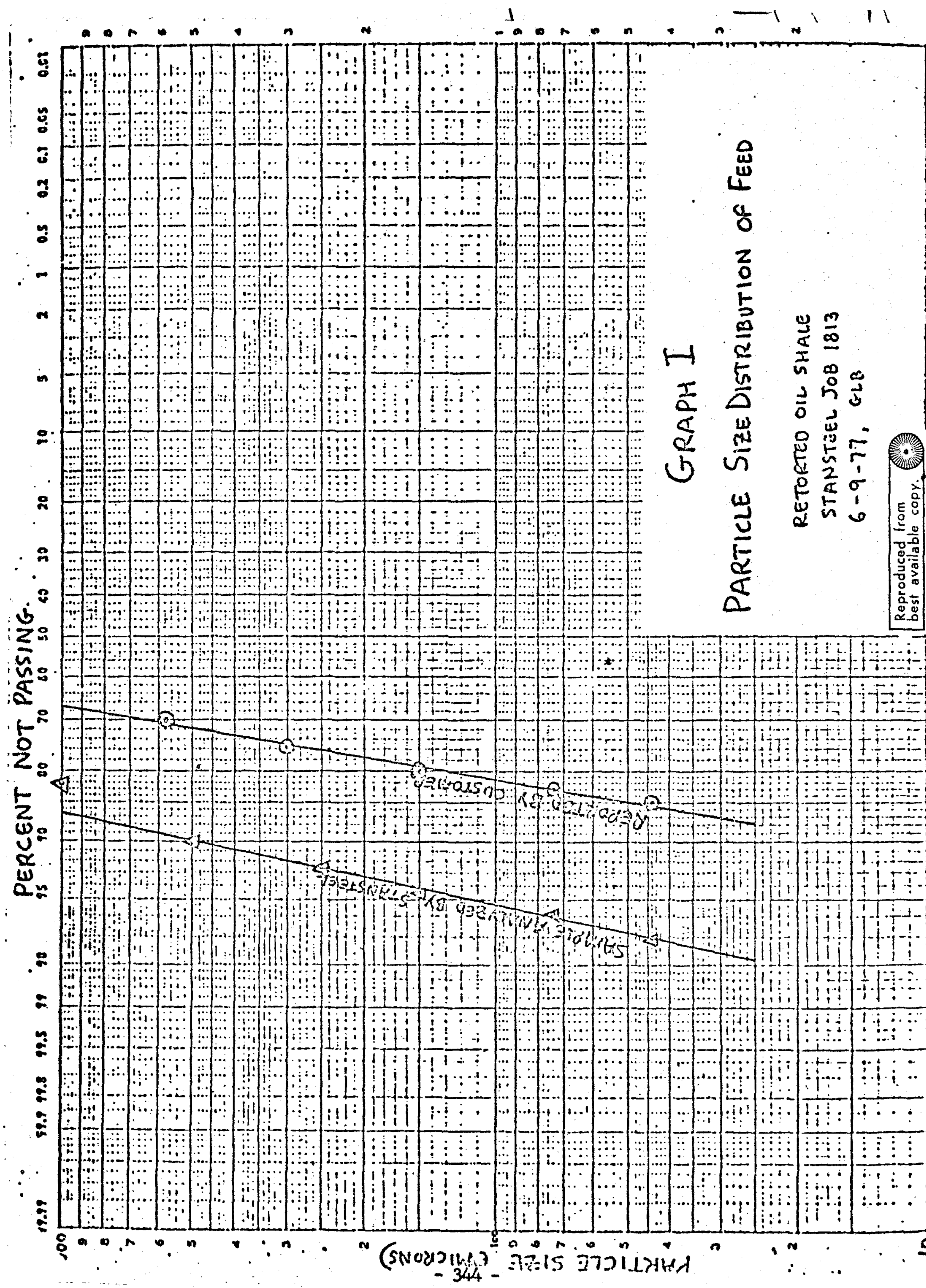


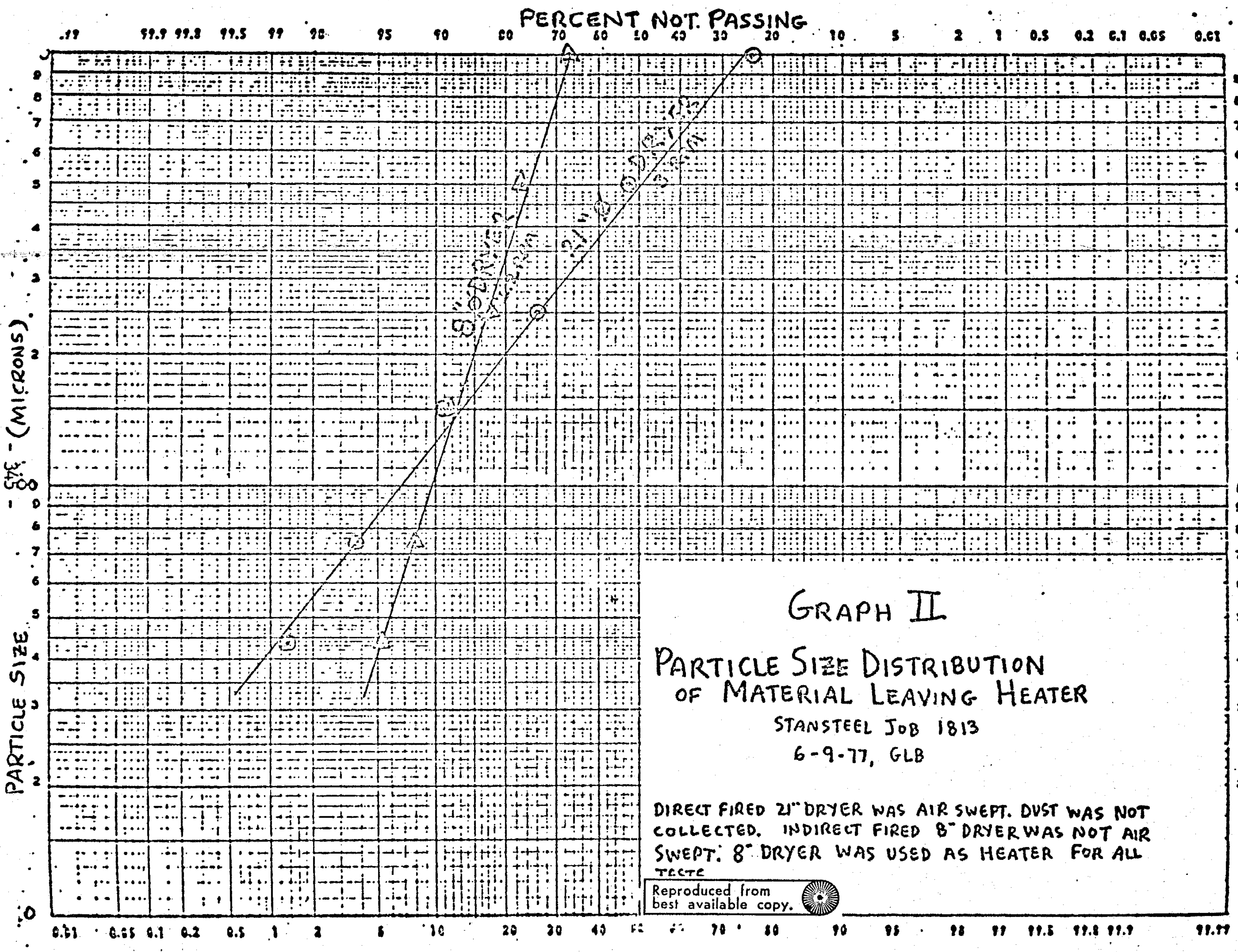
FIGURE 1

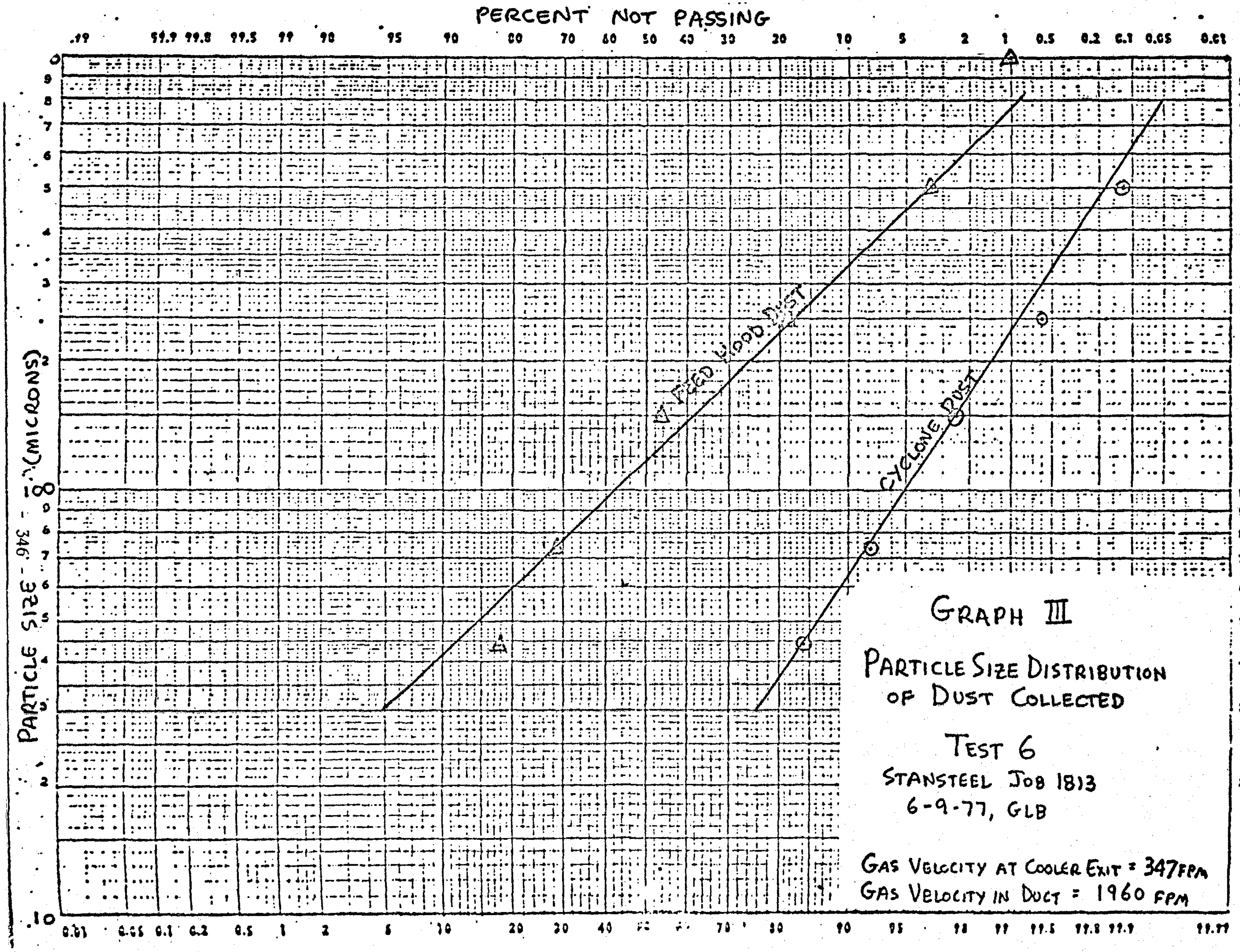
SKETCH OF EQUIPMENT

SHOWING THERMOCOUPLE PLACEMENT

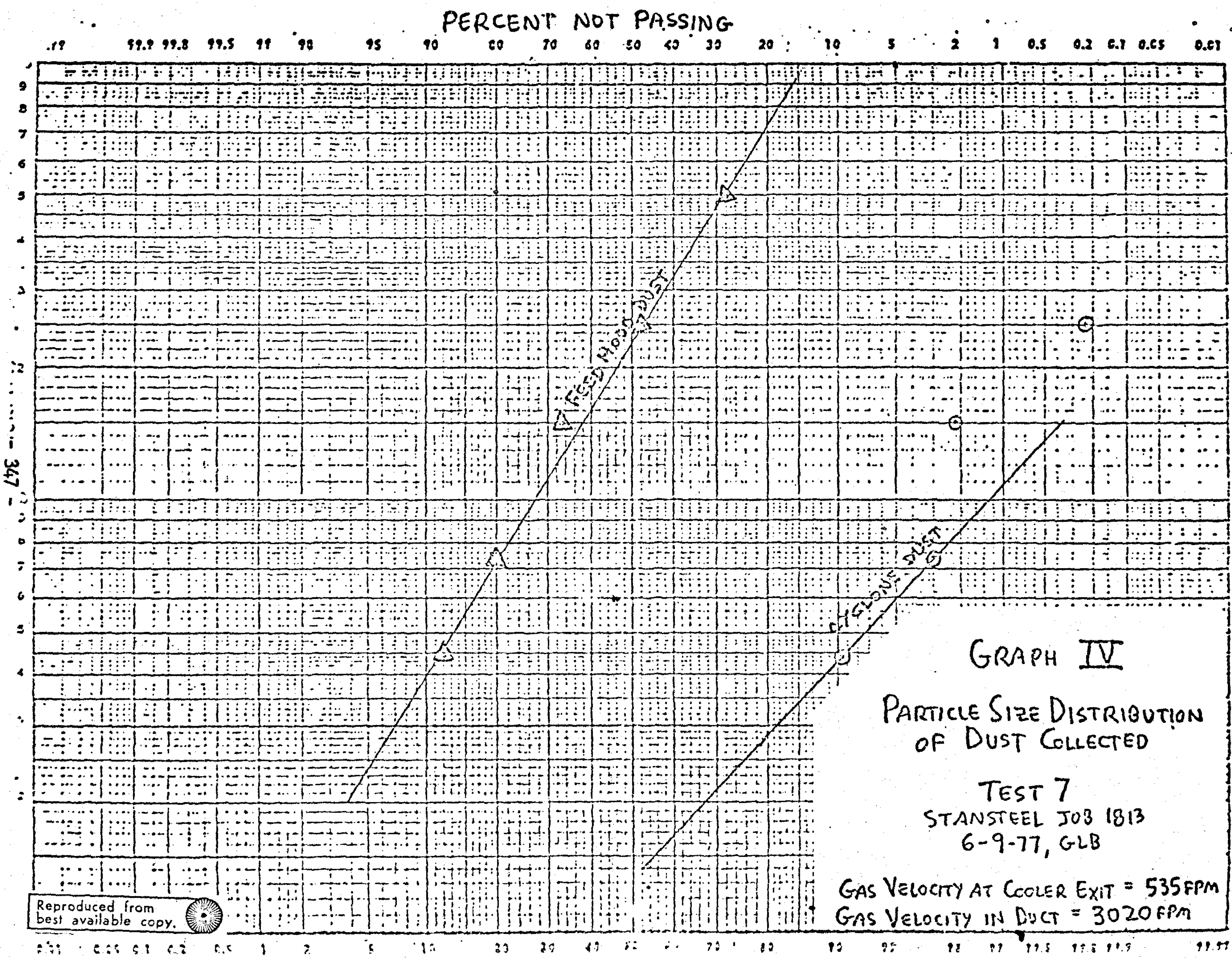
STANSTEEL JOB 1813
6/14/77, GLB

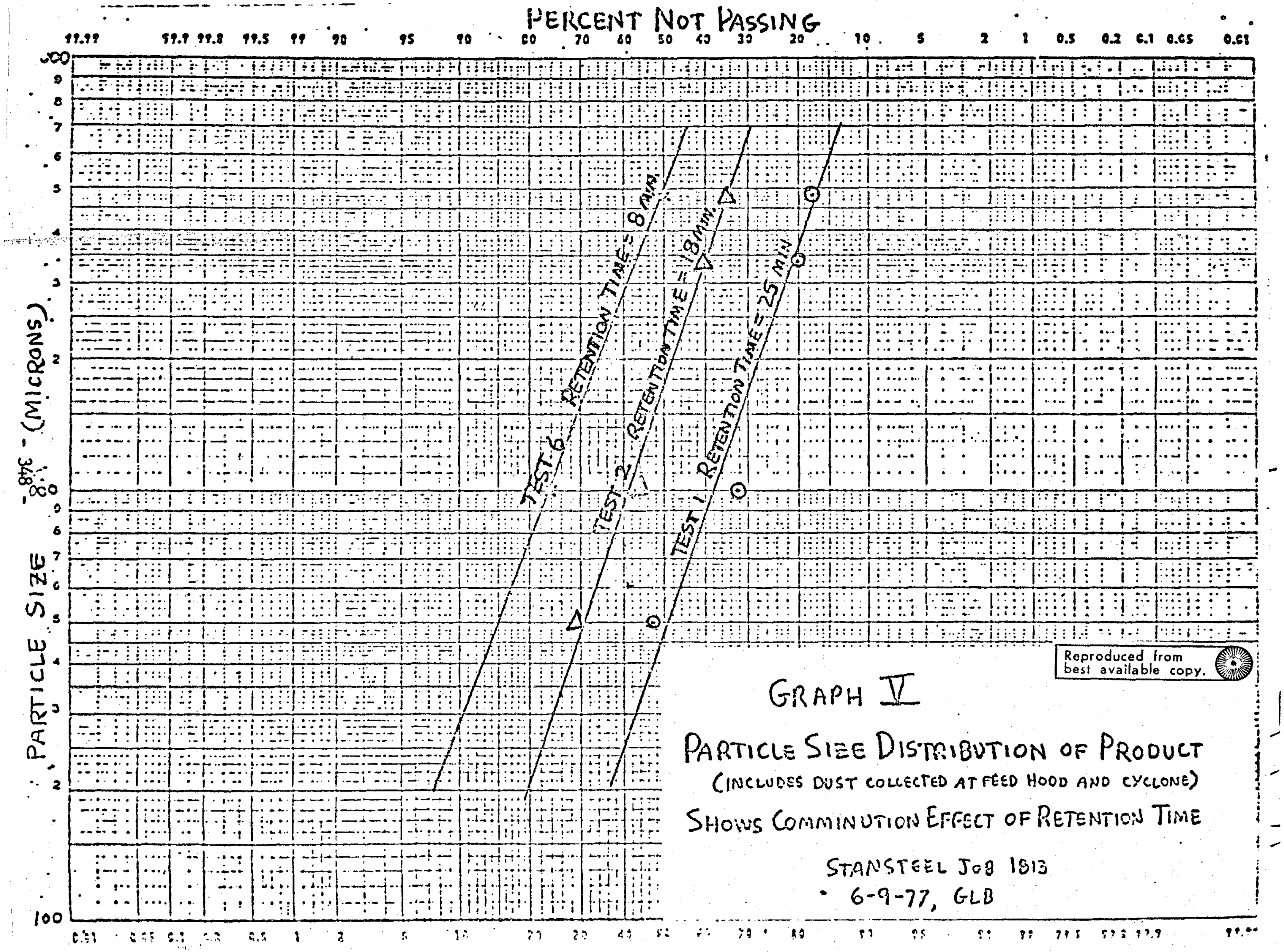

GRAPH I
PARTICLE SIZE DISTRIBUTION OF FEED
RETORTED OIL SHALE
STANSTEEL JOB 1813
6-9-77, GLB
PERCENT NOT PASSING
PARTICLE SIZE (MICRONS)
REPORTED BY CUSTOMER
SAMPLE ANALYSED BY STANSTEEL


PERCENT NOT PASSING
PARTICLE SIZE - (MICRONS)
8" DRYER
21" DRYER
GRAPH II
PARTICLE SIZE DISTRIBUTION OF MATERIAL LEAVING HEATER
STANSTEEL JOB 1813
6-9-77, GLB
DIRECT FIRED 21" DRYER WAS AIR SWEPT. DUST WAS NOT COLLECTED. INDIRECT FIRED 8" DRYER WAS NOT AIR SWEPT. 8" DRYER WAS USED AS HEATER FOR ALL TESTS



PERCENT NOT PASSING
PARTICLE SIZE - (MICRONS)
FEED WOOD DUST
CYCLONE DUST
GRAPH III
PARTICLE SIZE DISTRIBUTION
OF DUST COLLECTED
TEST 6
STANSTEEL JOB 1813
6-9-77, GLB
GAS VELOCITY AT COOLER EXIT = 347 FPM
GAS VELOCITY IN DUCT = 1960 FPM


PERCENT NOT PASSING
FEED HOOD DUST
CYCLONE DUST
GRAPH IV
PARTICLE SIZE DISTRIBUTION
OF DUST COLLECTED
TEST 7
STANSTEEL JOB 1813
6-9-77, GLB
GAS VELOCITY AT COOLER EXIT = 535 FPM
GAS VELOCITY IN DUCT = 3020 FPM



GRAPH V
PARTICLE SIZE DISTRIBUTION OF PRODUCT
(INCLUDES DUST COLLECTED AT FEED HOOD AND CYCLONE)
SHOWS COMMINUTION EFFECT OF RETENTION TIME

STANSTEEL JOB 1813
6-9-77, GLB

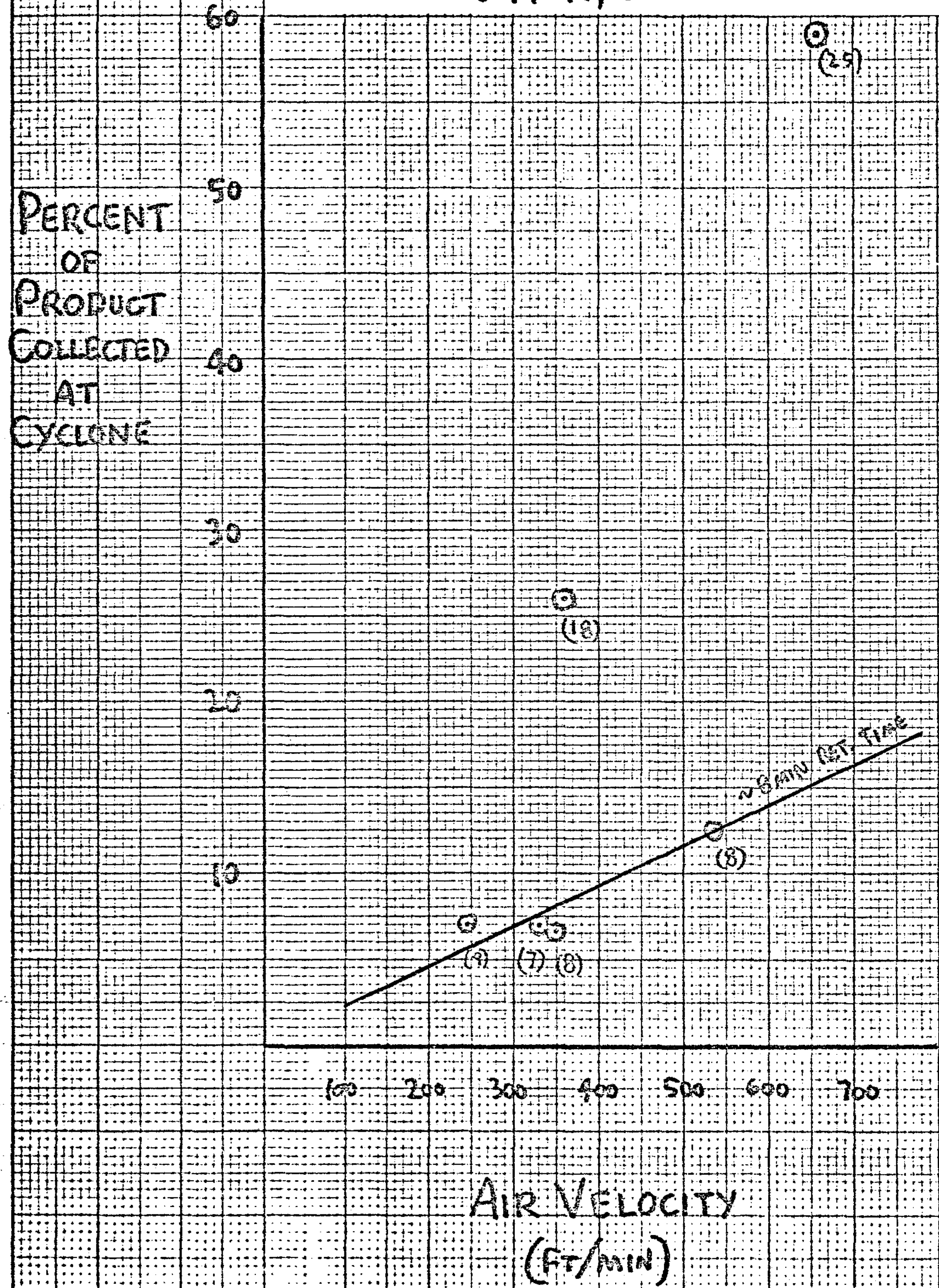
GRAPH VI

EFFECT OF AIR VELOCITY ON DUST CARRYOVER

(RETENTION TIME IN PARENTHESES)

STANSTEEL JOB 1813

6-14-77, GLB



STANSTEEL CORPORATION

ALLIS-CHALMERS
5001 SOUTH BOYLE AVENUE • LOS ANGELES, CALIFORNIA 90058 / 213-585-1231

PROCESS LABORATORY DATA SHEET

JOB NO. 1813 CUSTOMER CLEVELAND CLIFFS IDENTIFICATION
TEST NO. 1 DATE 6-1-77 MATERIAL OIL SHALE (RETORTED)
INVESTIGATOR GLD JR OBSERVER DH

UNIT CHARACTERISTICS:

PILOT COOLER SIZE: 15"ϕ×8' SHELL SPEED, RPM 10
FLOW COUNTER CURRENT PITCH, In./Ft. 1/16
DISCHARGE END DAM HEIGHT, In. NONE OTHER DRYER SLOPE = 1/16 RPM = 2.2

Time	START FEED 145	205	220	START TEST 225	235	250	STOP FEED 255	
Feed Rate, #/min.	1	1	1	2/3	2/3	2/3	2/3	
Moisture, wt % (in/out)								
Material Temp, °F (in/out)		/85	602/82	666/83	662/78	467/77	452/77	
Air Temp, °F (in/out)		83/95	82/89	82/90	73/87	77/100	71/102	
Pitot Temperature, °F		99	91	90	89	103	97	
Pitot Tube, In. W.C.	.80	.80	.80	.8	.80	.81		
ACFM at Pitot		800	794	793	793	808		
ACFM leaving Cooler		798	791	793	782	803		
LFM leaving Cooler		651	645	646	638	655		
Cyclone Δ P, In. W.C.	4.3	4.3	4.4	4.3	4.3	4.4		
Scrubber ΔP, In. W.C.								
Quantity Fed, lbs.	60							
Cooler Product, lbs.	8.1							
Cyclone Product, lbs.	4.8							
Cooler Retention, lbs.	22.2							
Cyclone Retention, lbs.	2.6							
Total Recovered, lbs.	37.7							

Bulk Density, $\#/ft^3$

Angle of Repose

20# fed from 225 to 255
Feed Chute to cooler plugged 235-240

AN ALLIS-CHALMERS SUBSIDIARY

STANSTEEL CORPORATION

ALLIS-CHALMERS

5001 SOUTH BOYLE AVENUE • LOS ANGELES, CALIFORNIA 90058 / 213-585-1231

PROCESS LABORATORY DATA SHEET

JOB NO. 1813   CUSTOMER CLEVELAND CLIFFS   IDENTIFICATION

TEST NO. 2   DATE 6-2-77   MATERIAL RETORTED OIL SHALE

INVESTIGATOR GLB, JR   OBSERVER BOB HEISLER

UNIT CHARACTERISTICS:

PILOT COOLER SIZE: 15"Ø x 8'   SHELL SPEED, RPM 10

FLOW COUNTERCURRENT   PITCH, In./Ft. 1/4

DISCHARGE END DAM HEIGHT, In. NONE   OTHER [illegible] RPM = 8

Time	START FEED 740	810	830	840				
Feed Rate, #/min.	3	3	3	3				
Moisture, wt % (in/out)								
Material Temp, °F (in/out)		602/75	562/75	487/75				
Air Temp, °F (in/out)	75/	75/103	75/109	75/128				
Pitot Temperature, °F		110	119	126				
Pitot Tube, In. W.C.		.23	.23	.23				
ACFM at Pitot		433	436	439				
ACFM leaving Cooler		428	428	[illegible]				
LFM leaving Cooler		349	349	359				
Cyclone ΔP, In. W.C.		1.4	1.5	1.4				
Scrubber ΔP, In. W.C.								
Quantity Fed, lbs.	168							
Cooler Product, lbs.	40.8							
Cyclone Product, lbs.	10.7							
Cooler Retention, lbs.	49.7							
Cyclone Retention, lbs.	4.1							
Total Recovered, lbs.	105.3							
MATL @ FEED HOOD FROM TESTS 1&2	≈90#							

Bulk Density, #/ft³ COOLER PRODUCT = 57.4

Angle of Repose

LIFTERS ARE FULL

AMBIENT AIR TEMP DB/WB = 75/69

2#/min FIRST 14 min, 3#/min

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PROCESS LABORATORY DATA SHEET

JOB NO. 1813 CUSTOMER CLEVELAND CLIFFS IDENTIFICATION

TEST NO. 3 DATE 6-2-77 MATERIAL RETORTED OIL SHALE

INVESTIGATOR GIB, JR OBSERVER BA

UNIT CHARACTERISTICS:

PILOT COOLER SIZE: 15"ø x 8' SHELL SPEED, RPM 10

FLOW COUNTER CURRENT PITCH, In./Ft. 1/4

DISCHARGE END DAM HEIGHT, In. NONE OTHER

Time	START FEED 1015	1025	1045	1055	1105	1115		
Feed Rate, #/min.	3	3	3	3	3	3		
Moisture, wt % (in/out)								
Material Temp, °F (in/out)		437/77	440/79	408/84	405/82	430/79		
Air Temp, °F (in/out)	77/	77/96	77/111	77/113	77/114	77/114		
Pitot Temperature, °F		96	109	111	112	113		
Pitot Tube, In. W.C.		.16	.21	.21	.20	.20		
ACFM at Pitot			413	414	404	405		
ACFM leaving Cooler			415	416	406	406		
LFM leaving Cooler			338	339	331	331		
Cyclone ΔP, In. W.C.		1.4	1.5	1.45	1.45			
Scrubber ΔP, In. W.C.								
Quantity Fed, lbs.	168							
Cooler Product, lbs.	106.7							
Cyclone Product, lbs.	7.7							
Cooler Retention, lbs.	29.8							
Cyclone Retention, lbs.	1.2							
Total Recovered, lbs.	164.9							
FEED HOOD DUST, LBS	19.5							

Bulk Density, #/ft³

Angle of Repose

DRYER @ 8 RPM 3/8" SLOPE

AMBIENT AIR TEMP DB/WB = 77/69

JR ZEROED PITOT @ 1040

FEED PLUGGED @ 1110

LARGE PRODUCT IS ≈110°F; PRODUCT IN PAIL IS 77°F @ 11:05

AN ALLIS-CHALMERS SUBSIDIARY

ALLIS-CHALMERS

STANSTEEL CORPORATION

5001 SOUTH BOYLE AVENUE • LOS ANGELES, CALIFORNIA 90058 / 213-585-1231

PROCESS LABORATORY DATA SHEET

JOB NO. 1813 CUSTOMER CLEVELAND CLIFFS IDENTIFICATION

TEST NO. 4 DATE 6-2-77 MATERIAL RETORTED OIL SHALE

INVESTIGATOR GLB, JR OBSERVER BH

UNIT CHARACTERISTICS:

PILOT COOLER SIZE: 15"ø x 8' SHELL SPEED, RPM 10

FLOW COUNTER CURRENT PITCH, In./Ft. 1/4

DISCHARGE END DAM HEIGHT, In. NONE OTHER DRYER SLOPE = 3/8 RPM = 6

Time	START FEED 1253	110	120	130	140	150		
Feed Rate, #/min.	3	3	3	3	3	3		
Moisture, wt % (in/out)								
Material Temp, °F (in/out)		342/87	421/86	412/87	470/87	442/87		
Air Temp, °F (in/out)		86/	86/120	86/113	86/119	86/117		
Pitot Temperature, °F			110	110	113	114		
Pitot Tube, In. W.C.		.11	.12		.11	.11		
ACFM at Pitot			313		300	300		
ACFM leaving Cooler			318		303.	302		
LFM leaving Cooler			259		247	246		
Cyclone Δ P, In. W.C.		1.2	1.2	1.2	1.2	1.2		
Scrubber ΔP, In. W.C.								
Quantity Fed, lbs.	168							
Cooler Product, lbs.	106.8							
Cyclone Product, lbs.	7.7							
Cooler Retention, lbs.	25.5							
Cyclone Retention, lbs.	.4							
Total Recovered, lbs.	169.4							
FEED HOOD, LBS.	18.8		DRYER RET. = 10.2#					

Bulk Density, #/ft³ COOLER PRODUCT = 62#/CF

Angle of Repose

AMBIENT AIR TEMP DB/WB 66/74

PRODUCT IN PAIL IS 82°F @ 1:50

A FEW LARGE ROCKS AS HIGH AS 140°F

AN ALLIS-CHALMERS SUBSIDIARY

ALLIS-CHALMERS

STANSTEEL CORPORATION

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PROCESS LABORATORY DATA SHEET

JOB NO. 1813 CUSTOMER Cleveland Cliffs IDENTIFICATION
TEST NO. 5 DATE 6-3-77 MATERIAL Retorted Oil Shale
INVESTIGATOR GLB, JR OBSERVER BH

UNIT CHARACTERISTICS:

PILOT COOLER SIZE: 15"ø x 8' SHELL SPEED, RPM 10
FLOW Countercurrent PITCH, In./Ft. 1/2
DISCHARGE END DAM HEIGHT, In. None OTHER Dryer slope: 1/2 RPM = 10

Time	745	800						
Feed Rate, #/min.	5	5						
Moisture, wt % (in/out)								
Material Temp, °F (in/out)	238/	503/70						
Air Temp, °F (in/out)	70/70	70/85						
Pitot Temperature, °F	70	85						
Pitot Tube, In. W.C.	.23	.23						
ACFM at Pitot	418	423						
ACFM leaving Cooler	418	423						
LFM leaving Cooler	340	345						
Cyclone Δ P, In. W.C.	1.4	1.5						
Scrubber ΔP, In. W.C.								
Quantity Fed, lbs.	75							
Cooler Product, lbs.	39.5							
Cyclone Product, lbs.	10.4							
Cooler Retention, lbs.	23.7	(Includes	Feed Hood)					
Cyclone Retention, lbs.								
Total Recovered, lbs.	73.6							

Bulk Density, $\#/ft^3$

Angle of Repose

Feed Chute Cleaned @ 755

Feed [illegible] at 800, Test Aborted

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STANSTEEL CORPORATION

ALLIS-CHALMERS

8001 SOUTH BOYLE AVENUE • LOS ANGELES, CALIFORNIA 90058 / 213-585-1231

PROCESS LABORATORY DATA SHEET

JOB NO. 1813 CUSTOMER CLEVELAND CLIFFS IDENTIFICATION ______
TEST NO. 6 DATE 6-3-71 MATERIAL RETORTED OIL SHALE
INVESTIGATOR GLB JR OBSERVER BH

UNIT CHARACTERISTICS:

PILOT COOLER SIZE: 15"Ø x 8' SHELL SPEED, RPM 10
FLOW COUNTERCURRENT PITCH, In./Ft. 1/2
DISCHARGE END DAM HEIGHT, In. NONE OTHER DRYER SLOPE = 3/8 RPM = 10

Time	855	910	920	930	940	950		
Feed Rate, #/min.	4	4	4	4	4	4		
Moisture, wt % (in/out)								
Material Temp, °F (in/out)	343/	350/71	455/71	390/71	500/72	460/71		
Air Temp, °F (in/out)	71/	71/102	71/118	71/115	71/113	71/117		
Pitot Temperature, °F		94	111	114	114	117		
Pitot Tube, In. W.C.	.22	.22	.22	.22	.22	.22		
ACFM at Pitot		417	424	425	425	426		
ACFM leaving Cooler		424	429	426	424	426		
LFM leaving Cooler		345	350	349	346	347		
Cyclone Δ P, In. W.C.	1.5	1.5	1.5	1.5	1.5	1.5		
Scrubber ΔP, In. W.C.								
Quantity Fed, lbs.	216							
Cooler Product, lbs.	120.7							
Cyclone Product, lbs.	8.2							
Cooler Retention, lbs.	31.8							
Cyclone Retention, lbs.	.8							
Total Recovered, lbs.	205.							
FEED HOOD, lbs	23.		DRYER RET = 20.8					
Bulk Density, #/ft³	COOLER PRODUCT = 60.0			CYCLONE = 50.6				
Angle of Repose								

AMBIENT AIR TEMP DB/WB = 71/65

PRODUCT IN PAIL IS 78°F @ 9:50
PRODUCT TEMP OF CHUNKS APPROACH 100°F

AN ALLIS-CHALMERS SUBSIDIARY

ALLIS-CHALMERS

STANSTEEL CORPORATION

5001 SOUTH BOYLE AVENUE • LOS ANGELES, CALIFORNIA 90058 / 213-583-1231

PROCESS LABORATORY DATA SHEET

JOB NO. 1813 | CUSTOMER Cleveland Cliffs | IDENTIFICATION
TEST NO. 7 | DATE 6-3-77 | MATERIAL Retorted oil shale
INVESTIGATOR GLD, JD | OBSERVER BH

UNIT CHARACTERISTICS:

PILOT COOLER SIZE: 15"Ø x 8' | SHELL SPEED, RPM 10
FLOW Countercurrent | PITCH, In./Ft. 1/4
DISCHARGE END DAM HEIGHT, In. NONE | OTHER Dryer: Slope = 3/8 RPM = 10

Time	1030	1040	1050	1100	1110			
Feed Rate, #/min.	4	4	4	4	4			
Moisture, wt % (in/out)								
Material Temp, °F (in/out)		385/77	354/79	487/77	476/80			
Air Temp, °F (in/out)		77/96	77/90	77/109	77/103			
Pitot Temperature, °F		98	95	110	110			
Pitot Tube, In. W.C.		.53	.52	.53				
ACFM at Pitot		650	642	657				
ACFM leaving Cooler		648	637	656				
LFM leaving Cooler		528	519	535				
Cyclone ΔP, In. W.C.		3.3	3.3	3.3				
Scrubber ΔP, In. W.C.								
Quantity Fed, lbs.	168							
Cooler Product, lbs.	66.3							
Cyclone Product, lbs.	8.2							
Cooler Retention, lbs.	30.8							
Cyclone Retention, lbs.	1.0							
Total Recovered, lbs.	159.2							
Feed Hopp. lbs.	27.4	Dryer feed	= 25.5					

Bulk Density, #/ft³ Cooler Product = 59.0 Cyclone = 51.9#/cf

Angle of Repose

Ambient Air Temp 28/88 71/89

Product in pail is 74°F ~ 110

Larger rocks up to 105°F

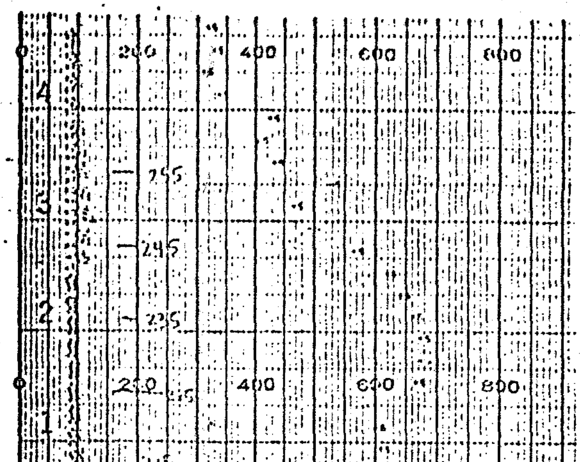
pH Scrubber = 10

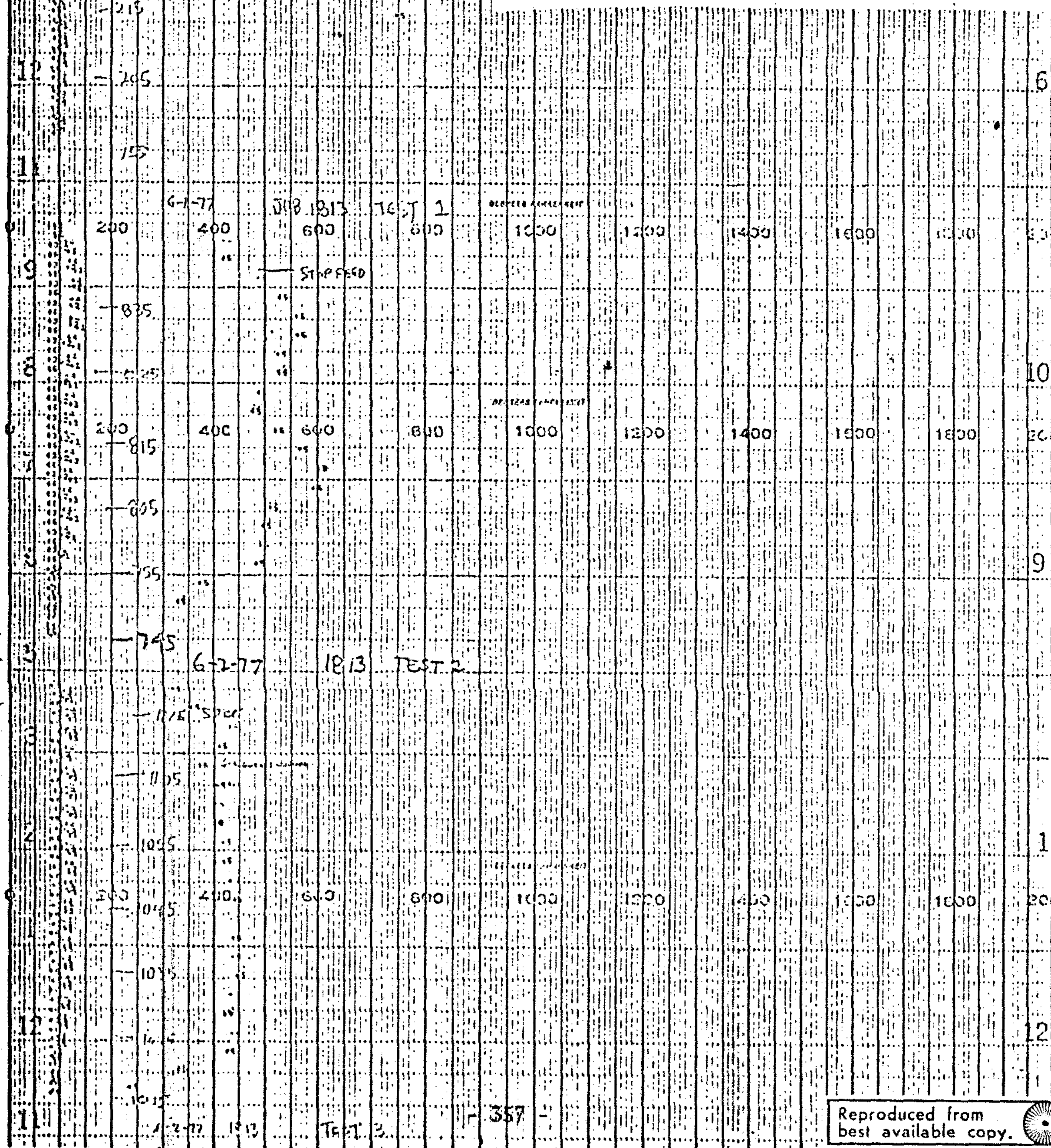
AN ALLIS-CHALMERS SUBSIDIARY

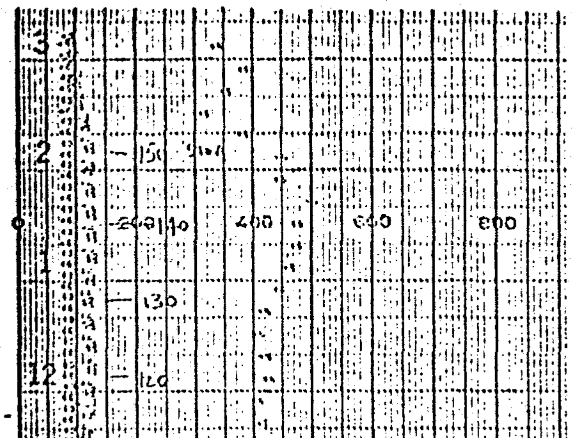
THERMOCOUPLE LOCATIONS

1. AIR IN DUCT ABOVE FEED HOOD
2. AIR IN COOLER AT EXIT
3. MATERIAL AT DISCHARGE OF COOLER
4. AIR IN DUCT AT PITOT
5. MATERIAL AT DISCHARGE OF HEATER
6. AMBIENT AIR

STANSTEEL CORPORATION
JOB 1813
6/14/77, GLB







THERMOCOUPLE LOCATIONS

1. AIR IN DUCT ABOVE FEED HOOD
2. AIR IN COOLER AT EXIT
3. MATERIAL AT DISCHARGE OF COOLER
4. AIR IN DUCT AT PITOT
5. MATERIAL AT DISCHARGE OF HEATER
6. AMBIENT AIR

STANSTEEL CORPORATION
JOB 1813
6/14/77, GLB

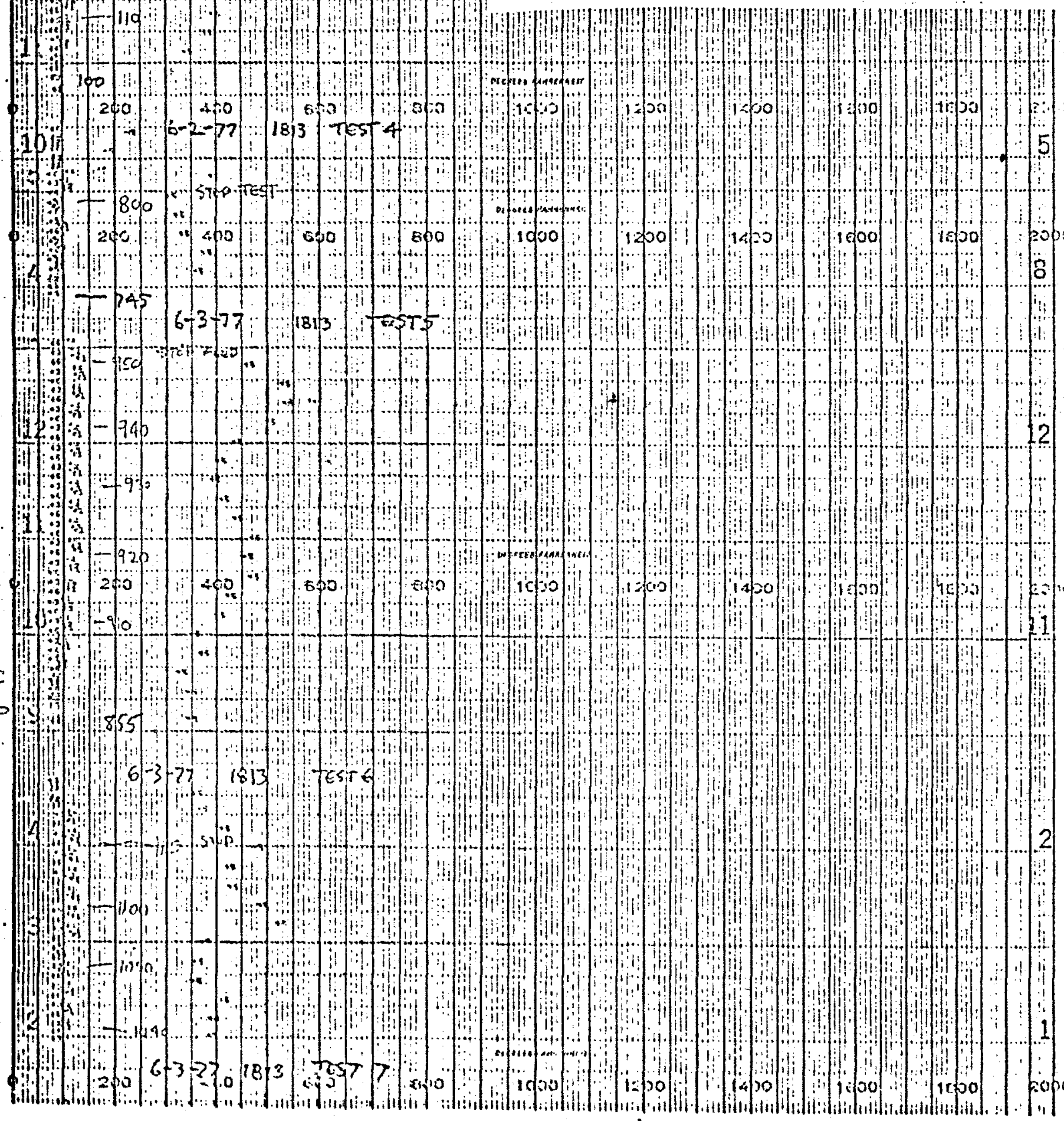


EXHIBIT A

Report

Prepared For

Mr. Ed Simonian
Stansteel Corporation
5001 South Boyle Avenue
Vernon, CA 90058

WEST COAST TECHNICAL SERVICE INC.

17605 Fabrica Way, Suite D
Cerritos, California 90701
213/921-9831
714/523-9200

Date: June 17, 1977

Job No.: 13003

P.O. No.: 28702

Your sample of extracted oil shale was received in our laboratory on June 3, 1977 for the determination of heat capacity from 100 - 400°F.

The sample was analyzed by differential scanning calorimetry with the following results.

T (°F)	C_p (Btu/lb/°F)
100	0.269
150	0.280
200	0.286
250	0.290
300	0.289
350	0.287
400	0.290

If we can be of any further service, please do not hesitate to contact us.

Respectfully submitted,

WEST COAST TECHNICAL SERVICE INC.

H. Dwight Fisher, Ph.D.
Vice President-Technical Director

HDF/kd

EXHIBIT B

RETORTED OIL SHALE DATA
SUPPLIED BY CLEVELAND CLIFFS

CHEMICAL COMPOSITION:

	39%	SiO_2
	6%	Sl_2O_3
	3%	$Fe_2\ O_3$
	14%	Ca O
	9%	Mg O
	.3%	SO_4
	2%	Na_2O
	7%	K_2O
	3%	Ca
Approx	20%	Unknown

SPECIFIC HEAT: $.2 - .26\ \frac{BTU}{\#°F}$

Particle distribution -

100%	minus	2-1/2"
96%	"	1-1/2"
76%	"	3/4"
53%	"	3/8"
40%		# 4
38%		# 8
35%		#16
30%		#30
25%		#50
20%		#100
17%		#200
15%		#325

EH Simonian

EXHIBIT C

PURCHASE ORDER

THE CLEVELAND-CLIFFS IRON COMPANY

ORDER NO. WD-77149

DATE May 16, 1977

REQ. NO.

VENDOR NO.

Stansteel Corporation
5001 Boyle Avenue
Los Angeles, California 90058
ATTN: Jorge Gerakios

MAIL 2 COPIES OF INVOICE AND SHIPPING NOTICE TO THE CLEVELAND-CLIFFS IRON CO., PURCHASING DEPARTMENT, AS INDICATED BELOW

- [ ] 1460 UNION COMMERCE BLDG. CLEVELAND, OHIO 44115
- [ ] ISHPEMING, MICH. 49849
- [ ] TACONITE, MINN. 55786
- X Box 1211, Rifle, Co. 816

- [ ] SHIP TO
- [X] INVOICE TO

THE CLEVELAND-CLIFFS IRON COMPANY
P.O. Box 1211
Rifle, Colorado 81650

PLEASE FURNISH THE FOLLOWING SUBJECT TO "CONDITIONS" ON BACK HEREOF WHICH ARE A PART OF THIS ORDER

QUANTITY	492-02-014	CCI CO. CODE NO.	PRICE
	Determine cooling characteristics of retorted oil shale and recommend equipment for cooling material. Estimate five (5) to six (6) days at $350/day not to exceed seven (7) days or $2,450 (Includes set up and take down expenses) Job 1813 Copies Made: Mr. Rodriguez ✓ Simonian		

F.O.B.	TERMS	VIA
[X] DESTINATION [ ] SHIPPING PT	NET 30 DAYS	

ATTENTION: Prices shown on this order are considered FIRM. Invoices with prices other than those shown will not be accepted. If not in agreement, please contact our Purch. Dept. immediately for permission to change.

The Cleveland-Cliffs Iron Company

By [signature]

APPENDIX G

SIEVE ANALYSES
CHEN & ASSOCIATES

chen and associates, inc.
CONSULTING ENGINEERS

SOIL & FOUNDATION ENGINEERING
96 S. ZUNI • DENVER, COLORADO 80223 • 303/744-7105
1924 EAST FIRST STREET • CASPER, WYOMING 82601 • 307/234-2126

May 13, 1977

Subject: Aggregate Testing, Retorted Oil Shale from Anvil Points, Garfield County, Colorado.

Job No. 14,583

The Cleveland-Cliffs Iron Company
P. O. Box 1211
Rifle, Colorado 81650

Attention: Mr. Robert A. Heisler

Gentlemen:

As requested, we tested a sample of retorted oil shale on May 10, 1977. One approximately 100-pound sample was submitted to our office on May 3, 1977. The sample was split with one-half returned to your office. The remaining sample was divided into 6 samples, of which 3 were tested for gradation and hydrometer analysis in accordance with ASTM procedures. The results are as follows:

Sieve Size	% Passing Sample #1	% Passing Sample #2	% Passing Sample #3
3	100.0	100.0	100.0
1½	92.2	96.8	93.7
3/4	62.6	75.6	55.9
3/8	42.2	54.2	35.9
#4	33.5	44.4	26.7
#8	33.3	44.0	26.5
#16	32.5	43.5	26.0
#30	31.5	42.6	25.4
#50	30.2	40.8	24.4
#100	28.6	38.5	23.4
#200	26.7	36.0	22.2
.005 mm	2.7	3.7	1.8
.002 mm	1.7	1.6	1.0

Calculations of the percentage passing the .074 mm was based on an average specific gravity of 2.65.

We have plotted the test results as shown on the attached figures to allow determination of percentage passing different sieve sizes.

The Cleveland-Cliffs Iron Company
May 13, 1977
Page 2

We will be available for testing of the remaining samples upon notification.

If you have any further questions, please call.

Very truly yours,

CHEN AND ASSOCIATES, INC.

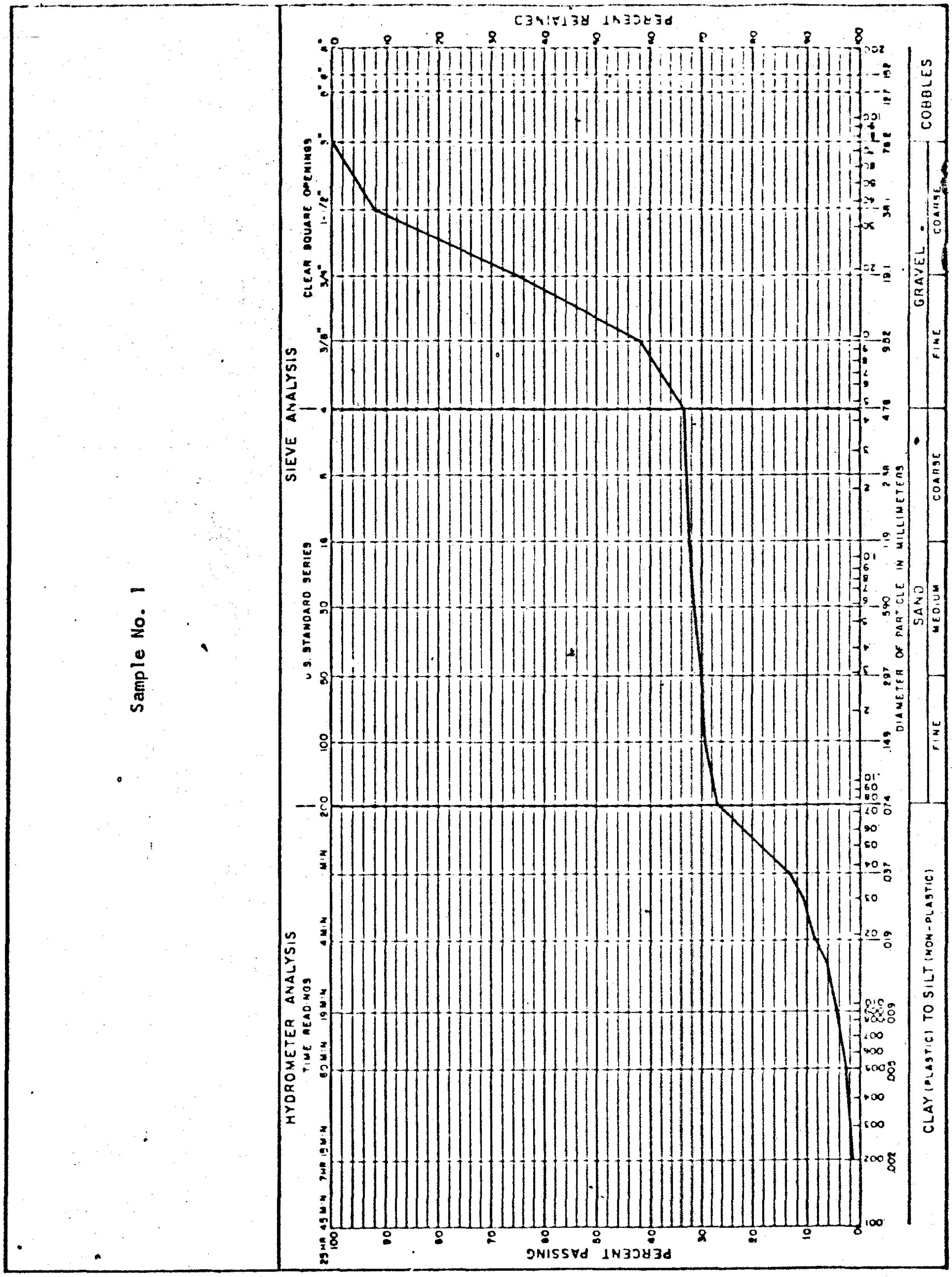
By

Hans Froeschle

HF/med
Rev. By: F
Enclosures

CHEN AND ASSOCIATES

Consulting Soil and Foundation Engineers



#14,583

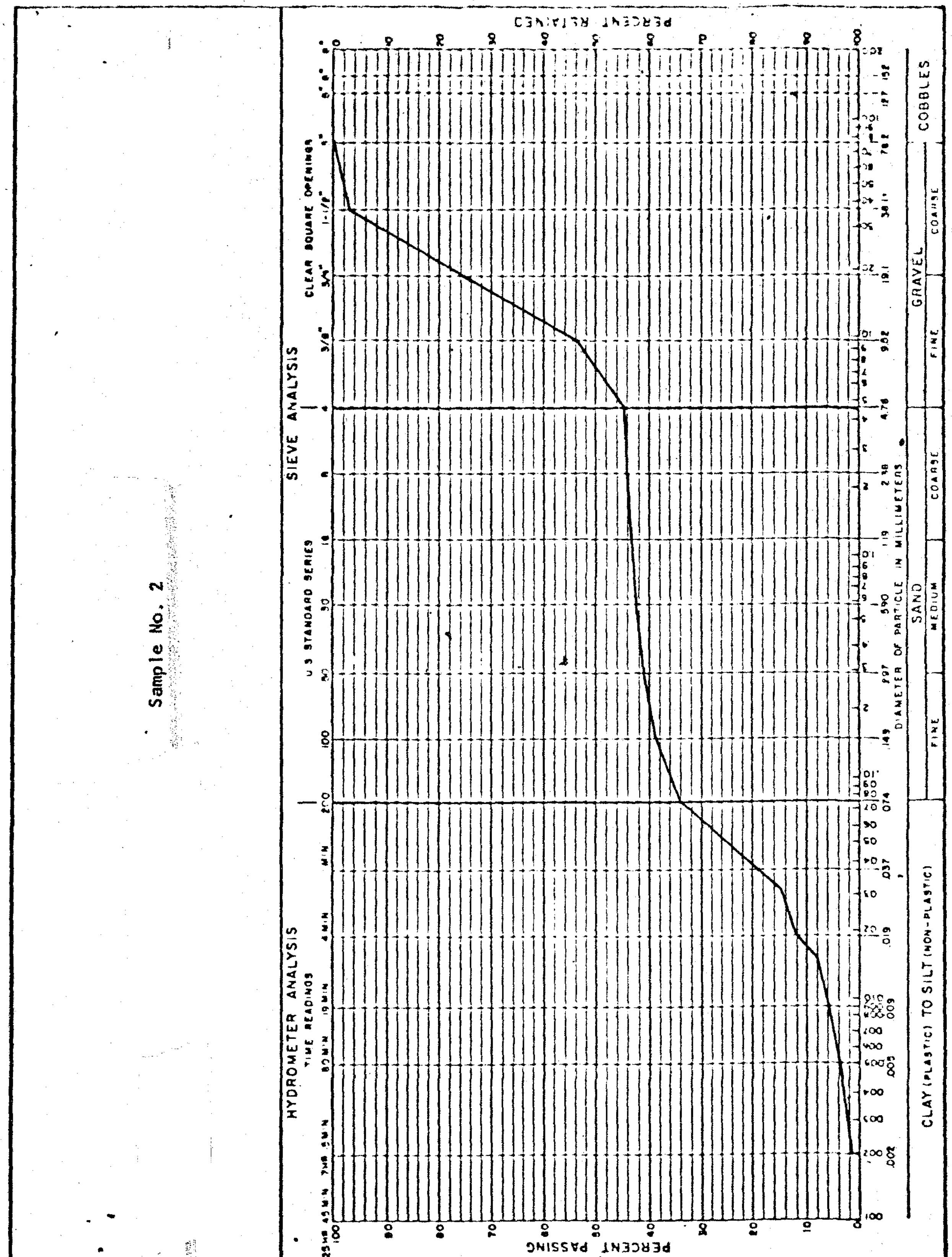
GRADATION TEST RESULTS

Fig. 1

CA-2A

CHEN AND ASSOCIATES

Consulting Soil and Foundation Engineers

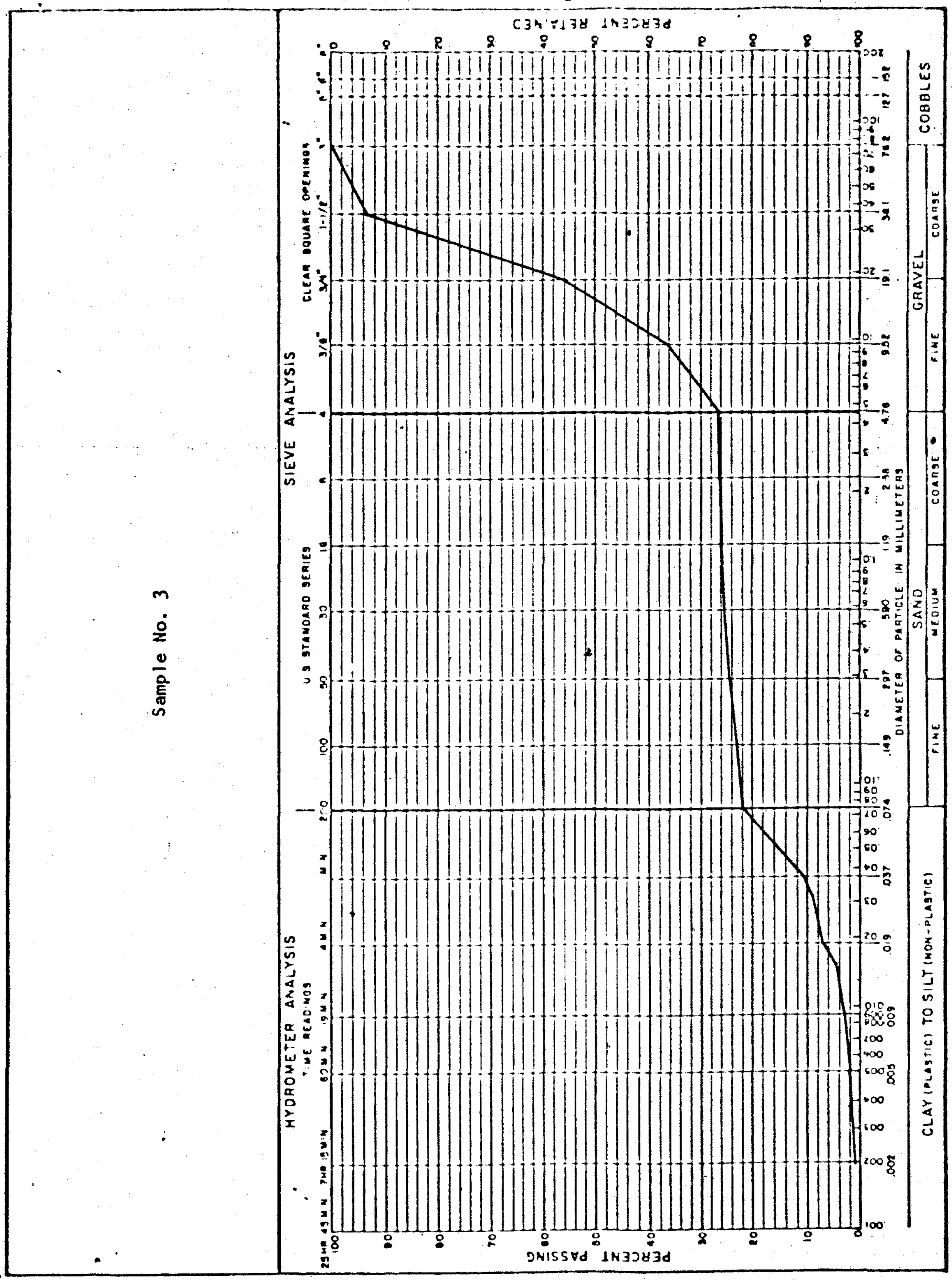


#14,583 GRADATION TEST RESULTS Fig. 2

CA-2A

CHEN AND ASSOCIATES

Consulting Soil and Foundation Engineers



#14,583

GRADATION TEST RESULTS

Fig. 3

CA-2A

chen and associates, inc.
CONSULTING ENGINEERS

SOIL & FOUNDATION ENGINEERING
96 S. ZUNI • DENVER, COLORADO 80223 • 303/744-7105
1924 EAST FIRST STREET • CASPER, WYOMING 82601 • 307/234-2126

June 7, 1977

Subject: Gradation Tests, Retorted Oil Shale from Anvil Points, Garfield County, Colorado.

Job No. 14,583

The Cleveland-Cliffs Iron Company
P. O. Box 1211
Rifle, Colorado 81650

Attention: Mr. Robert A. Heisler

Gentlemen:

We are forwarding the test results for the sieve and hydraulic analyses for specimens of retorted oil shale Nos. 4 and 5. We previously submitted the test results for Specimens No. 1, 2 and 3 in our letter dated May 13, 1977. The results for Specimens 4 and 5 are as follows:

Sieve Size	Specimen #4	Specimen #5
3"	100%	100%
1½"	94%	88%
3/4"	66%	53%
3/8"	46%	36%
#4	37%	29%
#8	37%	28%
#16	36%	27%
#30	34%	25%
#50	31%	23%
#100	29%	20%
#200	26%	18%
.005 mm	2%	2%
.002 mm	0.6%	0.6%

Calculation of the percentage passing the No. 200 screen was based on an average specific gravity of 2.65.

We have plotted the test results and included a copy on the attached figures.

If we can be of additional service, please call.

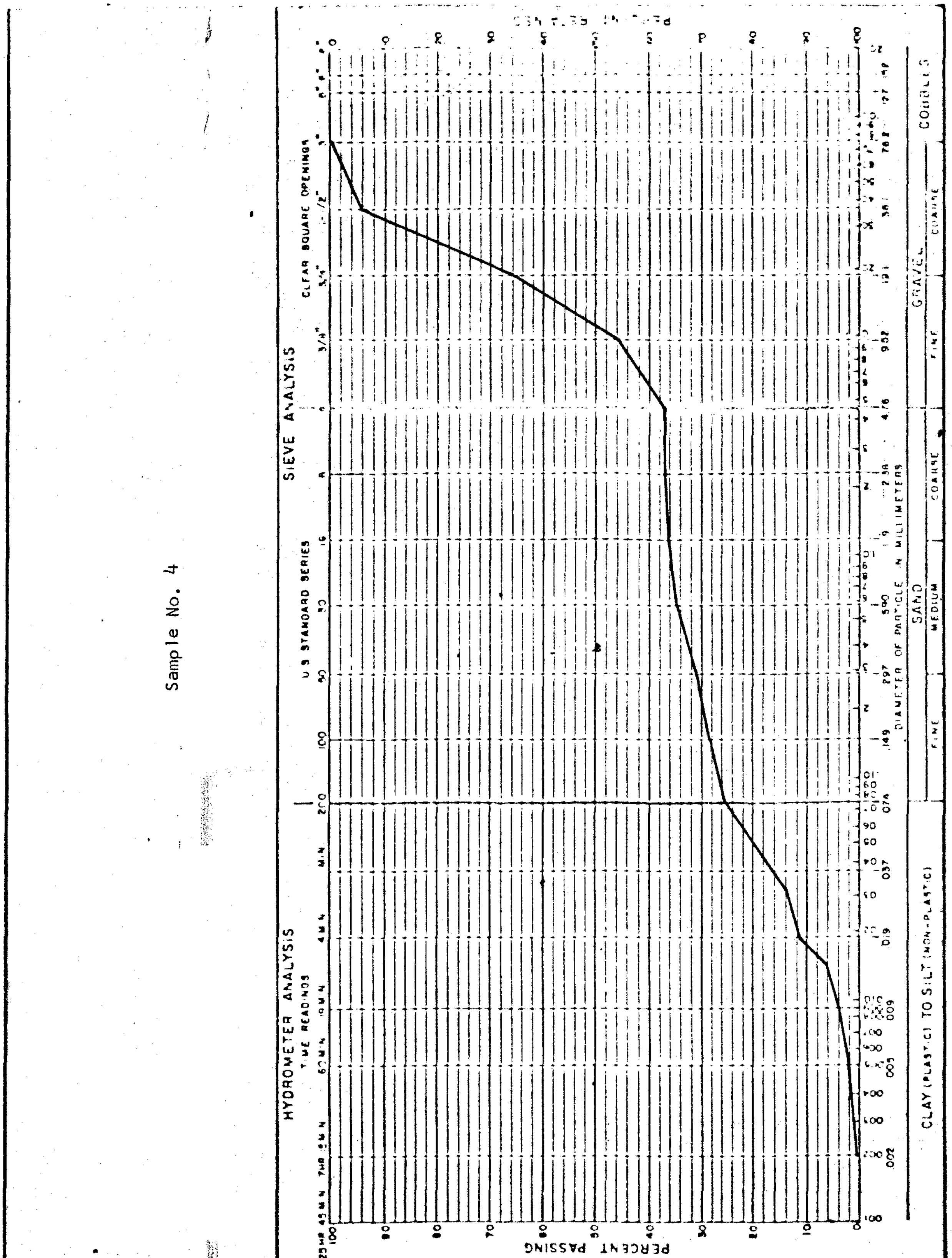
Sincerely,

CHEN AND ASSOCIATES, INC.

DONALD E. BRESSLER
REGISTERED
7524
PROFESSIONAL ENGINEER
STATE OF COLORADO

By [signature]
Hans Froeschle

HF/med
Rev. By: D.E.B.
Enclosures



#14,583

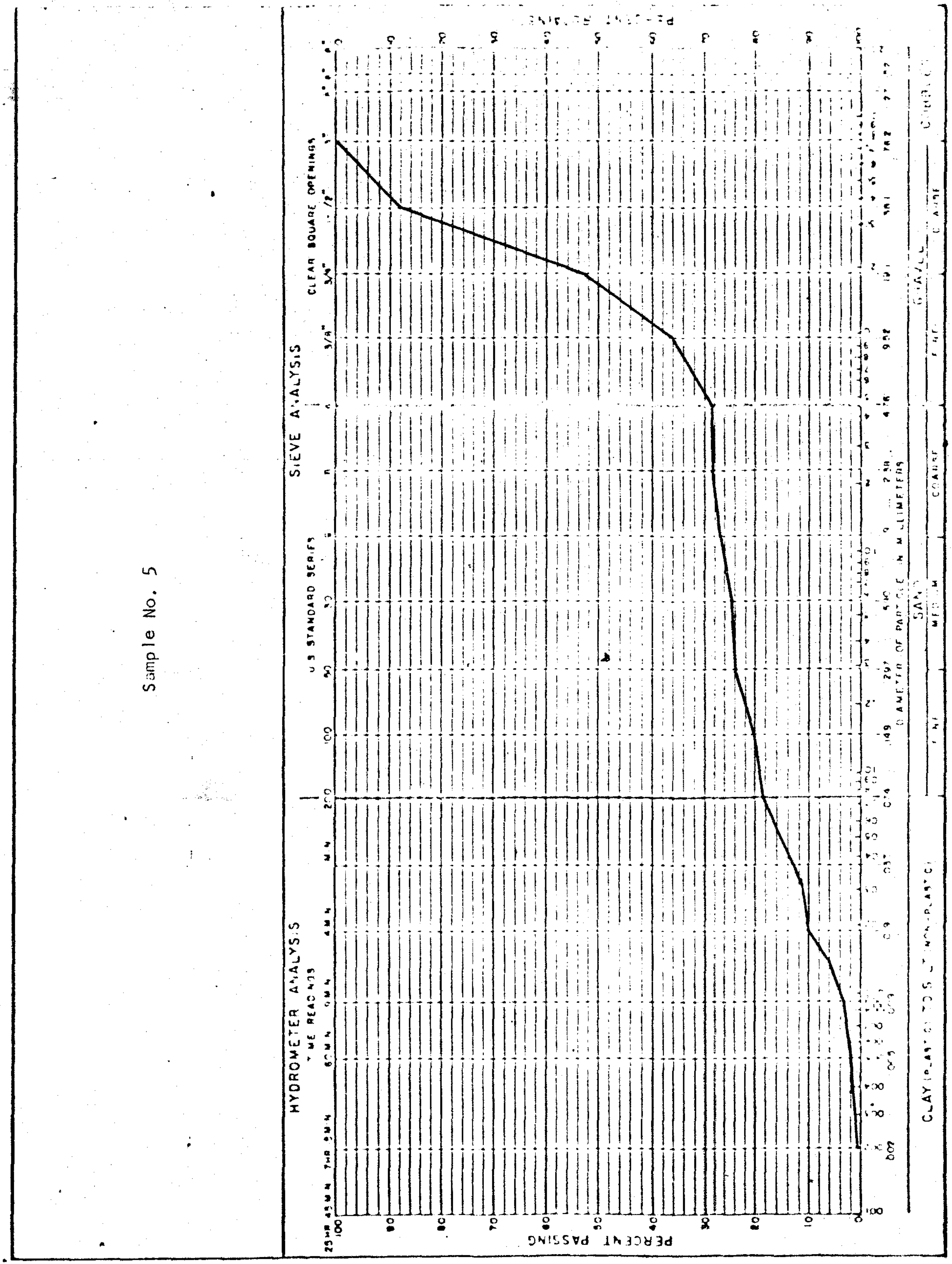
GRADATION TEST RESULTS

Fig. 1

CA-2A

CHEN AND ASSOCIATES

Consulting Soil and Foundation Engineers



#14,583

GRADATION TEST RESULTS

Fig. 2

CA 2A

chen and associates, inc.
CONSULTING ENGINEERS

SOIL & FOUNDATION ENGINEERING
96 S. ZUNI • DENVER, COLORADO 80223 • 303/744-7105
1924 EAST FIRST STREET • CASPER, WYOMING 82601 • 307/234-2126

June 22, 1977

Subject: Aggregate Testing, Retorted Oil Shale from Anvil Points, Garfield County, Colorado.

Job No. 14,583

The Cleveland-Cliffs Iron Company
P.O. Box 1211
Rifle, Colorado 81650

Attention: Mr. Robert A. Heisler

Gentlemen:

As requested, we performed gradation analysis on three samples of Retorted Oil Shale submitted to our office. Samples from runs 3 and 4 were tested for gradation above the #200 screen only and Sample #3 from the cyclone product was subjected to hydrometer test only. The results are as follows:

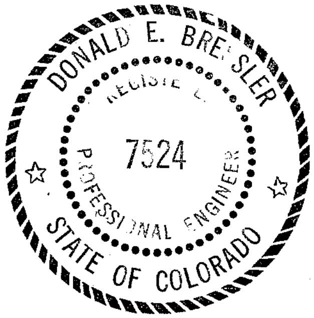
Sieve Size	% Passing Run #3 Cooler Product	% Passing Run #4 Product	% Passing Sample #3 Cyclone Product
3	100.0	100.0	
1½	92.0	96.0	
3/4	63.0	64.0	
3/8	41.0	40.0	
#4	28.0	27.0	
#8	22.0	22.0	
#16	11.0	19.0	
#30	4.0	11.0	
#50	2.4	9.0	
#100	1.9	9.0	
#200	1.7	9.0	95
.005 mm			84.0
.002 mm			5.0

Calculations of the percentage passing the .074 mm was based on an average specific gravity of 2.65.

We have plotted the test results as shown on the attached figures to allow determination of percentage passing different sieve sizes.

The Cleveland-Cliffs Iron Company
June 22, 1977
Page 2

If we can be of additional service, please call.



Sincerely,

CHEN AND ASSOCIATES, INC.

By ______________________
Hans Froeschle

HF/as
Rev. By: D.E.B.
Enclosures

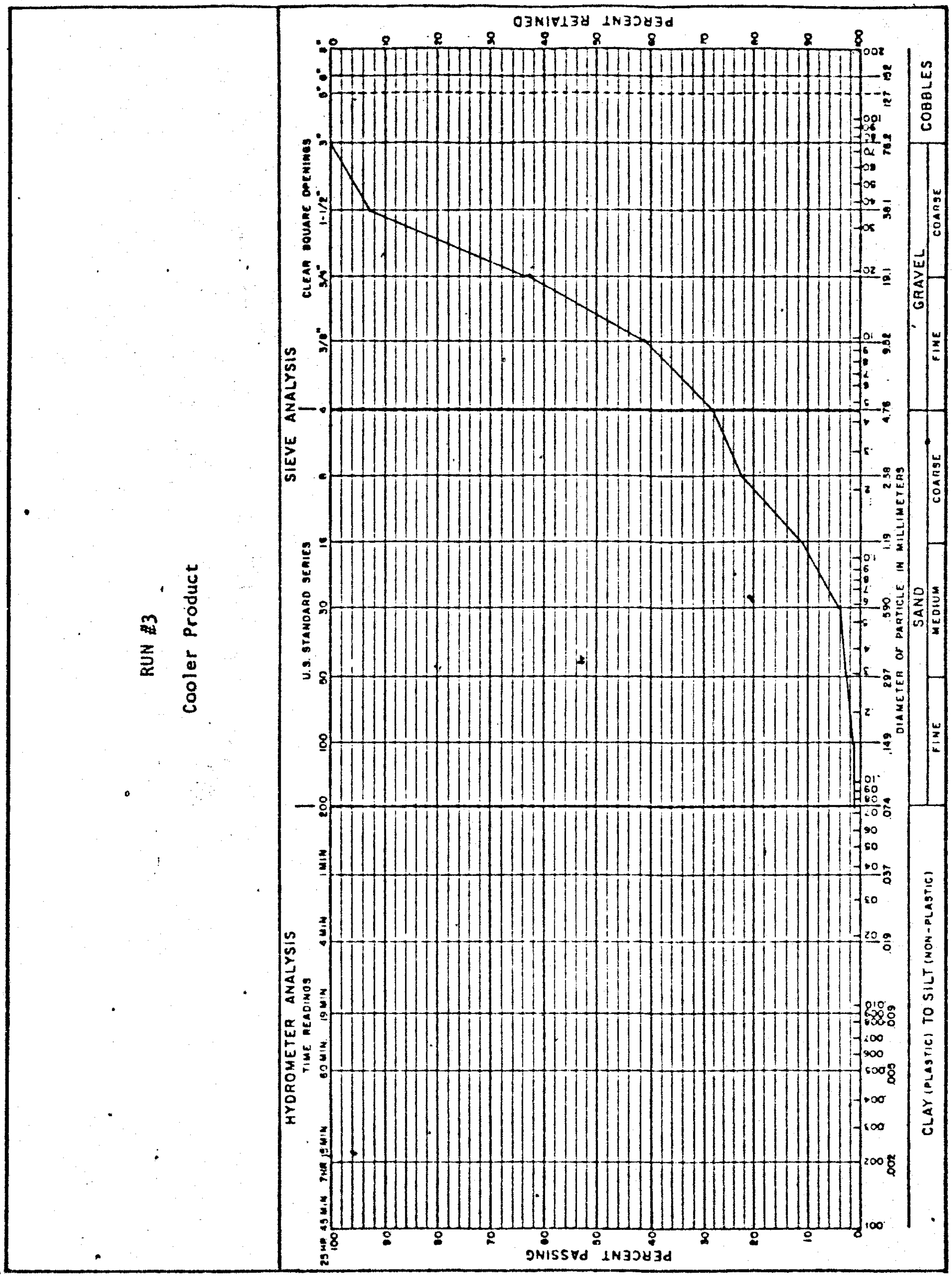


Fig. 1

6-22-77

#14,583

GRADATION TEST RESULTS

CHEN AND ASSOCIATES

Consulting Soil and Foundation Engineers

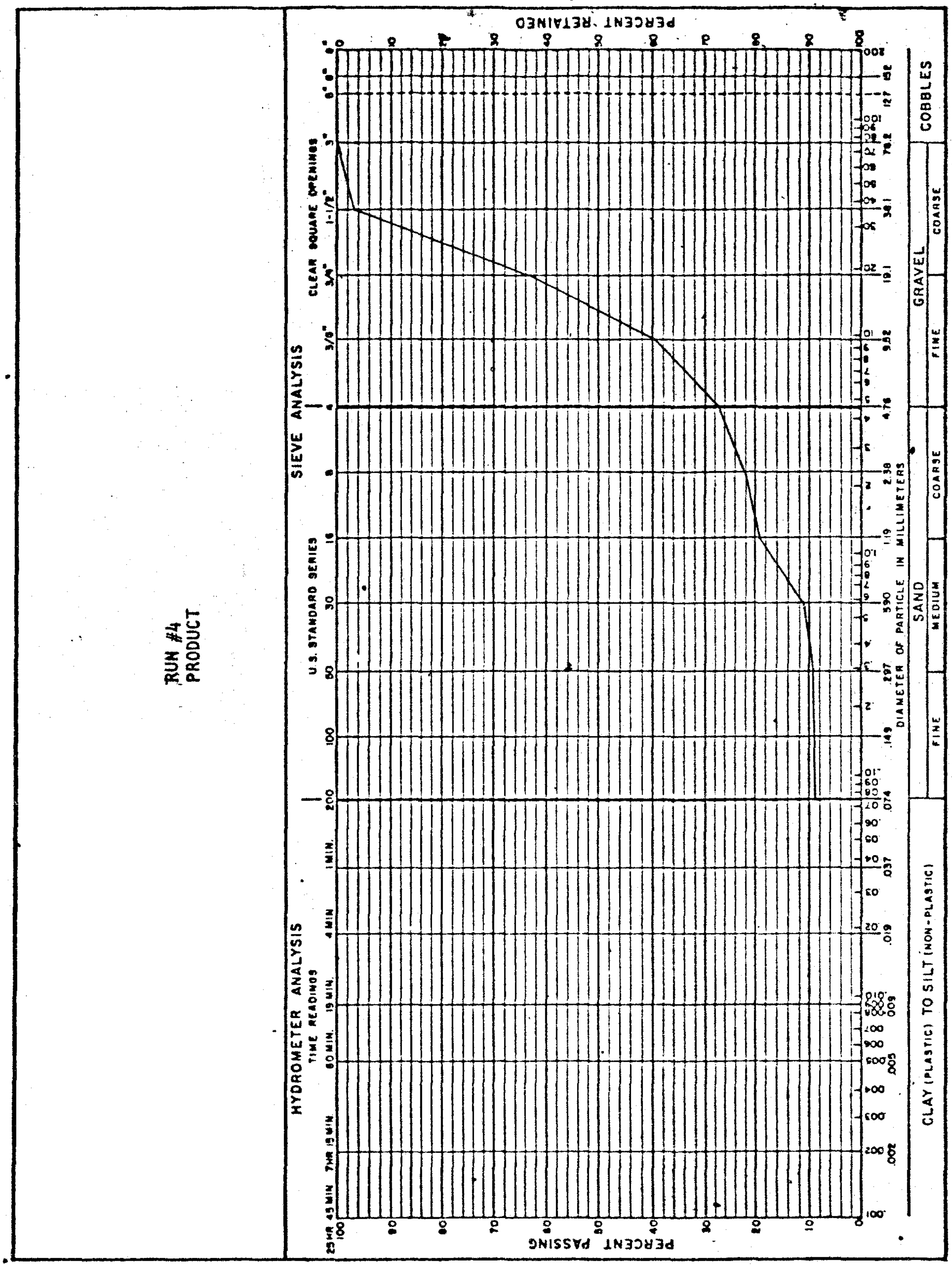


Fig. 2

#14,583 6-22-77

GRADATION TEST RESULTS

CA-2A

Chen and Associates

Consulting Soil and Foundation Engineers

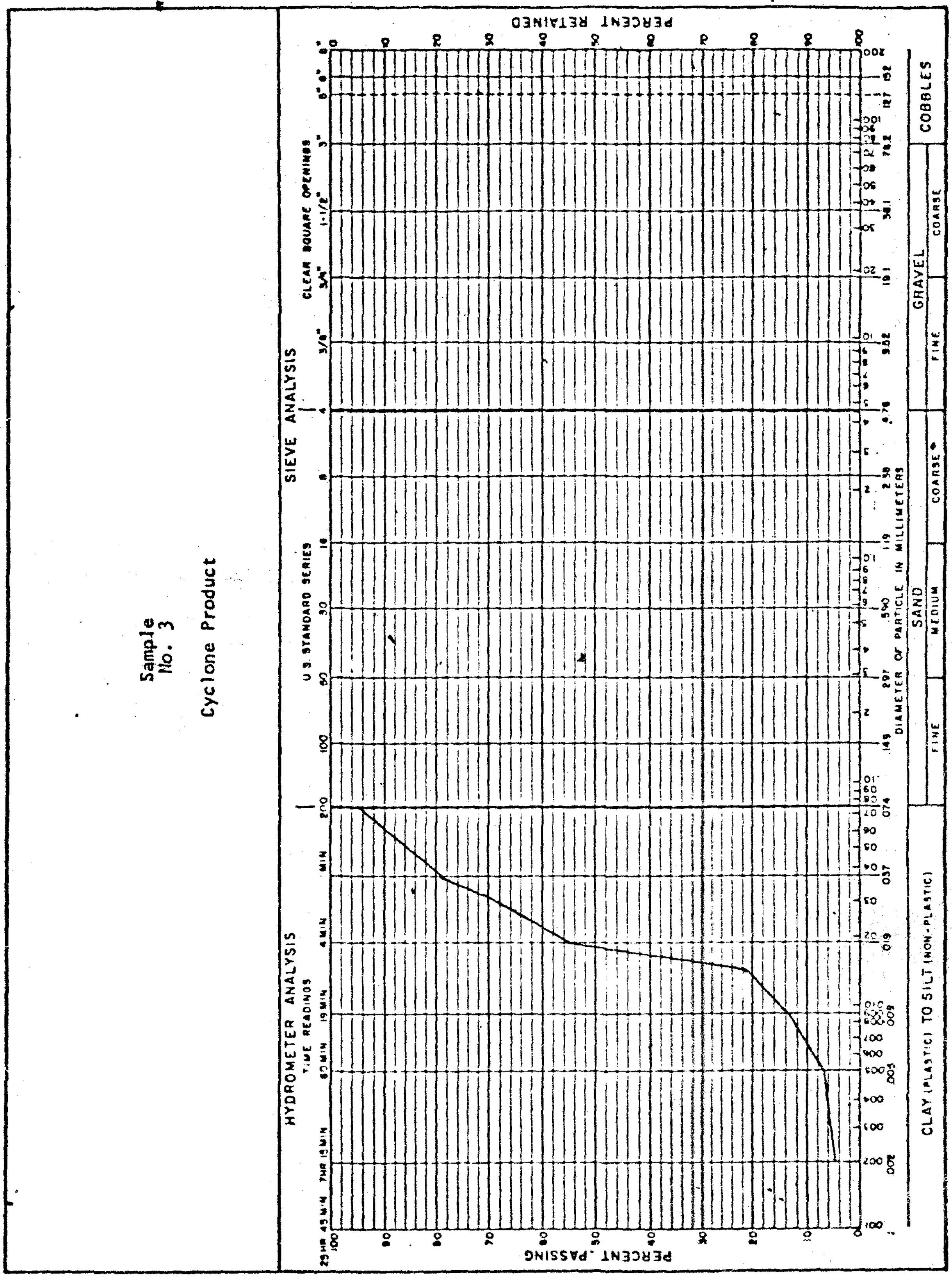


Fig. 3

6-22-77

#14,583

GRADATION TEST RESULTS

CA-2A