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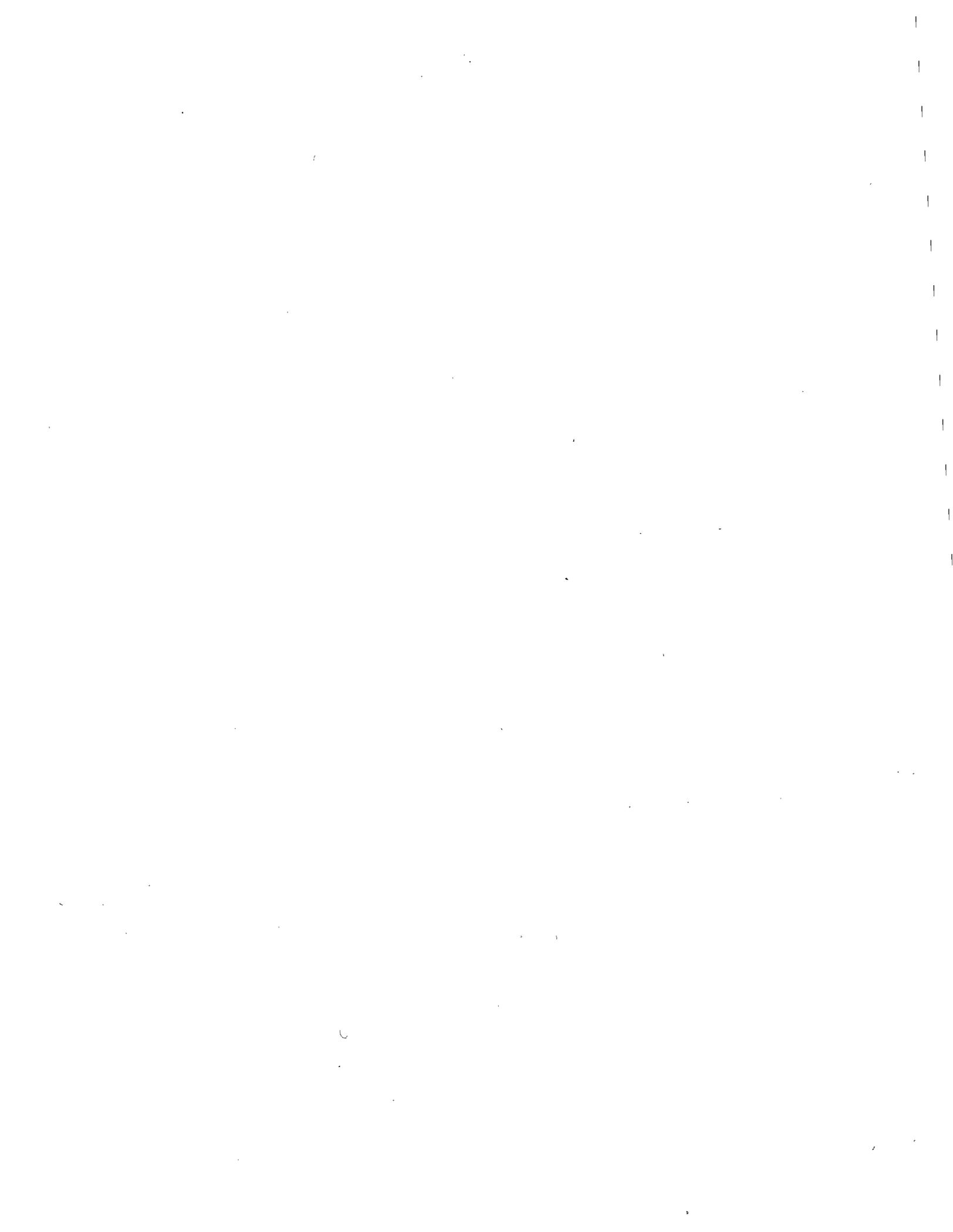
PROTOTYPE UNDERGROUND DEMONSTRATION MINE
IN OIL SHALE DEPOSITS

FINAL REPORT

Bureau of Mines Open File Report 69-79

March 1979

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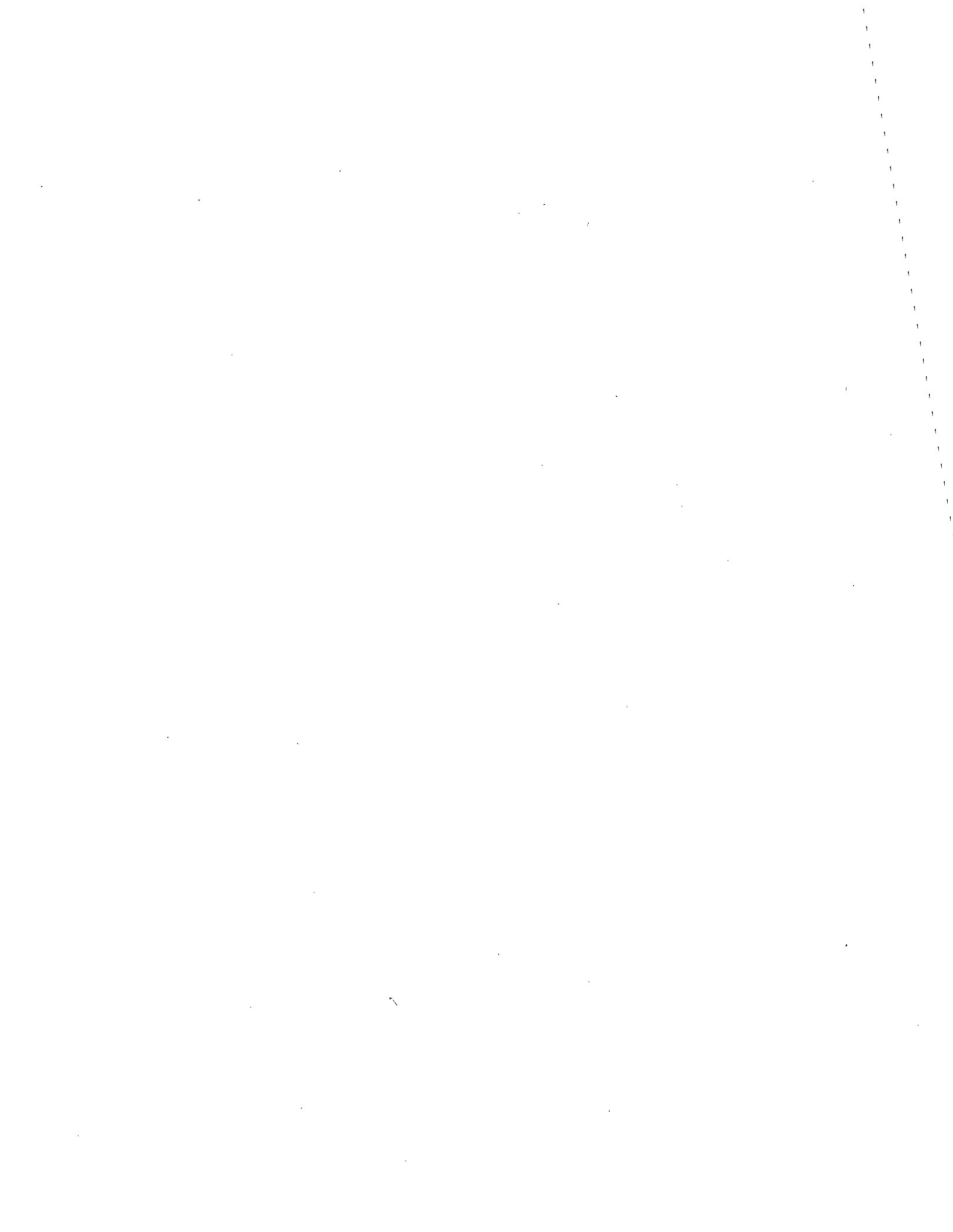


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16. Abstract (Limit: 200 words) This report contains a summary of designs, specifications, costs, and schedules developed for a prototype oil-shale demonstration mining program to be undertaken at the Horse Draw site in the Piceance Creek Basin, Rio Blanco County, Colo. The objective of the program is to assess the feasibility of extracting oil shale and associated saline minerals from deep, rich oil shale zones that occur within the central portion of the basin. Four mining systems were specified for demonstration. These include (1) chamber and pillar mining with backfill, (2) sublevel stoping with backfill, (3) sublevel stoping with full subsidence, and (4) block caving. Commercial-scale designs for each system were modified and scaled down as required to achieve program objectives. The demonstration mine layout and design includes specifications for our handling, backfilling, ventilagion, dewatering, maintenance, and supply systems. A feature of particular interest in the design is the combination of block caving and sublevel stoping methods into a single unit for demonstration of subsidence mining systems in deep oil shale deposits. The report also contains a detailed breakdown of projected capital and operating costs for the program.			
17. Document Analysis a. Descriptors Oil shale Demonstration mine Piceance Creek Basin Mining Backfilling b. Identifiers/Open-Ended Terms Subsidence Chamber and pillar Sublevel stoping Block caving Gassy mine Layout and design Capital and operating costs c. COSATI Field/Group 081			
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FOREWARD

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. J0265020. 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 Bureau's Denver Mining Research Center, with Mr. Robert L. Bolmer acting as the Technical Project Officer. Mr. B. G. Horton was the Contract Administrator for the Bureau of Mines.

This report is a summary of the work performed under Phases I, II, and III of the Contract during the period from July 1976 to February 1979. It was submitted in March 1979.

No patents or inventions have resulted from Contract work.

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

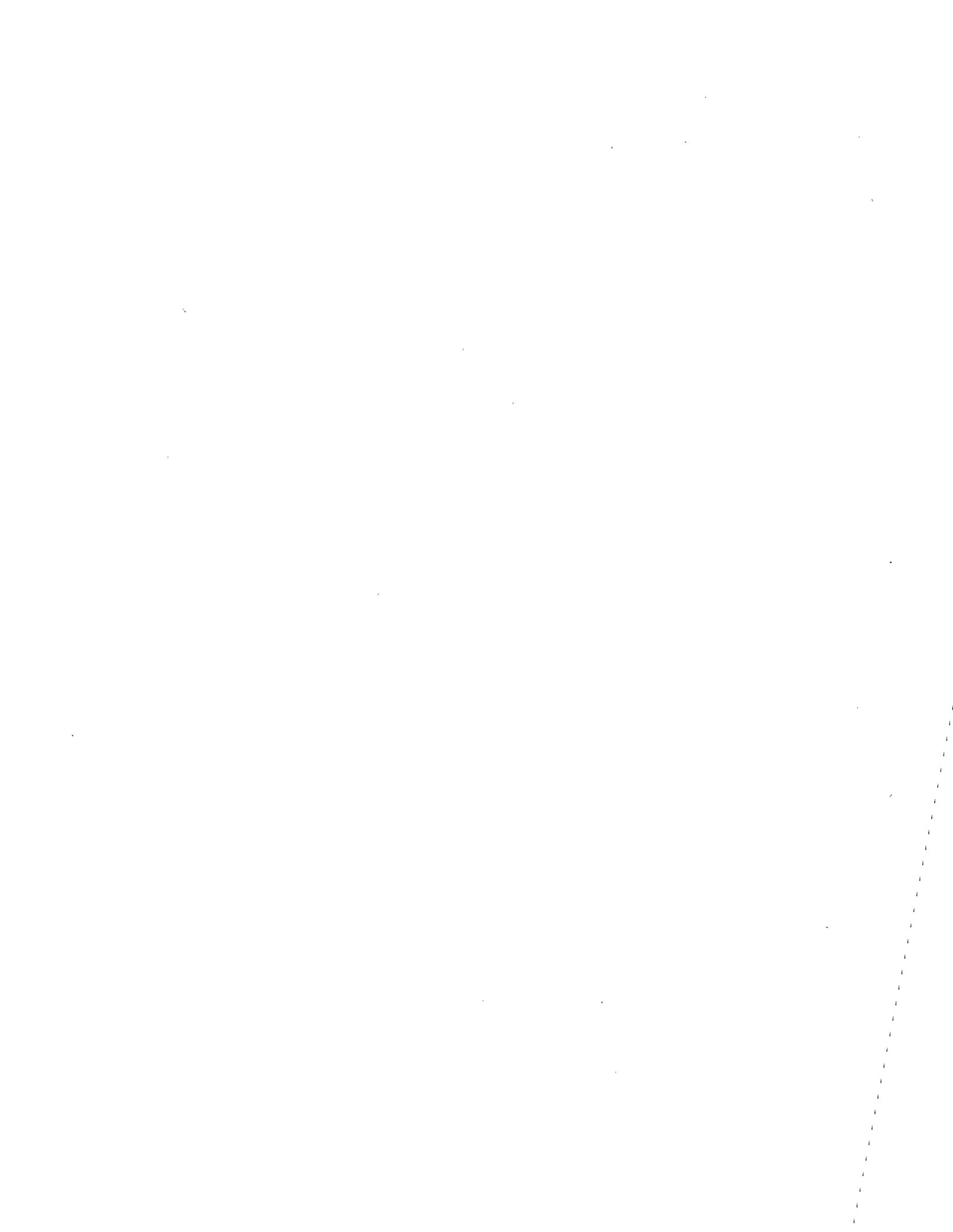


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1.0 INTRODUCTION

The following report on a Prototype Underground Demonstration Mine in deep oil shale deposits summarizes all work completed between July 1976 and February 1979 by The Cleveland-Cliffs Iron Company under Phases I, II, and III of U. S. Bureau of Mines Contract No. J0265020. Earlier contract studies by the Bureau of Mines had disclosed several promising new systems of mining the deeper and thicker deposits of oil shale and associated minerals in the Piceance Creek Basin of Colorado (1). However, the technical and economic feasibility of these conceptual methods could be verified only by actual field demonstration. The design, layout, and costing of a prototype underground demonstration mine for such purpose thus was the objective of this Contract. Although demonstration mining is no longer contemplated by the Bureau, the technical and financial information resulting from the Contract work and summarized here will benefit any future mining efforts by the Government or industry.

Phase I work consisted of adaptation of specified mining systems for demonstration purposes. Design and layout of the demonstration mine were addressed in Phase II, and detailed costing of the demonstration program was completed in Phase III.

1.1 SCOPE OF WORK - PHASE I

Design features of each mining system were reviewed to determine the minimum numbers of demonstration excavations required for adequate evaluation of each system. Rock strength data for the projected mining interval at the Horse Draw site were evaluated to determine maximum allowable roof spans and pillar dimensions for desired factors of safety. Using these criteria, the dimensions of stopes and support pillars were established for each demonstration unit. The extent of development work required to block out the stoping areas was determined.

Comprehensive engineering analyses were made of the mining functions required to perform the indicated development and mining in the demonstration units. All pertinent aspects of mining relating to cost and efficiency of the work to be performed were evaluated. Detailed estimates were prepared of such items as work cycles, labor and equipment requirements, equipment availability and maintenance, consumption of explosives and other expendable supplies, ventilation and dewatering requirements, task durations, rates of progress and productivity, power consumption, backfill tonnages, and costs of subcontracted services. From such data operating schedules were determined for mining of the four demonstration units. Results of the engineering studies were submitted to the U. S. Bureau of Mines as an interim Phase I Report.

1.2 SCOPE OF WORK - PHASE II

Supplementary engineering studies were undertaken during Phase II to develop a complete mine design program for demonstration of the scaled-down mining systems designed during Phase I. Specific topics addressed in Phase

II included orientation and layout of the demonstration mine, analysis of primary development requirements, mine systems design, and mine scheduling.

To enhance the overall cost effectiveness of the program, a common interval (the R-4 zone) was selected for demonstration of all four mining systems. Determination of an optimum vertical and horizontal orientation for mine layout was based on (1) the vertical distribution of zones of rich oil shale, nahcolite and dawsonitic shale; (2) the vertical distance separating the base of the lower aquifer from the mining interval; (3) the orientation of local geologic structures such as faults, slump zones, and fractures at the Horse Draw site; and (4) other site-specific information provided by the U. S. Bureau of Mines. The location for the production shaft was selected after evaluation of subsurface geologic data and surface site conditions.

Initial considerations in mine layout planning included design of the shaft pillar around the production shaft, the layout of main entries from the shaft toward the demonstration areas, and the specific arrangement of the demonstration units relative to the main entries and the production shaft. Main entries were projected on two levels (the upper level and the lower haulage level), bearing S20°W from the production shaft. The two demonstration units that incorporate backfilling for overburden support (chamber and pillar mining with backfill and sublevel stoping with backfill) were situated approximately 1,200 feet from the production shaft. The two subsidence mining systems (block caving and sublevel stoping with full subsidence) were located an additional 960 feet from the shaft. Such layout provides adequate separation between the production shaft and the demonstration units to ensure shaft stability.

After the main entries and demonstration units had been laid out, additional drifts and ramps were specified to provide access to the units for mining and for implementation of mine systems (ventilation, haulage, supply etc.). Excavation requirements in the shaft station areas for ventilation airways, sump fixtures and a pumping station, substations, shops, magazines, etc. were also considered. Designs and specifications for mine ventilation, crushing and conveyance, dewatering, backfilling, and other systems were developed concurrently.

All primary development requirements were broken down into constituent mining functions and subjected to engineering analyses, as in Phase I. The results of both analyses were combined to assess all program requirements. Schedules were developed depicting mine operation, manpower and equipment utilization, production and productivity, and mine ventilation requirements. An interim Phase II Report was submitted to the U. S. Bureau of Mines containing a detailed discussion of these studies and activities.

1.3 SCOPE OF WORK - PHASE III

A detailed analysis was made of all costs to be incurred in demonstrating the four mining systems. Capital costs and mine operating costs were evaluated separately. Operating costs were broken down by operational phases,

by tasks, and by mining functions. Costs of support activities, facilities, and manpower were identified and analyzed by source.

Total and unit costs were assembled for all labor, equipment, power, materials, and supply items. Costs were based on estimates and quotations received in calendar year 1978. Increases in costs were projected through calendar year 1980.

A mine operating schedule was developed in Phase II which will permit demonstration of the four mining systems with optimum efficiency and cost effectiveness. A comparative evaluation was made of capital and operating costs for one-shift-per-day and two-shifts-per-day modes of mine operation.

The Final Report contains a summary of all designs, specifications, technical analyses, and economic evaluations completed during the three project phases.

2.0 GEOTECHNICAL DESIGN DATA

The Horse Draw area in the north central portion of the Piceance Creek Basin originally was selected by the U. S. Bureau of Mines as the site for a prototype underground demonstration mine in oil shale deposits. The geology of the area was defined by surface mapping and by evaluation of core and geophysical data from four drill holes. Site-specific hydrologic information was obtained from jet tests and pump tests conducted in these holes. Selected intervals of core from two of the holes, 01-A and 02-A, were tested to determine values of rock strength for various zones of interest intersected by the drilling.

2.1 GEOLOGY

The purpose of the geological data collection program initiated in the latter part of 1975 was to determine stratigraphic and structural relationships of oil shale and associated minerals of the Green River Formation at the Horse Draw site. As a part of the program, surface geological features such as fracture orientation patterns and localized slump zones were identified.

2.1.1 Setting and Topography

The Piceance Creek Basin of northwestern Colorado is an asymmetrical, northwest-trending, structural basin that lies between the White River drainage system to the north and the Colorado River drainage system to the south (Figure 2.1). The land surface within the basin consists of a series of steep-sided ridges and valleys that generally slope downward from the margins toward the interior of the basin. Surface cover is generally sparse, consisting mainly of sage, juniper, and pinon pine in the lower areas.

Horse Draw is the main drainage feature of the northeastern quadrant of USEM Site No. 2 (Figure 2.2). It is a small, 2.5-mile intermittent drainage, tributary to Piceance Creek. The draw itself has several very short tributaries that end abruptly against steeply-rising slopes, particularly along the northwest bank. The surface erosional pattern is generally rugged except near the junction with Piceance Creek, where a small flood plain has developed. The surface features are indicative of infrequent periods of intense stream flow and erosion associated with spring runoff and late summer storms.

2.1.2 Stratigraphy

The Eocene Uinta Formation outcrops at the surface throughout the central part of the basin. Beneath the Uinta formation lies the thick Eocene Green River Formation composed of three distinct members: The Parachute Creek Member, the Garden Gulch Member, and the Douglas Creek Member. The Parachute Creek Member and the upper portion of the Garden Gulch Member contain zones of kerogenetic marlstone (oil shale) and associated carbonate saline minerals, such as nahcolite (NaHCO_3) and dawsonite ($\text{NaAl}(\text{OH})_2\text{CO}_3$). Zones of halite also are present in this interval, usually in close association with nahcolite.

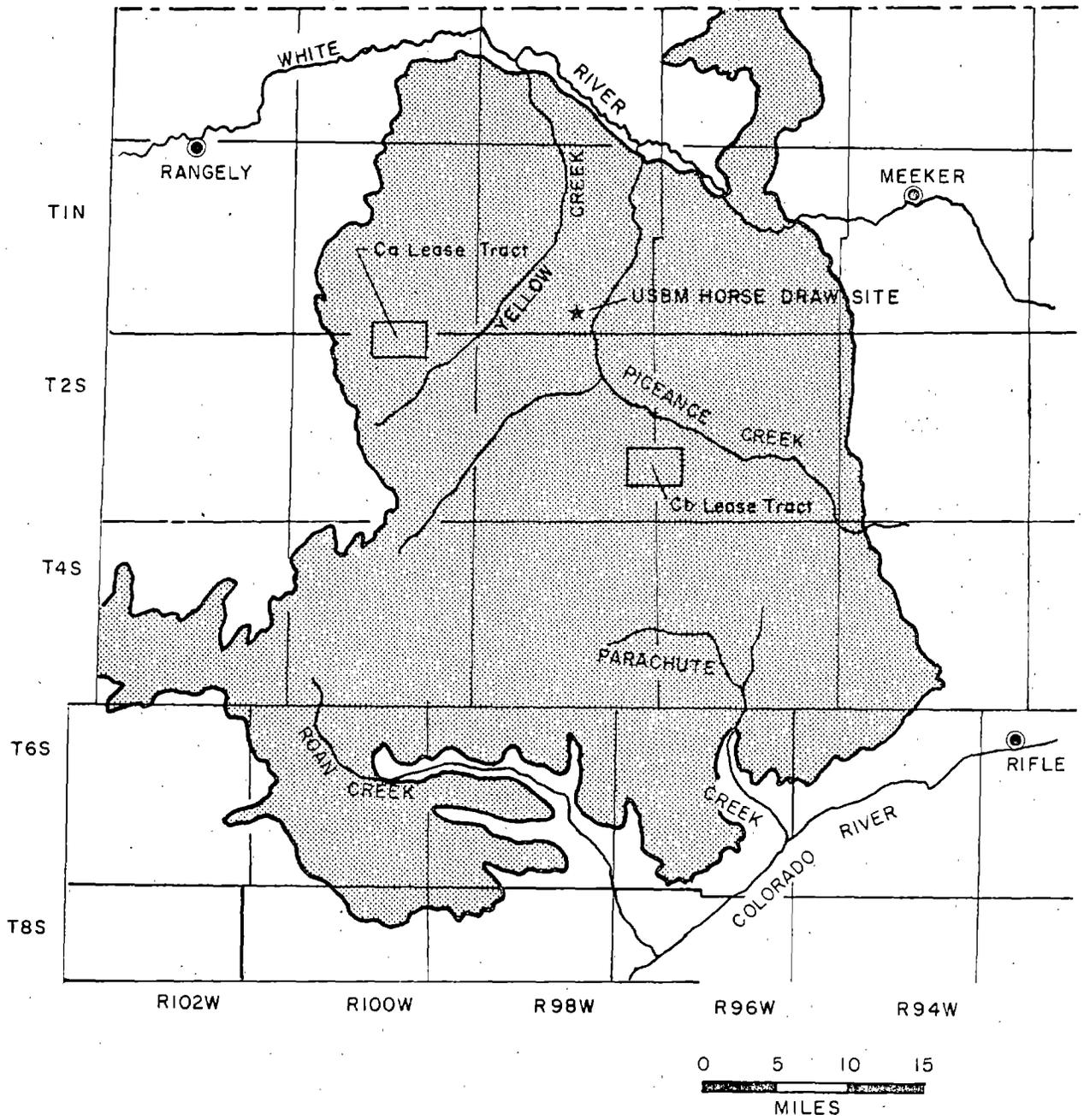


FIGURE 2.1

LOCATION MAP - PICEANCE CREEK BASIN

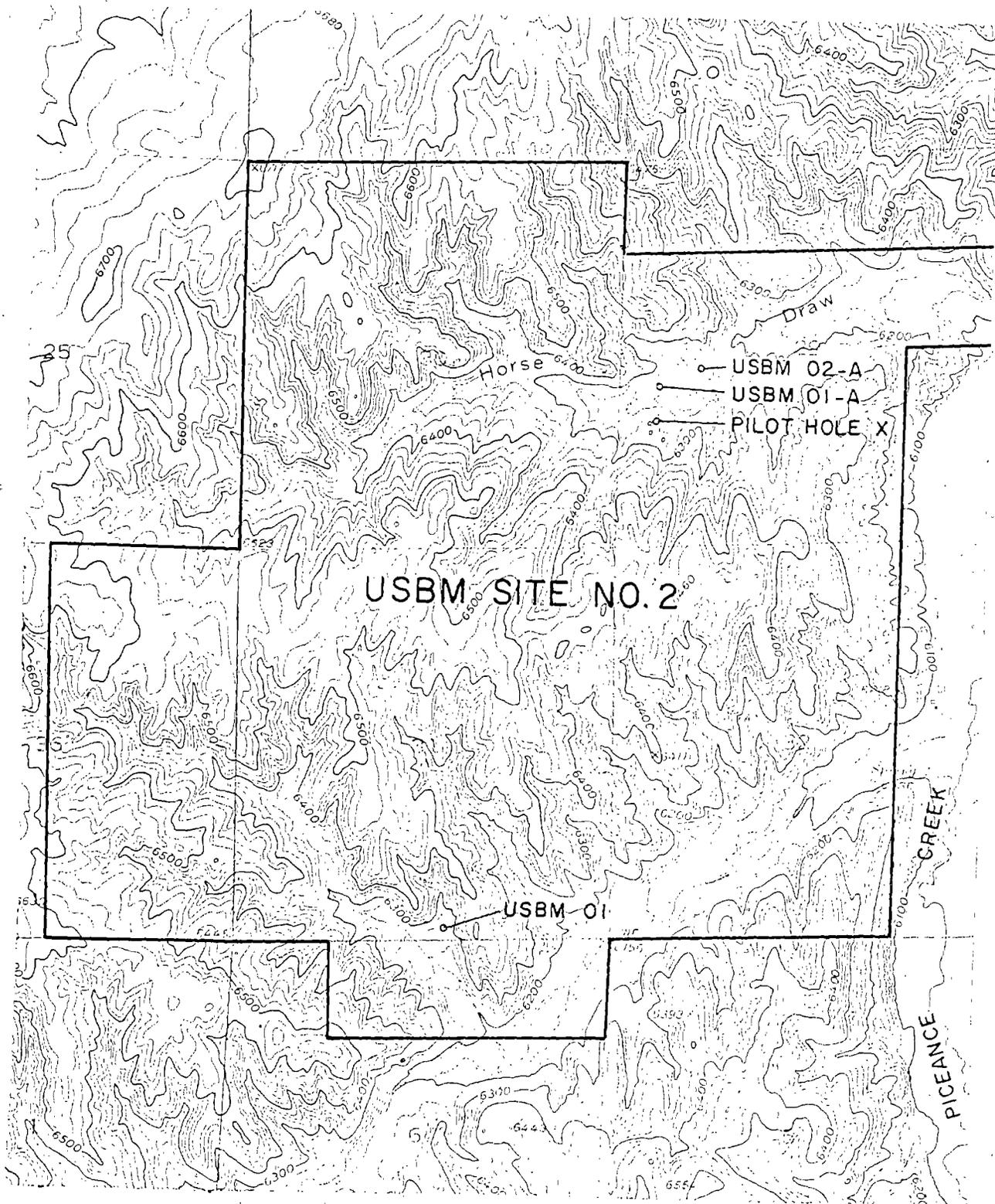


FIGURE 2.2
TOPOGRAPHIC MAP OF HORSE DRAW AREA
SHOWING USBM DRILL HOLES

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The stratigraphic profile at Horse Draw was determined from logs of four drill holes designated USBM 01, USBM 01-A, USBM 02-A, and Pilot Hole X. The locations of these holes are shown in Figure 2.2. All holes were collared in the Uinta Formation and were bottomed in the Garden Gulch Member of the Green River Formation. A cross-section through the holes showing the various marker beds and oil shale zones is presented in Figure 2.3. As indicated in the cross-section, the Uinta-Green River Formation contact is approximately 800 feet below the surface in this area of the basin. The Mahogany Zone extends from approximately 900 to 1,100 feet and the saline facies from about 1,100 to 2,400 feet. The saline oil shale section is further subdivided into rich and lean zones, designated R-0 through R-6 and L-0 through L-5, respectively. Saline minerals have been leached from the uppermost 300 feet of this section by ground water, forming the relatively permeable lower aquifer.

2.1.3 Structure

The most prominent structural feature in the north central basin area is the N70°W-S70°E trending graben that transects the area of interest about 1.6 miles south of the selected shaft site (Figure 2.4). This feature is an outlying expression of the Piceance Creek Dome to the southeast. Other parallel, but less persistent, tensional structural features lie to the southeast mostly along the northern flank of the dome.

A few zones of localized surface slump have been identified in the area of interest (Figure 2.4). Some jointing is present in the exposed surface rocks. One joint set, oriented N50°E and nearly vertical, coincides with major tributaries on the south side of Horse Draw and with the general direction of tension cracks interpreted from U. S. Bureau of Mines resistivity surveys.

2.2 HYDROLOGY

The Piceance Creek Basin may be classified as an arid to semiarid region with annual precipitation ranging from about 10 inches in the lower areas to as much as 25 inches in the higher plateaus. The direction of the main drainage system is generally north to northeast. Tributary drainages form a varied pattern from branching to structurally controlled (by faults, joints, and folds). Except for Piceance Creek and Yellow Creek all streams in the northwest portion of the basin are intermittent, flowing only during periods of spring runoff and late summer storms.

Water is a valuable resource in the area, and a significant portion of it is contained in two aquifers. The geohydrologic section at Horse Draw consists of an upper aquifer, a confining layer (Mahogany Zone), a lower aquifer, and the unleached saline zone (Figure 2.5). The geohydrologic model (illustrating the physical and chemical characteristics of the two aquifers) was defined by hydrological tests conducted during 1975 and 1976 by the U. S. Geological Survey, Water Resources Division. The upper aquifer is about 900 feet thick and

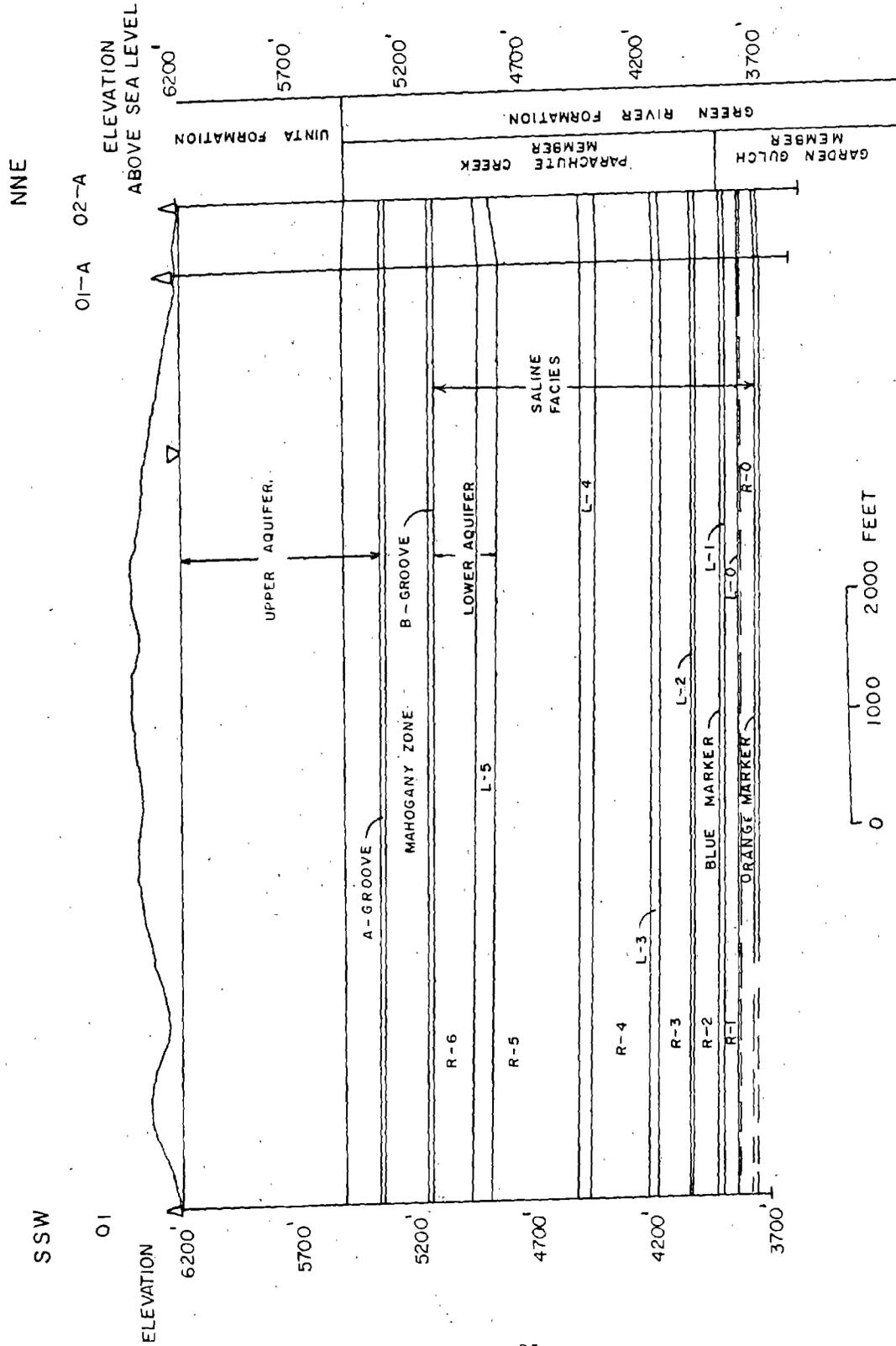


FIGURE 2.3

GEOLOGIC SECTION OF THE HORSE DRAW SITE

LEGEND

- ▲ — SURFACE CORE HOLE
- ▽ — WATER TABLE
- R-4, ETC. — RICH ZONE
- L-4, ETC. — LEAN ZONE

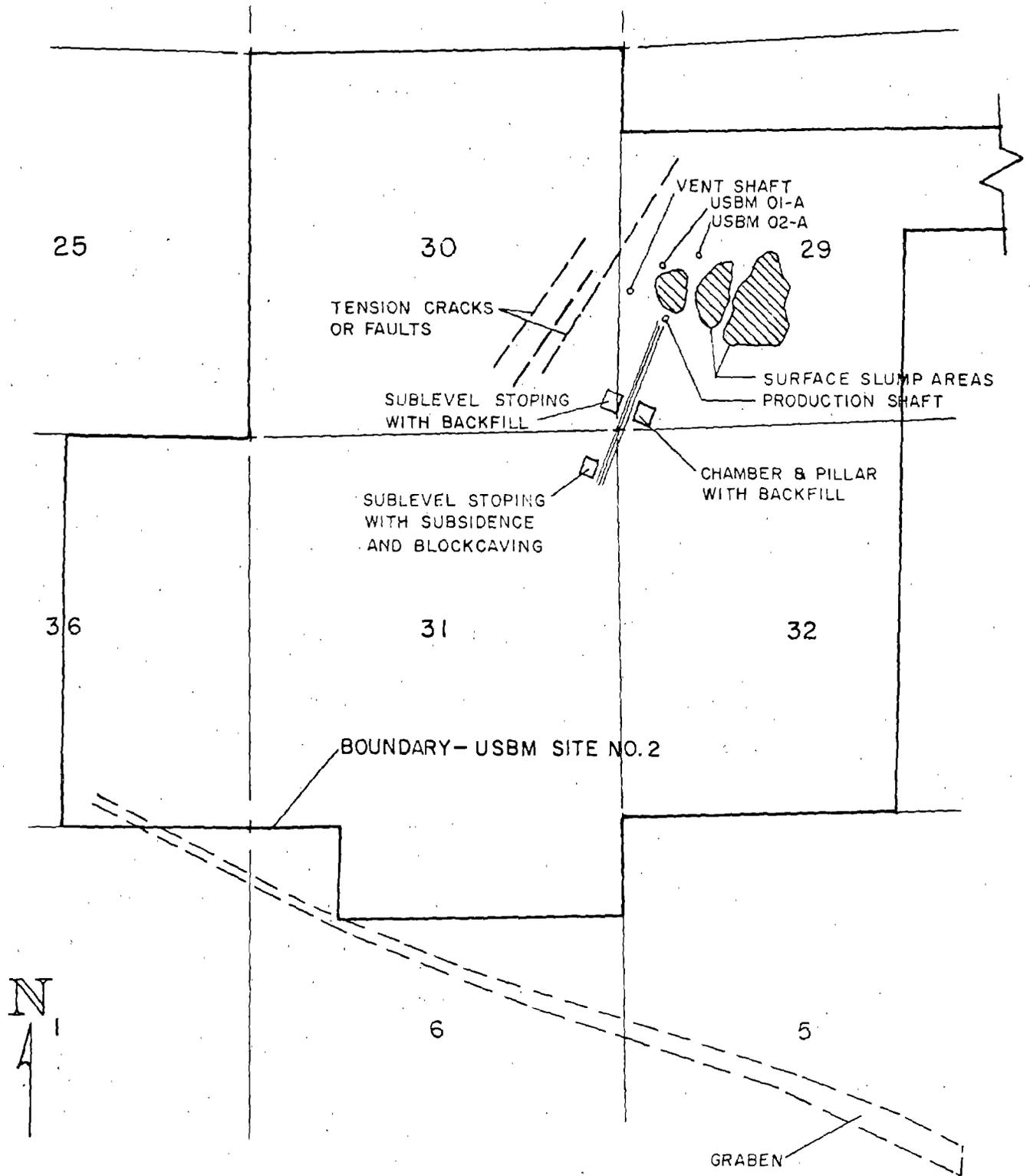


FIGURE 2.4

GEOLOGIC STRUCTURE AND MINE ORIENTATION

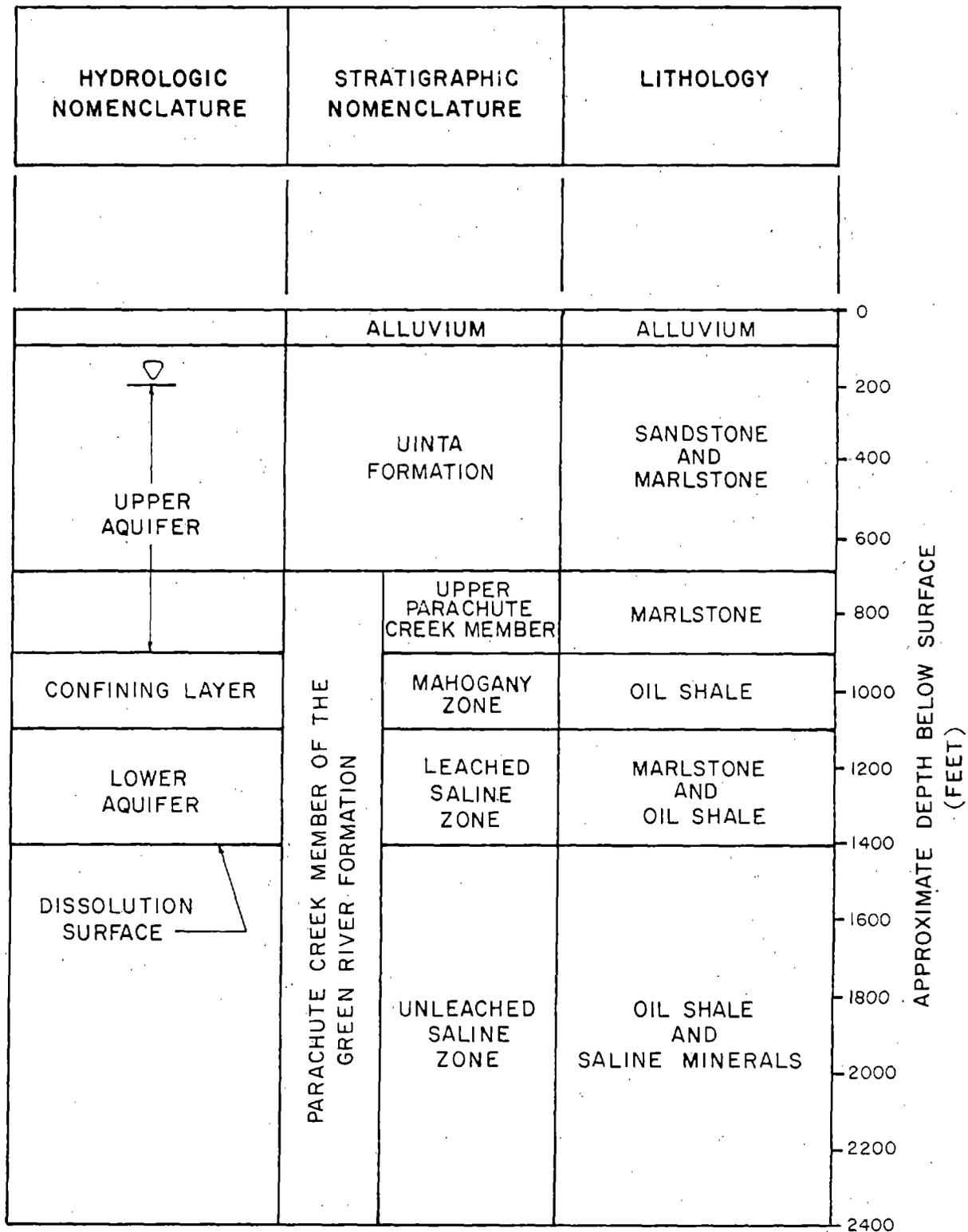


FIGURE 2.5
 GEOHYDROLOGIC MODEL OF HORSE DRAW SITE
 PICEANCE CREEK BASIN, COLORADO

consists mainly of sandstone and marlstone. A confining layer, composed of a rich oil shale zone approximately 200 feet thick, separates the two aquifers. The lower aquifer is about 300 feet thick and consists of marlstone and oil shale that have been leached of soluble minerals. The underlying unleached saline zone is more than 800 feet thick and consists of oil shale and saline minerals. This zone also is referred to as the high-resistivity zone and is virtually impermeable (3).

Four holes were drilled and tested at the demonstration mine site to develop a geohydrologic model. The first hole (USEM 01) was drilled at Fault Draw, south of Horse Draw. The hole ultimately served as an upper aquifer observation well only. Holes 01-A and 02-A subsequently were drilled at Horse Draw. Hydraulic heads in the upper and lower aquifer were monitored in 01-A during pump tests of 02-A. Finally, Pilot Hole X was drilled to provide site-specific hydrologic data for the bored shaft.

For the upper aquifer a value of transmissivity of 2,600 square feet per day and a storage coefficient of 2.5×10^{-3} , were determined (Hole 01-A). The storage coefficient is similar to the value used in the development of a regional ground-water model by Weeks et al., (8). Based on observations of drawdown in the pumped well (Hole 02-A), transmissivity of the lower aquifer is estimated to be 210 square feet per day.

Water in the upper aquifer and in the upper part of the lower aquifer has a concentration of dissolved constituents of about 1,500 mg/L and a specific conductance of about 2,000 micromhos per centimeter at 25°C. This is about the same dissolved-solids concentration as was measured during periods of low flow at the confluence of Horse Draw and Piceance Creek (8). Near the base of the lower aquifer the section contains sufficient soluble minerals to exhibit an increase in the specific conductance of water with depth. Shales in the unleached saline zone are virtually impermeable and water in this zone apparently is highly saline.

2.3 ROCK PROPERTIES DATA

Core from drill Holes 01-A and 02-A at the Horse Draw site was tested and analyzed to derive information on rock quality, strength, and creep characteristics. In addition to lithologic logs, core index logs were compiled to define rock quality. Core index defines rock quality by relating the length of broken and lost core and the number of joints found in the core to the drilled interval. Geophysical logs run in the holes included caliper, temperature, self-potential, resistivity, gamma ray, density, neutron, and 3-D velocity. A lithologic log of the proposed demonstration mining interval is presented in Figure 2.6, and oil shale assays of the interval are given in Figure 2.7.

To obtain rock strength parameters, laboratory tests of selected intervals of core from Holes 01-A and 02-A were performed. The types of tests performed

USBM 01-A
COLLAR EL. 6236'

USBM 02-A
COLLAR EL. 6224'

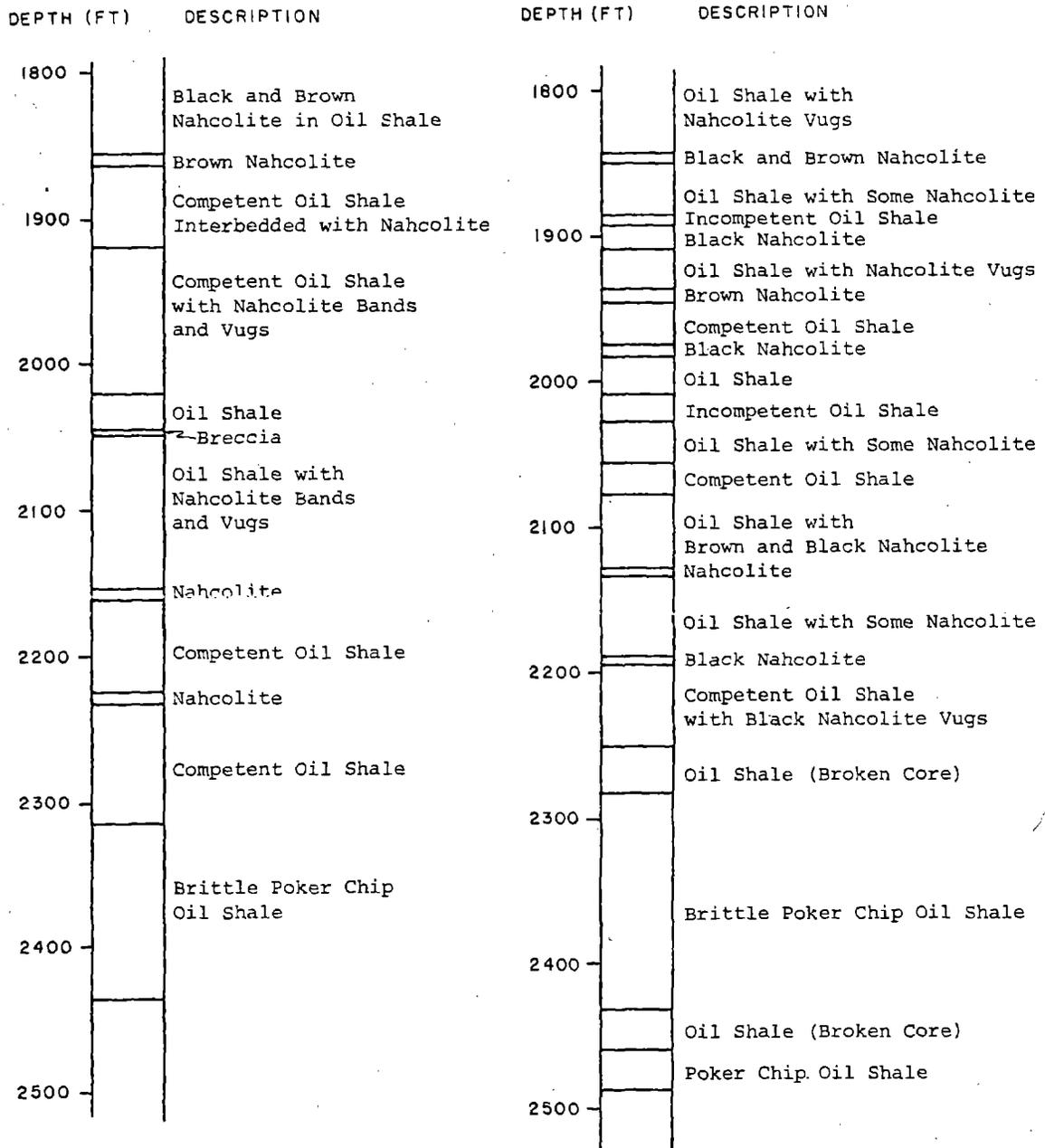


FIGURE 2.6
CORE LOG OF THE ZONE OF INTEREST

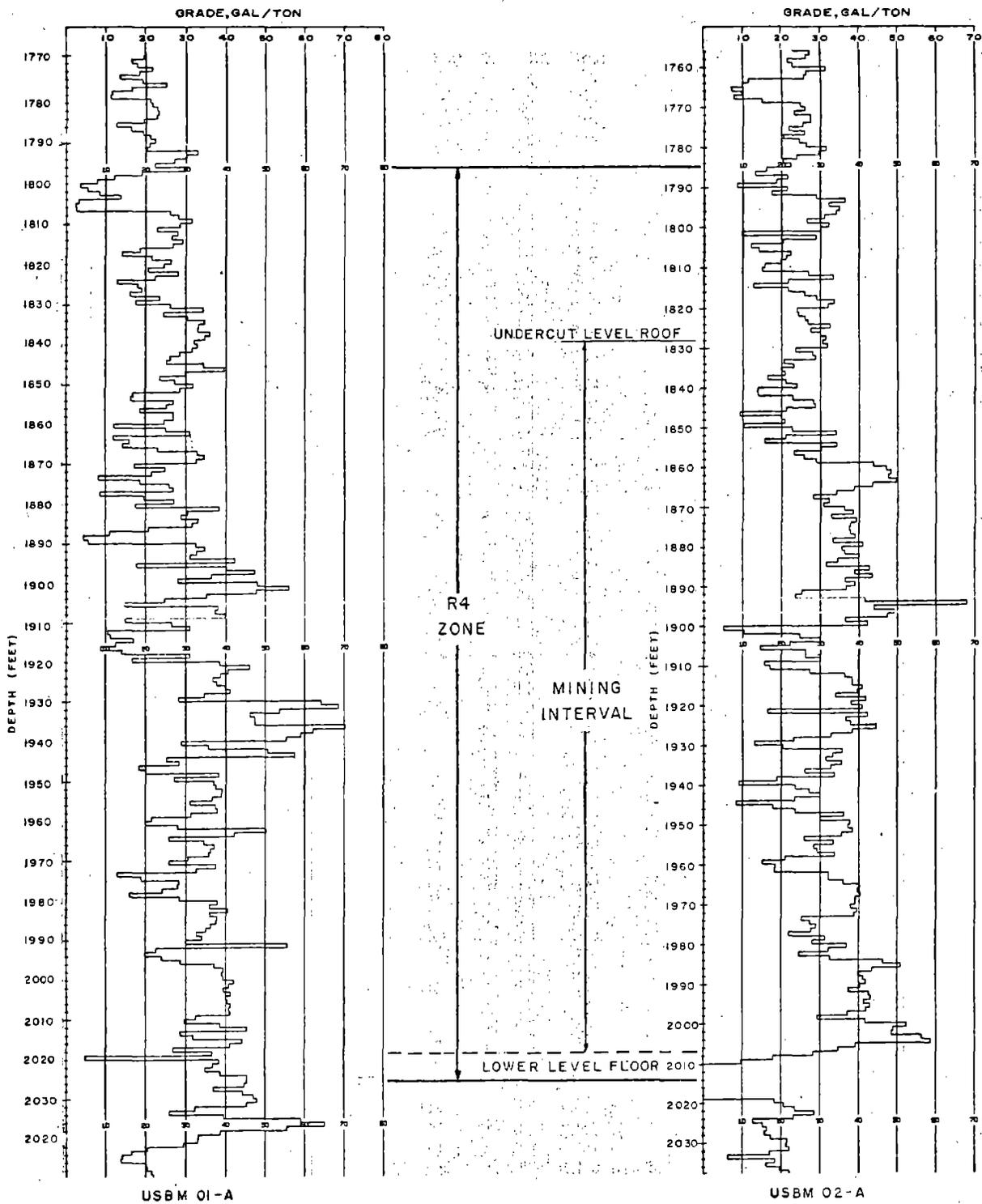


FIGURE 2.7
OIL SHALE ASSAYS

include uniaxial and triaxial compression, indirect tension (Brazilian), modulus of rupture, ultrasonic velocity, and creep. Average rock strength parameters determined by testing are presented in Table 2.1. Compressive strengths vary from a low value of 5,755 psi for the R-3 zone to a high value of 14,545 for the Mahogany Zone. Indirect tensile strengths vary from 643 psi for the R-3 zone to 1,570 for the Mahogany Zone. Ratios of the modulus of rupture to indirect tensile strength vary from about 3 to 6 in the L-2, R-3, and L-3 zones. Creep rates vary from about 23 microinches to approximately 360 microinches per inch per day, and are directly proportional to oil content and applied stress level.

2.4 DESIGN OF ROOF SPANS AND PILLAR DIMENSIONS

The dimensions of roof spans and pillars were determined using elastic beam theory and tributary area theory. Laboratory rock strength data used in design calculations are presented in Table 2.1. Laboratory and field tests have shown that the strength of pillars composed of dissimilar rock units usually is more nearly equal to the average strength of the units than to the strength of the weakest unit (5). Pillars were designed to have safety factors between 1.2 and 4, and roof spans were designed to have safety factors between 2.4 and 8. The design safety factor was modified in proportion to the required duration of stability of a specified opening. Openings which must remain serviceable for only a few months (i.e., openings to be back-filled) were designed with smaller safety factors than permanent openings. Determinations of the allowable dimensions of roof spans and pillars for the four demonstration units were based on rock properties as well as operational requirements. The effect of blast damage on pillar strength was considered in determining pillar dimensions; however, it has been assumed that a method of controlled blasting will be practiced. The effect of depth was considered in computing the dimensions of roof spans and pillars.

2.5 POTENTIAL INFLOWS RESULTING FROM INDUCED SUBSIDENCE

To predict the magnitude of seepage which could occur from demonstration of the block caving system, an analysis of rock mass disturbance as a result of block caving was performed. Rock strength data obtained from core specimens from Holes 01-A and 02-A were used to construct a model of the rock mass at the Horse Draw site (Figure 2.8).

Three cases were considered regarding the potential extent of caving.

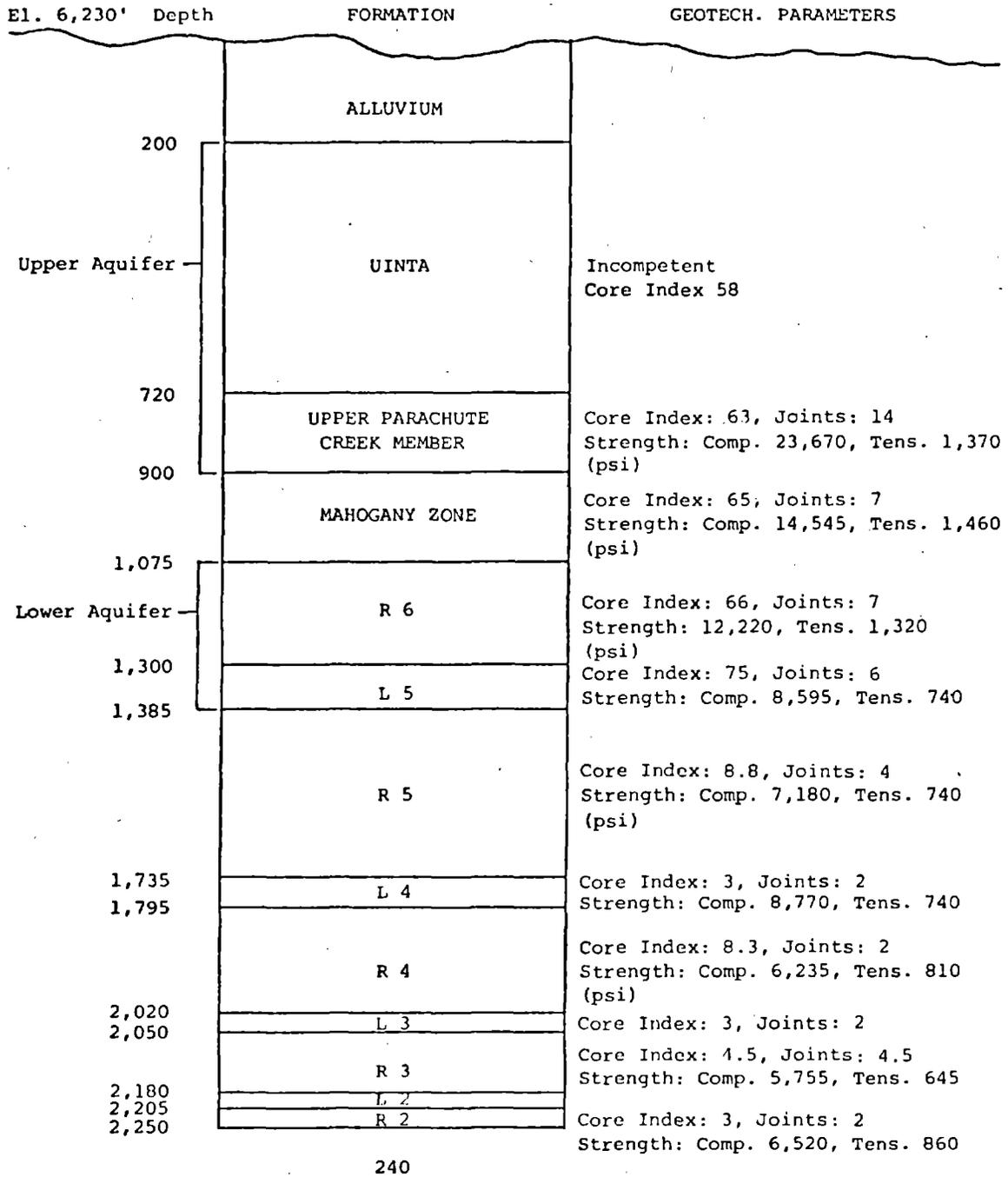
Case 1 (Best): By regulating the withdrawal of ore from the chamber, the caving zone will be induced to propagate upward to a predetermined horizon below the dissolution surface. When the horizon is intersected, ore withdrawal will be terminated resulting in cessation of caving. Rock mass disturbance will be confined to a zone surrounding the cave, and fractures induced within this zone will not extend upward into the lower aquifer.

TABLE 2.1

AVERAGE ROCK STRENGTH PARAMETERS FOR THE HORSE DRAW SITE *
 PICEANCE CREEK BASIN, COLORADO
 (From USIM Core Holes 01-A & 02-A)

Zone	Depth (feet)	Core Index (%)	Density (pcf)	Compressive Strength (psi)	Static Young's Modulus (x 10 ⁶ psi)	Static Poisson's Ratio	Indirect Tensile Strength (psi)	Modulus of Rupture (psi)	Friction Angle (deg)	Cohesion (psi)	Creep (μ in/in /day)
Overlying Strata	0- 890	58	144	23,670	2.48	0.21	1,370	-	-	-	-
Mahogany	890-1,080	65	128	14,545	1.19	0.24	1,570	-	-	-	-
R-6	1,100-1,290	63	120	12,220	1.12	0.27	1,320	-	-	-	-
R-5	1,380-1,730	11	129	7,175	0.95	0.31	740	3,820	-	-	78.7
L-4	1,730-1,790	3	133	8,770	1.27	0.29	731	-	-	-	22.9
R-4	1,790-2,020	8	123	6,233	0.55	0.27	812	3,240	40	1,167	135.3
L-3	2,020-2,060	3	116	5,862	0.34	0.36	857	3,540	43	1,412	-
R-3	2,060-2,195	4	127	5,755	0.63	0.26	643	2,840	37	1,177	99.9
L-2	2,195-2,215	4	130	4,648	0.59	0.18	780	3,370	-	-	143.4
R-2	2,215-2,305	18	126	6,518	0.45	0.28	860	-	38	1,377	358.9
R-1	2,325-2,385	50	138	7,952	0.96	0.18	1,037	-	20	2,798	-

* Testing by Twin Cities Mining Research Center



NOTE: Core Index greater than 50 indicates incompetent rock
Most joints dip at steep angles (greater than 65°)

FIGURE 2.8
GEOTECHNICAL MODEL FOR BLOCK CAVING ANALYSIS

This case represents an idealized condition which may be achieved if caving and fracturing occur as assumed. Subsidence would not be detectable on the surface.

Case 2 (Anticipated): Controlled ore withdrawal will be practiced as in Case 1. Extraction of ore will be terminated when the caving zone reaches a predetermined horizon below the dissolution surface. However, settlement of overlying beds will continue until pressures inside and outside the cave come to equilibrium. Fracturing may propagate far enough above the cave to intersect the lower aquifer (Figure 2.9). However, fracture propagation could proceed very slowly, depending on the dimensions of the cave and the properties of the rock mass. A minimal, detectable amount of surface subsidence is anticipated as a result of the Case 2 condition.

Case 3 (Worst): Efforts to control the upward extension of caving will prove to be inadequate and caving will continue beyond the predetermined horizon after ore withdrawal is terminated. Conceivably, caving could progress upward to the lower aquifer, and eventually to the upper aquifer. Uncontrollable proliferation of caving would induce large inflows of ground water into the mine and could lead to extensive surface subsidence.

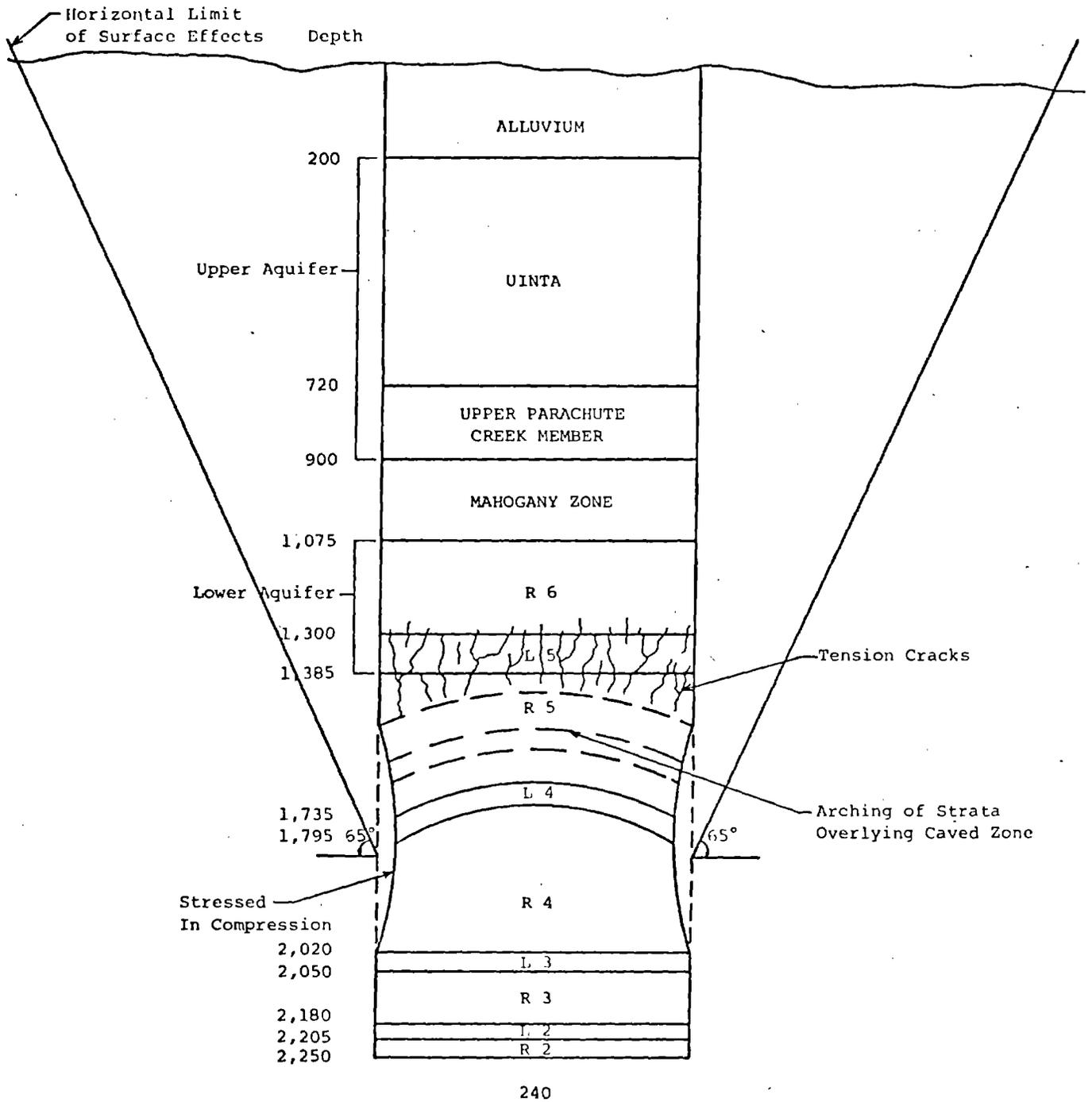
Estimates of water inflow rates for the three assumed conditions of rock mass disturbance were prepared by Golder Associates, (4). A geohydrological model was constructed for the Horse Draw site, utilizing the rock mass model discussed above and hydrologic data compiled from pump tests of Holes 01-A, 02-A, and Pilot Hole X. The Golder study was confined to the analysis of expected inflows into the block caving demonstration unit of the demonstration mine. A number of simplifying assumptions were made to facilitate the analysis as follows:

Restatement of the cases in simplified form (Figure 2.10)

Case A: Inflow is due to a cave whose upward penetration is almost to the base of the lower aquifer.

Case B: Inflow is due to a cave which fully penetrates the lower aquifer but which stops at the base of the Mahogany Zone.

Case C: Inflow is due to a cave which penetrates both aquifers to the level of the water table.



EXPECTED EFFECTS RESULTING FROM THE BLOCK CAVING AND SUBLEVEL STOPPING WITH FULL SUBSIDENCE METHODS.

FIGURE 2.9
ANTICIPATED FRACTURE PROPAGATION ABOVE CAVE ZONE

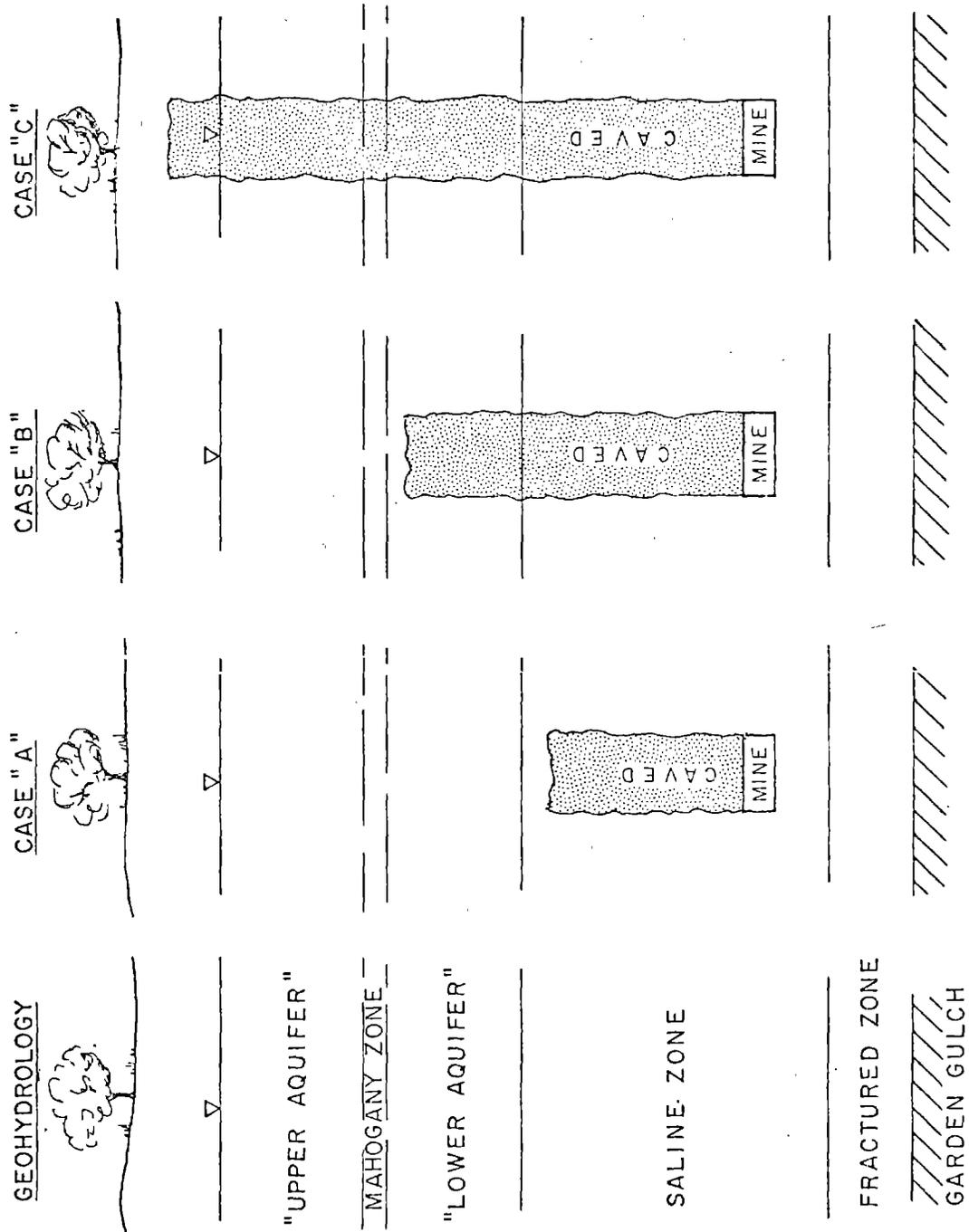


FIGURE 2.10
CAVING CASES CONSIDERED IN ANALYSIS
 (After Golder Associates (4))

Assumptions made in the analysis:

1. The cave will have the shape of a "pipe" of constant cross-sectional area.
2. Caved material will have a permeability of 30 feet per day (10^{-2} cm/sec.), which is equivalent to a highly fractured rock mass.
3. The mine will be at atmospheric pressure throughout.
4. Flow in all undisturbed strata is to be predominately horizontal, whereas in the caved zone flow will be vertical.
5. Inflow due to drainage from caved material is ignored, and caving will be instantaneous.
6. The materials in each layer of the geohydrologic model are homogeneous; that is, free from large local defects, faults, barriers, or major water conduits.

In each case the flow computation was performed as follows:

- a. Flow from the "aquifers" tapped by the caved materials is computed assuming that all caved material is at atmospheric pressure. The result is the maximum flow which can occur, ignoring flow due to pore space water release from within the caved material itself.
- b. The flow which can occur in a pipe whose diameter is that of the cave and which is filled with rubble of 30-feet-per-day permeability is computed, assuming that it drains under gravity (with unit hydraulic gradient). This defines the limit of the influence of head loss on total flow in the caved pipe.
- c. The maximum flow which can occur in the caved pipe when it is subject to the full static head of the formation is computed. This defines the upper limit of flow to the mine. (It turns out to be unimportant except for a very small caved-pipe radius.)

- d. The actual flow is computed for the range in which resistance of the pipe is a significant factor, and the results are graphed.

Inflow rates were computed at one hour after caving (which approximates the maximum inrush) and at 19 days after caving (approximate steady-state flow). Aquifer flows were computed from the standard constant-drawdown aquifer equations (7), and the flow in the caved pipe was computed from the standard Darcy flow equation. Results of analyses for the three cases are presented in Figure 2.11.

A number of conclusions were presented in the Golder study regarding the influence of mine design parameters on the potential severity and control of inflows to the mine.

- a. Each case differs by about one order of magnitude. Thus, controlled caving in the demonstration mine appears to be of critical importance.
- b. Flow rates, particularly early flow rates, are strongly influenced by the radius of the caved area. Thus, this factor is worthy of investigation in the demonstration mine.
- c. Sudden inrush flow rates are about double the steady-state, postcaving flows, suggesting that emergency pump capacity can be reduced by the provision of adequate water storage capacity in the mine.

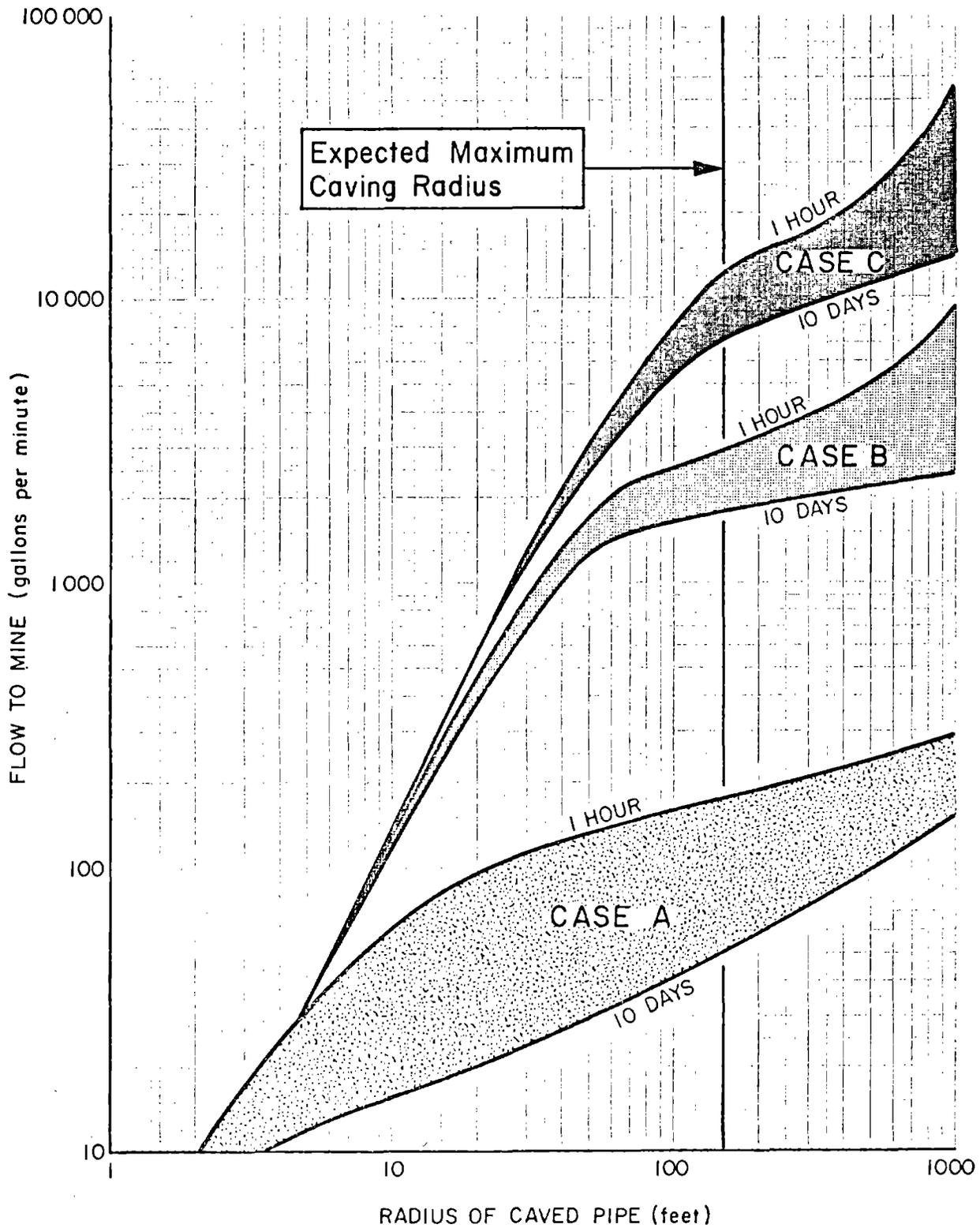


FIGURE 2.11
 FLOW TO MINE-SUMMARY
 (After Golder Associates (4))

3.0 MINE LAYOUT

3.1 SCALING DOWN OF THE FOUR MINING SYSTEMS

Conceptual designs of four specified mining systems were proposed to the U. S. Bureau of Mines for demonstration mining in a report prepared by Cameron Engineers (1). Plans and specifications for the commercial-scale designs of the four mining systems have been modified and scaled down for demonstration purposes. The main objective is to demonstrate the practicability and safety of the commercial designs, and to accomplish the task in the most economical fashion.

3.1.1 Functional Criteria

For the chamber and pillar mining with backfill unit and the sublevel stoping with backfill unit a minimum of three chambers and three stopes will be demonstrated. A unit of three openings (chambers or stopes) is the required minimum number to test the operational feasibility of stabilizing narrow pillars between openings using an alternating sequence of excavation and backfilling. If stress-strain data from the rib pillars indicate that anticipated loading conditions have not been achieved, an additional two chambers or two stopes can be mined to gain a more accurate representation of pillar stress conditions to be anticipated in commercial-scale mining operations.

The sublevel stoping with full subsidence unit will require two stopes for adequate demonstration. An essential aspect of proving the operational feasibility of this stoping method will be to demonstrate the practicality of secondary mining of pillars left during stope development. After the two stopes have been mined, crown and rib pillars will be blasted into the stope cavities. If the mining of one rib pillar fails to provide sufficient data to demonstrate the method adequately, contingency plans have been provided for excavating a third stope.

The layout of two stopes in the sublevel stoping with full subsidence unit was sized to match the dimensions of the block caving unit. The block caving unit has been located immediately above the sublevel stoping unit. Thus, the caved ground resulting from development of the block caving unit will be representative of caved overburden normally found above a full subsidence mining system and the drawpoints developed for the stopes also will serve as drawpoints for the caving system. The area affected by subsidence induced by the two systems will be minimal.

Since a block caving system has never been implemented to mine oil shale, the dimensions of the cave area required to amply demonstrate the method can only be estimated. The initial undercut will measure 225 feet by 240 feet, with contingency plans to expand the area to 300 feet by 365 feet if necessary. The initial dimensions of the block caving unit were established with the objective of minimizing the area of overburden to be affected by subsidence.

3.1.2 Operations Criteria

Operational criteria related to hoisting and ventilation capacities and the mine environment were stipulated by the Bureau of Mines. One 8-foot-diameter ventilation shaft and one 20-foot-diameter production and service shaft, with a hoisting capacity of 7,500 tons per 24-hour day, were specified. Gassy mine conditions, high rock temperatures, and permissible diesel-powered underground equipment also were specified.

Equipment that can be disassembled and lowered down a relatively small shaft was selected. Equipment size also was dictated by the necessity to operate in moderately small drifts. The ventilation system for each demonstration unit was designed to provide a safe working environment under gassy mine conditions and to conform to or exceed State and Federal standards.

3.1.3 Economic Criteria

Proper design and successful operation of the demonstration mine should provide results that can be applied to subsequent design and operation of commercial-scale mines. As stated before, the demonstration units were designed to minimize overall mining costs. Drifts were sized to minimize excavating costs, and the scale of each method to be demonstrated was reduced to an operable minimum consistent with the objective of obtaining reliable engineering data for commercial-scale mine design. A major reason for placing the block caving unit directly over the sublevel stoping with full subsidence unit was to minimize costs of demonstrating these two mining methods.

3.2 VERTICAL AND HORIZONTAL MINE LAYOUT

Four mining systems are to be demonstrated on a restricted scale within the lower (unleached) saline oil shale interval in the Parachute Creek Member of the Green River Formation. Initially, technical cost-benefit ratios were considered for single-level versus multiple-level demonstration mine layouts. The single-level layout concept was chosen to facilitate completion of the demonstration program in a cost-effective manner.

3.2.1 Factors Affecting Vertical Location

Two stratigraphic intervals within the unleached saline zone were considered for location of the single-level demonstration mine. These intervals are shown in Figure 3.1, labeled "Preliminary Mining Levels I and II." Detailed evaluation of geologic parameters indicated that Level I was most suitable for vertical location of the demonstration mine. Selection was based on thickness and grade of oil shale, persistence of nahcolite and dawsonite occurrences, rock temperatures, and site-specific physical rock characteristics and hydrologic conditions.

Thickness, oil shale grade, and assay values for nahcolite and dawsonite in the saline zone are listed in Table 3.1. The demonstration mine

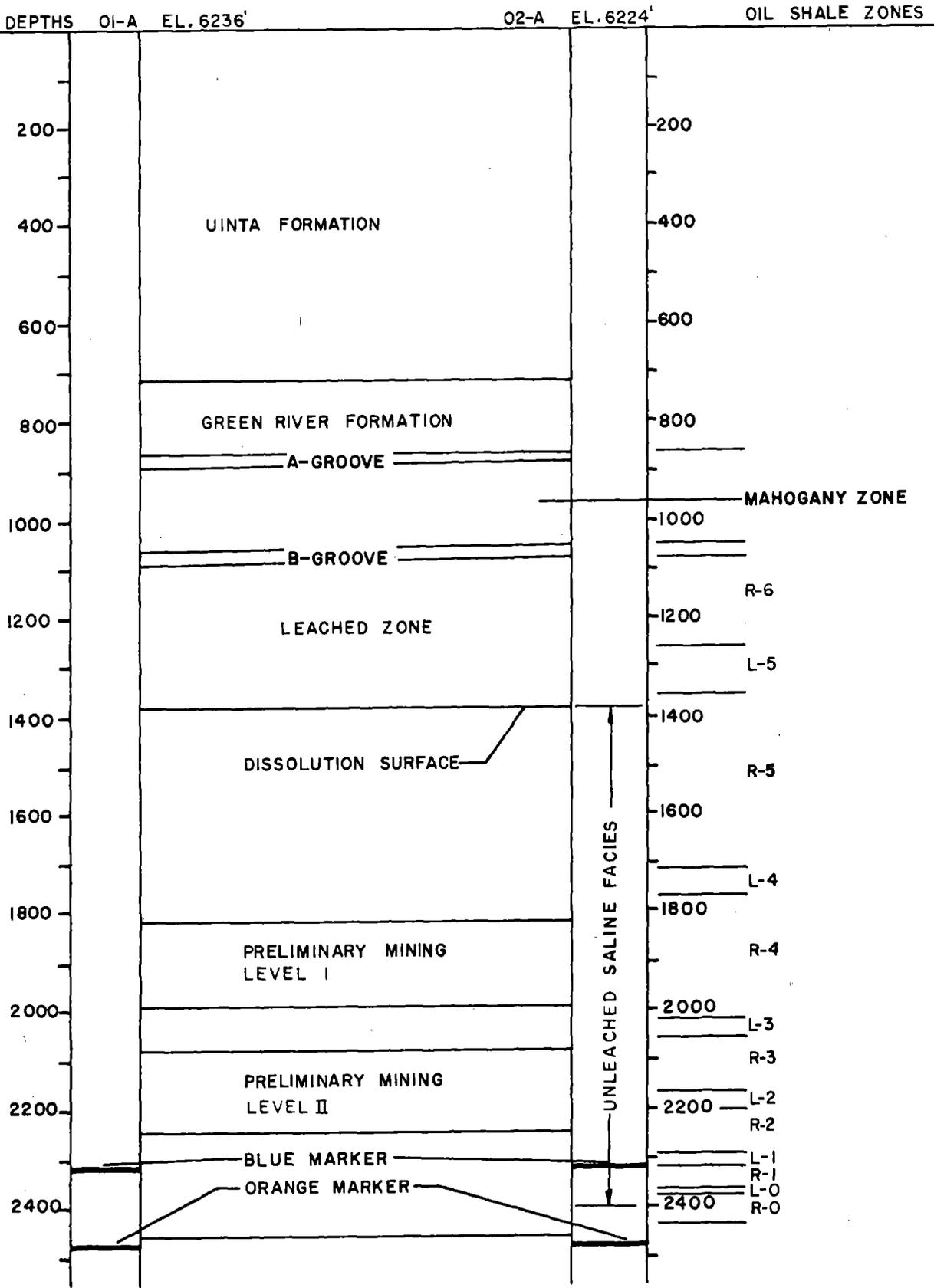


FIGURE 3.1
PRELIMINARY MINING LEVELS

TABLE 3.1

SALINE FACIES - ASSAY DATA*

Zone	Thickness (ft)	Hole 01-A			Hole 02-A		
		Oil Content (gpt)	Nahcolite (wt %)	Dawsonite (wt %)	Oil Content (gpt)	Nahcolite (wt %)	Dawsonite (wt %)
R6	190	21.40	-	-	25.82	-	-
L5	90	17.38	-	-	17.29	-	-
R5	350	22.22	-	-	23.87	-	-
L4	60	22.27	20.36	6.72	20.91	19.18	6.90
R4	232	30.12	20.96	6.74	30.58	17.65	7.48
L3	38	28.39	6.45	5.85	18.83	23.47	6.73
R3	130	27.30	28.81	6.45	26.43	23.34	6.51
L2	20	26.73	3.43	5.46	22.03	17.96	6.64
R2	90	31.42	9.57	3.25	33.28	2.20	1.98
L1	20	26.92	0.20	1.80	8.98	0.70	1.91
R1	60	14.17	2.57	4.27	27.33	0.00	1.03
L0	10	34.29	0.23	2.59	16.78	0.30	3.41
R0	50	24.49	1.11	3.80	25.92	2.00	4.04

* Oil yield assay data from Laramie Energy Technology Center
 Nahcolite and dawsonite content determined by gas-loss method (6)

will be located in the R-4 zone, which is representative of thick, rich, deep oil shale deposits. This zone contains sufficient concentrates of nahcolite and dawsonite to permit mining of bulk samples for testing and evaluation.

Physical properties data of selected intervals of core from the saline facies penetrated by Holes 01-A and 02-A are summarized in Table 2.1. Physical rock properties are relatively uniform through the unleached saline facies. Core index values indicate that the most competent rock within the interval occurs between the R-5 zone and the middle of the R-2 zone.

Hydrologic information obtained from Holes 01-A and 02-A indicates that a prolific aquifer is present above the Mahogany Zone and a second, less prolific aquifer is present in the leached interval of the saline facies. Because permeability in the unleached saline facies is very low, inflow of water into the demonstration mine located beneath the aquifers will be very small so long as mining operations are of a noncaving nature. Two of the four mining methods to be demonstrated will induce caving of strata above the mining interval. Consequently, a relatively thick barrier between the mine level and the dissolution surface will be required. Location of the demonstration mine in the R-4 zone will permit the retention of an approximate 400-foot vertical barrier between the mine and the lower aquifer.

Rock temperatures measured in Holes 01-A, 02-A, and Pilot Hole X are plotted against depth in Figure 3.2. Anticipated temperatures around openings in the demonstration mine will average about 90°F. Higher temperatures would be encountered if the lower horizon were selected for mining.

3.2.2 Horizontal Mine Layout

The horizontal layout of the demonstration mine was planned in three stages:

- (1) Location of the main (production and service) shaft
- (2) Orientation of access drifts
- (3) Overall mine layout

3.2.2.1 Main Shaft Location

The location of the 20-foot-diameter production shaft was dictated, in part, by the previously established position of the ventilation shaft. Other constraints and considerations imposed on the production shaft location were surface and subsurface geology, the proximity of an ore stockpile area, governmental regulations and rock mechanics criteria which established a minimum distance of separation of the two shafts, the relationship to mine orientation and layout, and the desire to minimize environmental disturbance.

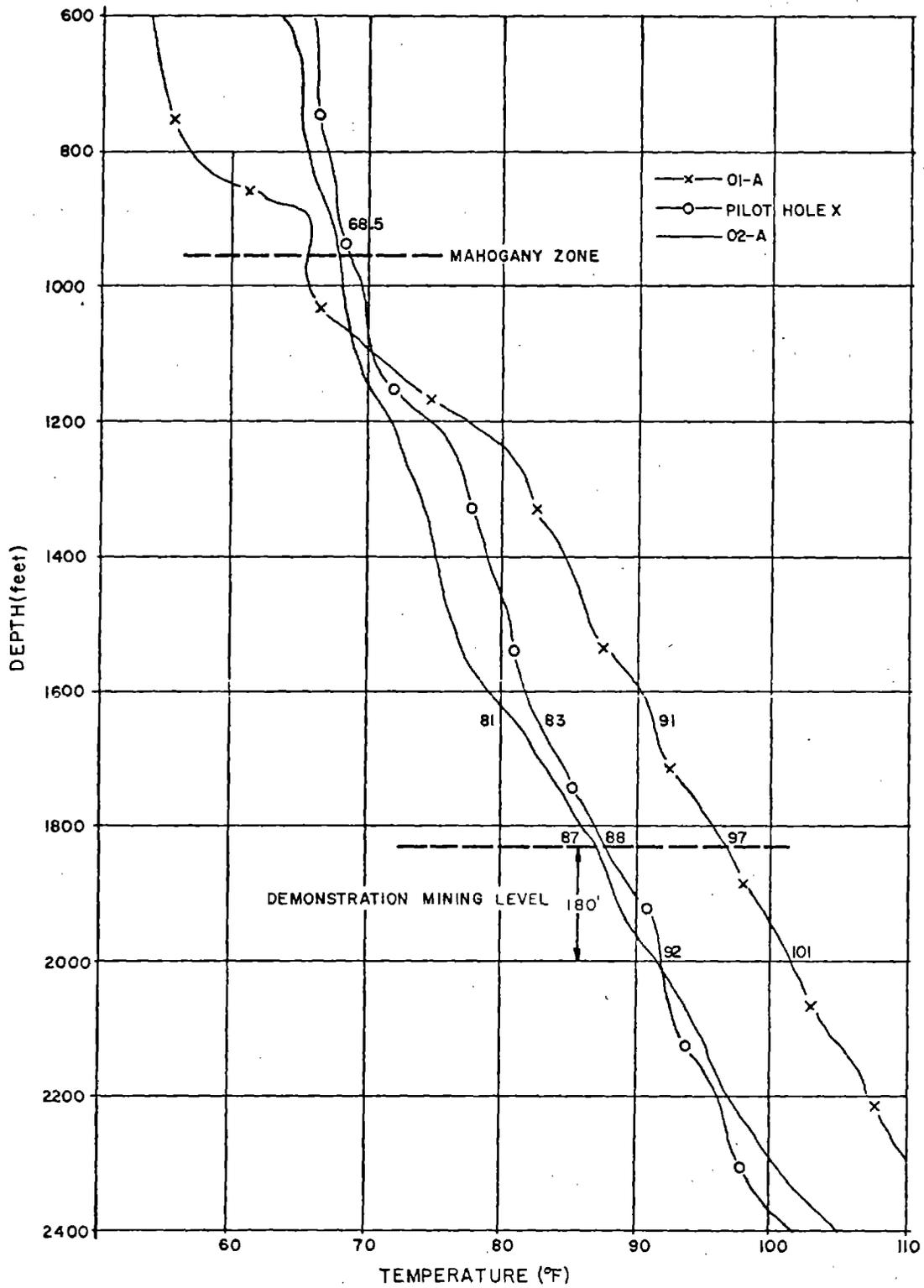


FIGURE 3.2
SUBSURFACE ROCK TEMPERATURES

The production shaft will be located 604 feet from the ventilation shaft on a bearing of S60°59'E. Coordinates of the shaft site are 230,247N, 1,211,875E, with a surface elevation of 6,298.1 feet. The production shaft has been located centrally on a large existing storage pad, adjacent to the ventilation shaft area. Utilization of this storage area will minimize further surface disturbance and will allow adequate space for locating ancillary structures and facilities required for shaft sinking and subsequent mining activity.

3.2.2.2 Orientation of Access Drifts

Mine orientation has been chosen to maximize the long-term stability of the main access drifts by orienting them normal to the direction of maximum horizontal stress and to the strike of the dominant joint set. In situ stress determinations have not been made at the demonstration mine site. However, the direction of maximum horizontal stress is thought to be aligned with the trend of the graben, approximately N70°W, which exists to the south of the Horse Draw site (Figure 2.4). In situ stress determinations by Wolff et al., (9) appear to confirm this preliminary assessment.

The main access drifts will be driven S20°W from the production shaft, as indicated in Figure 2.4. Areas of questionable ground stability to the northeast and northwest of the two shafts will be avoided. Driving the access drifts on this bearing will also facilitate mine drainage, as the openings will be extended slightly up-dip parallel to bedding from the production shaft.

3.2.2.3 Overall Mine Layout

Factors considered in design of the horizontal demonstration mine layout include:

- Dimensioning the unmined shaft pillar to provide adequate protection of the shaft.
- Segregation of units employing backfilling from caving systems in order to confine the area affected by subsidence.
- Input of sufficient flexibility in the layout design to permit either sequential or concurrent modes of mine development and operation.

A 1,200-foot-radius shaft pillar is planned to separate the production shaft from demonstration units employing backfilling. This dimension will accommodate an angle of draw of 60° above these units. A

2,160-foot radius will separate the shaft from units of unrestrained subsidence, based on a draw angle of 45°. Backfilling units will be separated from caving units by a 400-foot barrier pillar.

The mine layout will permit a combination of sequential and concurrent activities in the four demonstration units. The anticipated limiting factors for concurrent development and production mining are the specified ventilation and hoisting capabilities of the plant. To maximize operating efficiency and economy, a demonstration mining schedule incorporating sequential and concurrent development and operation of the demonstration units, subject to ventilation and hoisting constraints, is planned.

4.0 DEMONSTRATION MINE DESIGN

The proposed layout of the demonstration mine is illustrated in Figure 4.1. Access to the demonstration areas will be developed on three levels, as shown. Shaft stations will be constructed on the upper and lower mine levels. By driving an exhaust entry between each shaft station and the ventilation shaft, separate ventilation circuits will be established on the two levels. All dewatering and hoisting functions will be performed from the lower level station area. Multiple entry development has been projected throughout the mine to conform with Federal regulations for gassy mine conditions.

A 1,200-foot barrier will separate the production shaft from the closest demonstration mining areas. Methods to be demonstrated nearest the shaft (chamber and pillar mining with backfill and sublevel stoping with backfill) are those which will offer the least potential for disturbance of the surface and shaft area environments. Mining methods which may produce significant surface subsidence (sublevel stoping with full subsidence and block caving) will be removed a distance of 2,160 feet from the production shaft to ensure shaft stability. As mentioned previously, the superposition of the block caving unit over the sublevel stoping with full subsidence area will minimize the total surface area affected by subsidence.

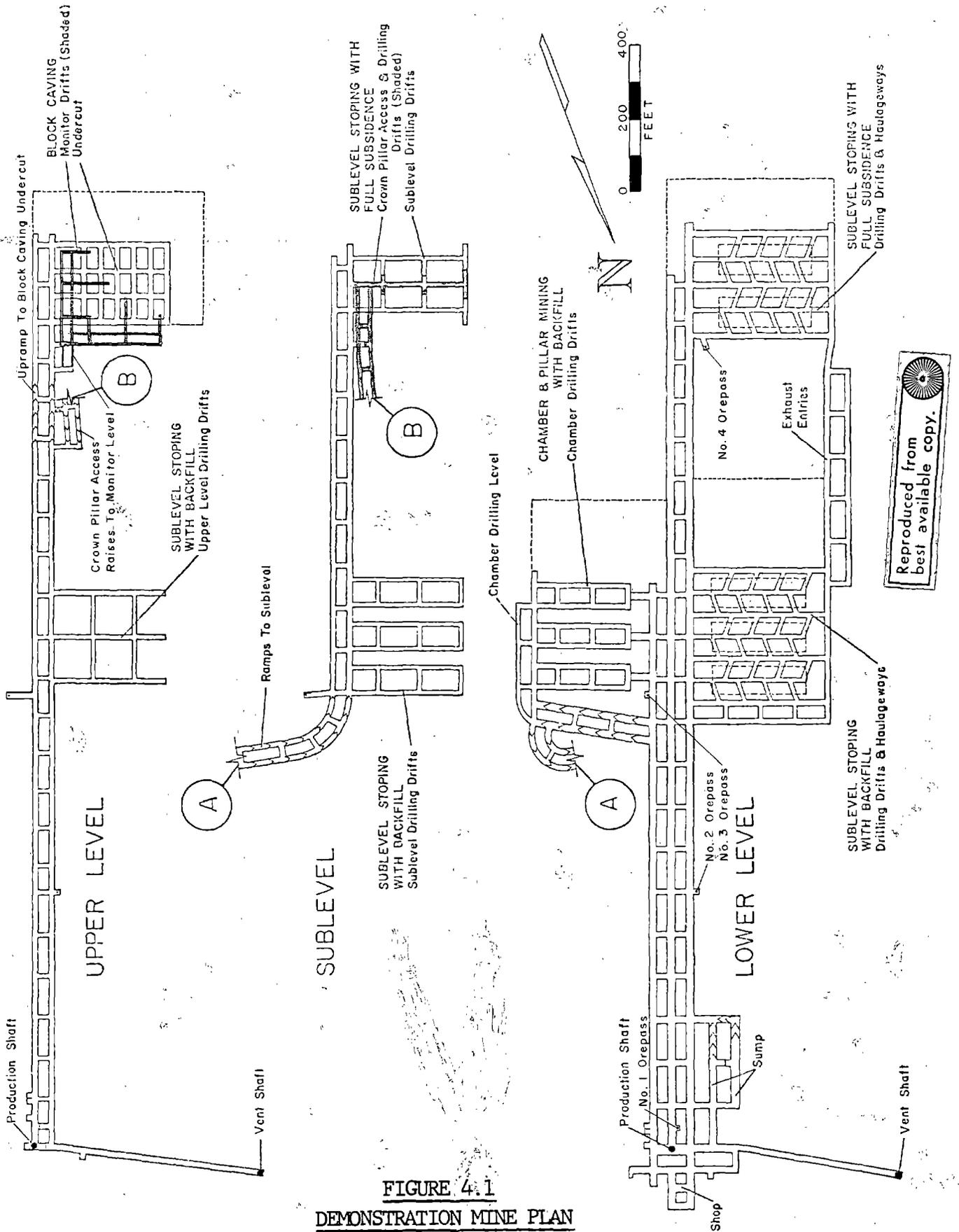
Shaft sinking and shaft equipping will be performed by others as defined in the Scope of Work. The shaft contractor also will excavate storage and skip-loading pockets and do sufficient drifting on both the upper and lower levels to provide working room for development start-up. Installation of station facilities such as skip loading, crushing, and shaft pumping equipment, will be completed prior to commencement of the demonstration mining program.

4.1 PRIMARY DEVELOPMENT

Primary development consists of excavating a number of openings in the vicinity of the upper and lower shaft stations for location of support facilities as well as driving drifts, ramps, and raises outward from the station areas to establish access to the demonstration areas.

Work classified under primary development will be performed in two separate stages (Figure 4.2). Stage I will include all development work to be completed prior to demonstration of the chamber and pillar with backfill and sublevel stoping with backfill systems (Figures 4.1 and 4.2). Stage I tasks include:

1. Driving single entries between the production and ventilation shafts to connect the two shafts on the upper and lower mining levels for ventilation.



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FIGURE 4.1
DEMONSTRATION MINE PLAN

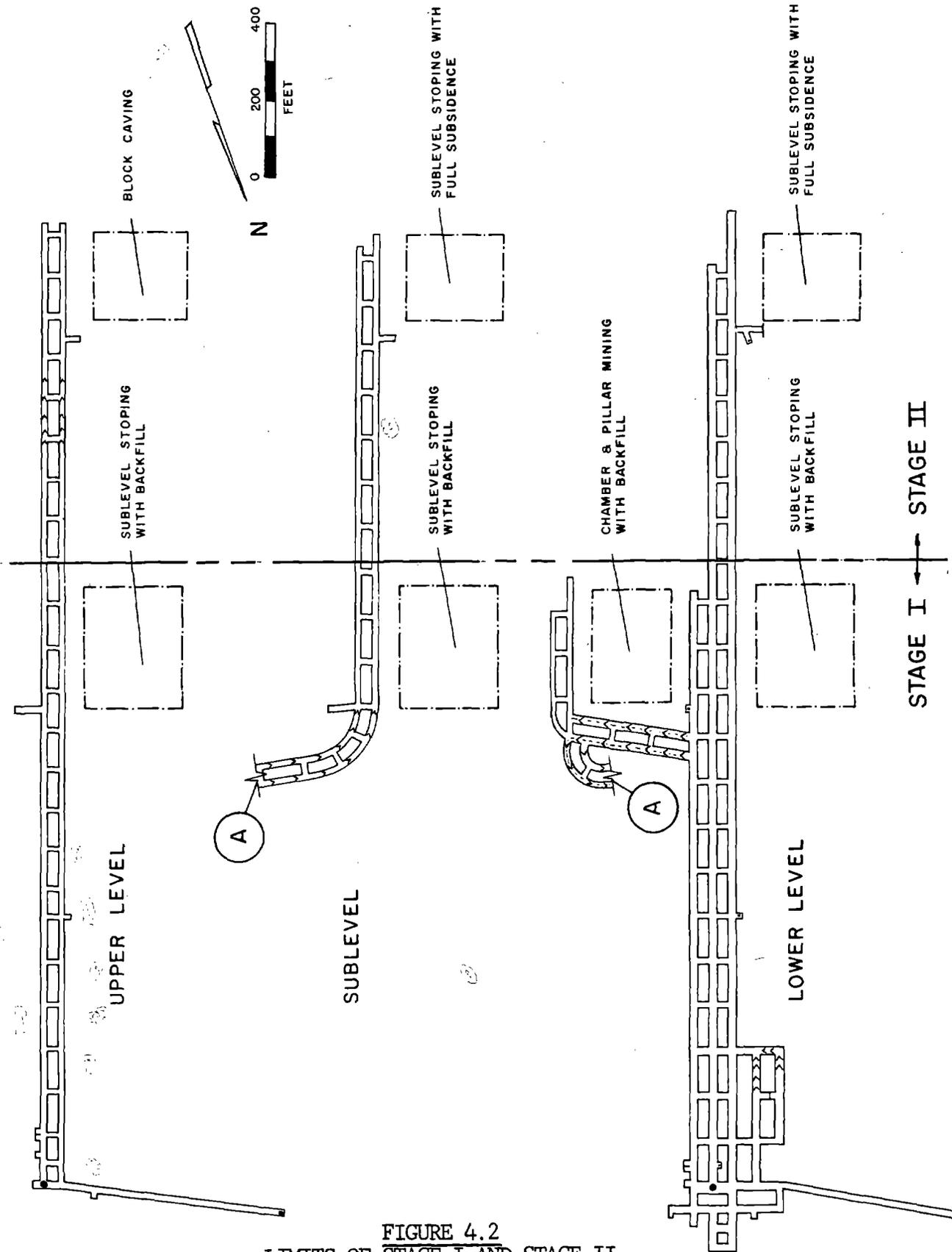


FIGURE 4.2
 LIMITS OF STAGE I AND STAGE II
 MAIN ENTRY DEVELOPMENT

2. Completing a number of openings around the upper and lower shaft station levels for location of station area facilities.
3. Extending main entries on the upper and lower mining levels from the shaft stations approximately 1,600 feet through the chamber and pillar and sublevel stoping with backfill demonstration areas.
4. Driving chamber drilling level and sublevel access ramps and drifts.
5. Boring three 52-inch-diameter orepasses from the lower to the upper mining level.

Stage II of primary development will be undertaken while development and mining of the first two demonstration units are in progress. Stage II development will provide access to areas reserved for demonstration of the sublevel stoping with full subsidence and block caving systems (Figures 4.1 and 4.2). Tasks to be performed during Stage II are as follows:

1. Extension of main entries approximately 800 feet on the lower level, sublevel, and upper level. Entries on the upper level will be ramped up 20 feet to the elevation of the block caving undercut level.
2. Extension of the exhaust entry serving the sublevel stoping with backfill unit to the area designated for demonstration of sublevel stoping with full subsidence.
3. Driving two parallel ramps downward from the upper level mains to the crown pillar drilling level.
4. Boring two inclined raises upward from the block caving undercut level to provide access to the two block caving monitor levels.
5. Boring a single orepass from the lower level to the upper block caving monitor level.

Table 4.1 summarizes dimensions and tonnages of all excavations to be completed during Stages I and II of primary development.

4.1.1 Lower Level

4.1.1.1 Stage I

The first task to be completed on the lower level will be to connect the ventilation and production shafts. A single 12-foot-high

TABLE 4.1

EXCAVATION SUMMARY
PRIMARY DEVELOPMENT

<u>Development Segment</u>	<u>Drift Size</u>	<u>Development Footage</u>	<u>Number Of Rounds</u>	<u>Tons Per Round</u>	<u>Stage I Tonnage</u>	<u>Stage II Tonnage</u>	<u>Total Tonnage</u>
Lower Level:							
Shaft Connection and Station Facilities	15' x 20'	1,800	180	207	37,260	-	37,260
	15' x 15'	280	28	155	4,340	-	4,340
	12' x 15'	560	56	125	7,000	-	7,000
Sump and Pump Room Main Entries	-	-	-	-	7,570	-	7,570
	15' x 20'	5,170	517	207	77,420	29,600	107,020
	15' x 15'	810	81	155	9,770	2,790	12,560
Chamber Drill Level Entries	12' x 15'	1,480	148	125	18,500	-	18,500
Exhaust Entry Extension	12' x 12'	1,390	139	100	-	13,900	13,900
Orepasses (4)	(52"-Diam.)	623	-	-	400	230	630
Sublevel:							
Ramp to Sublevel	12' x 15'	1,210	121	125	15,120	-	15,120
Main Entries	12' x 15'	2,690	269	125	10,370	23,250	33,620
Upper Level:							
Shaft Connection and Station Area	12' x 15'	1,020	102	125	12,760	-	12,750
Main Entries	12' x 15'	5,130	513	125	39,500	24,630	64,130
Crown Pillar Access	12' x 12'	1,070	107	100	-	10,700	10,700
Monitor Level Access	(52"-Diam.)	200	-	-	-	200	200
TOTALS					<u>240,000</u>	<u>105,300</u>	<u>345,300</u>

by 15-foot-wide entry will be driven between the two shafts for this purpose. While driving this entry, ventilation air will be supplied from surface via temporary 30-inch-diameter tubing suspended in the production shaft.

When permanent ventilation has been established, excavations for station area facilities will be undertaken. These include:

- Maintenance shops
- Sump and pump stations
- Orepass to upper level
- Fuel station
- Electrical substation
- Explosives magazine
- General storage area
- Lunchroom
- First aid station
- Supervisor's office

The sump will be excavated to a depth of 13 feet below the lower level floor to provide adequate storage capacity. Roof excavation will be required to provide clearance for the pump room and electrical substation to be located above the lower level. As individual station area excavations are completed, subcontractors will be engaged for major construction and equipment installation. The proposed shaft station layout is shown in Figure 4.3.

Three parallel entries, 15 feet high by 20 feet wide, will be driven from the shaft station area to the demonstration unit areas. Crosscuts, 15 feet wide, will be turned between the entries at intervals not exceeding 100 feet to comply with gassy mine regulations. Lower level main entries will be driven approximately 1,600 feet during Stage I.

Access to the chamber drilling level, 48 feet above the lower level main entries, will be gained by driving two parallel ramps, 12 feet high by 15 feet wide, up an approximate 12% grade. Chamber drilling level development will consist of driving a pair of 12-foot by 15-foot entries behind the three chambers of the demonstration unit (Figure 4.4). Crosscuts between ramps and between drilling level entries will be spaced at maximum intervals of 100 feet. Ramp and drilling level entry development will be undertaken after Stage I main entry drifting is completed.

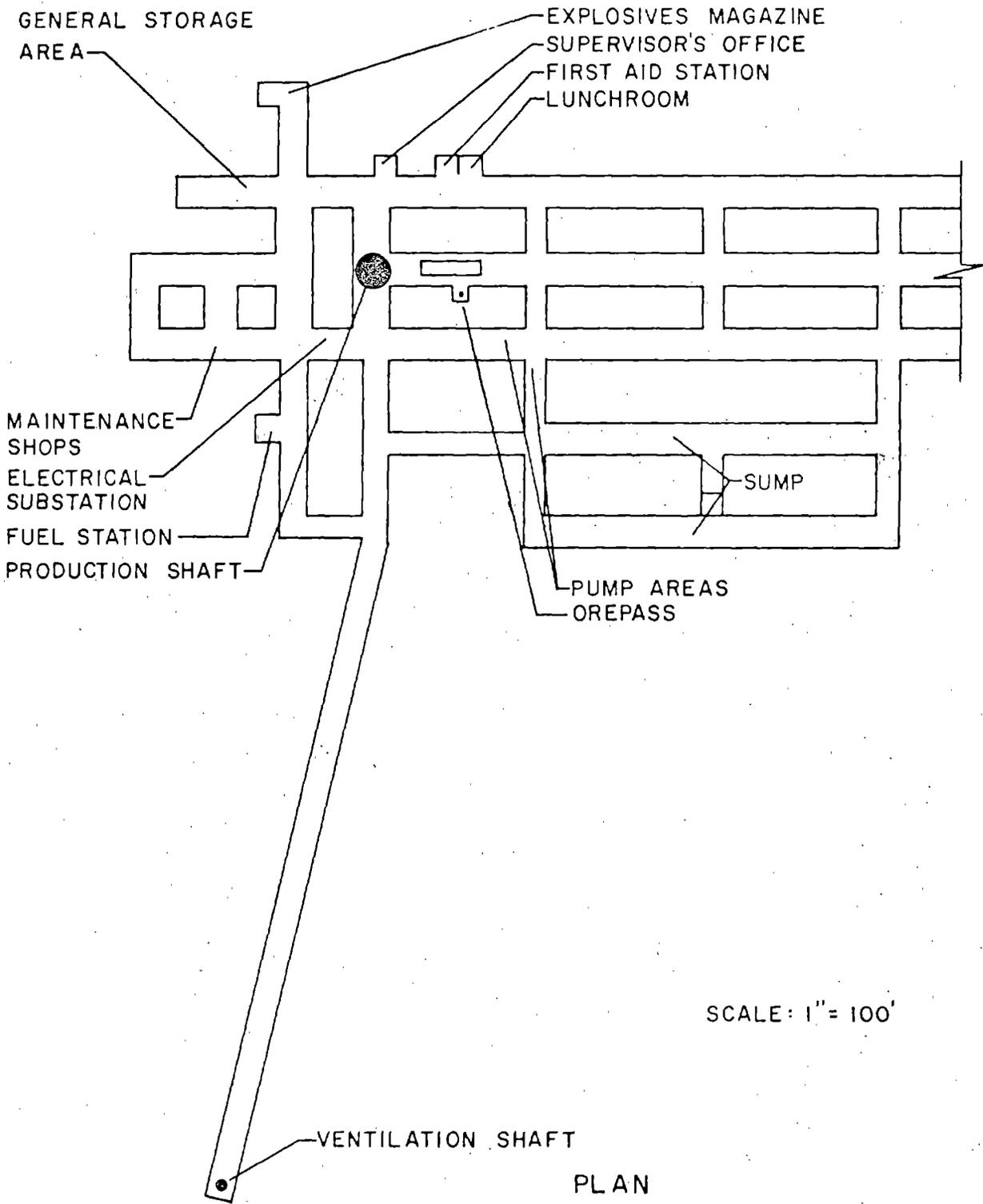


FIGURE 4.3
LOWER LEVEL SHAFT STATION LAYOUT

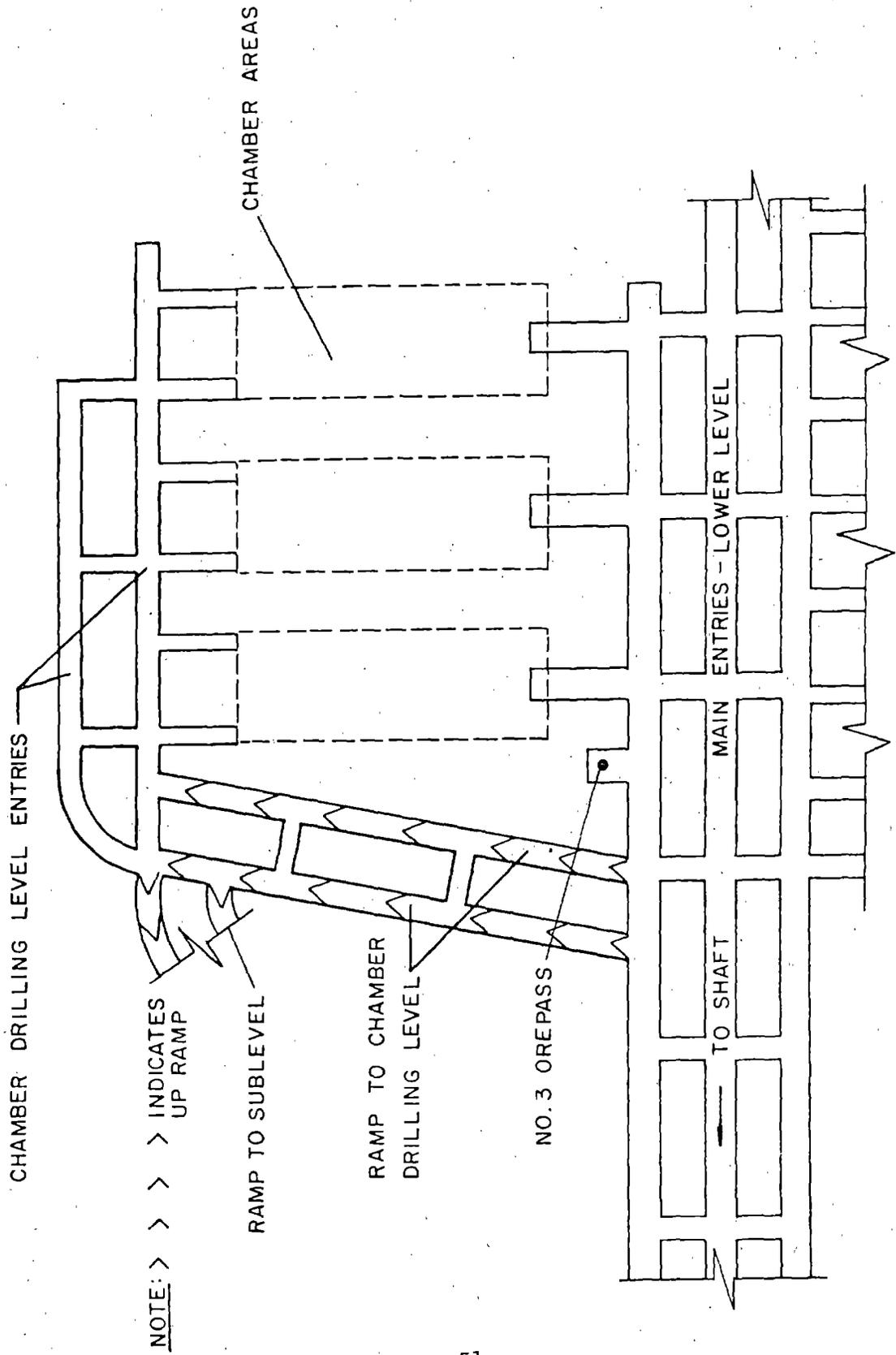


FIGURE 4.4
CHAMBER DRILLING LEVEL
RAMP AND ENTRIES

SCALE: 1"=100'

Vertical bored raises, 52 inches in diameter, will be used to transfer broken shale from the sublevel and upper level to the lower haulage level. The raises will be blind-bored upward from locations near conveyor loading points on the lower level. Orepasses Nos. 1, 2, and 3, each 132 feet long, will be completed during Stage I. Profile views of the orepasses are provided in Figure 4.5.

4.1.1.2 Stage II

Primary development work on the lower level will be resumed before extraction and backfilling operations are completed in the chamber and pillar and sublevel stoping with backfill demonstration units. The lower level main entries will be extended approximately 800 feet to the sublevel stoping with full subsidence demonstration area. Orepass No. 4 will be bored upward a distance of 227 feet to connect the two block caving monitor levels and intermediate levels with the lower level (Figure 4.5).

The ventilation exhaust entry, serving the sublevel stoping with backfill unit, will be extended to the sublevel stoping with full subsidence demonstration area. A two-entry system with crosscuts at maximum 100-foot intervals will be employed (Figure 4.1).

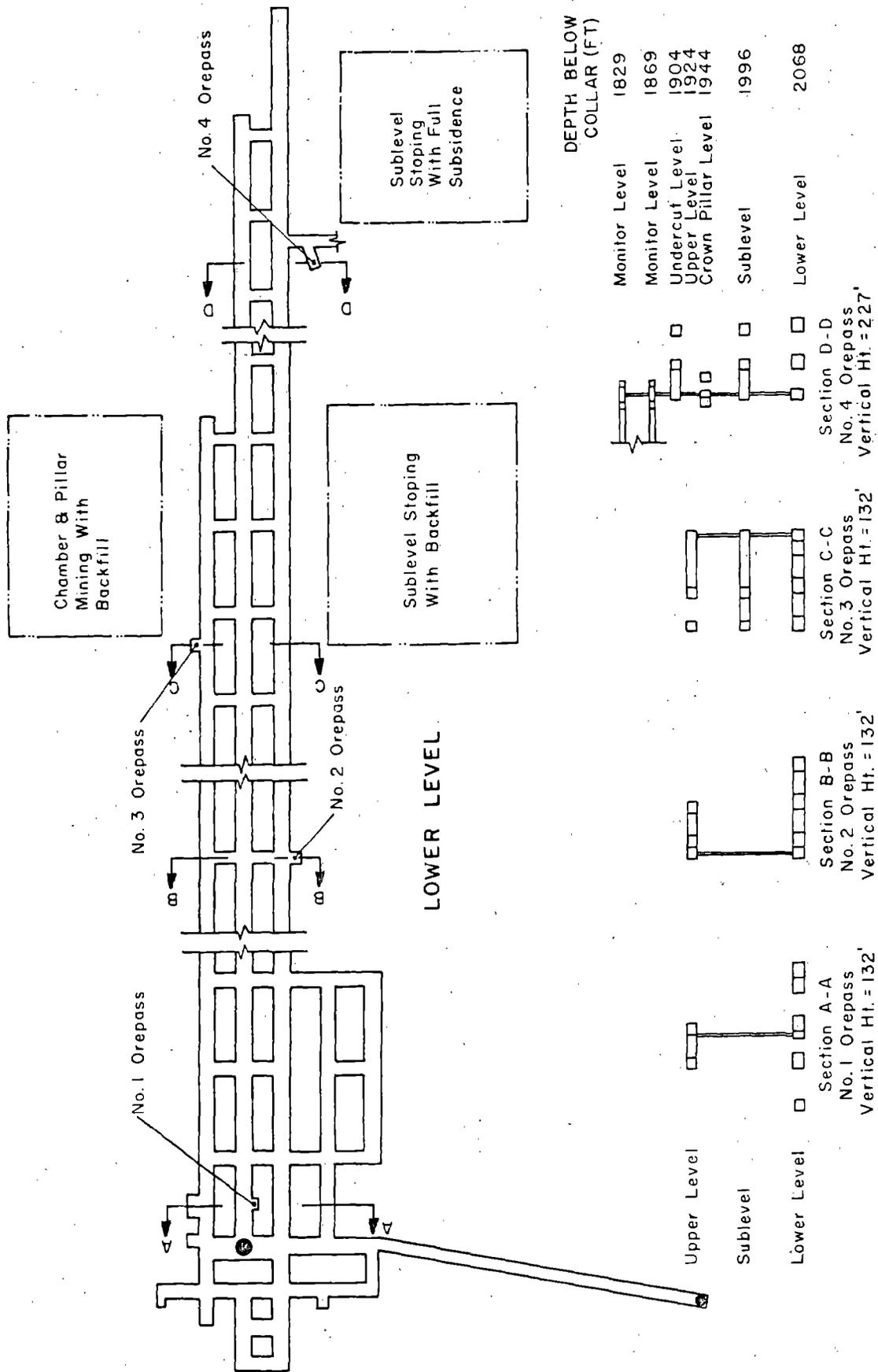
4.1.2 Sublevel

Access to the sublevel will be gained by continuation of ramps previously driven to the chamber drilling level. Once the sublevel elevation is reached (72 feet above the lower level floor), entries will be driven through the demonstration area (Figure 4.2). The sublevel entries will be located directly above the parallel with the lower level main entries and will provide access for development and production activities in the two sublevel stoping demonstration units. Ramp and entry development will consist of driving two parallel headings, 12 feet high by 15 feet wide, with crosscuts at intervals not exceeding 100 feet. As with lower level main entry development, two stages are assumed. In the first stage, sublevel entries will be driven to a point that will permit stope preparation and mining in the sublevel stoping with backfill system. In the second stage, entries will be extended approximately 800 feet to the sublevel stoping with full subsidence area.

4.1.3 Upper Level

4.1.3.1 Stage I

A single 12-foot high by 15-foot wide drift, approximately 550 feet long, will be driven to connect the ventilation and production shafts to establish a separate ventilation circuit on the upper level. Station area development work will require only minor excavations to provide openings to be used as a fuel station, lunchroom, supervisor's office, etc. (Figure 4.6).



SCALE: 1" = 200'

FIGURE 4.5
OREPASS LAYOUT

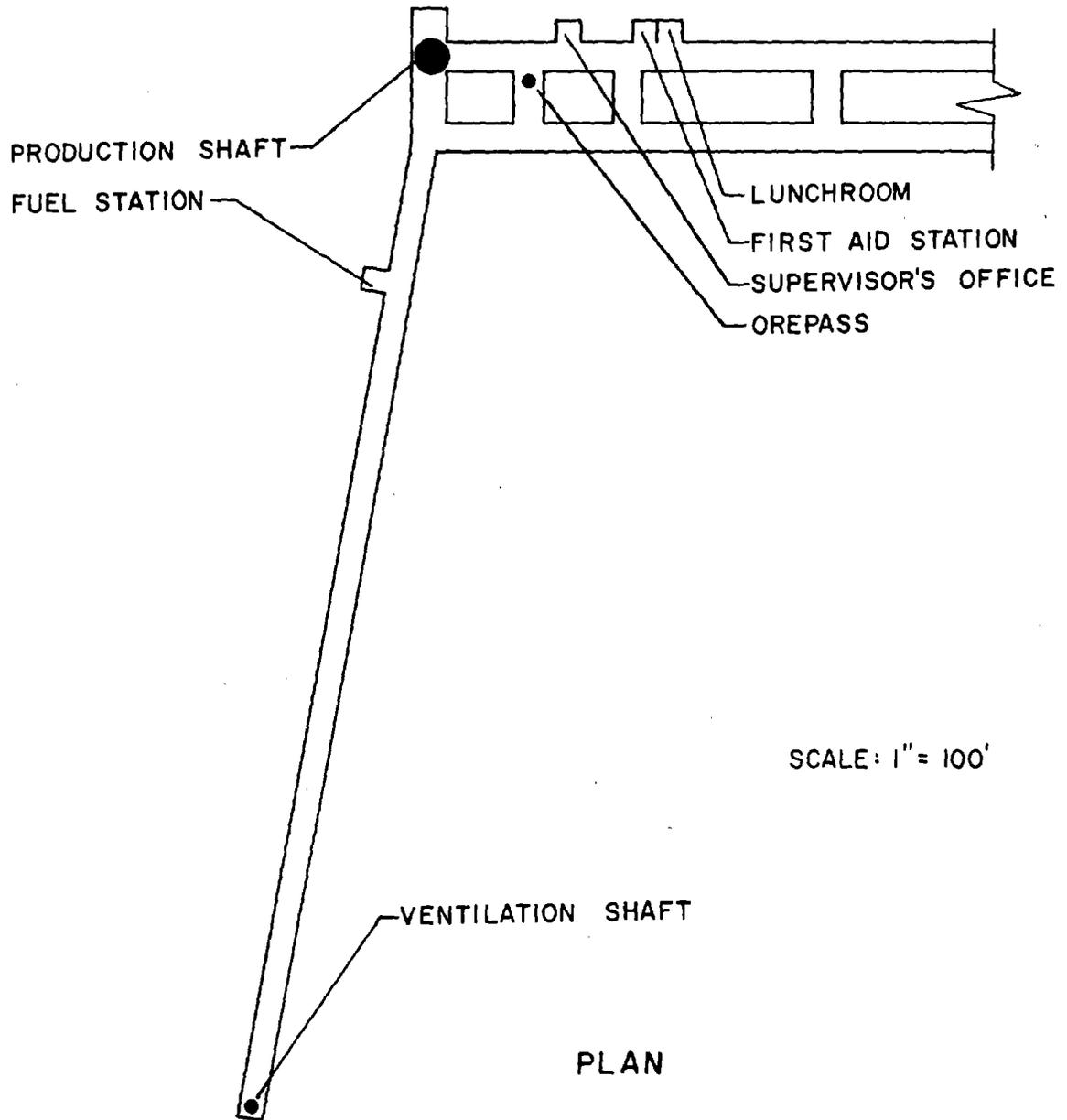


FIGURE 4.6
UPPER LEVEL SHAFT STATION LAYOUT

Two parallel entries, 12 feet high by 15 feet wide, will be driven from the shaft toward the demonstration mining areas a distance of about 1,600 feet. The entries will overlie main entries on the lower level and will provide access for upper level drilling drifts and crosscuts driven within the sublevel stoping with backfill unit. Primary development on the upper level is illustrated in Figure 4.2.

4.1.3.2 Stage II:

Upper level main entries will be extended to the designated area for demonstration of subsidence mining systems. Prior to reaching the block caving area, the upper level mains will be ramped up at 12% to the undercut elevation (Figure 4.7). Ramping will increase the vertical separation of the upper and lower levels from 144 feet in the vicinity of the chamber and pillar unit to 164 feet in the block caving undercut area.

Two inclined raises will be bored upward from the block caving undercut level to establish access to the monitor levels above the block caving demonstration unit (Figure 4.7). During monitor level development, one of the 52-inch-diameter raises will serve as a manway and the second will be used as an accessway for moving equipment and supplies.

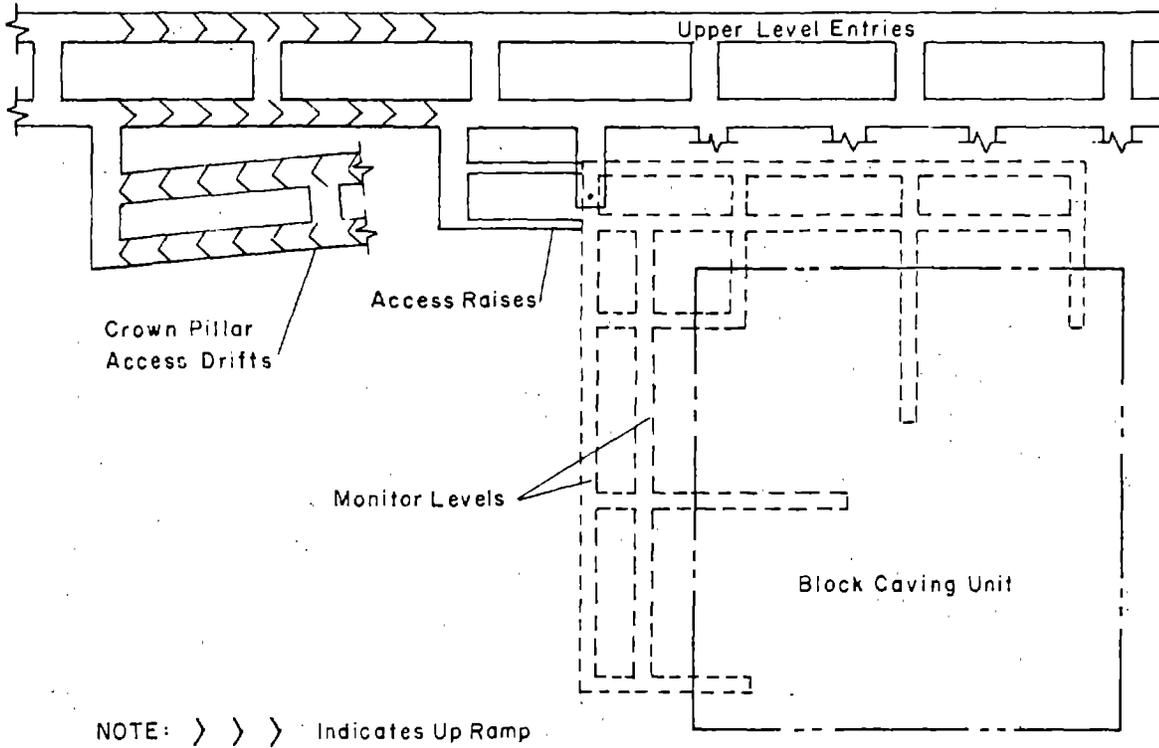
A single heading will be turned off of the upper level mains and two parallel ramps will be driven down a 10% grade to gain access to the crown pillar drilling level, 40 feet below the block caving undercut (Figure 4.7). The crown pillar access drifts will be excavated 12 feet high by 12 feet wide.

4.2 DEMONSTRATION MINING

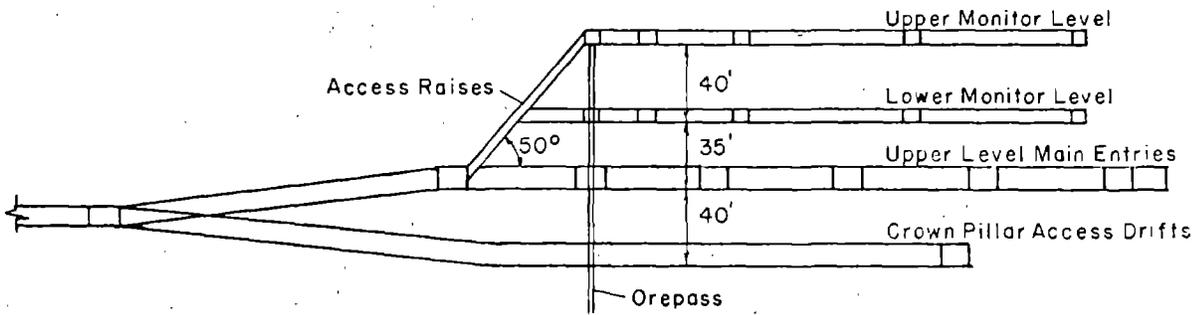
4.2.1 Chamber and Pillar Mining With Backfill

Chamber and pillar mining is a modified form of room and pillar mining in which drifts driven at right angles to the main-entry system are expanded by fan drilling into chambers. The method is particularly well suited to the practice of backfilling mined-out areas. Mining and backfilling of chambers in an alternating sequence provides lateral support for rib pillars left between chambers for overburden support. The presence of backfill material in adjacent chambers permits a reduction of rib pillar dimensions, thereby improving the extraction ratio.

The system will be demonstrated by mining three chambers as shown in Figure 4.8. The two end chambers of the demonstration unit will be mined first. Mining of the central chamber will not be undertaken until backfilling has been completed in both outer chambers. In a commercial mine, the 70-foot chamber spans and 40-foot rib pillars will provide a 64% in-panel extraction ratio. The mine-wide extraction ratio will be somewhat less, dependent upon the chamber length and the size of unmined barrier pillars. Extraction data projected for development and mining of the three chambers are summarized in Table 4.2.



PLAN



PROFILE

SCALE: 1" = 100'

FIGURE 4.7

UPPER LEVEL DEVELOPMENT - STAGE II

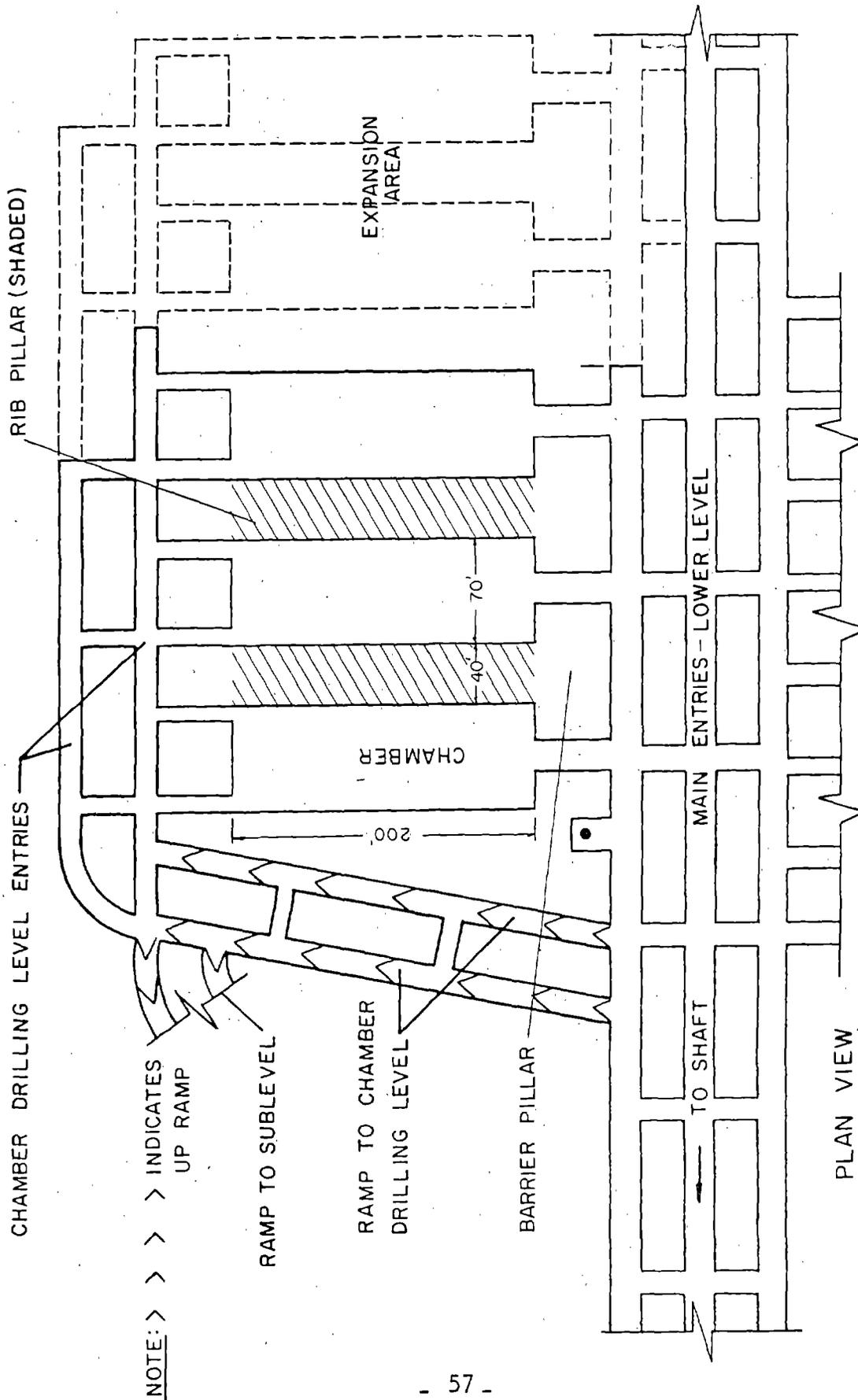


FIGURE 4.8

DEMONSTRATION UNIT
CHAMBER AND PILLAR MINING WITH BACKFILL

SCALE: 1"=100'

TABLE 4.2

EXCAVATION SUMMARY
CHAMBER AND PILLAR MINING WITH BACKFILL

<u>Openings</u>	<u>Section Dimensions</u>	<u>Total Footage</u>	<u>Number Of Rounds</u>	<u>Tons Per Round</u>	<u>Total Tonnage</u>
Unit Development:					
LHD Entries	15' x 20'	180	18	207	3,730
Chamber Drilling Entries and Crosscuts	12' x 12'	1,914	192	100	19,140
Slot Raises	(52" Diam.)	99	-	-	100
Chamber Slots	60' x 70'	30	-	-	6,230
Subtotal					<u>29,200</u>
	<u>Overall Chamber Dimensions</u>		<u>Number Of Chambers</u>	<u>Tons Per Chamber</u>	
Chamber Mining:	60' x 70' x 200'		3	50,255	<u>150,770</u>
Chambers					<u>179,970</u>
TOTAL					

Measurements of stress changes in rib pillars and deflections of chamber roof spans will be obtained during mining of the three chambers to evaluate rock mass behavior. If stress measurements show the rib pillars to be underloaded after mining three chambers, two additional chambers can be mined to achieve pillar stress levels representative of a commercial-scale operation. An area of suitable dimensions has been reserved for this contingency in the demonstration mine plan (Figure 4.8).

4.2.1.1 Unit Development

Three 15-foot-high by 20-foot-wide LHD entries will be turned off the lower level mains and driven 60 feet along the projected chamber centerlines. LHD entries will serve as haulways from the chambers during slot development and chamber mining.

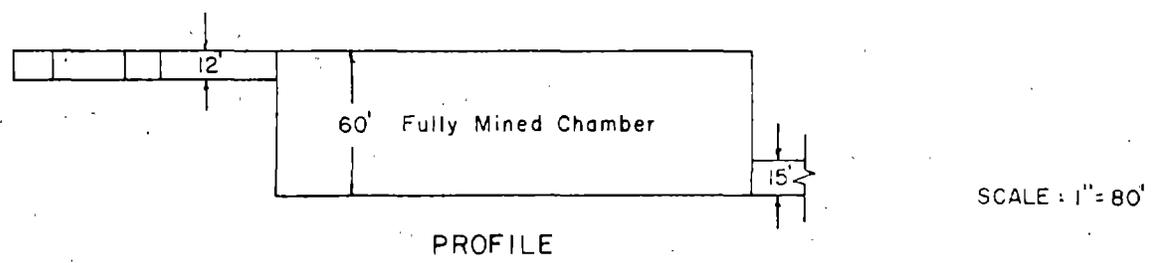
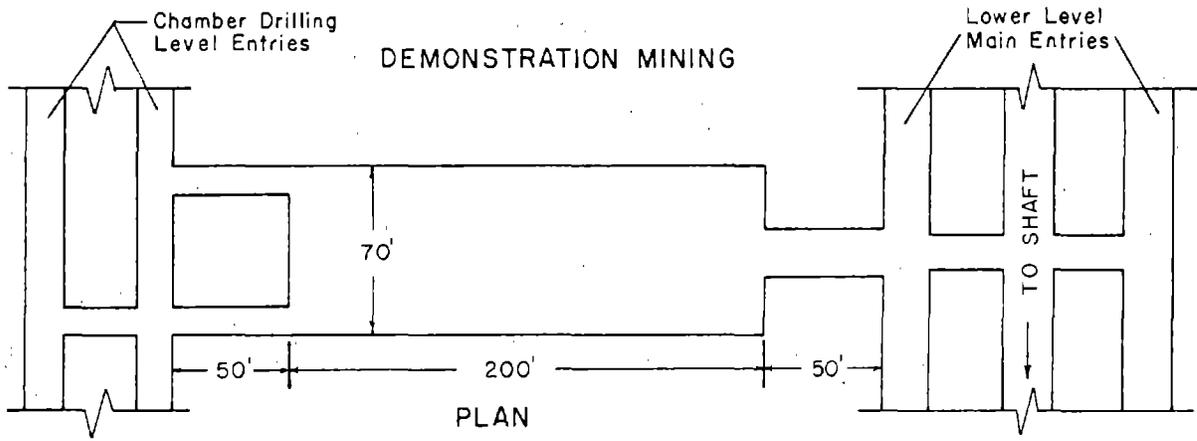
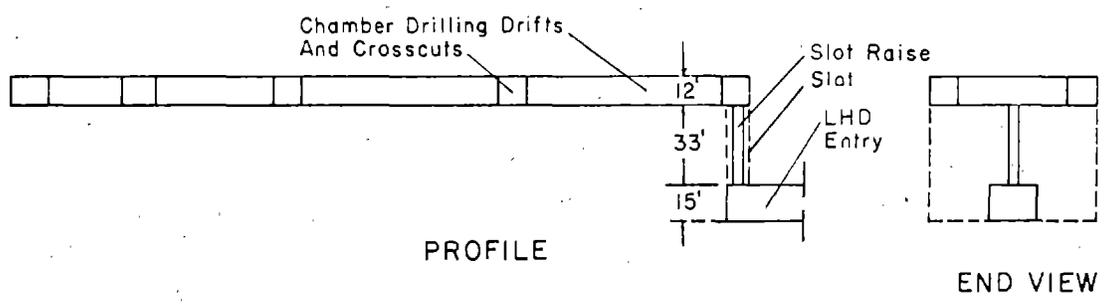
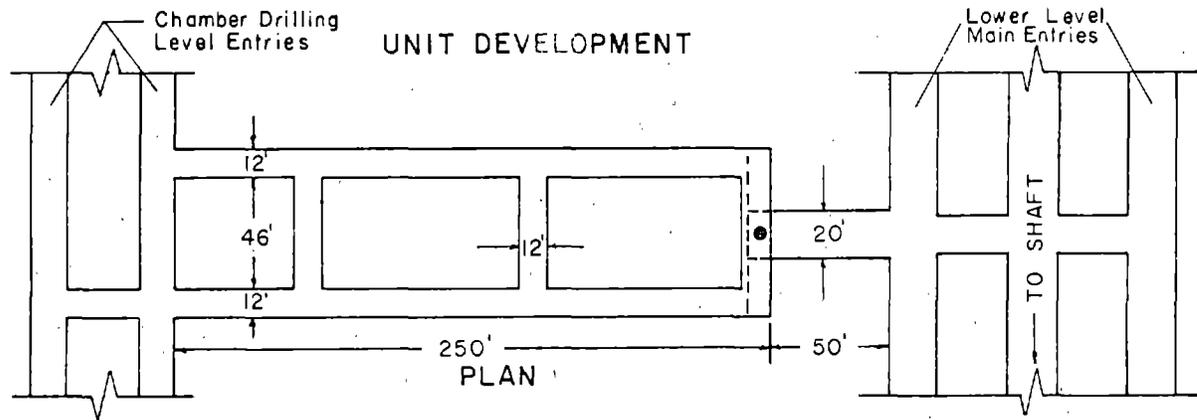
Two drilling drifts will be driven into each chamber area from the chamber drilling level entries. Drilling drifts will be driven with connecting crosscuts spaced not more than 100 feet apart to comply with regulations governing development in gassy mines. Drifts and crosscuts on the chamber drilling level, 48 feet above the lower level, will be 12 feet high by 12 feet wide. The layout and dimensions of drilling drifts and LHD entries are shown in Figure 4.9.

A 52-inch-diameter raise will be blind-bored upward from the LHD entry in each chamber area to intersect the center of the crosscut connecting the ends of the overlying drilling drifts, as shown in Figure 4.9. The raise will provide blasthole relief for slot development. When completed, the slot openings will measure 70 feet wide, 60 feet high, and 10 feet along the chamber axis. The development of the slot is illustrated subsequently in Figure 5.2, page 88.

4.2.1.2 Unit Demonstration

Unit development will be completed in all three chamber areas before mining of the first two chambers is begun. The chambers will be excavated by drilling asymmetric fans of blastholes from the chamber drilling drifts. When mining is completed, the chambers will be 200 feet long, 60 feet high, and 70 feet wide. Views of a fully-mined chamber are provided in Figure 4.9.

Alluvium will be used to backfill the mined-out chambers. The backfill material will be introduced into the mine through a 30-inch-diameter borehole intersecting the upper mine level. Assuming a fill density of 100 pounds per cubic foot, each chamber ultimately will contain an estimated 42,000 tons of alluvium. Movement of LHD's over the emplaced backfill material will be sufficient to produce partial compaction.



SCALE : 1" = 80'

FIGURE 4.9
PLAN AND PROFILE VIEWS OF DEVELOPMENT
CHAMBER AND PILLAR MINING WITH BACKFILL

4.2.2 Sublevel Stopping With Backfill

Sublevel stopping is a high volume, low-cost, open-stopping method which is well suited for mining fairly regular ore bodies composed of both competent ore and host rock. Open-stope production is accomplished by long-hole drilling from levels and sublevels, and blasting in successive slices. The implementation of backfilling with sublevel stopping improves ground control. However, when compared to sublevel stopping with full subsidence, backfilling results in reduced extraction due to the unrecovered ore left in pillars.

To demonstrate the sublevel stopping with backfill system, three stopes will be excavated (Figure 4.10). A contingency area will be left to expand the demonstration unit to five stopes if measurements indicate that rib pillars between the three stopes are underloaded. The relatively thin, 40-foot rib pillars that separate the stopes are designed for maximum resource extraction. The design of pillars will require that stopes be mined in an alternating sequence. The central stope will not be excavated until the mined-out stopes on either side have been backfilled. The 80-foot stope spans and 40-foot intervening rib pillars will result in an approximate 67% in-panel extraction ratio.

4.2.2.1 Unit Development

Extensive drifting will be done to develop the necessary openings for drilling, loading, haulage, and ventilation on the stope floor level (Table 4.3, Figure 4.10). Four LHD haulageways and three stope floor drilling drifts will be driven from the lower level main entries. Loading crosscuts will be turned at 70° angles from the stope floor drifts to the left and right to connect with adjacent LHD haulageways. A ventilation exhaust entry will be driven around two sides of the demonstration area to collect exhaust air from the LHD haulways.

Stope drawpoints will be situated at the intersections of stope floor drifts and loading crosscuts. Dimensions of both sets of openings will be 12 by 12 feet. The exhaust entry and LHD haulageways will be 12 feet high and 15 feet wide.

Six drilling drifts will be driven from the sublevel mains along the boundaries of the stopes a distance of 300 feet (Figure 4.11). Three ventilation crosscuts will be required between each pair of drilling drifts.

The upper level will be developed by driving three drilling drifts along the center of each stope at the roof level. The drilling drifts will be connected by ventilation crosscuts at approximate 100-foot intervals. Drifts and crosscuts will be 12 by 12 feet in section.

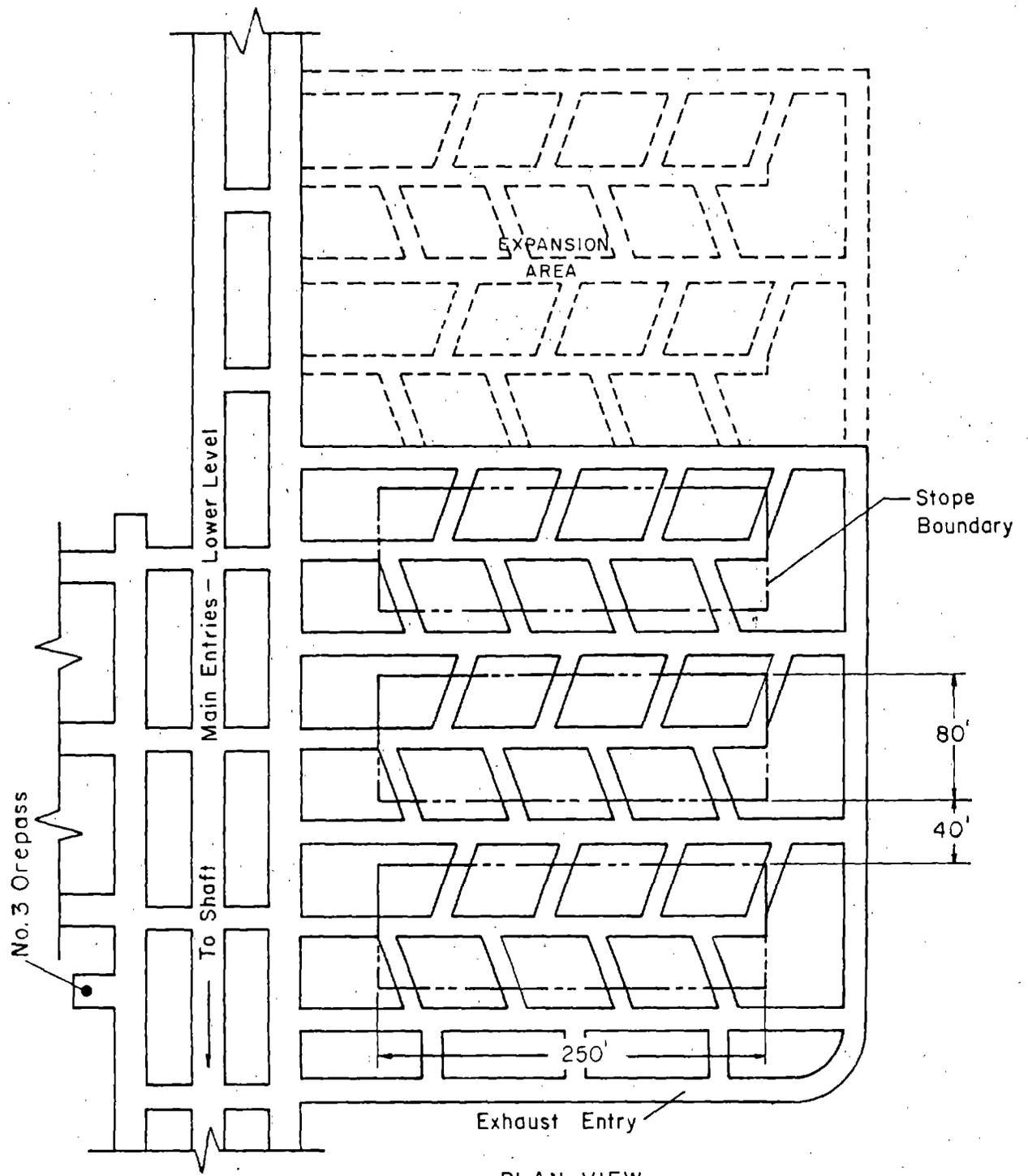


FIGURE 4.10
DEMONSTRATION UNIT
SUBLEVEL STOPPING WITH BACKFILL

SCALE: 1" = 100'

TABLE 4.3

EXCAVATION SUMMARY
SUBLEVEL STOPPING WITH BACKFILLING

<u>Openings</u>	<u>Section Dimensions</u>	<u>Total Footage</u>	<u>Number Of Rounds</u>	<u>Tons Per Round</u>	<u>Total Tonnage</u>
Unit Development:					
LHD Haulageways	12' x 15'	1,750	175	125	21,880
Exhaust Entry	12' x 15'	420	42	125	5,250
Stope Floor Drifts	12' x 12'	900	90	100	9,000
Loading Crosscuts	12' x 12'	1,188	119	100	11,880
Exhaust Entry Crosscuts	12' x 12'	90	9	100	900
Sublevel Drilling Drifts and Crosscuts	12' x 12'	2,304	231	100	23,040
Upper Level Drilling Drifts and Crosscuts	12' x 12;' (52"-Diam.)	1,332	133	100	13,320
Slot Raises	132' x 80'	396	-	-	400
Slots	132' x 80'	30	-	-	14,710
Subtotal					<u>100,380</u>
Stope Mining:					
Stopes	Overall Stope Dimensions		Number Of Stopes	Tons Per Stope	
	132' x 80' x 250'		3	130,550	<u>391,650</u>
TOTAL					<u>492,030</u>

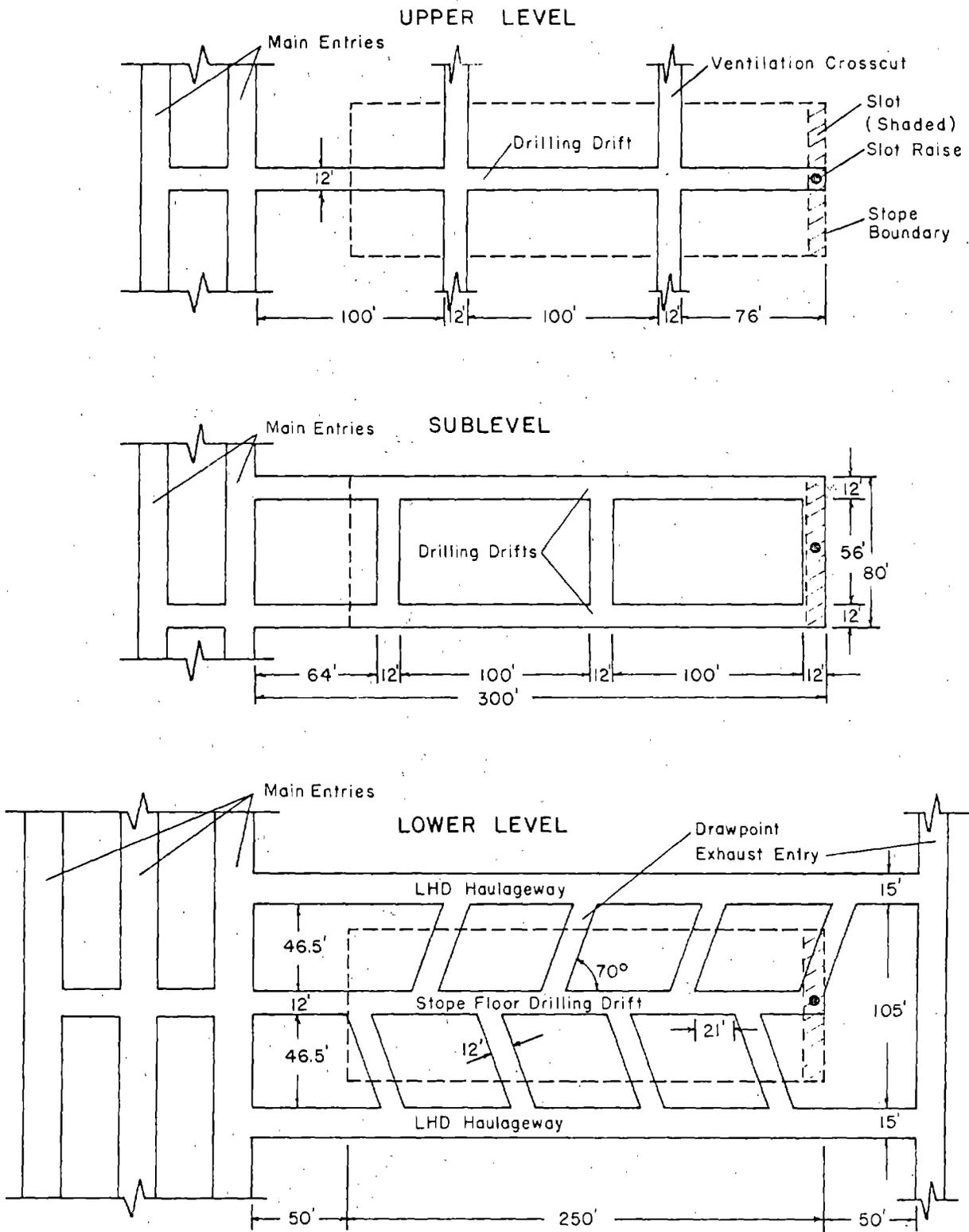
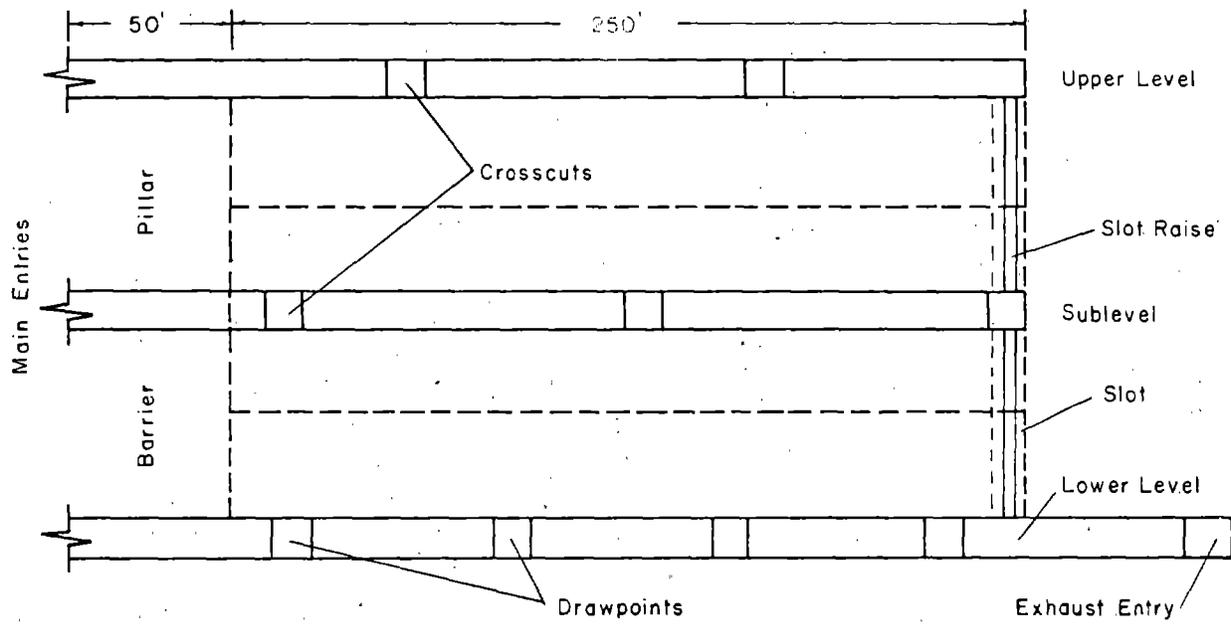
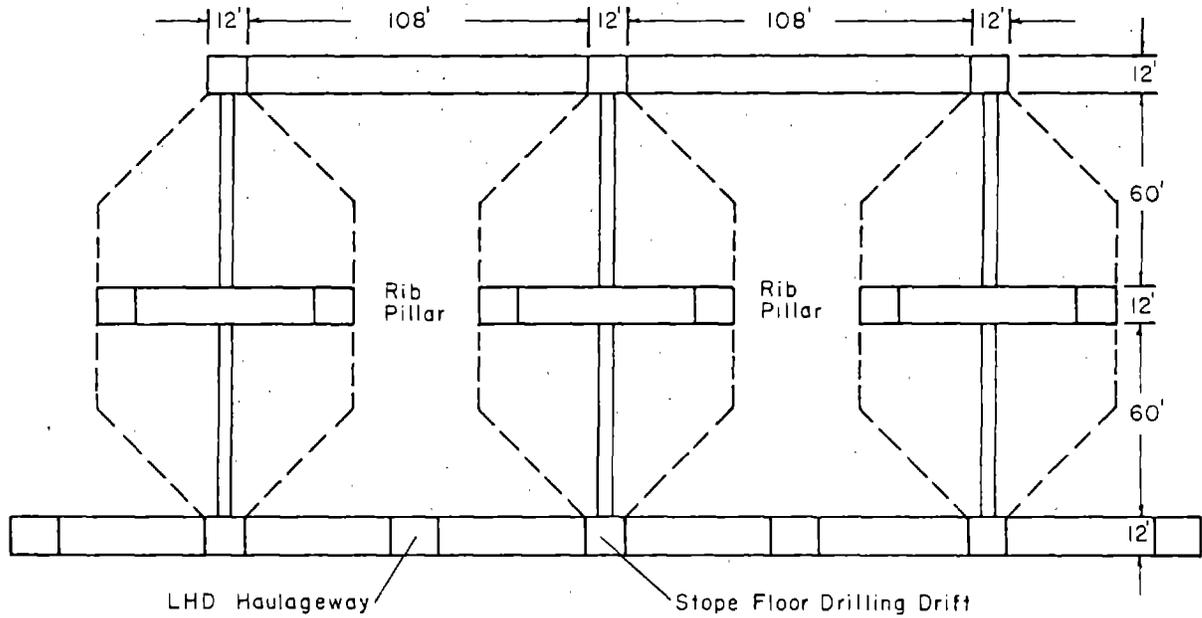


FIGURE 4.11
PLAN VIEWS OF DEVELOPMENT
SUBLEVEL STOPING WITH BACKFILL

SCALE: 1" = 80'



PROFILE



END VIEW

FIGURE 4.12

VERTICAL SECTIONS OF DEVELOPMENT OPENINGS
SUBLEVEL STOPPING WITH BACKFILL

The 108-foot spacing between upper level drilling drifts exceeds the MSHA 100-foot standard for development in gassy mines. A variance may be required in order to proceed with upper level development. In a commercial system, a second stoping interval would be developed above the first interval, and development plans for lower and upper levels would be identical. As a result, the drift spacing in a commercial system would meet MSHA requirements.

A 52-inch-diameter raise will be blind bored upward from the stope floor drilling drift in each stoping area (Figures 4.11 and 4.12). Subsequent expansion of the raises by drilling and blasting will establish 10-foot-wide slots across the ends of the stoping areas. The slots will extend from the stope floors to the upper level drilling drifts above. Development of a slot is illustrated subsequently in Figure 5.4, page 91.

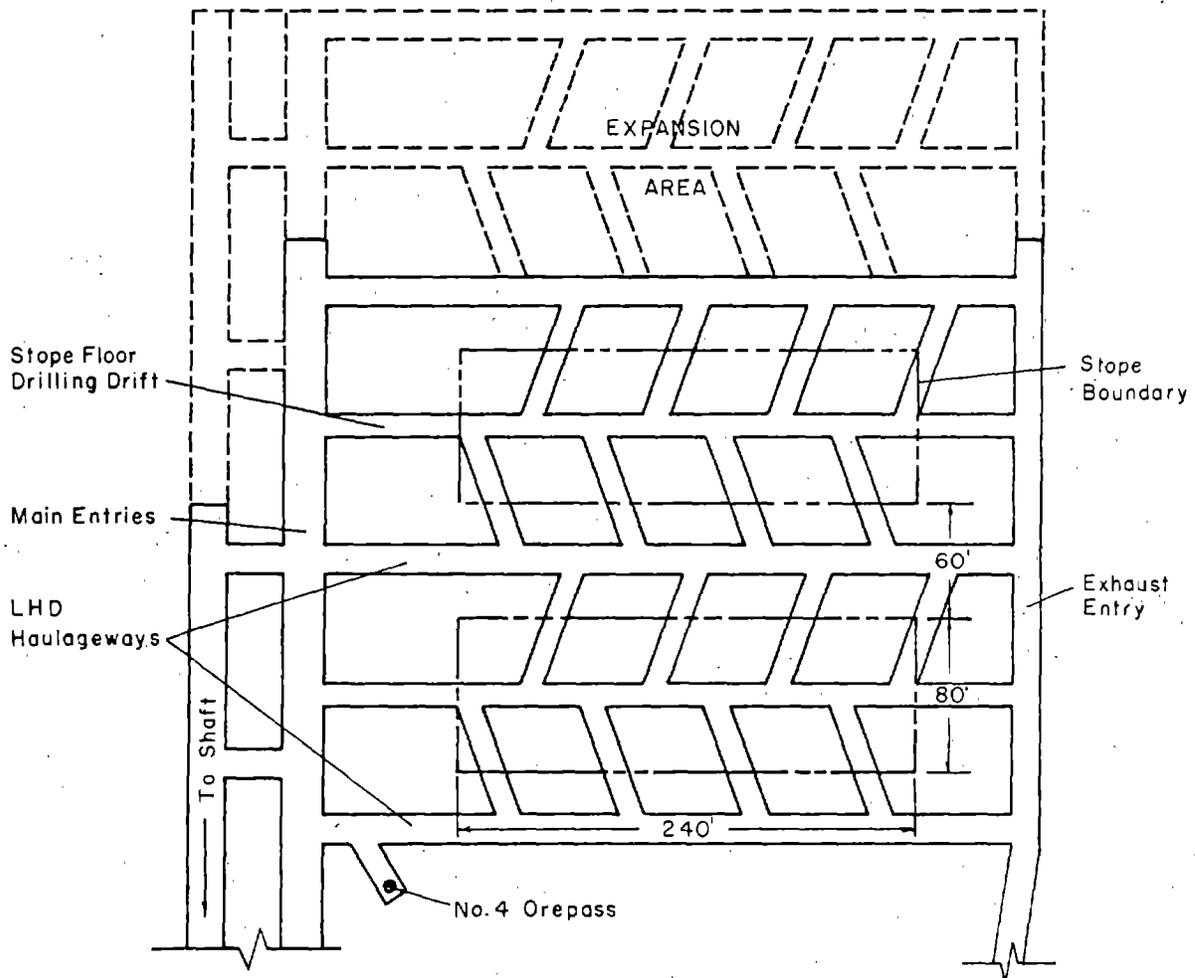
4.2.2.2 Unit Demonstration

The two outer stopes of the demonstration unit will be mined concurrently. Excavation will proceed from the slot, located at the far end of each stope, and retreat toward the main entries. Fan drilling will be conducted simultaneously from the stope floor, sublevel, and upper drilling levels, with fans spaced 10 feet apart along the length of a stope. Approximately 130,000 tons of oil shale will be mined from each stope (Table 4.3). The central stope will not be mined until the stopes on either side have been backfilled. As before, alluvium will be employed as the backfill material. Assuming a fill density of 100 pounds per cubic foot, 80,000 tons of material will be required to fill each stope.

4.2.3 Sublevel Stopping With Full Subsidence

Sublevel stopping with full subsidence is a high volume, low-cost mining method designed to achieve near-total resource recovery. The method is well suited to mining fairly regular ore bodies having both competent ore and host rock. In a commercial operation successively deeper levels are developed with crown pillars separating the roof horizons of new stopes from the floors of worked-out stopes above. After a stope is excavated the surrounding crown and rib pillars are drilled and blasted, and the ore collapses into the stope cavity to be recovered through the drawpoints.

To demonstrate this method, two stopes measuring 80 feet wide, 124 feet high, and 240 feet long will be mined by fan drilling from stope floor drifts and ring drilling from sublevel drilling drifts (Figure 4.13). The stopes will be separated by a 60-foot-wide rib pillar. A 40-foot-thick crown pillar will span the stopes, isolating them from an overcut level. The overcut (which also will serve as the undercut level for the block caving unit) will simulate an overlying level of worked-out stopes.



PLAN
(LOWER LEVEL)

SCALE: 1" = 100'

FIGURE 4.13
DEMONSTRATION UNIT
SUBLEVEL STOPPING WITH FULL SUBSIDENCE

Two drilling drifts will be driven within the rib pillar during unit development to facilitate pillar extraction subsequent to stope mining. A 70-foot barrier pillar will be left at the front of the stopes for protection of main entries on the various levels, and a 50-foot barrier at the rear of the stopes will remain to isolate the exhaust entry on the lower level.

An additional area has been reserved as a contingency for mining a third stope. From the standpoint of demonstrating the sublevel stoping with full subsidence method, mining two stopes probably will provide an adequate insight into the method's practicability. However, if the undercut level in the overlying block caving demonstration unit proves to be inadequate to induce suitable caving, expansion of the undercut level will be necessary. In that event, the third stope would be mined to provide additional drawpoints for the expanded block caving unit.

4.2.3.1 Unit Development

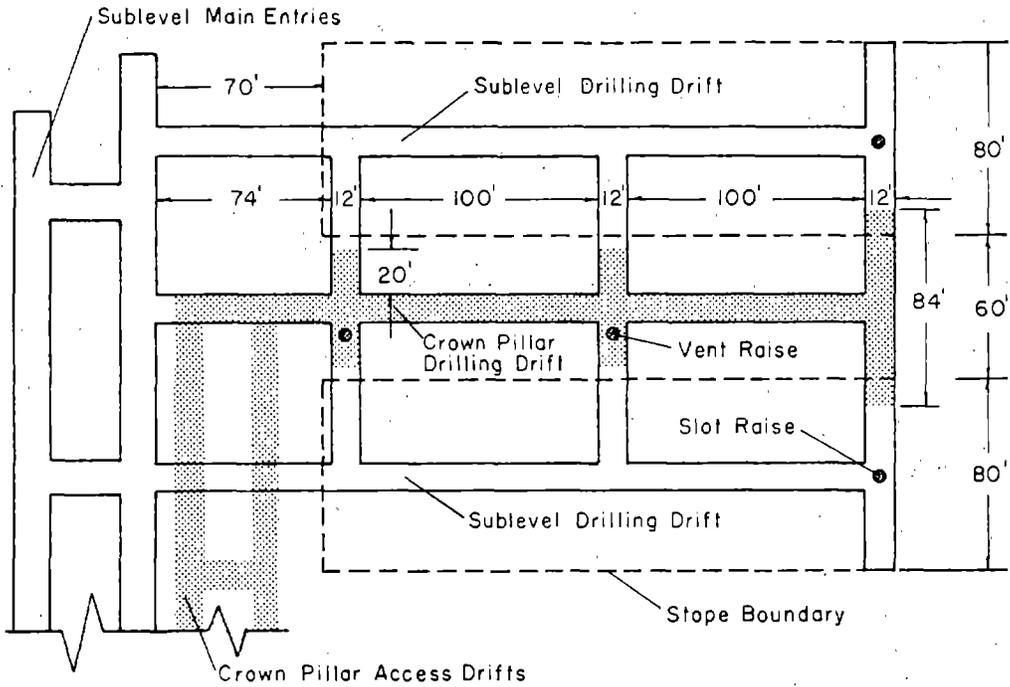
Development on the haulage level will consist of driving two stope floor drifts and three LHD haulageways (each 360 feet long), 16 drawpoint crosscuts and an exhaust entry, shown in Figure 4.13. Crosscuts will be turned at angles of 70° from the stope floor drifts and driven 60 feet to connect with adjacent LHD haulageways. Stope floor drifts and drawpoint crosscuts will be 12 feet high by 12 feet wide with crosscuts widened to 17 feet at the haulageway intersections to improve LHD maneuverability. LHD haulageways and the exhaust entry will be 12 feet high by 15 feet wide. The exhaust entry is an extension of the exhaust entry servicing the sublevel stoping with backfill unit (Figure 4.1).

On the sublevel three drilling drifts, each 310 feet in length, and three connecting crosscuts will be driven, as shown in Figure 4.14. The crosscut joining the ends of the drilling drifts will establish access for slot development in both stopes and will be 220 feet long. Other crosscuts, turned for ventilation purposes only, will be 128 feet long. All drifts and crosscuts on the sublevel will be 12 by 12 feet in section.

The crown pillar drilling level will be situated 124 feet above the stope floor. Development on this level will consist of a single 310-foot-long drift and assorted crosscuts, all 12 by 12 feet in section (Figure 4.14). The 84-foot-long crosscut on the end of the crown pillar drilling drift will function as a ventilation return airway from the stopes once the slots have been developed. The stub crosscuts ultimately will serve as drilling stations during rib pillar extraction.

As the crown pillar drilling drift is advanced, ventilation air will be supplied through two ventilation raises (Figure 4.15). The 52-inch-diameter raises will be blind bored upward from the sublevel to intersect the 20-foot-long stub crosscuts on the crown pillar drilling level.

CROWN PILLAR DRILLING LEVEL (SHADED)
AND SUBLEVEL



LOWER LEVEL

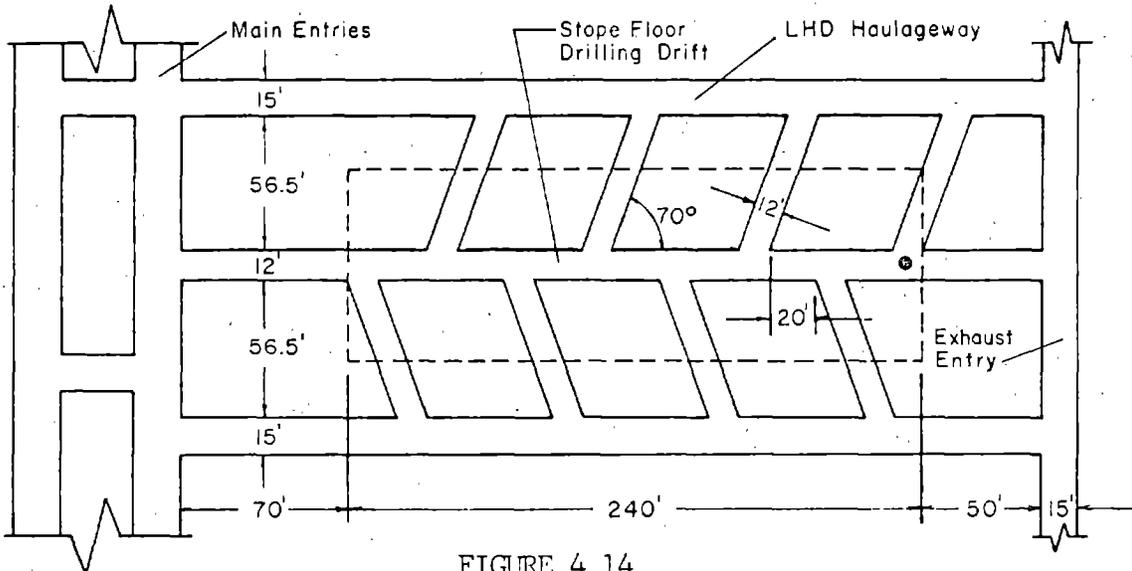
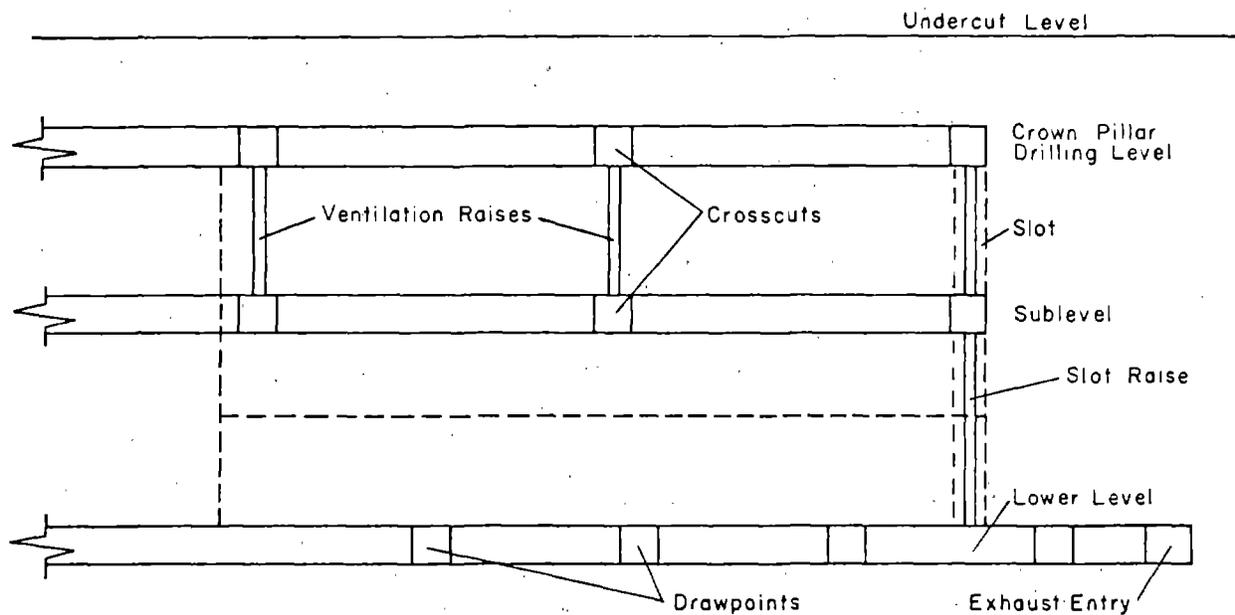
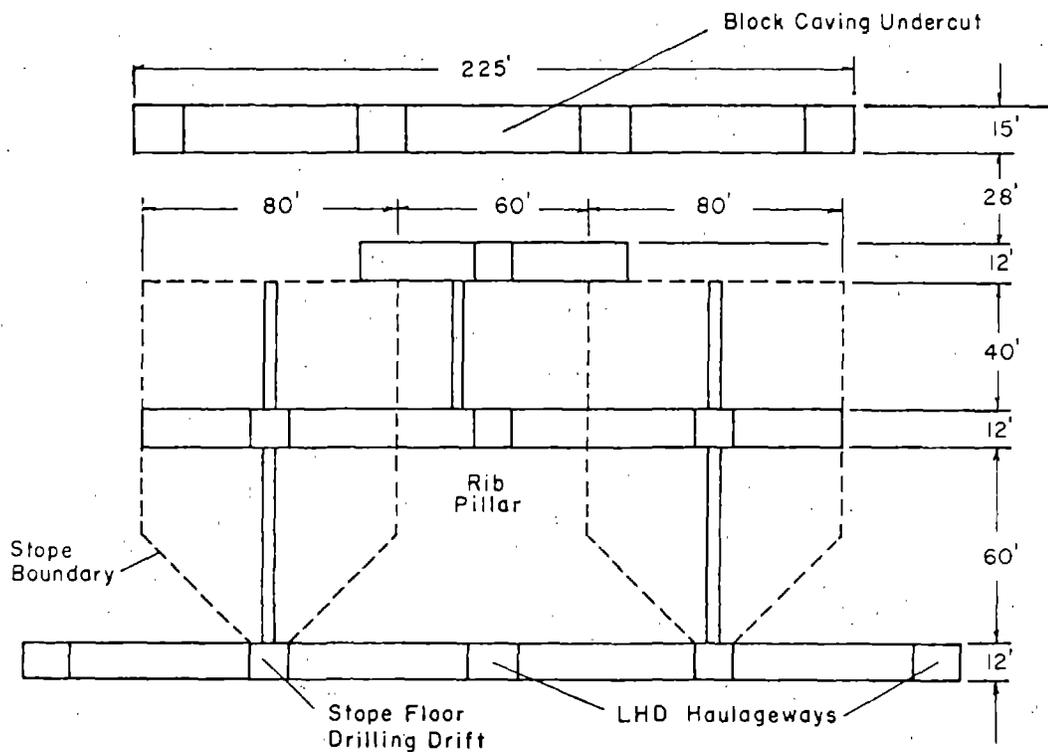


FIGURE 4.14
PLAN VIEWS OF DEVELOPMENT
SUBLEVEL STOPING WITH FULL SUBSIDENCE



PROFILE



END VIEW

FIGURE 4.15

VERTICAL SECTIONS OF DEVELOPMENT OPENINGS
SUBLEVEL STOPING WITH FULL SUBSIDENCE

Slot raises, 52 inches in diameter and 112 feet long, will be blind bored upward along the rear wall of each stope. The raises will extend the full height of the stopes and will provide blasthole relief for slot development.

Required drifting and anticipated tonnages to be excavated during stope development and mining are listed in Table 4.4.

4.2.3.2 Unit Demonstration

Mining in the two stopes will be performed concurrently. The lower 34 feet of the stopes will be excavated by fan drilling and blasting from the stope floor, and the remaining 78 feet will be mined by ring drilling and blasting from sublevel drilling drifts. Mining will be conducted in a retreating sequence from the slots at the rear of the stopes toward the main entries.

After both stopes have been mined, the crown and rib pillars will be rubblized in stages. First, a series of long blastholes will be drilled into the crown pillar above one stope from the crown pillar drilling level; the crown pillar will be rubblized in two rounds. Then, the rib pillar and the crown pillar above the second stope will be drilled simultaneously from the sublevel and crown pillar drilling drift and excavated in a sequence of three blast rounds (Section 5.1.2.5).

In a commercial operation a third stage of pillar recovery normally is performed. The bottom portion of the rib pillar, containing the haulageway and stope drawpoints, is blasted and recovered. This stage of recovery is not practical for demonstration, as the drawpoints must remain serviceable for demonstration mining of the block caving unit.

4.2.4 Block Caving

Block caving is a low-cost underground mining method suitable for mining large, thick ore bodies. The major requirement for a successful block caving operation is that the ore body be sufficiently weak or fractured to prevent the rock mass from breaking into large pieces, arching across the chamber, or tending to cave in a sporadic and catastrophic manner.

Demonstration of the block caving method will consist of developing an undercut by conventional room-and-pillar mining and then blasting the chain pillars to initiate caving of the roof (Figure 4.16). If caving is unsatisfactory the undercut can be expanded on two sides to an area measuring 330 feet by 365 feet.

The block caving unit will serve as an overcut for the sublevel stoping with full subsidence unit. After the 40-foot crown pillar separating the two units has been rubblized, broken shale from the block caving unit can be drawn down by mucking from the stope drawpoints. Controlled withdrawal of

TABLE 4.4

EXCAVATION SUMMARY
SUBLEVEL STOPPING WITH FULL SUBSIDENCE

<u>Openings</u>	<u>Section Dimensions</u>	<u>Total Footage</u>	<u>Number Of Rounds</u>	<u>Tons Per Round</u>	<u>Total Tonnage</u>
Unit Development:					
LHD Haulageways	12' x 15'	1,080	108	125	13,500
Exhaust Entry	12' x 15'	295	30	125	3,690
Stope Floor Drifts	12' x 12'	720	72	100	7,200
Loading Crosscuts	12' x 12'	960	96	100	9,600
Sublevel Drilling Drifts and Crosscuts	12' x 12'	1,346	135	100	13,460
Crown Pillar Drilling Drifts and Crosscuts	12' x 12'	410	41	100	4,100
Ventilation Raises	(52" Diam.)	80	-	-	80
Slot Raises	(52" Diam.)	224	-	-	230
Slots	112' x 80'	20	-	-	9,240
					<u>61,100</u>
Stope Mining:					
Stopes	Overall Stope Dimensions		Number Of Stopes	Tons Per Stope	
	112' x 80' x 240'		2	66,330*	<u>132,660*</u>
					<u>193,760</u>

* (66,330 tons extracted per stope = 120,700 tons broken in each stope - 54,370 tons retained in each stope.)

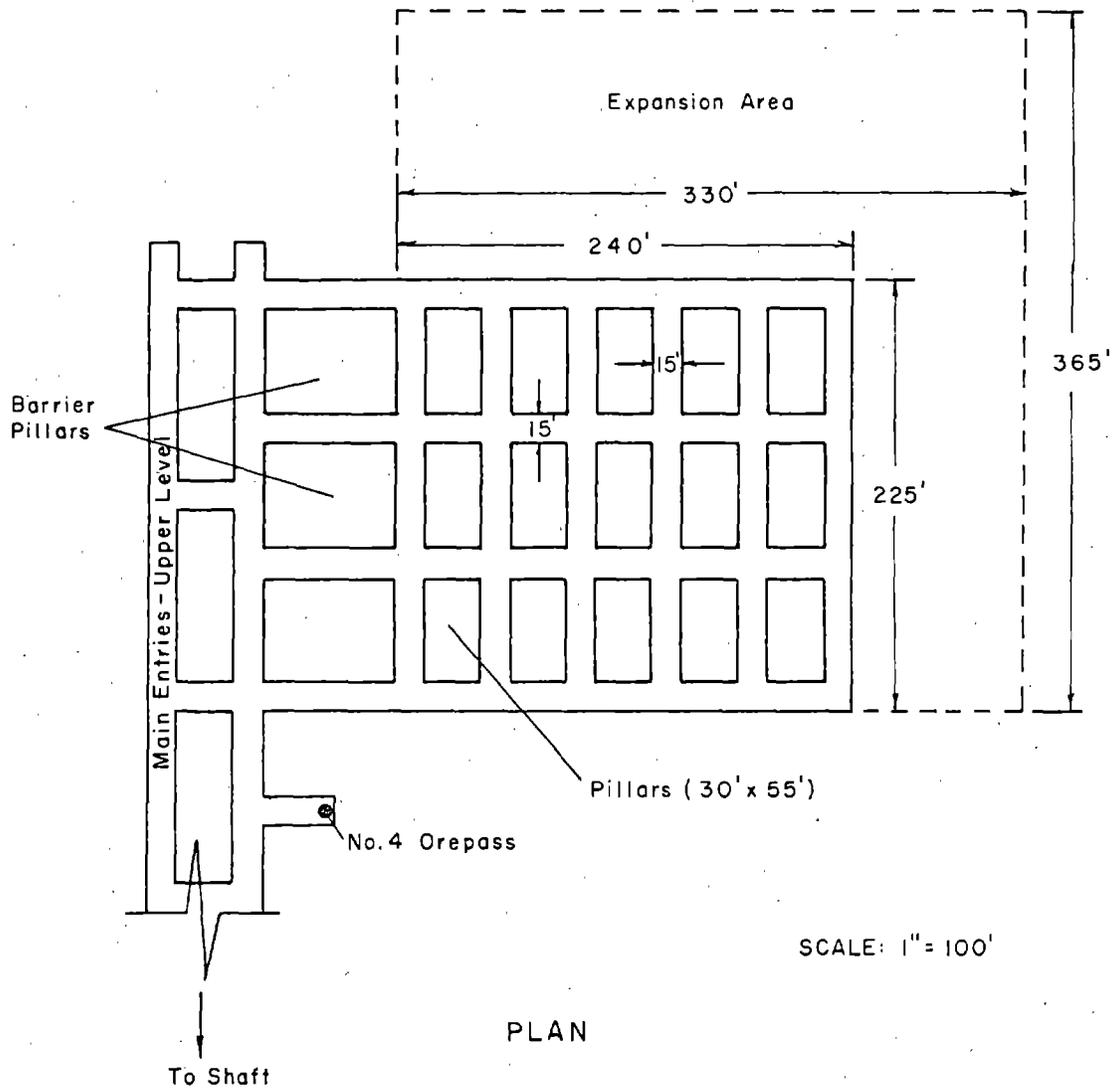


FIGURE 4.16
UNDERCUT LAYOUT - BLOCK CAVING

material from the stopes represents a workable method of regulating the extent and rate of caving. A summary of extraction data for development openings within the demonstration unit is presented in Table 4.5. The total amount of material to be withdrawn from the cave is indeterminate in advance of mining.

4.2.4.1 Unit Development

Two monitor levels will be established at elevations 35 feet and 75 feet above the block caving undercut level (Figure 4.17). Drifts will be driven along two sides of the block on each level. Crosscuts driven from several points along the drifts will extend into the block and will be used to observe the initiation and propagation of caving. All monitor drifts and crosscuts will be driven 8 feet high and 8 feet wide.

The dimensions of the block caving demonstration unit are determined by the size of the undercut. Entries and crosscuts in the undercut will be 15 feet by 15 feet in section. Pillars in the undercut area will be 30 feet by 55 feet. Extraction in the 225-foot by 240-foot area will be about 50% during development.

A series of perimeter cutoff holes will be drilled into the roof of the undercut level. The blastholes will be 3 inches in diameter, 25 feet deep, and spaced on 10-foot centers (closer spacing at the corners). A total of 100 holes will be drilled to complete the pattern.

To complete the undercut, pillars will be drilled and blasted in rows of three. As each row of pillars is shot, the perimeter cutoff holes tributary to the row of pillars will also be blasted to aid and initiate caving. A total of five blast rounds will be required to rubblize all 15 pillars and to shoot all 100 perimeter relief holes. Blasting will be conducted in a retreating sequence.

4.2.4.2 Unit Demonstration

Mining of the block caving demonstration unit will consist of mucking broken shale from the stope drawpoints beneath the block. Removal of material from the stopes will be closely supervised to assess the potential for controlling the upward propagation of the cave. Caving characteristics will be closely observed from the two monitor levels during the demonstration to determine:

1. The rate of cave propagation above the undercut level and the geometry of the cave.
2. The degree of success achieved in controlling the rate of caving.

TABLE 4.5

EXCAVATION SUMMARY
BLOCK CAVING

<u>Openings</u>	<u>Section Dimensions</u>	<u>Total Footage</u>	<u>Number Of Rounds</u>	<u>Tons Per Round</u>	<u>Total Tonnage</u>
Unit Development: Monitor Level Drifts and Crosscuts	8' x 8'	3,136	523	26.5	13,850
Room-and-Pillar Entries and Crosscuts	15' x 15'	2,230	223	155	<u>34,570</u>
TOTAL (Development Only)					<u><u>48,420</u></u>

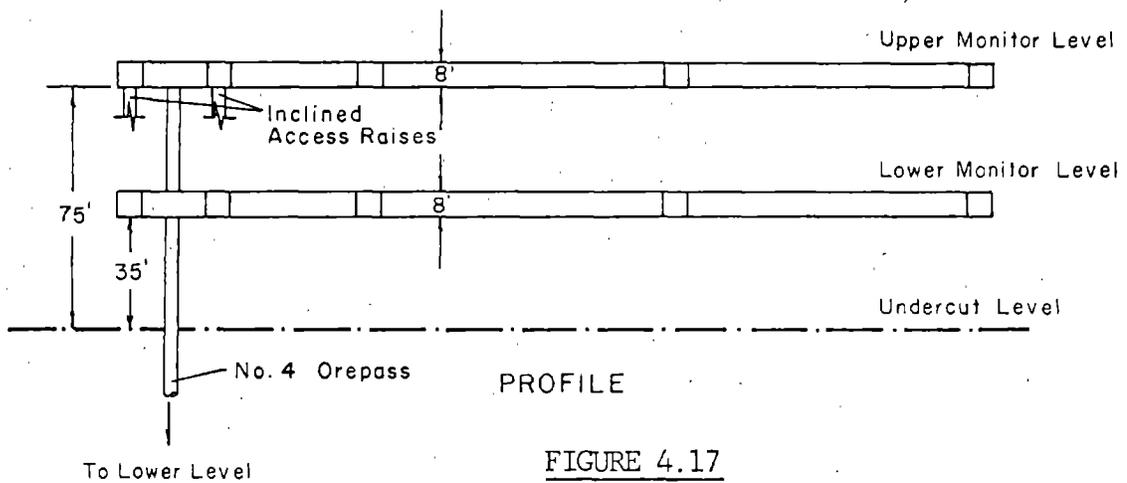
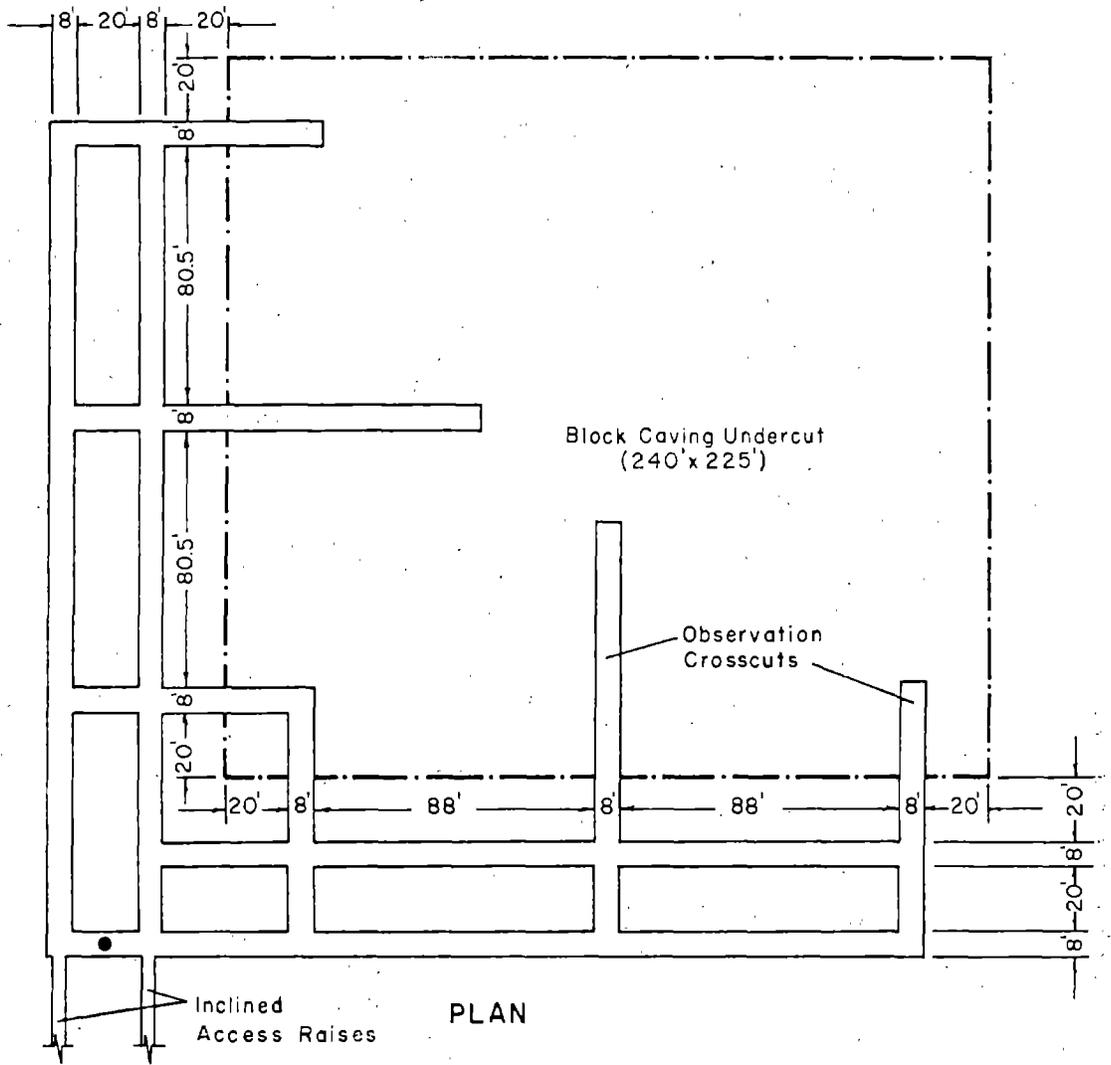


FIGURE 4.17
MONITOR LEVEL LAYOUT
BLOCK CAVING

3. The size consist of caved material.
4. The final configuration of the cave area at the conclusion of the demonstration.

All of these factors are pertinent for assessment of the overall suitability of mining deep oil shale deposits by the block caving method.

5.0 ENGINEERING ANALYSES OF MINING FUNCTIONS

Comprehensive engineering evaluations have been performed of the task sequence planned for various projected stages of primary development, unit development, and demonstration mining. The work cycles required to complete each mining task have been broken down into a number of basic functions for analysis, as follows:

1. Drilling and blasting
2. Loading and hauling
3. Heading preparation, including scaling, roof bolting, and cleanup

Raise boring and backfilling, which constitute fundamentally different activities from the typical "drill-blast-muck" cycle, were analyzed as separate functions.

In determining manshift allocations and requirements, an estimate of available productive work time of 400 minutes per shift has been used. Available productive work time per shift represents 100 percent utilization of available work time. It is a theoretical number and does not represent the expected productive work time. In order to determine realistic estimates of costs and productivity, a judgement factor has been applied to account for all unexpected delays (avoidable or unavoidable) by increasing all cycle times by 18 percent.

5.1 DRILLING AND BLASTING

5.1.1 Drifts and Ramps

Extensive drifting will be required to drive the 8.1 miles of ramps, entries, crosscuts, and drilling drifts of various dimensions specified in the demonstration mine design. Drilling and blasting to be performed during primary development and unit development consists almost exclusively of drift work; some roof and floor excavation will be required for construction of shaft station-area facilities on the lower level. A breakdown of footage requirements for each segment of development is provided in Table 5.1.

Development headings will be drilled by two-man crews. Diesel-powered, two-boom drill jumbos, equipped with independent-rotation percussion drifters will be used to drive all drifts and ramps except the 8-foot by 8-foot monitor drifts of the block caving unit. The jumbos also will be capable of drilling the near-vertical holes required to complete sump and roof excavations in the lower-level shaft station area.

TABLE 5.1

DRIFTS AND RAMPS
SUMMARY OF DEVELOPMENT FOOTAGES

	CROSS-SECTIONS					TOTAL
	<u>15' x 20'</u>	<u>15' x 15'</u>	<u>12' x 15'</u>	<u>12' x 12'</u>	<u>8' x 8'</u>	
Primary Development	6,970'	1,090'	12,090'	2,340'	-	22,490'
Chamber and Pillar With Backfilling	180	-	-	1,914	-	2,094
Sublevel Stopping With Backfilling	-	-	2,170	5,814	-	7,984
Sublevel Stopping With Full Subsidence	-	-	1,375	3,488	-	4,863
Block Caving	-	<u>2,230</u>	-	-	<u>3,136</u>	<u>5,366</u>
TOTALS	<u>7,150'</u>	<u>3,320'</u>	<u>15,635</u>	<u>13,556'</u>	<u>3,136'</u>	<u>42,797'</u>

Various drift dimensions have been specified in mine planning in order to achieve a cost-effective design consistent with specific operational requirements. Drill patterns designed for use in drifts of differing dimensions are shown in Figure 5.1. Drift rounds have been designed to optimize the length of pull and the fragmentation characteristics of each round, and also to minimize blast damage of the surrounding rock.

With the exception of the 14-hole pattern used to drive the block caving monitor drifts, all drill rounds will consist of a combination of 1.5- and 2-inch-diameter holes, all approximately 11 feet long. Perimeter holes will be 1.5 inches in diameter to minimize blast damage to the drift ribs and roof; inner holes and lifters will be 2 inches in diameter. These patterns are designed to yield an average advance of 10 feet per round. The round designs shown in Figure 5.1 have been used as a basis for computing drilling and charging cycles, tonnages, explosives consumption, manpower requirements, etc.

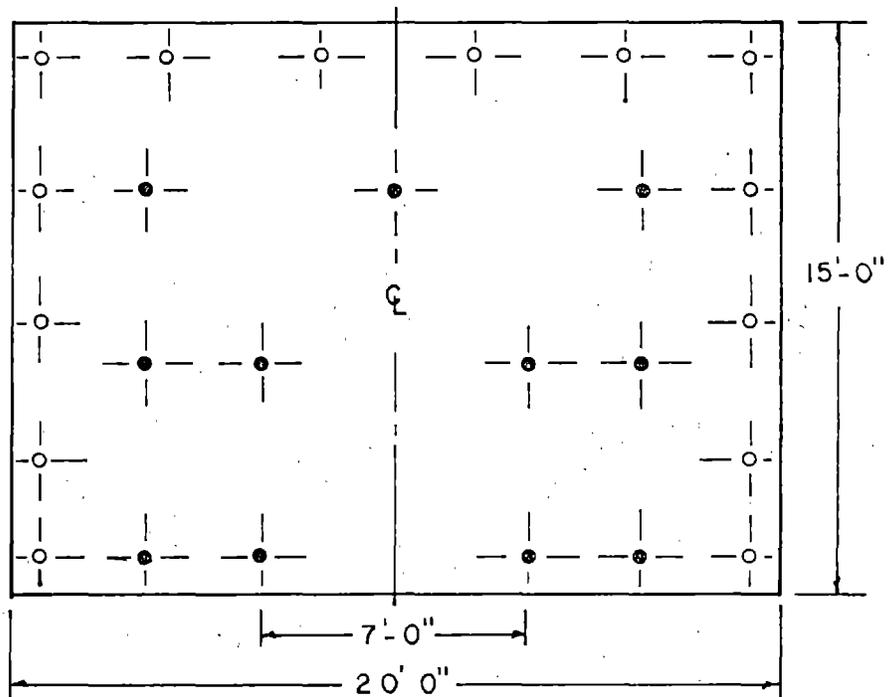
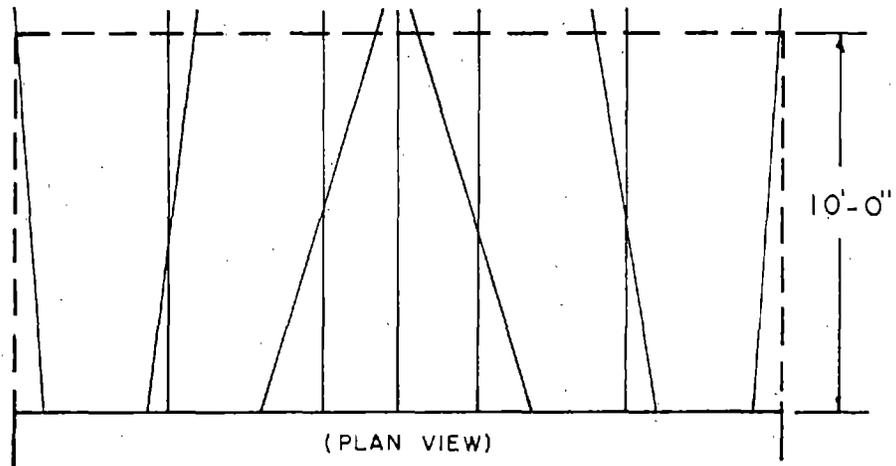
The block caving monitor drifts will be driven by two-man crews with jackleg drills. The 14-hole pattern will consist of 7-foot-long holes, 1.5 inches in diameter, and is designed to average 6 feet of advance per round.

Blastholes in drift rounds will be primed with a high-strength primer and a millisecond delay electric blasting cap and will be charged with a mixture of ammonium nitrate and fuel oil (AN/FO). Though not considered permissible, AN/FO will likely be a common explosive used in oil shale mines. A variance permitting its use will be required if the shale mines are classified as gassy.

Charging will be performed by two-man crews. Using a hand-held charging lance, the AN/FO prills will be blown into the holes from a pressure vessel of 500-pound capacity. Charging equipment will be mounted on a diesel carrier with a platform for use when charging upper holes. Portable charging equipment will be used in the monitor drifts. Table 5.2 provides a summary of drilling and charging data for drift rounds of all required dimensions.

Normally blasting will be performed at the end of each shift. On-shift blasting will be practical only under certain conditions. During early stages of primary development when the number of advancing headings is small, on-shift blasting may be employed to reduce nonproductive time between blasts. On-shift blasting also may be conducted to expedite development work in remote areas of the mine where access is easily controlled and ventilation is suitable.

Drilling and charging cycle times will vary between headings of different sizes. Calculated cycle times for the drilling and charging functions listed in Table 5.2 are based on operational experience and on evaluations performed for other mine design projects. Sample calculations of

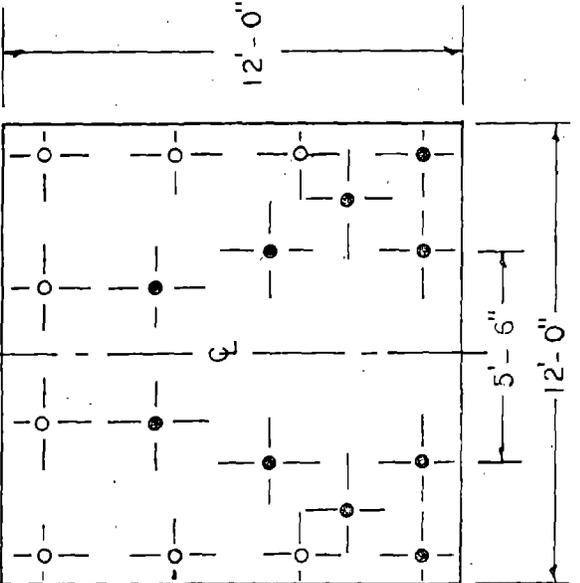
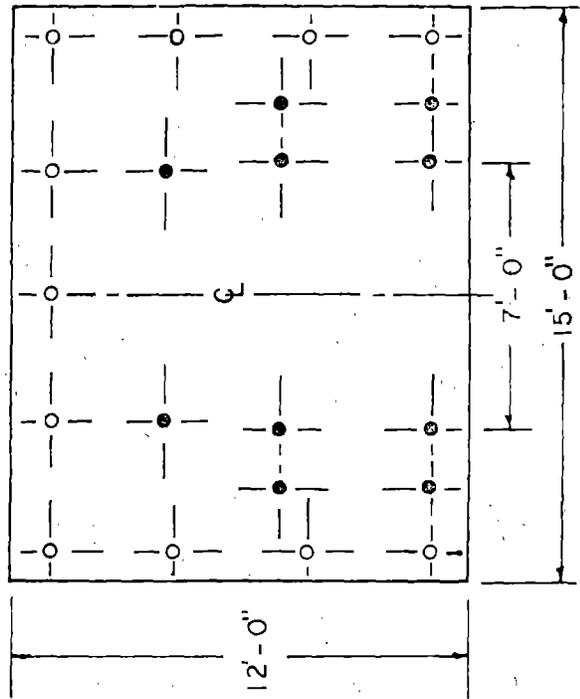
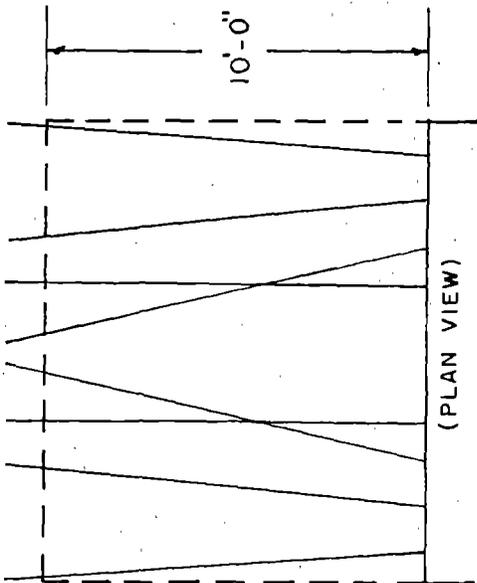
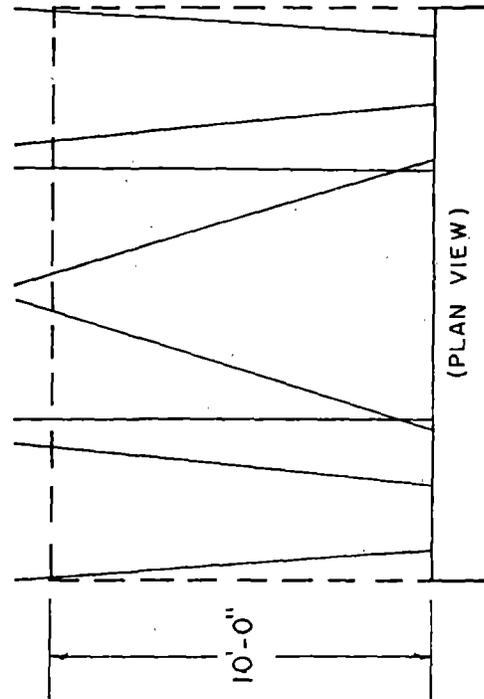


25-HOLE DRILL PATTERN

- 1.5" Diameter Hole
- 2" Diameter Hole

FIGURE 5.1
DRIFT ROUND DESIGNS

FIGURE 5.1 (Continued)

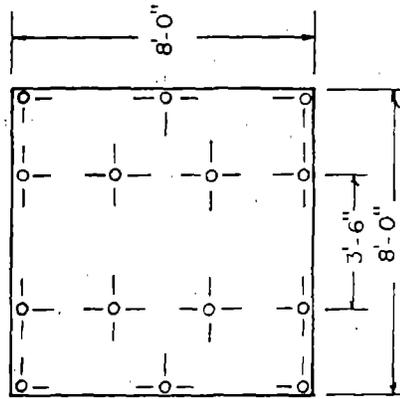
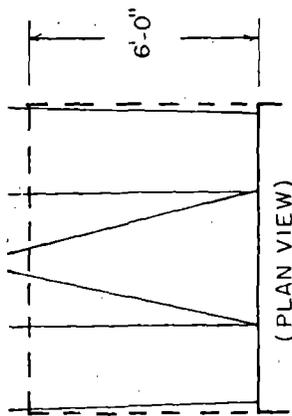
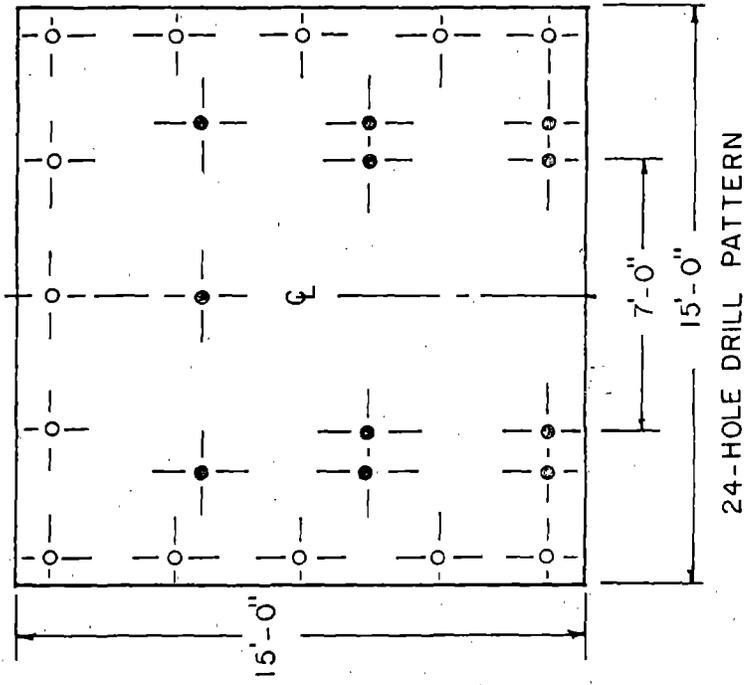
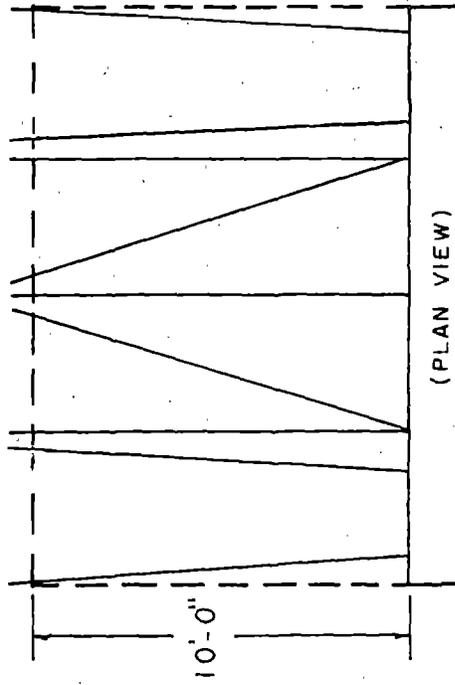


21-HOLE DRILL PATTERN

○ 1.5" Diameter Hole
● 2" Diameter Hole

18-HOLE DRILL PATTERN

FIGURE 5.1 (Continued)



- 1.5" Diameter Hole
- 2" Diameter Hole

TABLE 5.2

DRIFT ROUNDS - DRILLING AND BLASTING DATA

Heading Size (height & width)	Advance Per Round (ft)	Holes Per Round	Tons Per Round	Explosives Per Round AN/FO (lbs)	Explosives Per Round Primer (lbs)	Explosives Factor (lbs/ton)	Cycle Times Drilling (min)	Cycle Times Charging (min)
15' x 20'	10	25	207	175	12.5	0.91	198	113
15' x 15'	10	24	155	170	12.0	1.17	191	111
12' x 15'	10	21	125	150	10.5	1.28	181	103
12' x 12'	10	18	100	130	9.0	1.39	160	96
8' x 8'	6	14	26.5	49	7.0	2.09	144	85

typical drilling and charging cycles (for a 15-foot by 20-foot heading) are as follows:

Drilling Cycle:

Single-Hole Cycle

Position boom and collar hole	1.5 min.
Drill 11 feet at 3 fpm	3.7 "
Clean hole and retract	0.5 "
	<u>5.7 min/hole</u>

Move in and set up	20 min
Mark face	10 "
Muck for lifters	10 "
Drill 13 holes x 5.7 min/hole ÷ 0.9 (interference factor)	82 "
Tear down and move out	15 "
Service jumbo (15 min/shift prorated)	8 "
Miscellaneous delays (45 min/shift prorated)	23 "
Theoretical cycle time	<u>168 min</u>
Efficiency factor (18% of theoretical cycle time)	<u>30 "</u>
 Total drilling cycle time	 198 min

Charging and Blasting Cycle:

Single-Hole Cycle

Make Primer	0.5 min
Insert primer and charge hole	1.0 "
Interconnect leg wires	0.5 "
	<u>2.0 min/hole</u>

Move in and set up	10 min
Charge 25 holes x 2.0 min/hole	50 "
Run leads to blasting trunk line	10 "
Tear down and move out	5 "
Delays (20 min/shift prorated)	6 "
Theoretical cycle time	<u>96 min</u>
Efficiency factor (18% of theoretical cycle time)	<u>17 "</u>
 Total charging and blasting cycle time	 113 min

5.1.2 Demonstration Excavations

Blast rounds for the excavation of chambers and stopes and the recovery of pillars are specific designs utilizing long-hole fan and ring drilling patterns. In contrast to the drift rounds described earlier, the present category includes rounds designed to perform specialized mining tasks which will result in the breakage of large volumes of rock per round. One or more of these specific round designs will be employed in mining each of the demonstration units. In the following discussion, these blast rounds will be described in an order corresponding to the sequence planned for mining the various units. A summary of rounds and tonnage data for demonstration excavations is presented in Table 5.3.

5.1.2.1 Slot and Chamber Rounds - Chamber and Pillar Mining With Backfill Unit

Slot development is the final task of unit preparation to be completed prior to demonstration mining of the chambers. In each chamber, a 33-foot-long, 52-inch-diameter slot raise will be bored between the LHD entry and the overlying crosscut on the chamber drilling level (Figure 4.9). The slot raise will be expanded by conventional drilling and blasting to establish a 10-foot-wide slot opening extending the full height and width (60 feet by 70 feet) of the chamber. Drilling will be performed by a ring-drilling jumbo from the overlying crosscut.

Expansion of each slot raise will be accomplished by two blast rounds, as indicated in Figure 5.2. The initial enlargement round will consist of 14 holes drilled on 5-foot centers around the raise in a 10-foot by 20-foot pattern. These holes will be 3 inches in diameter and approximately 30 feet in length. Twelve additional 3-inch blastholes will compose the second enlargement round. Holes of the second round will be approximately 49 feet long extending from the drilling level to the chamber floor.

The 30-foot holes of the initial slot round will be charged with an average of 22 feet of AN/FO, a high-strength primer and an electric cap. The explosive column in the longer, 49-foot holes of the second slot enlargement round will be 35 feet long and will be double primed to ensure complete detonation. Two-inch-I.D. tubes will be inserted in the rib holes at either end of the slot prior to charging. Two-inch-diameter tubes, when placed in rib holes and filled with AN/FO, will result in reduced blast damage to the relatively thin rib pillars. This result is achieved by use of less explosive, as well as by the occurrence of a decoupling effect between the explosive and the drill hole wall.

Chamber mining will be accomplished by drilling and blasting eight-blasthole fans into the chamber area from each of the two chamber drilling drifts (Figure 5.3). The 3-inch-diameter blastholes will

TABLE 5.3

DEMONSTRATION UNITS - DRILLING AND BLASTING STATISTICS

	Round	Cycle Times (min.)		Number of Rounds	Tons	
		Drilling	Charging		Per Round	Total
<u>Chamber and Pillar Mining with Backfill:</u>						
1. Slots	First	570	93	3	420	1,260
	Second	755	157	3	1,655	4,970
2. Chambers	First	762	202	3	2,645	7,940
	Other	860	332	27	5,290	142,830
<u>Sublevel Stopping with Backfill:</u>						
1. Slots	First	950	266	3	505	1,520
	Second	1,926	498	3	4,405	13,220
2. Stopes	First	358	80	6	1,080	6,480
A. Stope Floor	Other	385	160	33	2,160	71,280
B. Sublevel	Each	892	400	36	6,560	236,160
C. Upper Level	Each	326	146	36	2,160	77,760
<u>Sublevel Stopping with Full Subsidence:</u>						
1. Slots	First	1,051	265	2	590	1,180
	Second	1,334	356	2	4,030	8,060
2. Stopes	First	313	71	2	1,080	2,160*
A. Stope Floor	Other	365	142	22	2,160	47,520*
B. Sublevel	First	854	219	2	4,070	8,140*
	Other	985	438	22	8,345	183,590*
3. Pillar Recovery	First	4,342	1,561	1	26,365	26,365*
A. First Crown Pillar	Second	3,788	1,458	1	26,485	26,485*
B. Rib Pillar and Second Crown Pillar	First	6,008	2,289	1	39,850	39,850*
	Second	10,444	3,915	1	71,990	71,990*
	Third	6,329	2,365	1	42,420	42,420*
<u>Block Caving:</u>						
1. Pillar Blasting	Each	611	266	5	5,120	25,600*
2. Perimeter Outoff Holes	Each	432	245	5	-	-

* Tonnes not fully withdrawn from stopes.

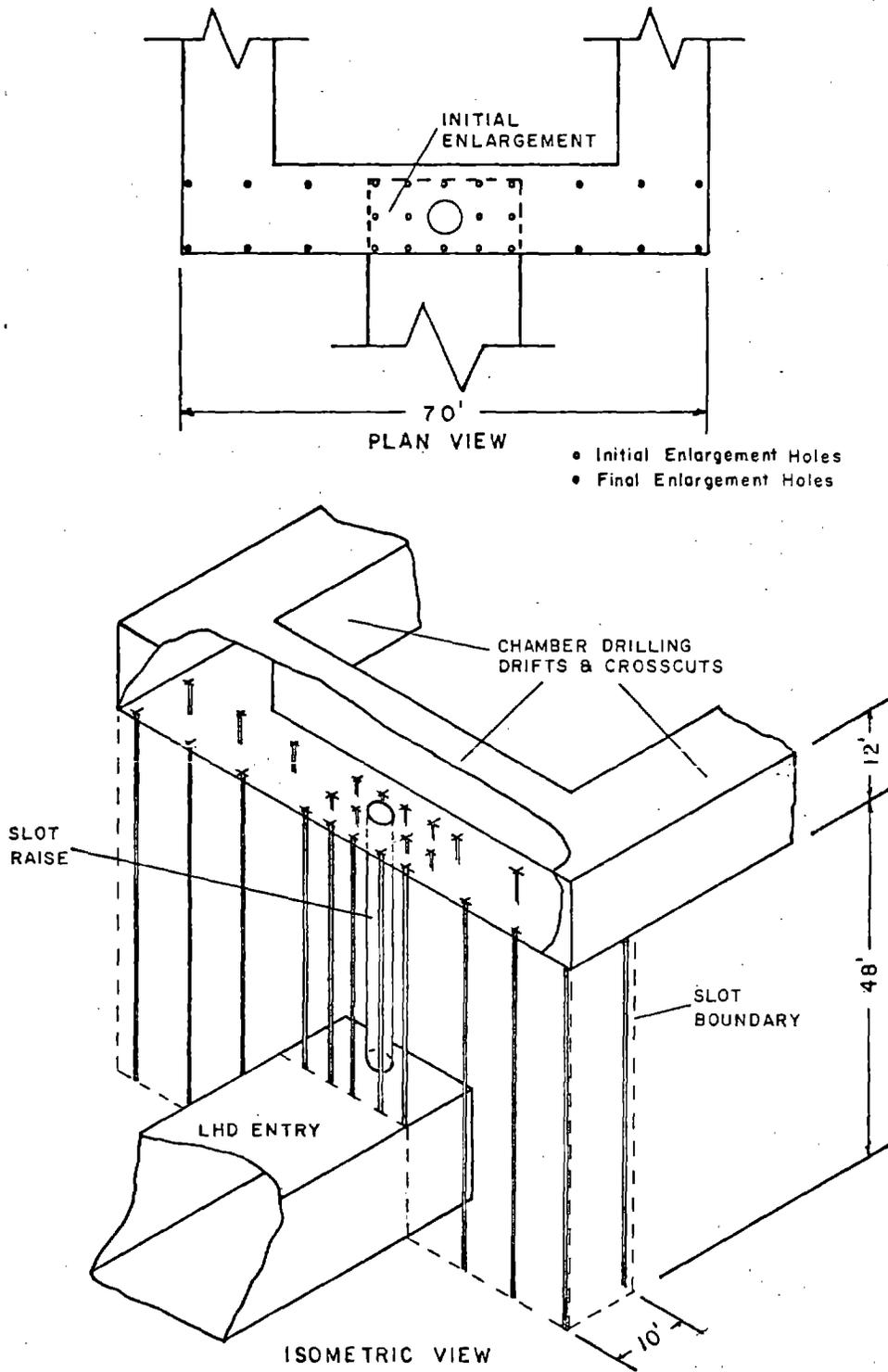
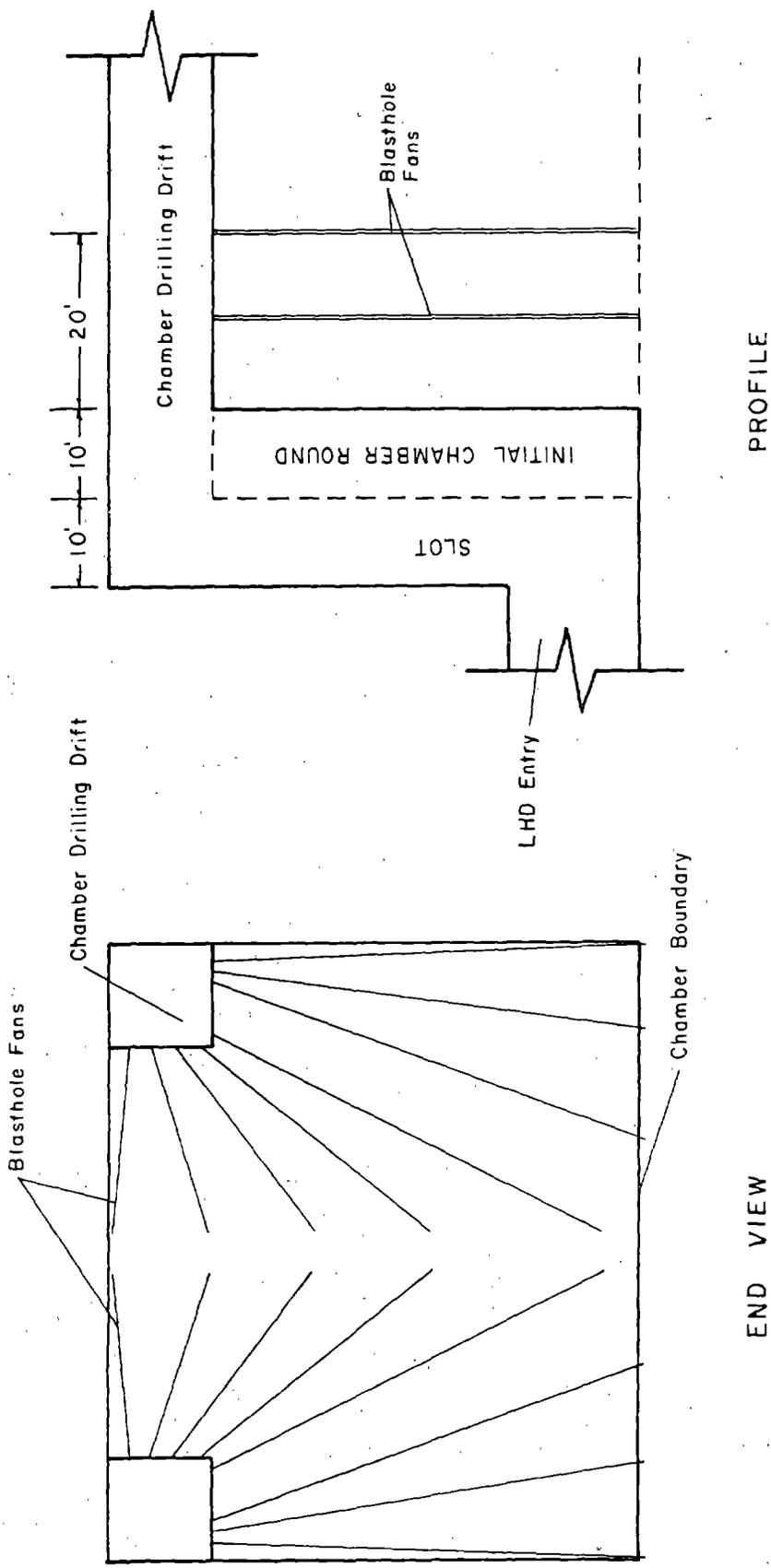


FIGURE 5.2
SLOT DRILLING PATTERN
CHAMBER AND PILLAR MINING



PROFILE

END VIEW

FIGURE 5.3
 CHAMBER DRILLING PATTERN
 CHAMBER AND PILLAR MINING

vary from about 22 to 54 feet in length. The initial chamber mining round will consist of a single fan of holes drilled from each chamber drilling drift and subsequent chamber rounds will consist of two fans radiating from either drilling entry. The burden between fans will be 10 feet. A ring-drilling jumbo, equipped with two independent-rotation percussion drill machines and operated by a two-man crew, can drill two 90° fans simultaneously from a single setup. Drilling positions will alternate between the left and right drilling drifts to complete each round.

The initial round to be blasted will consist of a single pair of fans, advancing the chamber face 10 feet and providing expansion room for subsequent larger blasts. Blasting two pairs of fans will result in a chamber face advance of 20 feet. Blastholes will be loaded with AN/FO using pneumatic charging equipment. Short holes, containing less than 20 feet of explosive, will be primed with a single, high-strength primer and an electric cap. Longer columns will require double priming. Two-inch-I.D. tubes will be inserted in rib holes to minimize blast damage sustained by the rib pillars.

5.1.2.2 Slot and Stope Round - Sublevel Stopping With Backfill Unit

A 52-inch diameter bored raise will be enlarged to form a slot opening at the rear of each stopping area to facilitate subsequent stope blasting. The initial enlargement round will consist of 24 3-inch blastholes drilled from the stope floor and from the sublevel. All holes in the initial round will be approximately 30 feet long and will be oriented parallel to the raise-bore centerline (Figure 5.4). Blasting will produce an opening measuring 10 feet square extending from the stope floor upward approximately 102 feet.

Final enlargement of the slot to the full width and height of the stope (80 by 132 feet) will require drilling 3-inch blastholes from the stope floor, sublevel, and upper level (Figure 5.4). From the stope floor drilling drift, two symmetrical fans will be drilled, each consisting of six blastholes from 31 to 48 feet in length. From the sublevel, a series of 32 vertical upholes and downholes will be completed. Holes in the sublevel pattern will range between 21 and 28 feet in length. Drilling for the final enlargement round will be concluded with two six-hole fans and eight vertical holes drilled from the upper level. The vertical holes will be approximately 30 feet long and the fan holes will range from 32 to 45 feet in length. The final enlargement round will be blasted sequentially, with the lower level fans to be shot first and the upper level holes shot last. As before, holes with charged columns of AN/FO longer than 20 feet will be double primed to ensure complete detonation. Rib holes collared on the sublevel will be lined with 2-inch-I.D. tubes prior to charging.

The drilling patterns which will be employed to excavate the three stopes are illustrated in Figure 5.5. Fan drilling will be performed concurrently from drilling drifts on the stope floor, the sublevel and

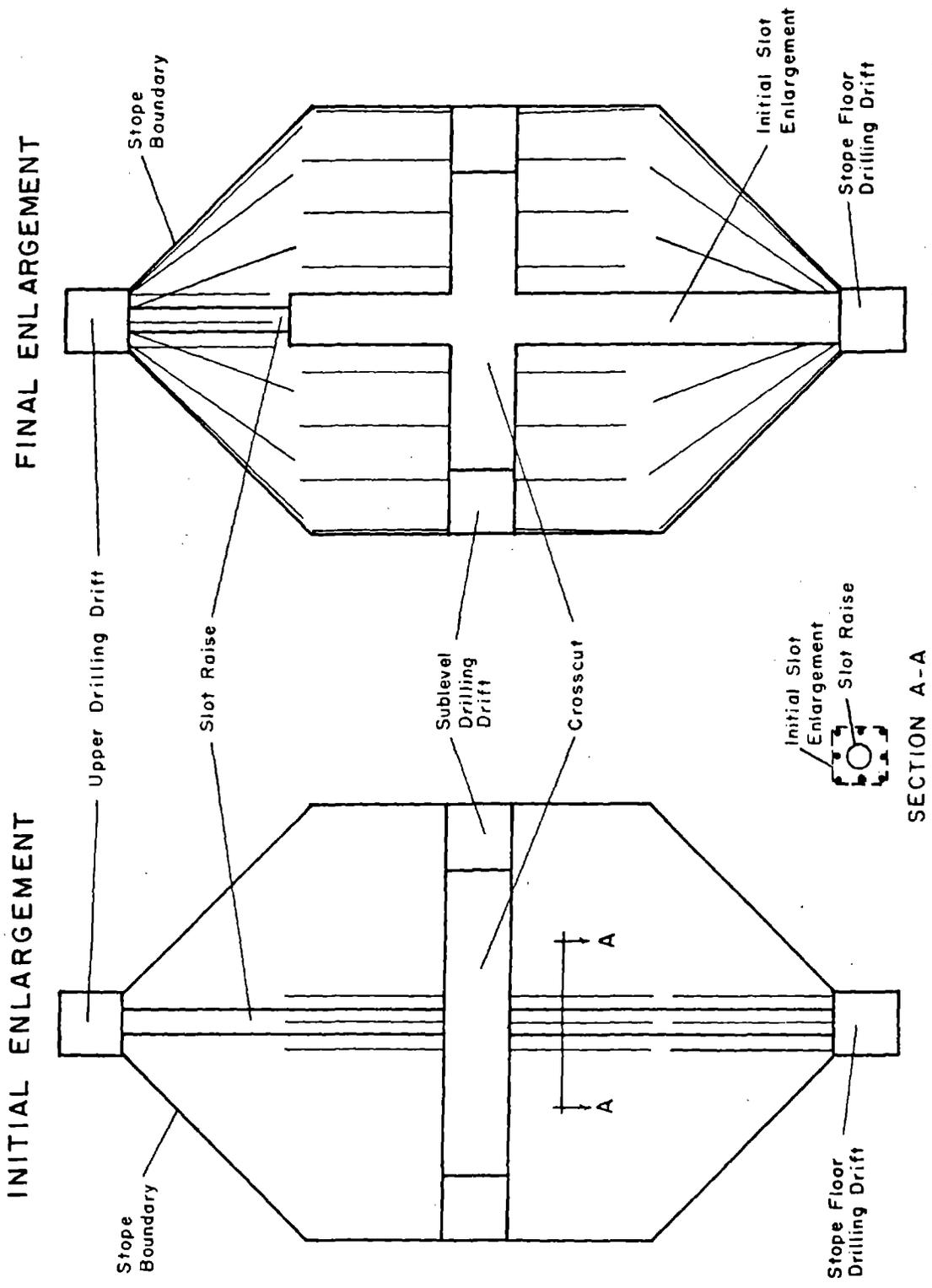
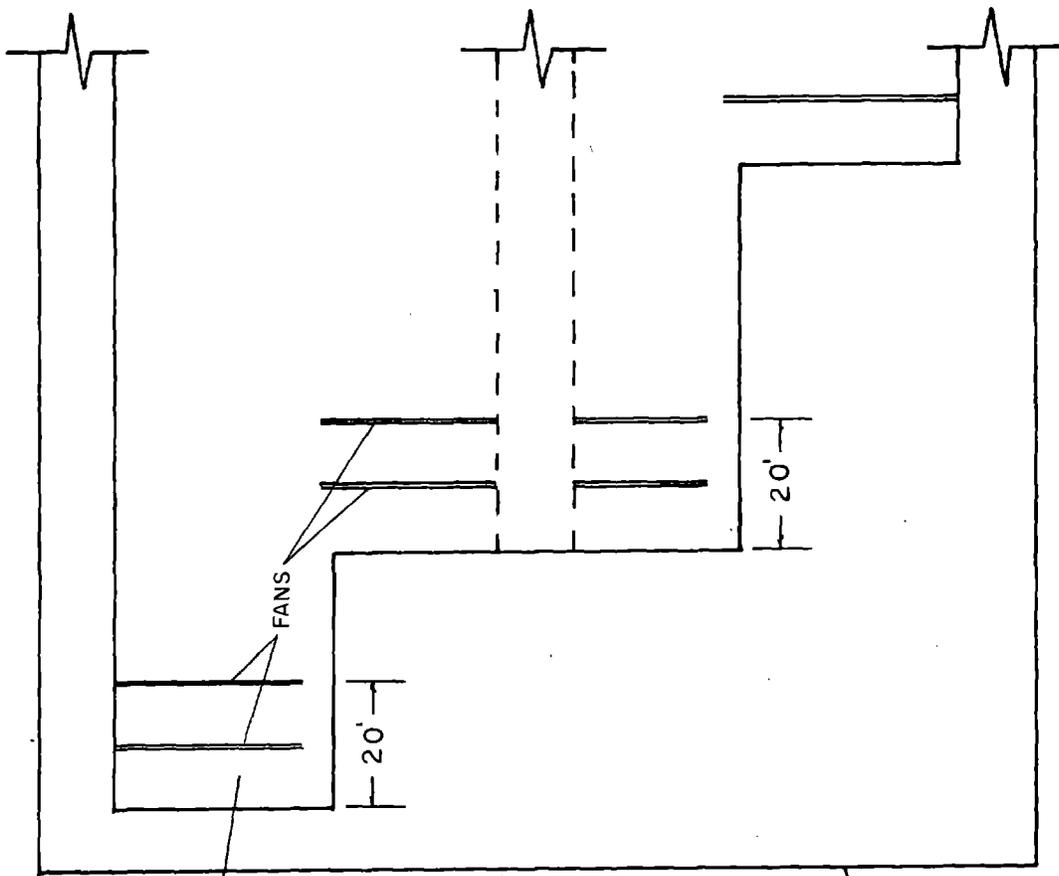
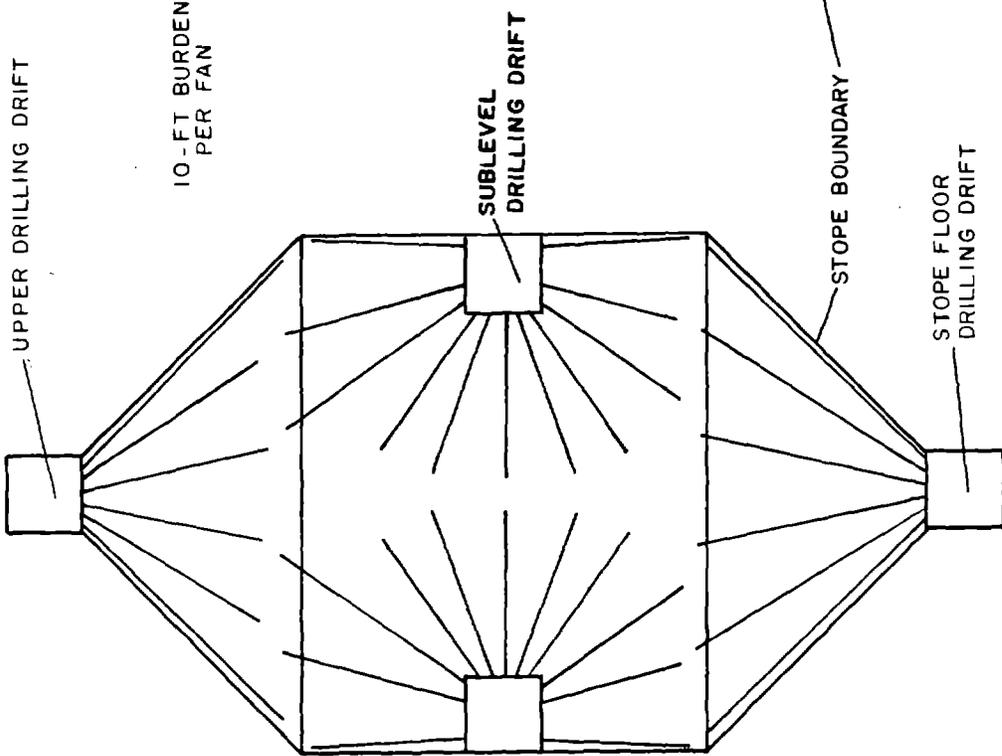


FIGURE 5.4
SLOT DRILLING PATTERN
SUBLEVEL STOPPING WITH BACKFILL



LONGITUDINAL SECTION



END SECTION

FIGURE 5.5
STOPE DRILLING PATTERN
SUBLEVEL STOPPING WITH BACKFILL

the upper level. However, blasting of rounds drilled from the different levels will be performed independently. The retreating stope faces on the three levels will be offset from one another by approximately 50 feet to minimize encroachment of the muckpile on stope floor drilling positions. The offset will provide an additional free face when blasting sublevel and upper level rounds.

The lower 35 feet of the stope will be drilled from the stope floor. Each six-hole fan will consist of two 38-foot, 42-foot, and 50-foot-long holes. The initial two rounds to be blasted from the stope floor will be single-fan rounds to establish expansion room for subsequent and larger blasts. Thereafter, two fans will be blasted concurrently, advancing the lower stope face 20 feet per round.

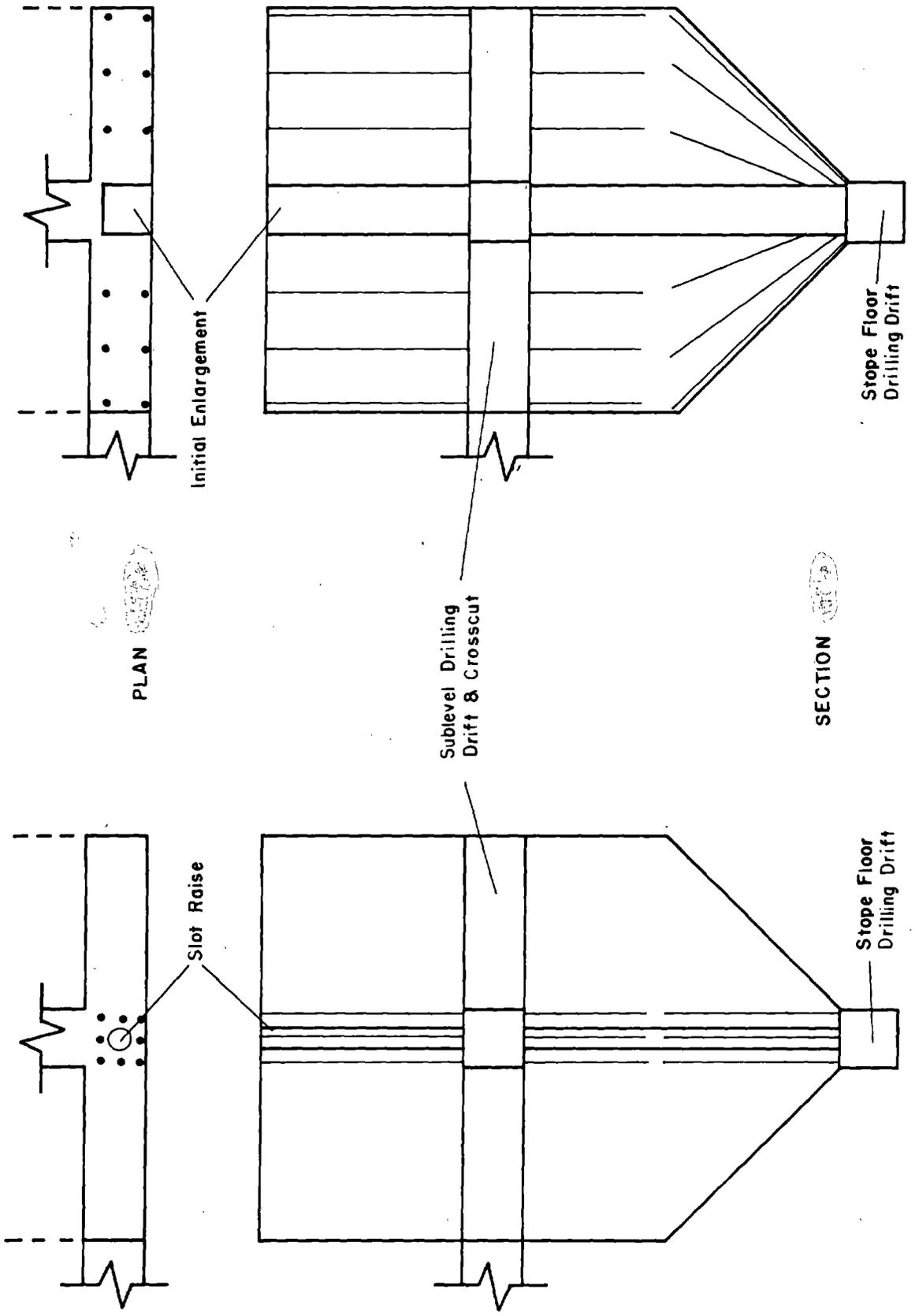
Each round drilled on the sublevel will consist of two pairs of 11-hole fans, drilled from opposite sides of the stope. The fans will be made up of blastholes of various lengths from 21 to 34 feet. As in chamber blasting, 2-inch-diameter tubes will be inserted into rib holes of each fan prior to charging to minimize the blast damage sustained by rib pillars.

All rounds on the upper level will be composed of two six-blasthole fans, each one consisting of two 30-foot, 35-foot, and 40-foot-long holes.

5.1.2.3 Slot and Stope Rounds - Sublevel Stopping With Full Subsidence Unit

Fifty-two-inch-diameter raises in the two stopping areas will provide blasthole relief during the subsequent excavations of slot openings (Figure 4.15). Initially, each raise will be enlarged to form an opening 10 feet square by drilling and blasting eight 34-foot-long upholes from the stope floor and eight upholes and eight downholes, 40 feet and 28 feet respectively, from the sublevel (Figure 5.6). The dimensions of the opening will be extended in a second stage of drilling and blasting to complete slot development. From the stope floor two fans of drillholes, ranging in lengths from 36 to 48 feet, will be blasted to enlarge the lower portion of the opening. From the sublevel 24 vertical blastholes drilled in the pattern indicated in Figure 5.6 will complete the enlargement round. Upholes drilled from the sublevel will be either single or double primed, depending upon the lengths of the AN/FO columns. The stope floor holes will be blasted first, to permit expansion of the volume of rock rubblized during the larger second shot above. All holes in the two enlargement rounds will be 3 inches in diameter.

Excavation of the stopes will be accomplished by drilling six-hole fans from the stope floor and 20-hole rings from the sublevel (Figure 5.7). The burden between adjacent fans or rings will be 10 feet. Lengths of holes will range from 35 to 48 feet in the fans, and from 27 to 52 feet in the rings. All holes will be 3 inches in diameter.



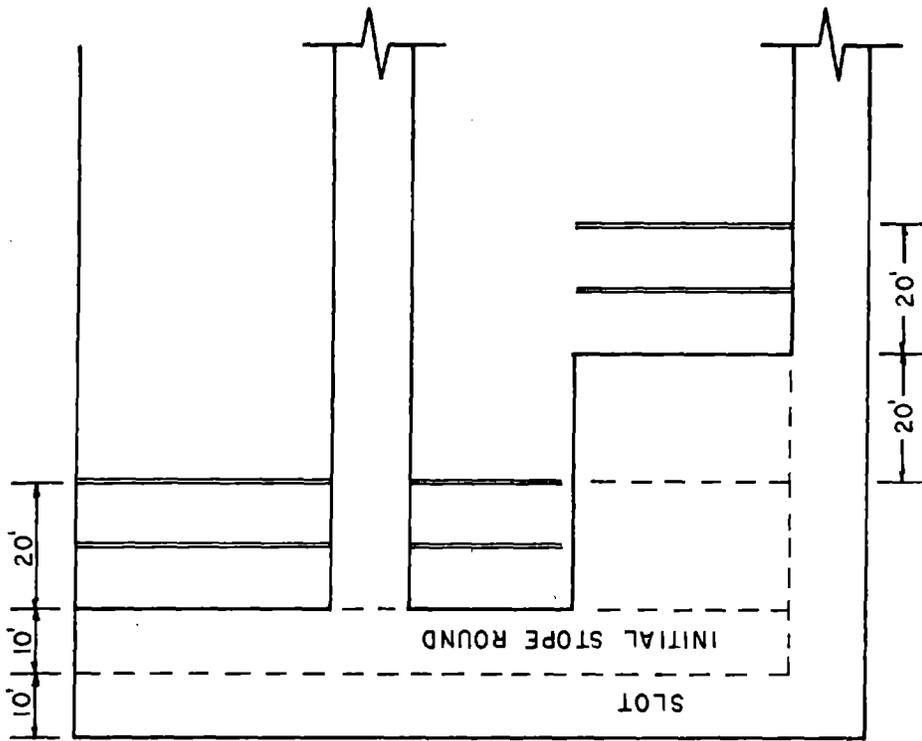
PLAN

SECTION

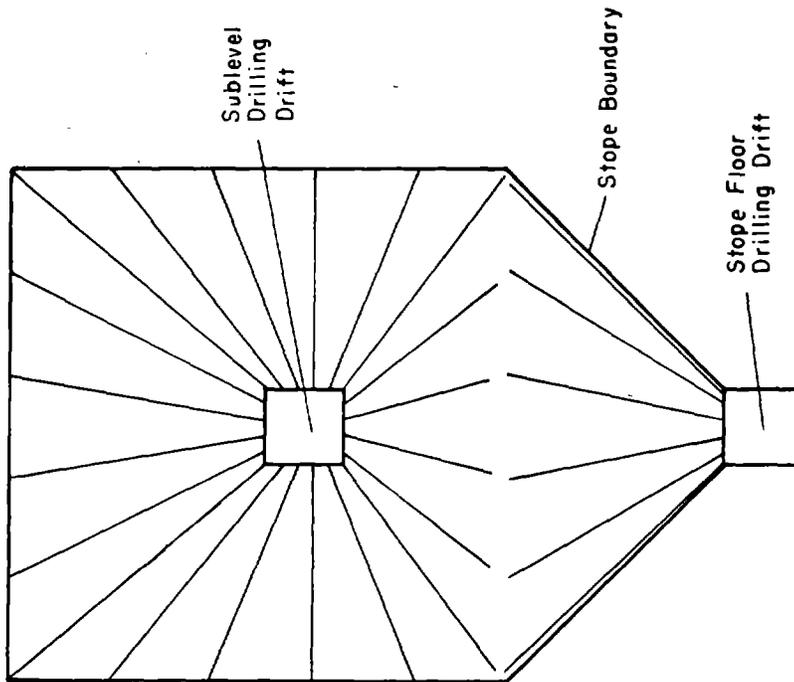
INITIAL ENLARGEMENT

FINAL ENLARGEMENT

FIGURE 5.6
 SLOT DRILLING PATTERN
 SUBLEVEL STOPPING WITH FULL SUBSIDENCE



PROFILE



END VIEW

FIGURE 5.7
 SLOPE DRILLING PATTERN
 SUBLEVEL STOPPING WITH FULL SUBSIDENCE

During excavation of the stopes, the free face above the stope floor level will be advanced about 50 feet ahead of the sublevel. The first round to be blasted on each level will involve a single ring or fan. In all subsequent rounds, two rings or fans will be blasted concurrently, yielding a face advance of 20 feet per round. The procedure for charging the holes will be as stated earlier.

5.1.2.4 Pillar Blasting on the Undercut Level - Block Caving Unit

Conventional room-and-pillar mining on the block caving undercut level will be conducted to block out five rows of three pillars each (Figure 4.16). When this development work has been completed, the pillars will be drilled and blasted in a retreating sequence. Concurrently, a series of perimeter cut-off holes will be drilled into the roof of the undercut and blasted. The purpose of these tasks is to initiate caving of the overlying block in an orderly and predictable manner.

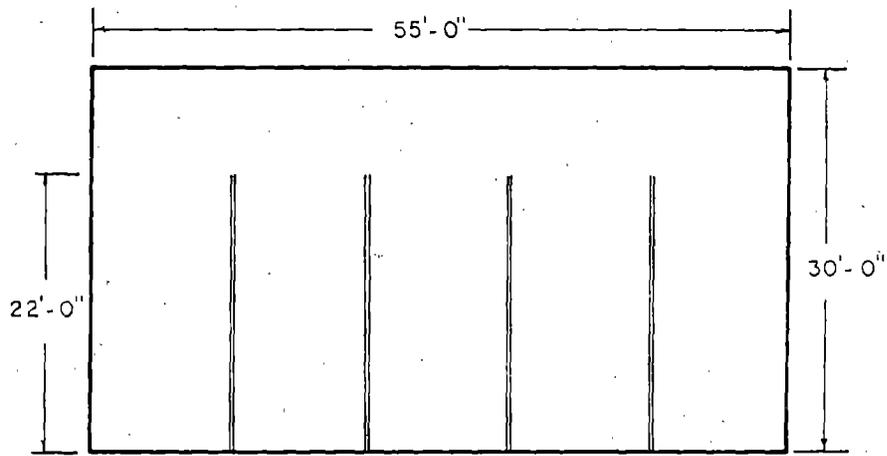
Five blast rounds will be employed to rubblize pillars and to establish the perimeter of the caving block. One row of three pillars will be drilled and charged per round. A pattern consisting of four vertical rows of three 3-inch blastholes will be drilled into each pillar by a two-man crew using a ring-drilling jumbo (Figure 5.8). Each blasthole will be approximately 22 feet long and will contain a column of AN/FO approximately 16 feet long when charged.

The perimeter cut-off holes will be 25 feet long, 3 inches in diameter and will be charged with 20-foot columns of AN/FO. A total of 100 blastholes will be required to complete the perimeter pattern. Holes will be drilled on 10-foot centers except at the corners of the excavation where a closer spacing will be required.

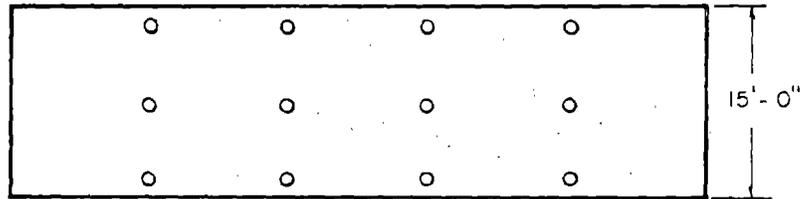
5.1.2.5 Pillar Recovery - Sublevel Stopping With Full Subsidence Unit

Pillar recovery will be accomplished in two stages. The first stage will consist of drilling and blasting the crown pillar spanning one stope. The crown pillar above the second stope and the 60-foot-wide rib pillar situated between the stopes will be extracted together during the second stage.

Two rounds will be blasted to recover the first crown pillar. The pillar will be mined by drilling six-hole fans in the pattern shown in Figure 5.9. The holes will range in length from 50 to 107 feet. A two-man crew operating a ring-drilling jumbo will be able to drill two fans of blastholes simultaneously from the crown pillar drilling drift. The two blast rounds will consist of 13 fans and 12 fans of holes, respectively.



PLAN



ELEVATION

SCALE: 1" = 15'

FIGURE 5.8
PILLAR BLAST ROUND
UNDERCUT LEVEL-BLOCK CAVING UNIT

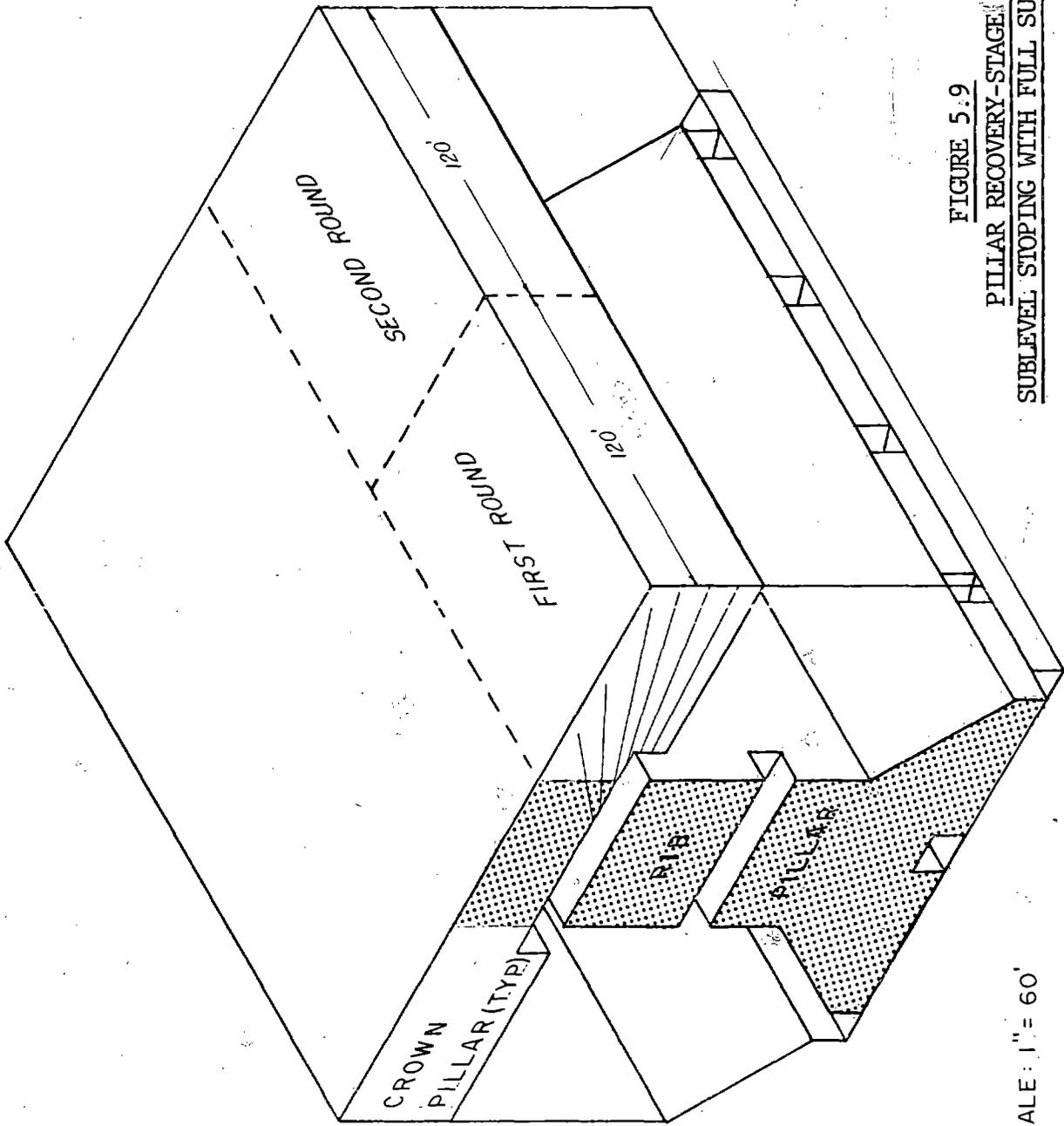


FIGURE 5.9
 PILLAR RECOVERY-STAGE I
 SUBLEVEL STOPPING WITH FULL SUBSIDENCE

SCALE: 1" = 60'

Fragmentation of the second crown pillar and the rib pillar will be undertaken in three separate sections. The sections will be rubblized in a retreating sequence, beginning with the 62-foot-long segment at the rear of the demonstration unit. Respective lengths of the second and third segments will be 112 and 66 feet.

In the first section a total of seven crown pillar fans and 10 rib pillar fans will be drilled by two-man crews operating ring-drilling jumbos. The pattern for drilling the crown pillar fans will be the same pattern used to mine the first crown pillar. As before, the six-hole fans will be spaced 10 feet apart. The rib pillar fans will be made up of five eight-hole fans drilled from stub crosscuts on either side of the crown pillar drilling drifts and five 16-hole fans drilled from crosscuts on the sublevel (Figure 5.10). The eight-hole fans in the upper portion of the rib pillar will consist of 3-inch-diameter blastholes ranging from 23 feet to 52 feet in length. The 16-hole fans in the lower portion of the pillar will be made up of 3-inch holes from 26 to 60 feet long. Both groups of fans will be spaced 10 feet apart through the 60-foot width of the pillar.

The second or middle pillar section to be mined will require 11 crown pillar fans, five upper rib pillar fans, and five rib pillar fans. Each of the 11 fans drilled into the crown pillar will contain six holes ranging from 50 to 107 feet in length. Each of the upper rib pillar fans will consist of 15 holes from 23 feet to 52 feet in length, and the rib pillar rings will be made up of 30 holes 26 feet to 60 feet long.

Mining the third and final section of the crown and rib pillars will require seven crown pillar fans and 10 rib pillar fans. The seven crown pillar fans will be drilled using the six-hole pattern previously described. Each of the upper rib pillar fans will contain nine holes, from 23 feet to 52 feet long, and each of the lower rib pillar fans will consist of 18 holes ranging from 26 feet to 60 feet in length.

5.2 LOADING AND HAULING

5.2.1 Primary Development

Five-yard-capacity load-haul-dump units (LHD's) will be used to haul material from headings and orepasses during primary development. During excavation of the shaft-station area on the lower level, LHD's will discharge material into a skid-mounted, feeder-breaker type crusher positioned at the edge of the shaft storage pocket. When the lower-level main entries have been extended a distance of 450 feet from the shaft, the feeder-breaker will be repositioned at the head of a 345-foot-long fixed belt conveyor. Thereafter, the belt will be extended in 345-foot increments to avoid extreme LHD haul distances.

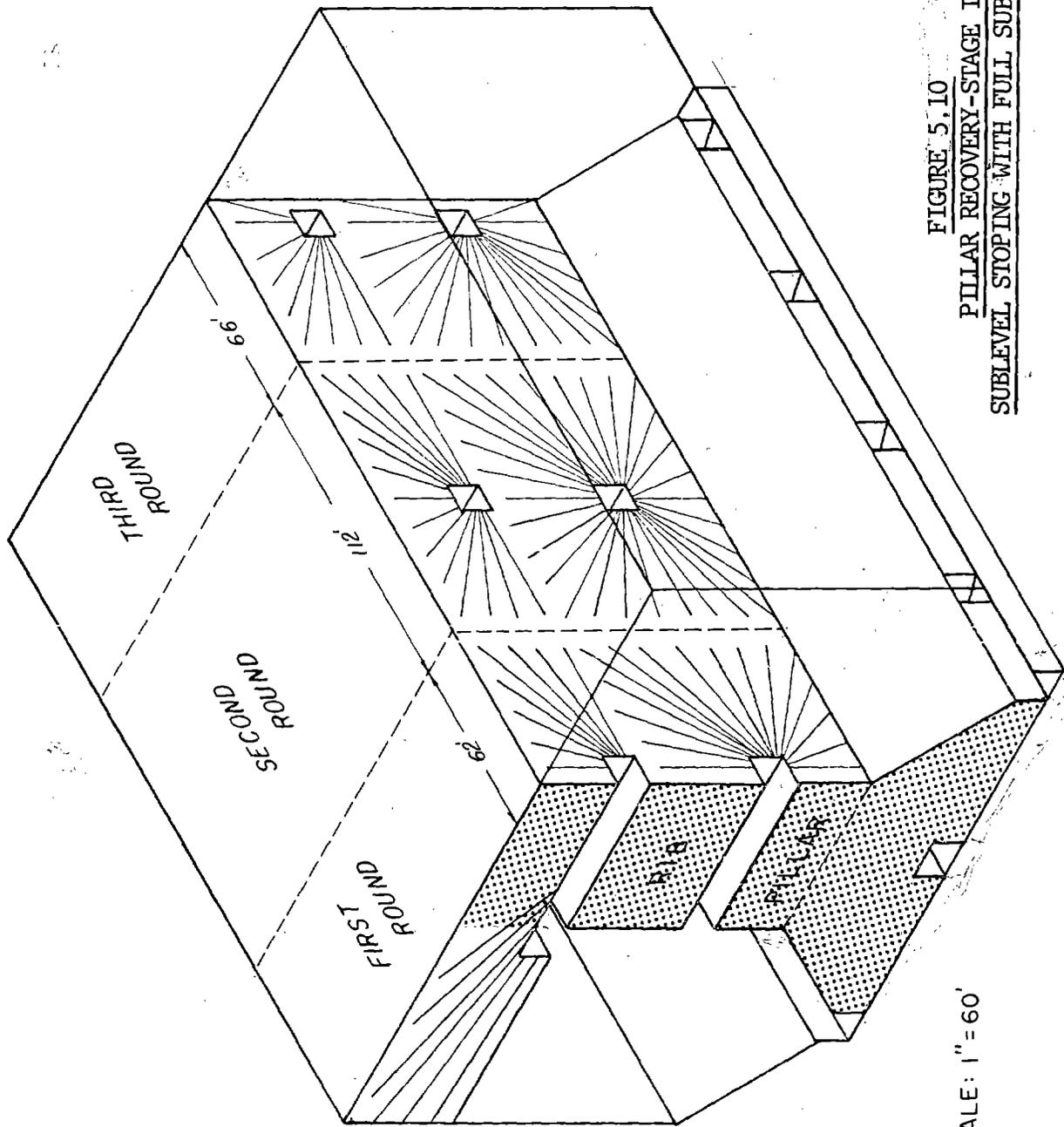


FIGURE 5.10
PILLAR RECOVERY-STAGE II
SUBLEVEL STOPPING WITH FULL SUBSIDENCE

SCALE: 1" = 60'

During development work in the vicinity of the upper-level shaft station, material will be delivered from headings to #1 orepass which will empty directly into the shaft storage pocket. Three additional orepasses will be completed between the upper and lower levels for removal of material produced during development work above the (lower) haulage level. Material from #2, #3, and #4 orepasses will be picked up on the haulage level by LHD's and transferred to the belt feeder.

Cycle times for LHD's are dependent upon the haulage distance. Total time requirements for mucking a heading will vary with both the haulage distance and the heading dimensions, i.e., the tonnage broken per round. Table 5.4 lists calculated LHD work cycles for various haul distances from 100 to 1,000 feet. The time required to muck a typical heading (15 feet high by 20 feet wide, located 500 feet from the dump point) is determined as shown in the following example:

Mucking Cycle:

Load-Haul-Dump Cycle

Doze muckpile	0.09 min
Fill bucket	0.16 "
Back and turn (twice)	0.39 "
Haul 500 feet (4.5 mph)	1.26 "
Dump bucket	0.08 "
Return 500 feet (5 mph)	1.14 "
Service machine (15 min/shift prorated)	0.16 "
Miscellaneous delays (30 min/shift prorated)	0.32 "
Theoretical cycle time	3.60 min
Efficiency factor (18% of theoretical cycle time)	0.65 "
LHD cycle time	4.25 min

207 tons/heading ÷ 5 tons/LHD trip = 42 trips

42 trips x 4.25 min/trip = 178.5 minutes

To calculate LHD requirements when mucking on development ramps, cycle times in Table 5.4 were increased by 20%. For a ramp heading with dimensions and haul distance specified in the foregoing example, the mucking cycle would be 214.2 minutes.

5.2.2 Demonstration Units

Cycle times have been calculated for the various loading and hauling tasks to be performed during development and mining of the four demonstration units. Load-haul-dump and mucking cycle times for each task

TABLE 5.4

TYPICAL LHD WORK CYCLES

(Five-Cubic-Yard LHD Unit)

	Haul Distance									
	<u>100'</u>	<u>200'</u>	<u>300'</u>	<u>400'</u>	<u>500'</u>	<u>600'</u>	<u>700'</u>	<u>800'</u>	<u>900'</u>	<u>1,000'</u>
Doze Muckpile	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09
Fill Bucket	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.16
Back & Turn (two/cycle)	0.39	0.39	0.39	0.39	0.39	0.39	0.39	0.39	0.39	0.39
Dump Bucket	0.08	0.08	0.08	0.08	0.08	0.08	0.08	0.08	0.08	0.08
Haul (4.5 mph)	0.25	0.51	0.76	1.01	1.26	1.52	1.77	2.02	2.27	2.53
Return (5 mph)	0.23	0.45	0.68	0.91	1.14	1.36	1.59	1.82	2.05	2.27
Miscellaneous Delay (30 min/shift prorated)	0.12	0.17	0.22	0.27	0.32	0.37	0.42	0.47	0.52	0.57
Service (15 min/shift prorated)	0.06	0.09	0.11	0.13	0.16	0.18	0.21	0.23	0.25	0.28
Theoretical Cycle	1.38	1.94	2.49	3.04	3.60	4.15	4.71	5.26	5.81	6.37
Efficiency Factor (18%)	0.24	0.34	0.44	0.54	0.65	0.75	0.85	0.95	1.05	1.15
TOTAL LHD CYCLE (Min.)	1.62	2.28	2.93	3.58	4.25	4.90	5.56	6.21	6.86	7.52

Average load per trip: 5 tons
Add 20% to cycle for mucking on ramp

have been listed in Table 5.5, together with corresponding tonnage and haul distance data. Sample calculations for loading and hauling cycles are described in Section 5.2.1.

Mucking cycle times in Table 5.5 were determined for the assignment of a single LHD per heading, with the exceptions of the tasks noted. Two or three LHD units will be employed in mucking the large-tonnage stope and chamber rounds in order to reduce the duration of these work cycles. One-half-yard LHD's will be utilized in development on the two block caving monitor levels. Five-yard LHD's will be used for all other mucking applications throughout the mine.

Muck from development headings on levels above the lower haulage level will be moved to orepasses for disposition on the haulage level. The material must be loaded a second time and transferred from the ore pocket to the belt feeder. In such cases, two haul distances and LHD cycles are included in the calculation of mucking cycle times. For tasks requiring two-stage haulage, separate haul distance values have been listed in the Table.

The mining plan for extraction of broken shale from the sublevel stoping with full subsidence unit calls for partial withdrawal of material from the stopes at specified intervals in the task sequence. As the stopes are excavated, sufficient material will be removed to accommodate expansion of the shale broken during rib and crown pillar recovery. The remainder of the material will be retained in the stopes and, with the newly blasted pillar rubble, will support the undercut caving block above. After pillars in the sublevel stoping unit have been rubblized, the column of the overlying caving block will be monitored as the broken material is withdrawn. Caution must be exercised to avoid drawing the stopes down too rapidly. Development of a void between rubblized and uncaved material within the block as a result of rapid drawdown could induce caving of an uncontrolled and catastrophic nature. Mucking cycles for this portion of the task sequence will be dependent upon the tonnages removed per increment. The total tonnage to be extracted from the stopes during demonstration of the block caving system cannot be estimated before the fact.

5.3 HEADING PREPARATION

Heading preparation consists of (1) scaling down loose rock from the roof and ribs of a working place; (2) cleanup of material after scaling; and (3) roof bolting. Because the three activities are arranged sequentially within each mining cycle, it is convenient to discuss them as elements of a single mining function. The method used to calculate the various heading preparation cycle times listed in Table 5.6 is detailed in the following example (15-foot-high by 20-foot-wide heading):

TABLE 5.5

DEMONSTRATION UNITS - MUCKING CYCLE

	<u>Tons Per Round</u>	<u>Avg. Haul Distance (ft)</u>	<u>LHD Unit Cycle (min)</u>	<u>Trips Per Round</u>	<u>Number Of LHD's</u>	<u>Mucking Cycle (min)</u>
<u>Chamber and Pillar Mining With Backfill</u>						
Unit Development						
LHD Entries	207	500	4.25	42	1	179
Drilling Drifts	100	800	6.21	20	1	125
Slots						
Bored Raise	35	225	2.46	7	1	18
First Expansion	420	225	2.46	84	1	207
Second Expansion	1,655	225	2.46	331	1	815
Chamber Mining						
First Round	2,465	300	2.93	529	2	775
Other Rounds	5,290	300	2.93	1,058	2	1,550
<u>Sublevel Stoping With Backfill</u>						
Unit Development						
LHD Haulageways	125	600	4.90	25	1	123
Exhaust Entry	125	600	4.90	25	1	123
Stope Floor Drifts	100	600	4.90	20	1	98
Loading X-Cuts	100	600	4.90	20	1	98
Exhaust Entry X-Cuts	100	600	4.90	20	1	98
Sublevel Drilling						
Drifts and X-Cuts	100	400 + 200	5.86	20	1	118
Upper Level Drilling						
Drifts and X-Cuts	100	400 + 200	5.86	20	1	118
Slots						
Bored Raise	133	500	4.25	27	1	115
First Expansion	505	500	4.25	101	1	430
Second Expansion	4,405	500	4.25	881	2	1,873
Stope Mining						
Stope Floor						
First Round	1,080	400	3.58	216	2	387
Other Rounds	2,160	400	3.58	432	2	774
Sublevel	6,560	400	3.58	1,312	2	2,349
Upper Level	2,160	400	3.58	432	2	774
<u>Sublevel Stoping With Full Subsidence</u>						
Unit Development						
LHD Haulageways	125	600	4.90	25	1	123
Exhaust Entry	125	600	4.90	25	1	123
Stope Floor Drifts	100	600	4.90	25	1	123

TABLE 5.5 (Continued)

	<u>Tons Per Round</u>	<u>Avg. Haul Distance (ft)</u>	<u>LHD Unit Cycle (min)</u>	<u>Trips Per Round</u>	<u>Number Of LHD's</u>	<u>Mucking Cycle (min)</u>
<u>Sublevel Stopping With Full Subsidence (Continued)</u>						
Loading X-Cuts	100	600	4.90	20	1	98
Sublevel Drilling Drifts and X-Cuts	100	400 + 200	5.86	20	1	118
Crown Pillar Drifts and X-Cuts	100	400 + 200	5.86	20	1	118
Vent. Raises	80	400 + 200	5.86	16	1	94
Slots						
Bored Raise	115	500	4.25	23	1	98
First Expansion	590	500	4.25	118	1	502
Second Expansion	4,030	500	4.25	806	2	1,713
<u>Stope Mining</u>						
Stopes						
Stope Floor						
First Round	1,080	400	3.58	†	2	*
Other Rounds	2,160	400	3.58	†	2	*
Sublevel						
First Round	4,070	400	3.58	†	2	*
Other Rounds	8,345	400	3.58	†	2	*
First Crown Pillar						
First Round	18,540	400	3.58	†	2	*
Second Round	18,540	400	3.58	†	2	*
Second Crown Pillar and Rib Pillar						
First Round	39,850	400	3.58	†	2	*
Second Round	71,990	400	3.58	†	2	*
Third Round	42,420	400	3.58	†	2	*
<u>Block Caving</u>						
Unit Development						
Monitor Level Drifts and X-Cuts	26.5	300 + 200	5.21	53	1	277
Undercut Entries and X-Cuts	155	400 + 200	5.86	31	1	182
Pillar Blasting	5,120	400	3.58	†	2	*

(* Only part of this material will be extracted from the stopes during demonstration mining. Consequently, mucking cycle times cannot be computed.)

(† Material from the undercut and crown and rib pillars will be drawn down through the stopes. Mucking cycles cannot be evaluated on a "per round" basis.)

TABLE 5.6

DRIFT ROUNDS
HEADING PREPARATION

<u>Heading Size</u> <u>(height & width)</u>	<u>Scaling</u> <u>Cycle</u> <u>(min)</u>	<u>Cleanup</u> <u>Cycle</u> <u>(min)</u>	<u>Roof Bolting</u> <u>Cycle</u> <u>(min)</u>	<u>Number of Bolts</u> <u>Per Unit Advance</u>	<u>Average Heading</u> <u>Preparation Cycle</u> <u>(min)</u>
15' x 20'	79	20	148	16 per 20'	174
15' x 15'	75	20	125	12 per 20'	158
12' x 15'	72	20	125	12 per 20'	155
12' x 12'	66	20	125	12 per 20'	149
8' x 8'	30	20	11	2 per 6'	61

Scaling Cycle:

Move into heading	15 min
Scale face, ribs, and roof	30 "
Move out of heading	10 "
Service scaler (15 min/shift prorated)	3 "
Miscellaneous delays (45 min/shift prorated)	9 "
Theoretical cycle time	<u>67 min</u>
Efficiency factor (18% of theoretical cycle time)	<u>12 "</u>
Total scaling cycle time	79 min

Roof Bolting Cycle:

Single-Bolt Cycle

Spot on hole	0.5 min
Collar hole and drill 6.5 feet	2.1 "
Retract drill	0.3 "
Insert bolt and tighten	1.2 "
	<u>4.1 min</u>
Move in and set up	20 min
Install 16 bolts x 4.1 minutes	66 "
Tear down and move out	15 "
Obtain bolting supplies (20 min/shift prorated)	7 "
Service bolter (15 min/shift prorated)	6 "
Miscellaneous delays (30 min/shift prorated)	11 "
Theoretical cycle time	<u>125 min.</u>
Efficiency factor (18% of theoretical cycle time)	<u>23 "</u>
Total roof-bolting cycle time	148 min

Average Heading Preparation Cycle:

Scaling Cycle	79 min
Cleanup cycle	20 "
Roof-bolting cycle (148 ÷ 2)	<u>74 "</u>
Total Heading preparation cycle	173 min

In drift and ramp headings, a small mechanical scaling unit will be used to pry and rake down loose rock from the drift face, ribs, and roof to ensure the safety of the working area. The scaling unit will be a modified backhoe equipped with a ripper tooth, operated by one man. Cycle time for scaling in development headings will vary with the heading dimensions (Table 5.6). A five-yard LHD will be assigned to clean up loose scattered material after scaling. The average cycle time for cleanup in all development headings is estimated to be 20 minutes per round.

Six-foot-long roof bolts will be installed on a 5-foot-grid pattern in the roof of development headings. Bolting will be performed after each 20-foot increment of heading advance, i.e., after every other round. The use of point-anchor type bolts has been assumed in this study. Bolting will be accomplished with a conventional single-boom roof-bolting jumbo operated by a two-man crew. Both the number of bolts installed per cycle and the duration of a bolting cycle will vary with the width of the heading (Table 5.6).

A jack-leg drill will be used for roof bolting in the monitor levels. Two bolts will be installed per each 6-foot increment of advance.

Scaling and roof-bolting cycles will be performed in all openings driven during primary development. Within the demonstration units, heading preparation will be reduced due primarily to the elimination of roof-bolting requirements for some drilling drifts within stope boundaries.

In the chamber and pillar demonstration unit, LHD entries and chamber drilling drifts will be scaled, cleaned up and bolted in the manner specified for primary development headings. Heading preparation will also be performed following each slot and chamber round. Inside the chambers, a mechanical unit consisting of a hydraulic or pneumatic breaker mounted on a crane boom, will be used for scaling. A crane-mounted platform will be employed to bolt the 60-foot-high roof from the lower level. Since drilling drift roofs were bolted during the chamber development stage, only a 46-foot-wide exposure of chamber roof will require bolting after each blast. With bolts installed on 5-foot centers, each 20 feet of chamber advance will need 36 bolts to complete the pattern.

Cycle times for scaling and bolting the brow following slot excavation are estimated to be 200 minutes each. After the initial chamber round (advance 10 feet) an estimated 124 minutes will be required for scaling and 223 minutes for roof bolting (18 bolts). Thereafter, each full-size chamber round will require an allocation of 209 minutes for scaling and 367 minutes for roof bolting.

In the sublevel stoping with backfill demonstration unit, all development headings will be scaled and cleaned up after mucking. All development openings, except for stope floor and sublevel drilling drift segments within stope boundaries, will be bolted. Scaling and bolting will not be performed within the

stopes during demonstration mining. The methods devised for mining and back-filling the stopes will preclude the exposure of crews or equipment to un-secured areas of roof and ribs.

In the sublevel stoping with full subsidence units, the stope floor drifts, sublevel drilling drifts, and crown pillar drilling drifts will be bolted through the barrier pillar only. Segments of these drilling drifts driven within the stope boundaries will be scaled and cleaned, but will not be bolted. All permanent development openings on the lower level (LHD haulageways, loading crosscuts, exhaust entry) will be bolted. No scaling or bolting will be required in the stopes during stope mining or pillar extraction.

Considerations governing heading preparation requirements in the block caving demonstration unit differ from the other units. Due to the restricted access to the monitor levels, hand scaling will be performed in these relatively small headings by a two-man crew. Cleanup work will be done with the one-half-yard LHD assigned to muck the headings. The roof will be bolted in all monitor level drifts and crosscuts, except for segments of the observation drifts extending into the caving block.

In the block caving undercut, a modified backhoe and five-yard LHD will be used for scaling and cleanup. Only the 70-foot-long entries passing through the barrier will be roof bolted.

5.4 RAISE BORING

An electric/hydraulic raise boring machine will be used to blind bore a number of raises within the demonstration mine. The raises will serve a variety of purposes as: orepasses, slot raises, ventilation conduits, and inclined accessways. A total of sixteen 52-inch-diameter raises of various lengths will be bored during different stages of mine development work (Table 5.7). The raise boring crew will consist of three men. A typical raise boring cycle, developed for a 132-foot-long raise, is as follows:

Raise Boring Cycle:

Form and pour concrete pad*	200 min
Set up raise boring equipment	400 "
Bore raise at 3 feet per hour	2,640 "
Theoretical cycle time	3,240 min
Efficiency factor (18% of theoretical cycle time)	584 "
Total raise boring cycle time	3,824 min

* Two-day period required for curing is not included in cycle.

TABLE 5.7
RAISE BORING DATA

<u>Task Description</u>	<u>Number of Raises</u>	<u>Length (ft)</u>	<u>Cycle Time (min)</u>
Primary Development:			
Stage I - Orepasses	3	132	3,824
Stage II - Orepass	1	227	6,066
Stage II - Accessways	2	100	3,068
Unit Development:			
Chamber Slot Raises	3	33	1,487
Backfill Stope Slot Raises	3	132	3,824
Subsidence Stope Slot Raises	2	112	3,352
Subsidence Stope Vent Raises	2	40	1,652

5.5 BACKFILLING

Two of the four mining systems to be demonstrated (chamber and pillar mining with backfill and sublevel stoping with backfill) incorporate backfilling as an integral design feature. By providing lateral support for thin pillars left between mined out chambers or stopes, emplaced backfill will permit increased extraction from mining areas and will limit subsidence of ground above the production openings.

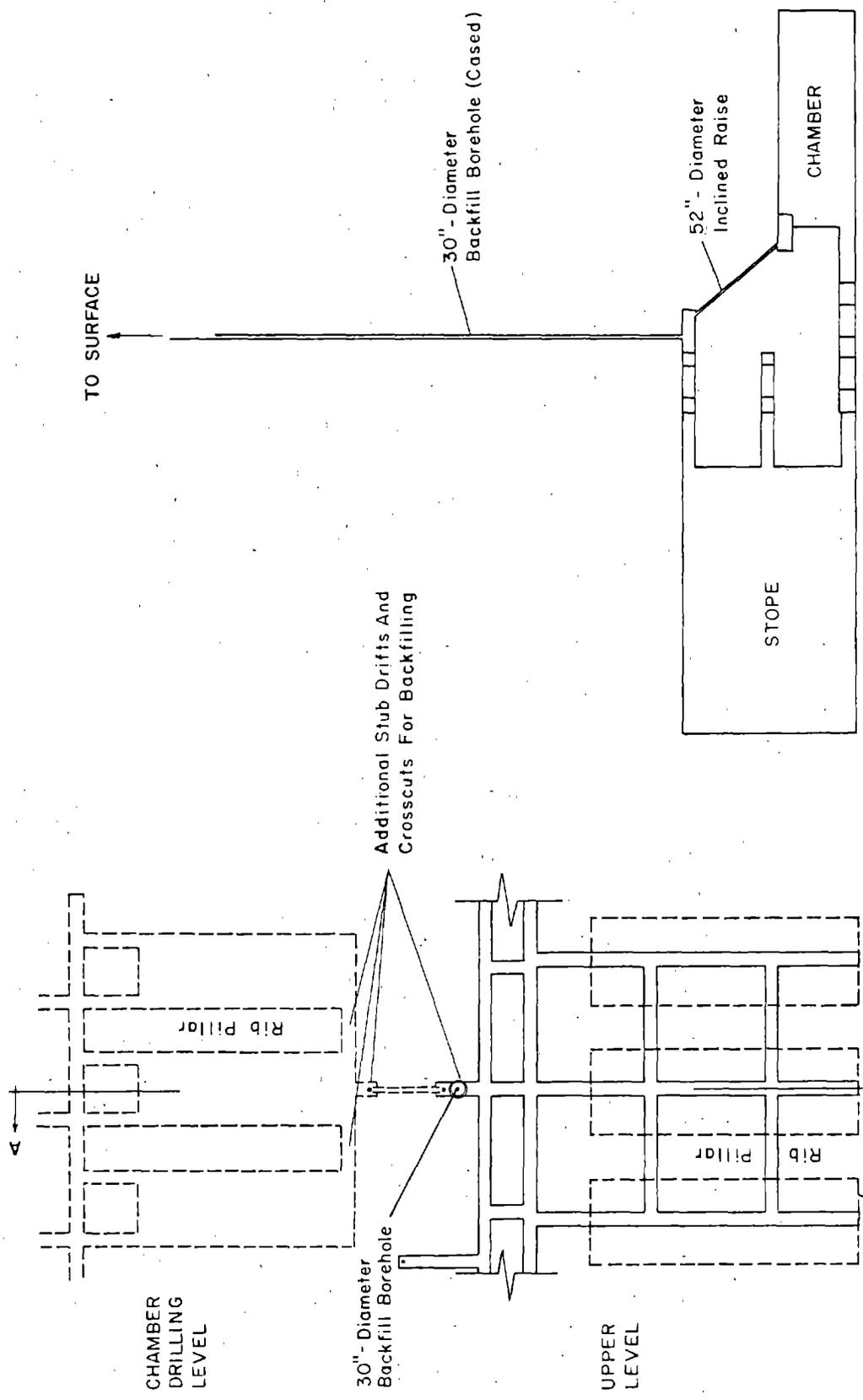
Alluvial material from the site surface will be used as backfill material. Alluvium will be crushed and screened prior to transfer into the mine through a 30-inch-diameter borehole. The borehole will be drilled to intersect the upper level main entries approximately 2,000 feet below the surface. The borehole will be fully cased and grouted to prevent ingress of ground water from the two overlying aquifers.

Preparation of alluvial material on the surface and completion of the backfill borehole to the upper mine level have been considered as subcontract items, and costs for these services have been estimated on a subcontract basis. However, engineering analysis of the backfilling function was performed to ensure that an efficient and cost-effective system can be implemented.

Figure 5.11 shows the optimum position for termination of the surface borehole inside the mine. This location permits the greatest possible reduction in haul distances between the borehole and fill stopes with the least amount of additional development work. Development openings required to implement this scheme for backfilling are indicated in Figure 5.11. Approximately 100 feet of additional drifting on the chamber drilling level and 20 feet of drifting on the upper level (all 12- by 12-foot openings) will be required. A single 52-inch-diameter raise, approximately 100 feet long, will be bored between the two levels.

Costs for this increment of development work also were estimated on a subcontract basis. None of the excavation or construction work specified above in support of the actual performance of backfilling has been included in the mine operating schedule (Section 6.1). However, the additional underground development requirements probably will be performed by the mining contractor, with minor adjustments in the operating schedule. A five-man crew working on the chamber drilling level and the upper level could complete this work in 15 shifts or less.

Backfilling was analyzed in the same manner described for loading and hauling (Section 5.2). Five-yard LHD units used to muck the headings and stopes will also be utilized for backfilling. In calculating the time required to fill each stope for scheduling purposes, an average haul distance of 400 feet was assumed. On this basis, two LHD's, working full time, will require 37.5 shifts to fill each of the three chambers with an estimated 42,000 tons of material. Two LHD's will be engaged for 94 shifts to fill each of the three stopes with an estimated 104,900 tons of alluvium.



SCALE: 1" = 150'

SECTION A-A

PLAN VIEW

FIGURE 5.11
SUGGESTED DEVELOPMENT OPENINGS FOR BACKFILLING

6.0 DEMONSTRATION MINE SCHEDULING

To prepare the mine operating schedule, operational and economic constraints outlined in Section 3.1 were evaluated together with specific task requirements established during the engineering analysis of mining functions. Supporting data were also compiled in schedule form for the purpose of locating and eliminating any conflicting requirements or allocations of manpower and equipment in the operating schedule and to examine the compatibility of period of peak productivity with respect to the stipulated hoisting and ventilation capacities.

As specified by the U. S. Bureau of Mines, alternate schedules were prepared for one-shift-per-day and two-shifts-per-day modes of operation. Each six-month period shown in the schedules represents 125 actual working days.

6.1 MINE OPERATING SCHEDULE

The engineering analysis of mining tasks and functions provided a detailed breakdown of work cycles, manpower, equipment, and excavation tonnages necessary to complete each mining task. These statistics constituted the basic data required for development of the mine operating schedule (Figure 6.1).

Scheduling was subject to constraints imposed by (1) the preestablished hoisting and ventilation capacities; (2) equipment availability; and (3) the undesirability of large manpower fluctuations.

For greatest efficiency and cost effectiveness, a combination of sequential and concurrent scheduling of tasks best fulfilled the program goals. This approach provided for effective utilization of manpower and equipment, and also permitted achievement of relatively consistent rates of productivity throughout the program's duration.

Items not addressed in the scope of work include the following:

- (1) Subcontract requirements for implementing the backfilling system (construction and operation of a surface aggregate preparation plant, drilling and casing of one 30-inch-diameter backfill borehole from surface, and extension of underground openings to expedite haulage of fill material)
- (2) Subcontract construction and installation of ancillary facilities in the shaft station area
- (3) Installation, maintenance, and monitoring of instrumentation for research of rock properties and design characteristics of demonstration units

Mining Task

YEARS (One - Shift - Per - Day)

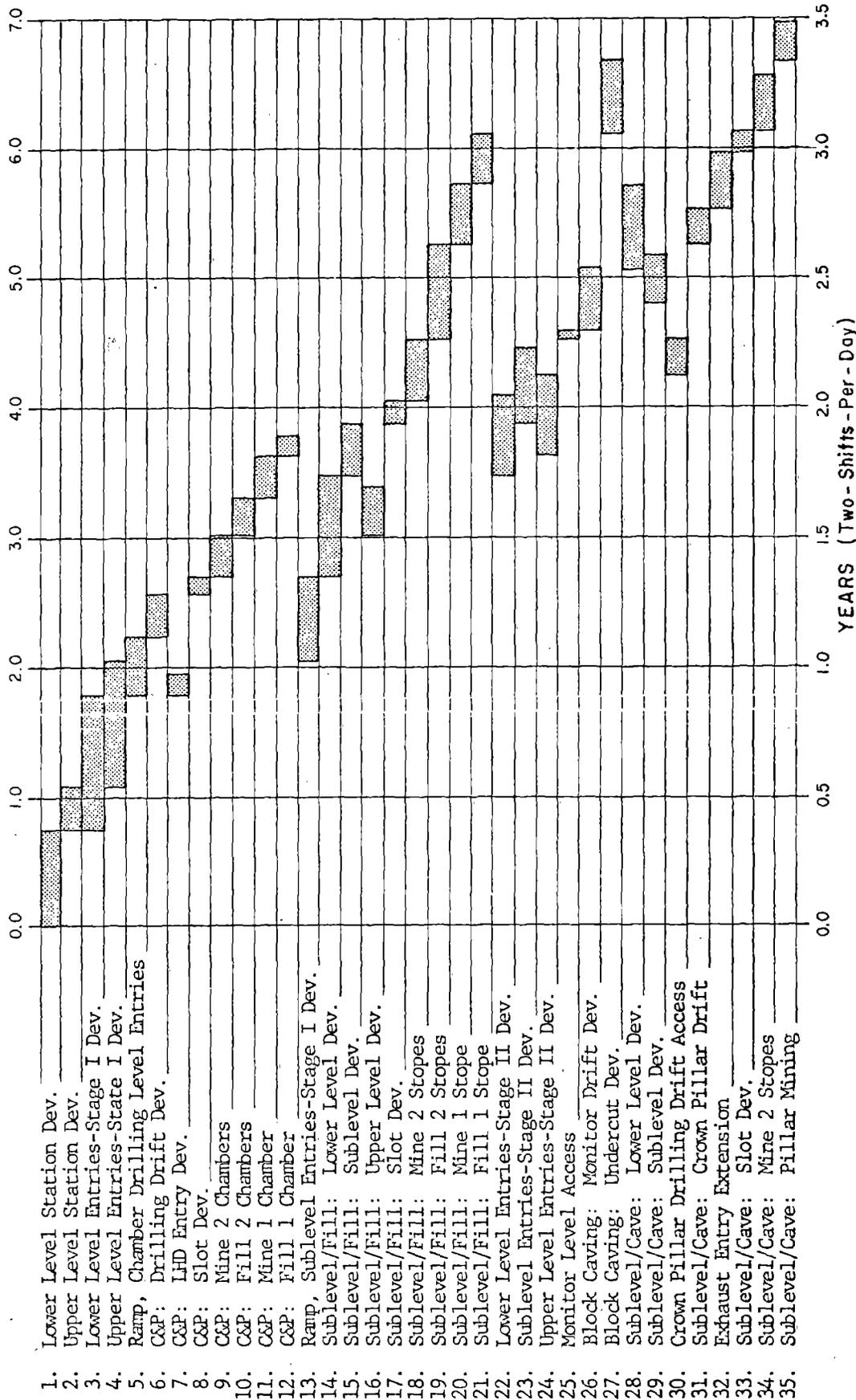


FIGURE 6.1
MINE OPERATING SCHEDULE

These nonscheduled subcontract and research activities will be performed concurrently with the scheduled operating program.

The scale above the bar graph in Figure 6.1 indicates elapsed time for a one-shift-per-day operation. The scale along the lower margin represents timing for a two-shifts-per-day schedule.

6.2 MANPOWER SCHEDULES

A comprehensive evaluation of all mining tasks, maintenance commitments, support systems, and supervisory requirements was made to provide a breakdown of projected total manpower requirements for the demonstration mining program. For the purpose of tabulating data, manpower requirements have been divided into the following descriptive categories:

Mine Direct: Hourly paid employees actively engaged in development and production mining such as drill operators, powder men, LHD operators, rock bolters, etc.

Mine Indirect: Additional hourly paid personnel who are actively engaged in and around the mine, whose time is chargeable directly to the mine operation. This category includes hoistmen and skip tenders plus varying numbers of utility men and maintenance personnel.

Surface: Hourly paid employees actively engaged on the surface in support of the mining activity. A one-shift-per-day schedule requires one truck driver, one heavy equipment operator, two utility men, and a second truck driver for periods of high productivity. A two-shifts-per-day schedule requires an additional equipment operator and truck driver.

Staff: Supervisory, administrative, technical, and security personnel. A one-shift-per-day mining schedule requires one superintendent, one mine captain, one master mechanic, one mine foreman, one maintenance supervisor, two engineers, one secretary-clerk, one warehouseman, and four security people (watchmen). A two-shifts-per-day schedule requires an additional foreman, maintenance supervisor, and timekeeper.

Direct operating labor requirements (category 'Mine Direct') were determined on a task-by-task basis. Bar charts assembled for detailed study of individual task requirements during the engineering analysis of mining functions were used to estimate operating labor commitments. The data are summarized in Table 6.1, which lists direct operating labor manshifts by function and task, together with the average crew size and the duration of each task. For most program segments, an operating crew of constant size was planned. In cases where the size of the crew will change during the completion of a task the crew size is reported in decimal form. For ease of correlation, the format used to prepare Table 6.1 is similar to the layout of the mine operating schedule (Figure 6.1).

Manpower requirements in Table 6.1 are subdivided into manshifts allocated for drilling and blasting, mucking and cleanup, scaling and roof bolting, and nonproductive time. A significant portion (about 29%) of total direct manpower is nonproductive because the demonstration scale program, with its limited number of working areas, is not conducive to uninterrupted, machine-paced work cycles. However, many of the manhours classed as nonproductive will probably be utilized on necessary deadwork not addressed in this report.

A second synopsis of direct operating labor requirements is provided in Table 6.2, which shows the distribution of direct manpower between major program segments.

Manpower requirements for a one-shift-per-day operation are summarized in Tables 6.3 and 6.4. Table 6.3 shows the distribution of estimated manshift requirements among the various personnel categories for each six-month period of operation (125 working days). The actual numbers of personnel in these categories are projected to vary with the scheduled task commitments during a six-month period. The numbers of personnel listed in Table 6.4 represent maximum (not average) manpower requirements for each category and operating period. Tables 6.5 and 6.6 provide similar tabulations of manshift and manpower estimates for a two-shifts-per-day mode of operation.

6.3 PRODUCTION AND PRODUCTIVITY

Estimates of production and productivity for one- and two-shifts-per-day mine operation are given in Tables 6.7 and 6.8. The column titles are defined as follows:

- (1) Gross Tons: tonnage mined during a six-month period
- (2) Tons Per Operating Day: gross tons divided by 125 working days per six-month period
- (3) Tons Per Manshift at the Face: gross tons divided by the number of direct manshifts (mining functions only)
- (4) Tons Per Manshift Underground: gross tons divided by the sum of direct and indirect manshifts (mining and support functions)
- (5) Tons Per Manshift Overall: gross tons divided by the total number of manshifts (underground, surface, and staff)

Production figures (tons per operating day) are averages for each six-month period. As such, they provide a general indication of variations

TABLE 6.1

DIRECT OPERATING LABOR - BY TASK

Mining Task	Average Crew Size (Men)	Task Duration (Shifts)	MANSHIFTS				TOTAL
			Drilling & Blasting	Mucking & Cleanup	Scaling & Bolting	Non-productive	
1. Lower Level Station Dev.	6.3	184	526	96	150	385	1,157
2. Upper Level Station Dev.	4	83	118	19	39	156	332
3. Lower Level Entries-Stage I Dev.	6	262	735	151	244	442	1,572
4. Upper Level Entries-Stage I Dev.	4	248	449	150	156	237	992
5. Ramp, Chamber Drilling Level Entries	4	112	211	51	73	113	448
6. C&P: Drilling Drift Dev.	6	85	246	69	92	103	510
7. C&P: LHD Entry Dev.	2	38	28	9	10	28	75
8. C&P: Slot Dev.	3.6	27	58	8	6	25	97
9. C&P: Mine 2 Chambers	6	83	118	148	47	185	498
10. C&P: Fill 2 Chambers	3	75	-	150	-	75	225
11. C&P: Mine 1 Chamber	4	79	59	73	24	160	316
12. C&P: Fill 1 Chamber	3	38	-	75	-	38	113
13. Ramp, Sublevel Entries-Stage I Dev.	4	157	290	88	100	150	628
14. Sublevel/Fill: Lower Level Dev.	7	198	587	140	187	472	1,386
15. Sublevel/Fill: Sublevel Dev.	6	99	296	79	47	172	594
16. Sublevel/Fill: Upper Level Dev.	5	93	171	46	64	184	465
17. Sublevel/Fill: Slot Dev.	5	43	143	32	-	40	215
18. Sublevel/Fill: Mine 2 Stopes	9	117	282	467	-	304	1,053
19. Sublevel/Fill: Fill 2 Stopes	3	188	-	376	-	188	564
20. Sublevel/Fill: Mine 1 Stope	5	116	141	233	-	206	580
21. Sublevel/Fill: Fill 1 Stope	3	94	-	188	-	94	282
22. Lower Level Entries-Stage II Dev.	4	150	295	56	90	158	599
23. Sublevel Entries-Stage II Dev.	4	145	264	86	92	138	580
24. Upper Level Entries-Stage II Dev.	4	153	279	97	97	139	612
25. Monitor Level Access	3	16	46	1	-	1	48
26. Block Caving: Monitor Drift Dev.	8.0	122	599	181	67	127	974
27. Block Caving: Undercut Dev.	5.5	140	388	113	54	213	768
28. Sublevel/Cave: Lower Level Dev.	5.2	162	414	100	134	192	840
29. Sublevel/Cave: Sublevel Dev.	5	91	173	47	29	206	455
30. Crown Pillar Drilling Drift Access	4	73	137	42	51	62	292
31. Sublevel/Cave: Crown Pillar Drift	3	60	77	12	9	82	180
32. Exhaust Entry Extension	4	120	178	68	66	168	480
33. Sublevel/Cave: Slot Dev.	4	33	81	21	-	30	132
34. Sublevel/Cave: Mine 2 Stopes	6	107	237	23	-	171	645
35. Sublevel/Cave: Pillar Mining	4.2	75	215	-	-	102	317
TOTALS			7,841	3,709	1,928	5,546	19,024

TABLE 6.2

BREAKDOWN OF DIRECT OPERATING LABOR BY MAJOR PROGRAM SEGMENTS

(Manshifts)

<u>Program Segment</u>	<u>Drilling & Blasting</u>	<u>Mucking & Cleanup</u>	<u>Scaling & Bolting</u>	<u>Subtotal</u>	<u>Non- productive</u>	<u>Total Direct Manshifts</u>
PRIMARY DEVELOPMENT						
Stage 1	2,329	555	762	3,646	1,483	5,129
Stage 2	1,199	350	396	1,945	666	2,611
CHAMBER & PILLAR MINING WITH BACKFILL	509	532	179	1,220	614	1,834
SUBLEVEL STOPPING WITH BACKFILL	1,620	1,561	298	3,479	1,660	5,139
SUBLEVEL STOPPING WITH SUBSIDENCE	1,197	417	172	1,786	783	2,569
BLOCK CAVING	<u>987</u>	<u>294</u>	<u>121</u>	1,402	<u>340</u>	<u>1,742</u>
TOTALS	7,841	3,709	1,928		5,546	19,024

TABLE 6.3

MANPOWER REQUIREMENTS
(One-Shift-Per-Day Operation)

(Manshifts)

<u>Year</u>	<u>Mine Direct</u>	<u>Mine Indirect</u>	<u>Total Mine</u>	<u>Surface</u>	<u>Total Mine & Surface (Hourly)</u>	<u>Staff</u>	<u>Grand Total</u>
0.0-0.5	552	725	1,277	375	1,652	1,625	3,277
0.5-1.0	1,264	1,100	2,364	500	2,864	1,625	4,489
1.0-1.5	1,257	1,145	2,402	500	2,902	1,625	4,527
1.5-2.0	1,210	1,141	2,351	500	2,851	1,625	4,476
2.0-2.5	1,136	1,253	2,389	500	2,889	1,625	4,514
2.5-3.0	1,411	1,362	2,773	500	3,273	1,625	4,898
3.0-3.5	1,789	1,450	3,239	500	3,739	1,625	5,364
3.5-4.0	1,954	1,550	3,504	500	4,004	1,625	5,629
4.0-4.5	2,122	1,885	4,007	625	4,632	1,625	6,257
4.5-5.0	1,552	1,558	3,110	500	3,610	1,625	5,235
5.0-5.5	1,581	1,501	3,082	500	3,582	1,625	5,207
5.5-6.0	1,264	1,430	2,694	500	3,194	1,625	4,819
6.0-6.5	1,314	1,346	2,660	500	3,160	1,625	4,785
6.5-7.0	<u>618</u>	<u>981</u>	<u>1,599</u>	<u>500</u>	<u>2,099</u>	<u>1,625</u>	<u>3,724</u>
TOTAL	19,024	18,427	37,451	7,000	44,451	22,750	67,201

TABLE 6.4
MANPOWER REQUIREMENTS
 (One-Shift-Per-Day Operation)

(Number of Men)

<u>Year</u>	<u>Mine Direct</u>	<u>Mine Indirect</u>	<u>Total Mine</u>	<u>Surface</u>	<u>Total Mine & Surface (Hourly)</u>	<u>Staff</u>	<u>Grand Total</u>
0.0-0.5	6	6	12	3	15	13	28
0.5-1.0	11	9	20	4	24	13	37
1.0-1.5	11	10	21	4	25	13	38
1.5-2.0	10	10	20	4	24	13	37
2.0-2.5	10	11	21	4	25	13	38
2.5-3.0	13	11	24	4	28	13	41
3.0-3.5	16	12	28	4	32	13	45
3.5-4.0	17	13	30	4	34	13	47
4.0-4.5	17	16	33	5	38	13	51
4.5-5.0	16	13	29	4	33	13	46
5.0-5.5	15	12	27	4	31	13	44
5.5-6.0	15	12	27	4	31	13	44
6.0-6.5	12	11	23	4	27	13	40
6.5-7.0	6	8	14	4	18	13	31

TABLE 6.5

MANPOWER REQUIREMENTS
(Two-Shifts-Per-Day Operation)

(Manshifts)

<u>Year</u>	<u>Mine Direct</u>	<u>Mine Indirect</u>	<u>Total Mine</u>	<u>Surface</u>	<u>Total Mine & Surface (Hourly)</u>	<u>Staff</u>	<u>Grand Total</u>
0.0-0.5	1,816	1,825	3,641	625	4,266	2,000	6,266
0.5-1.0	2,467	2,286	4,753	750	5,503	2,000	7,503
1.0-1.5	2,547	2,615	5,162	750	5,912	2,000	7,912
1.5-2.0	3,743	3,000	6,743	750	7,493	2,000	9,493
2.0-2.5	3,674	3,443	7,117	875	7,992	2,000	9,992
2.5-3.0	2,845	2,931	5,776	750	6,526	2,000	8,526
3.0-3.5	<u>1,932</u>	<u>2,327</u>	<u>4,259</u>	<u>750</u>	<u>5,009</u>	<u>2,000</u>	<u>7,009</u>
TOTAL	19,024	18,427	37,451	5,250	42,701	14,000	56,701

TABLE 6.6
MANPOWER REQUIREMENTS
 (Two-Shifts-Per-Day Operation)
 (Number of Men)

<u>Year</u>	<u>Mine Direct</u>	<u>Mine Indirect</u>	<u>Total Mine</u>	<u>Surface</u>	<u>Total Mine & Surface (Hourly)</u>	<u>Staff</u>	<u>Grand Total</u>
0.0-0.5	20	18	38	5	43	16	59
0.5-1.0	20	20	40	6	46	16	62
1.0-1.5	26	22	48	6	54	16	70
1.5-2.0	34	26	60	6	66	16	82
2.0-2.5	34	32	66	8	74	16	90
2.5-3.0	30	24	54	6	60	16	76
3.0-3.5	24	22	46	6	54	16	70

TABLE 6.7

PRODUCTION AND PRODUCTIVITY SCHEDULE
(One-Shift-Per-Day Operation)

<u>Year</u>	<u>Gross Tons</u>	<u>Tons Per Operating Day</u>	<u>Tons Per Manshift At The Face</u>	<u>Tons Per Manshift Underground</u>	<u>Tons Per Manshift Overall</u>
0.0-0.5	38,250	306	69	30	12
0.5-1.0	50,220	402	40	21	11
1.0-1.5	61,540	492	49	26	14
1.5-2.0	56,260	450	46	24	13
2.0-2.5	44,920	359	40	19	10
2.5-3.0	134,160	1,073	95	48	27
3.0-3.5	78,740	630	44	24	15
3.5-4.0	100,250	802	51	29	18
4.0-4.5	298,210	2,386	141	74	48
4.5-5.0	29,360	235	19	9	6
5.0-5.5	100,220	802	63	33	19
5.5-6.0	91,510	732	72	34	19
6.0-6.5	150,590	1,205	115	57	31
6.5-7.0	<u>25,250</u>	<u>202</u>	<u>41</u>	<u>16</u>	<u>7</u>
TOTAL	1,259,480				
Average		720	66	34	19

TABLE 6.8

PRODUCTION AND PRODUCTIVITY SCHEDULE
(Two-Shifts-Per-Day Operation)

<u>Year</u>	<u>Gross Tons</u>	<u>Tons Per Operating Day</u>	<u>Tons Per Manshift At The Face</u>	<u>Tons Per Manshift Underground</u>	<u>Tons Per Manshift Overall</u>
0.0-0.5	88,470	708	49	24	14
0.5-1.0	117,800	942	48	25	16
1.0-1.5	179,080	1,433	70	35	23
1.5-2.0	178,990	1,432	48	27	19
2.0-2.5	327,570	2,621	89	46	33
2.5-3.0	191,730	1,534	67	33	22
3.0-3.5	<u>175,840</u>	<u>1,407</u>	<u>91</u>	<u>41</u>	<u>25</u>
TOTAL	1,259,480				
Average		1,440	66	34	22

in production as work proceeds through development and production phases of each demonstration method. Production will approach the mine hoisting capacity (2,500 tons per shift) during years 4.0-4.5 (Table 6.7) when two stopes are mined concurrently in the sublevel stoping with backfill unit.

Productivity (tons per manshift) will vary according to the types of activities in progress. Productivity generally will be low during the early years when only development mining is performed. During later periods concurrent development and/or backfilling operations will tend to offset the high rates of productivity achieved in stope mining. Maximum productivity will be achieved when two stopes are mined concurrently in the sublevel stoping with backfill demonstration unit.

6.4 EQUIPMENT SCHEDULE

Table 6.9 indicates predicted variations in mobile equipment requirements through the course of the demonstration program. Projected utilization and the sequence of equipment acquisition are shown for one-shift and two-shifts-per-day modes of operation. For costing purposes it has been assumed that all mobile equipment will be procured on a permanent basis.

The demonstration mining program will require a relatively large number of LHD units. Two factors may be cited contributing to these requirements: (1) low anticipated mechanical availability of about 55%; and (2) nonproductive LHD time associated with development mining cycles and periods of restricted activity on the upper level.

6.5 VENTILATION SCHEDULE

Preliminary evidence from drill hole logs and more recent evidence from ventilation shaft equipping activities at the Horse Draw site suggest that methane (CH_4) and possibly hydrogen sulfide (H_2S) may be encountered within the proposed mining zone (communication with USBM personnel). Potential concentrations and rates of liberation of these gases into the demonstration mine environment are not known. Gassy conditions have been assumed throughout the mine, and ventilation requirements have been calculated to satisfy State and Federal Regulations for mines classified as gassy.

Table 6.10 lists the projected maximum ventilation requirements for each six-month period of mine operation. Requirements are based on an assumed shaft-to-face loss of 33% and include allowances for shaft station uses. Near-total ventilation capacity will be required during mining of the sublevel stoping with backfill unit.

On a task basis, ventilation requirements are the same for one-shift and two-shifts schedules. Air requirements during the early part of the

TABLE 6.9

MOBILE EQUIPMENT SCHEDULE

EQUIPMENT:	YEARS (One-Shift-Per-Day)														3.5		
	0.0	1.0	2.0	3.0	4.0	5.0	6.0	7.0	0.0	1.0	1.5	2.0	2.5	3.0			
<u>Mining:</u>																	
Face Jumbos	2/2*	2/3	2/3	2/4	1/4	2/4	3/4	3/4	2/4	2/4	3/4	2/4	3/4	2/4	1/4	1/4	
LHD Units (5 Yd)	1/2	2/4	2/4	3/6	4/8	4/9	5/9	5/9	4/9	4/9	3/3	3/3	4/9	4/9	4/9	3/9	
Ring Jumbo	-	-	-	-	1/2	1/2	3/3	3/3	3/3	3/3	3/3	3/3	3/3	2/3	2/3	3/3	
Low Heading Scaler	1/2	2/2	2/2	2/3	2/3	2/3	3/3	3/3	3/3	1/3	3/3	2/3	3/3	2/3	1/3	1/3	
Low Heading Bolter	2/2	2/2	2/2	2/3	2/3	2/3	2/3	2/3	2/3	1/3	3/3	2/3	3/3	2/3	1/3	1/3	
High Heading Scaler	-	-	-	-	1/1	1/1	1/1	1/1	-	-	-	-	-	-	-	-	
High Heading Bolter	-	-	-	-	1/1	1/1	1/1	1/1	-	-	-	-	-	-	-	-	
Powder Truck	1/1	2/2	2/2	2/2	2/2	2/2	2/2	2/2	2/2	2/2	2/2	2/2	2/2	2/2	2/2	2/2	
LHD Units (1/2 Yd)	-	-	-	-	-	-	-	-	-	2/4	2/4	-	-	-	-	-	
<u>Accessory:</u>																	
Utility Vehicle	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
U.L. Maintenance Vehicle	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
L.L. Maintenance Vehicle	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
U.L. Supervisor's Vehicle	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
L.L. Supervisor's Vehicle	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
<u>Surface:</u>																	
Dump Truck	1	1	1	1	1	1	1	1	2	1	1	1	1	1	2	1	
Grader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Front End Loader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Water Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
1-Ton Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
1/2 Ton Pickup	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	
Superintendent's Vehicle	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Ambulance	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	

* Number of units in use/total number of units

TABLE 6.10

VENTILATION REQUIREMENT SCHEDULE

<u>One-Shift-Per-Day</u>		<u>Two-Shifts-Per-Day</u>	
<u>Year</u>	<u>Cfm</u>	<u>Year</u>	<u>Cfm</u>
0.0-0.5	95,000	0.0-0.5	95,000
0.5-1.0	95,000	0.5-1.0	130,000
1.0-1.5	95,000	1.0-1.5	170,000
1.5-2.0	130,000	1.5-2.0	205,000
2.0-2.5	95,000	2.0-2.5	219,000
2.5-3.0	170,000	2.5-3.0	165,000
3.0-3.5	205,000	3.0-3.5	155,000
3.5-4.0	205,000		
4.0-4.5	219,000		
4.5-5.0	165,000		
5.0-5.5	165,000		
5.5-6.0	165,000		
6.0-6.5	155,000		
6.5-7.0	100,000		

program (approximately 100,000 cfm) are about half of the maximum requirements, which occur near the middle of the operating schedule. By lowering the main fan setting for this initial period of operation, a savings in energy (horsepower) consumption will be achieved.

7.0 MINE VENTILATION

The purpose of mine ventilation is to provide sufficient air of suitable quality to underground workings to dilute airborne dust, diesel exhaust fumes, and other noxious gases to harmless levels, thus ensuring a mine environment that does not endanger the health and safety of mine personnel. The essential considerations of ventilation system design begin with the quantity and quality of fresh air introduced into the workings, including airflow patterns, temperatures, and velocities, together with progressively changing levels of contaminants within the system and, finally, the amount, characteristics, and composition of air exhausted back to the surface.

Colorado State Law requires that a minimum of 75 cfm of fresh air be supplied for each unit of horsepower rating applied to all diesel-powered underground equipment. Additionally, mine ventilation must be designed so that diesel exhaust gases do not accumulate in any area but, instead, are exhausted to the surface by the most direct route. Federal and State Regulations also specify that mine air of minimum acceptable quality must contain no less than 20% oxygen, and the concentration of toxic gases shall not exceed the following limits:

	<u>Parts Per Million</u>
° Carbon Dioxide (CO ₂)	5,000
° Carbon Monoxide (CO)	50
° Nitrogen Dioxide (NO ₂)	5
° Oxides of Nitrogen (NO _x)	25
° Aldehydes (RCHO)	10
° Hydrogen Sulfide (H ₂ S)	10

In addition to the aforementioned regulations which are applicable to all mines, the Bureau of Mines has specified minimum ventilation requirements for various types of permissible underground diesel equipment operating in mines classified as gassy.

Primary ventilation will be supplied via one 20-foot-diameter production (intake) shaft and one 8-foot-diameter exhaust shaft. The exhaust shaft will be equipped with a surface-mounted, 600 horsepower fan capable of exhausting mine air at an approximate maximum rate of 220,000 cubic feet per minute. Underground, airflow will be directed into various working areas by brattices, overcasts, secondary fans, etc. Gassy mine conditions and an average rock temperature of 90° were assumed for mine layout planning and determination of ventilation requirements.

7.1 GENERAL

7.1.1 Ventilation Requirements

Based on optimum mine scheduling, the quantity of fresh air necessary to maintain acceptable air quality in the demonstration mine will range from 95,000 cfm to 219,000 cfm. A maximum flow of 95,000 cfm of fresh air will be required during initial phases of the operation when only a small amount of development work is in progress. In Year 4, while sublevel Stage II development, upper level Stage II development, and demonstration mining in the sublevel stoping with backfill unit are being performed concurrently, 219,000 cfm must be supplied. Air requirements for each six-month period of mine operation are shown in Table 6.10. The quantities listed in the Table reflect the combined ventilation requirements for all areas within the mine that are being worked concurrently.

7.1.2 Gassy Mine Restrictions:

Gassy mine conditions may not be encountered, but have been assumed to exist as a realistic (worst case) basis for mine design. Guidelines applied in mine design, which meet or exceed State and Federal Regulations, include:

- ° The main fan will be located on the surface.
- ° Pumps and electrical transformers will operate in a split of intake air.
- ° All development headings will be advanced in multiples of two or more with crosscuts between headings at intervals not exceeding 100 feet.
- ° Auxiliary fans and ventilation tubing will be used in development headings to assure positive airflow beyond the last open crosscut.
- ° Developed mine openings that are not in use will be ventilated or sealed off.
- ° Ventilating air from all working areas will be discharged into isolated return airways. In most instances, air will not be reused. However, in multiple-face operations, face-to-face reuse of ventilating air will be permitted as long as acceptable air quality is maintained.
- ° The quantity of air flowing through the last open crosscut will be at least 6,000 cfm.

- ° Permanent ventilation structures will be constructed of noncombustible materials.
- ° Temporary ventilation structures will be constructed of fire-resistant materials.
- ° Only permissible equipment will be operated beyond the last open crosscut.

7.1.3 High Temperature Considerations:

Temperature logs obtained in three core holes drilled at the Horse Draw site indicate the average rock temperature in the planned mining zone to be about 90°F (Figure 3.2). In addition to the high rock temperature, other sources of heat that will affect the underground environment include the following:

- ° Adiabatic compression in the shaft
- ° Equipment radiation and exhaust
- ° Chemical processes such as blasting and oxidation
- ° Body metabolism
- ° Rock movement
- ° Mine water
- ° Air friction and turbulence
- ° Heat from service lines

Of these heat sources, only adiabatic compression and diesel equipment are significant. The ventilation system is designed to discharge air heated by diesel equipment directly into exhaust airways. Anticipated wet bulb globe temperatures within the demonstration mine have been calculated and are presented in Table 7.1. The relatively dry ventilating air will result in evaporative cooling which will aid in maintenance of an acceptable mine environment.

The maximum summer temperatures shown in Table 7.1 will occur for only about two hours of each 24-hour period during the hottest summer days. These temperatures are not considered extreme since most mine work will be machine paced. Spot air conditioning, such as air conditioned cabs on equipment, may be desirable.

TABLE 7.1

EXPECTED WET BULB GLOBE TEMPERATURES (OF)

<u>Demonstration Unit</u>	<u>SUMMER</u>				<u>WINTER</u>							
	<u>Average Work Area Surface*</u>	<u>Exhaust Area</u>	<u>Maximum Work Area Surface</u>	<u>Exhaust Area</u>	<u>Average Work Area Surface</u>	<u>Exhaust Area</u>	<u>Minimum Work Area Surface</u>	<u>Exhaust Area</u>				
Chamber & Pillar Mining With Backfill	55	72.3	92.6	77	87.7	108	21	52.3	72.6	-9	34.7	55
Sublevel Stopping With Backfill	55	73.5	96.9	77	87.7	111.1	21	54.4	77.8	-9	38.4	61.8
Sublevel Stopping With Full Subsidence	55	74.5	92.5	77	87.7	105.7	21	62	80	-9	51.2	69.2
Block Caving	55	74.5	92.5	77	87.7	105.7	21	62	80	-9	51.2	69.2

Wet Bulb Globe Temperature (WBGT) = 0.7 WB + 0.3 DB

Where WB = Wet Bulb
DB = Dry Bulb

* Surface temperatures from average values measured at nearby Federal oil shale lease tracts.

7.1.4 Ventilation Equipment and Structures

Auxiliary fans will be used throughout the mine to direct flows of air into and within development and production areas. Both rigid and flexible ventilation tubing will be used in conjunction with the fans to carry air to the work areas. The nature of the application will govern the size of fan and kind of ducting required. Small 10-horsepower fans, for example, will be used to sweep individual development faces while large 75-horsepower fans will be used to exhaust air from demonstration mining areas. Medium sized 40-horsepower fans will be used to direct air into areas of development activity.

Permanent brattices will be used to seal crosscuts between fresh and exhaust airways. These structures will be made of sheet metal or other noncombustible material. Some will be installed with walk-through doors for foot traffic. A flame-resistant foam will be used as a sealant between brattice and rock walls to minimize leakage.

Temporary brattices will serve to direct ventilating air in and within development areas by blocking openings through which air could short circuit. These structures will be constructed of flame-resistant material. Some may be replaced with permanent ventilation structures to become a part of the overall mine ventilation system after development in an area is completed.

Ventilation doors will be installed at locations in the mine where traffic must move between air courses. The doors will be of welded metal construction and will generally be built in pairs to serve as air locks. Appropriate opening and closing devices will be provided. Chinking and foam sealant will be used as needed to minimize leakage.

Fixed opening regulators will be used in the exhaust drifts of each demonstration mining unit and in other mine airways, as needed, to proportion the ventilating air allotted to each area of the mine. Permanent regulators will be of welded steel construction.

Ventilation overcasts will be required where one air course must cross another, and will be of two designs. One type will be constructed of curved, culvert-type steel pipe and will be sufficiently large in cross-section to permit flow of 100,000 cfm or more of ventilating air unassisted by fans. A second type of overpass will consist of a section of large-diameter, rigid tubing plus an exhaust fan, suspended from the mine roof as a single unit.

7.2 VENTILATION OF SHAFT STATIONS

7.2.1 Lower Level Station

The quantity of fresh air directed to the lower level will vary with the number and types of activities in progress. Of the total amount

delivered, 15,000 cfm will be required to ventilate the station area. The airflow pattern in the station area is described in Figure 7.1. One split of fresh air from the production shaft will ventilate the magazine, storage area, shop, and fuel station before being discharged to the exhaust shaft. Another split of air will ventilate the substation and pump room. Regulators will control the volumes of air split from the main air stream. An air lock, consisting of two air doors, will isolate fresh air splits from return mine air.

Fans will be provided in the maintenance shop to maintain a positive airflow pattern and ventilation will be sufficient to ensure that air is replaced every 15 minutes. The magazine also will be provided with positive airflow to prevent buildup of heat and noxious gases. The conveyor discharge point and skip-loading pocket will be equipped with dust suppression devices. Exhaust fans and tubing will be provided for removal of airborne dust from these areas.

7.2.2 Upper Level Station

The upper level will be ventilated with varying amounts of fresh air depending on the extent of activities in progress. The upper level station area will require approximately 5,000 cfm for adequate ventilation. The airflow pattern in the upper level station area is presented in Figure 7.2. The lunchroom, office, and first-aid station will receive air from the main fresh air course directed toward the working area. A small split of air will be diverted across the orepass to clear dust generated by LHD dumping.

7.3 LOWER LEVEL VENTILATION

Fresh air for ventilation of lower level development and demonstration mining areas will enter from the production shaft, travel to various working areas through established air courses, and ultimately will discharge to the surface via the 8-foot-diameter exhaust shaft. Lower level air requirements will be minimal during primary development when main entry headings are being driven and maximal during mining of two stopes in the sublevel stoping with backfill demonstration unit. Air requirements for each segment of mining on the lower level are shown in Table 7.2. A safety factor of 1.5 was applied in all airflow calculations to compensate for losses between the shaft and points of usage.

7.3.1 Development

A general ventilation scheme for all development work on the lower level is shown in Figure 7.3. Although the areas depicted will be developed sequentially, each area will be ventilated in the manner indicated as it is worked.

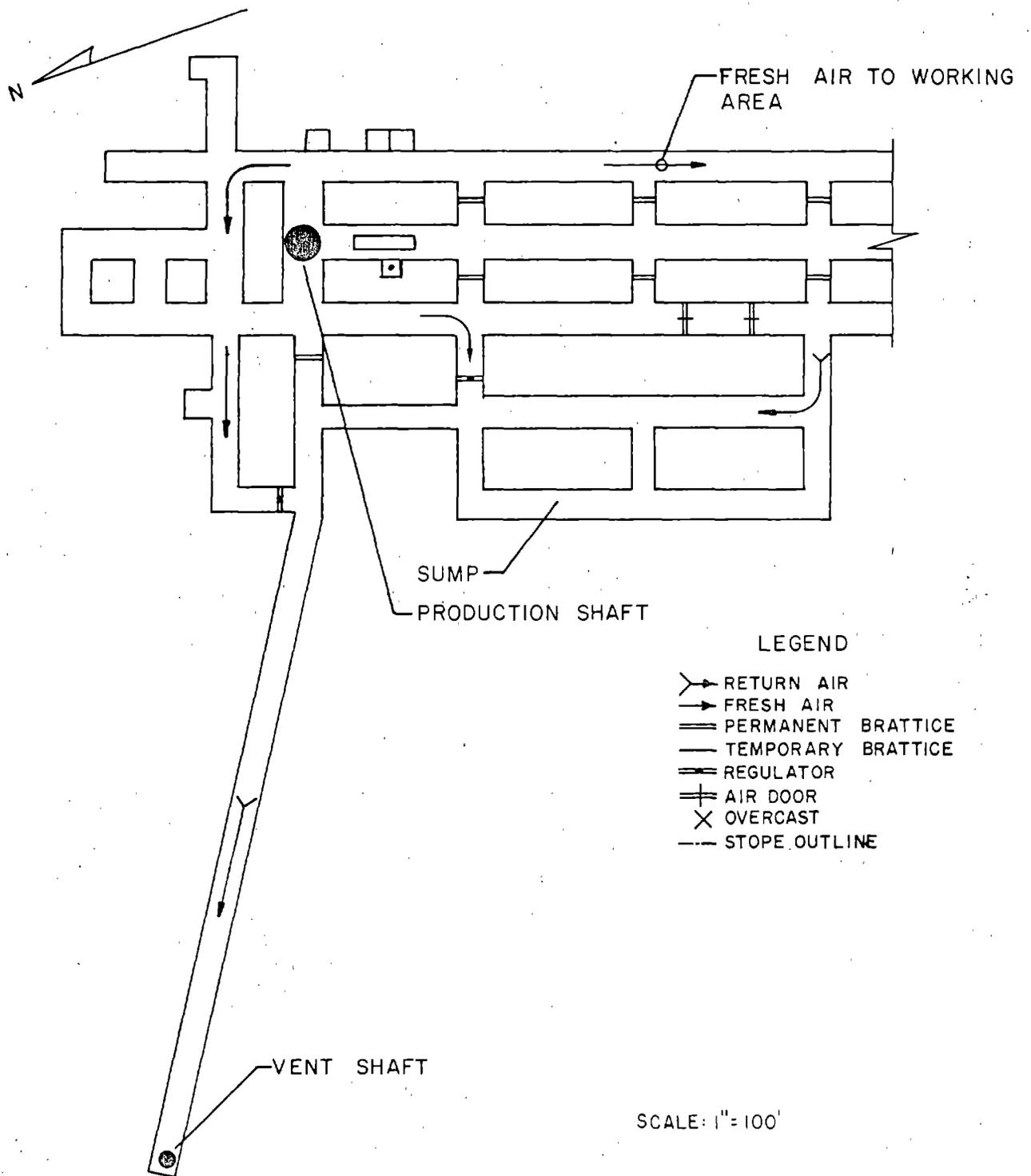


FIGURE 7.1

VENTILATION PLAN - LOWER LEVEL STATION

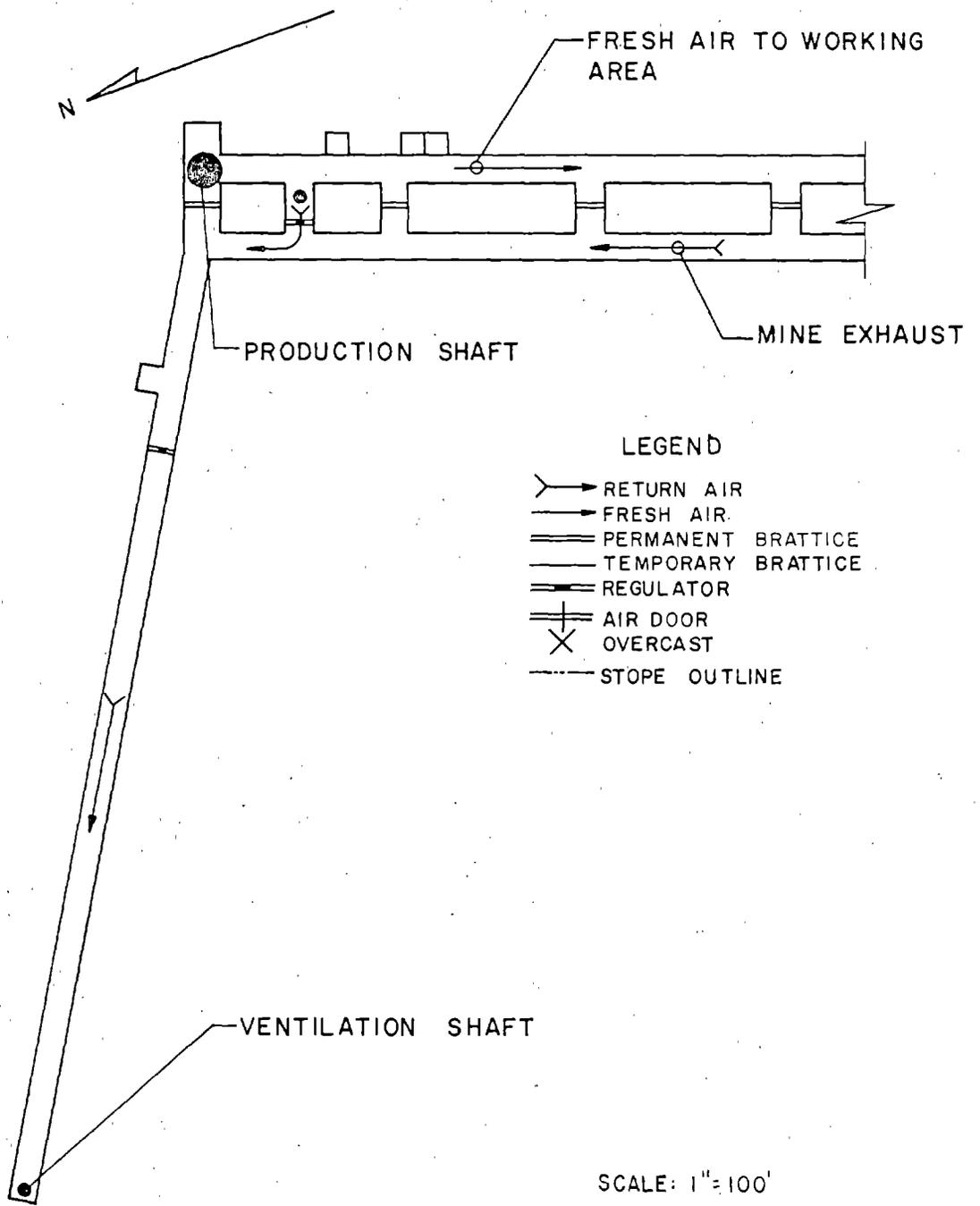


FIGURE 7.2

VENTILATION PLAN - UPPER LEVEL STATION

TABLE 7.2

LOWER LEVEL VENTILATION REQUIREMENTS
(ft³/min)

Shaft Station	15,000
Development:	
Station Area	37,500
Main Entries	37,500
Chamber Access Drifts	37,500
Ramp and Chamber Drilling Level	37,500
Sublevel Stoping With Backfill Unit	75,000
Sublevel Stoping With Full Subsidence Unit	37,500
Exhaust Ventilation Airway	37,500
Production:	
Chamber and Pillar Unit	
2 Chambers	75,000
1 Chamber	37,500
Sublevel Stoping With Backfill Unit	
2 Stopes	111,000
1 Stope	55,500
Sublevel Stoping With Full Subsidence Unit	85,500
Block Caving Unit	76,500

- LEGEND**
- ↖ RETURN AIR
 - ↗ FRESH AIR
 - ▬ PERMANENT BRATTICE
 - ▬ TEMPORARY BRATTICE
 - ▬ REGULATOR
 - ▬ AIR DOOR
 - ⊗ OVERCAST
 - - - STOPE OUTLINE

CHAMBER AND PILLAR MINING WITH BACKFILL

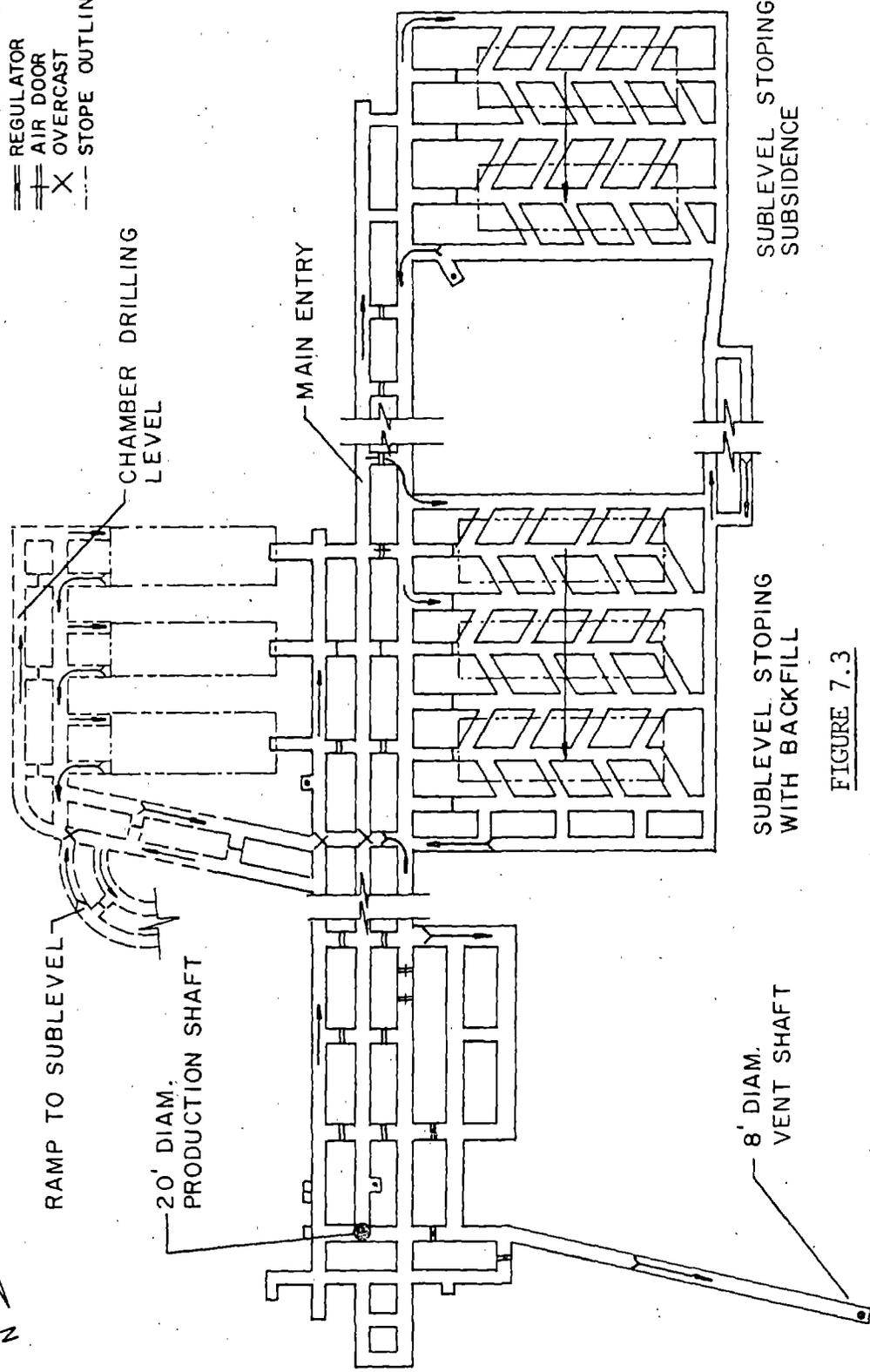


FIGURE 7.3

VENTILATION PLAN
LOWER LEVEL - DEVELOPMENT PHASE

SCALE: 1"=200'

Main Entries: Fresh air will be directed to the development headings through the east entry and will be discharged via the west entry. All crosscuts between the east and west entries, except those nearest to the working faces, will be blocked with brattices to establish positive airflow through the last open crosscut. Small booster fans and tubing will be used to sweep headings beyond the last open crosscut with fresh air.

Chamber Drilling Level Entries: The two ramps leading to the chamber drilling level will serve as intake and exhaust airways for entries on this level. Crosscuts between the ramps will be bratticed to prevent short circuiting of ventilation air. Fresh air will be diverted from the east main entry. A 75-horsepower fan will be used to exhaust air from the chamber drilling level through a 48-inch-diameter rigid-tube overpass into the west main entry return airway (Figure 7.3).

Chamber and Pillar Unit: LHD entries, 60 feet in length, will be driven from the main entries into the chamber mining areas. As each entry is driven, it will be ventilated with a small fan and tubing.

Sublevel Stopping With Backfill Unit: Lower level development in this unit consists of driving multiple headings through the demonstration area to establish drilling drifts, LHD haulageways, and a ventilation exhaust entry; all are interconnected by crosscuts. During development the headings will be ventilated by airflow from face to face. Temporary brattices will be used to block crosscuts, thereby forcing ventilating air through the working areas. Fresh air will be directed into the areas from the main headings by two 40-horsepower fans. Smaller 10-horsepower fans and tubing will be used to sweep faces with fresh air beyond the last open crosscut.

Sublevel Stopping With Full Subsidence Unit: Lower level development in the unit will be similar to that for the sublevel stopping with backfill unit, though fewer headings will be driven. The manner of ventilating the area during development is the same as described in the previous subsection except that one 40-horsepower fan, instead of two, will be used to direct required fresh air into the workings. In this instance, intake air from the mains will pass over the crusher before entering the development area. Dust suppression and possibly dust collection equipment will be used to ensure a relatively dust-free air supply.

Exhaust Ventilation Airway: This development task consists of driving two drifts to connect the two sublevel stopping units and is essential for exhaust ventilation during demonstration mining in the sublevel stopping with full subsidence unit. As the headings are driven, fresh air will be supplied through one of the drifts and exhausted via the second. A fan will be used to direct air into the development area and smaller fans with tubing will be used to sweep the faces beyond the last open crosscut.

7.3.2 Demonstration Mining

A general ventilation scheme, applicable to demonstration areas on the lower level, is shown in Figure 7.4. Although the units will not be mined simultaneously, each unit will be ventilated in the manner indicated. Air quantity requirements were detailed previously in Table 7.2.

Chamber and Pillar Unit: As the chambers are excavated, ventilating air will enter the mining area through the LHD entries in the front and discharge into exhaust entries at the rear of the unit (Figure 7.4). Airflow will be controlled by regulators situated in exhaust courses at the back of each chamber. Fans located in the LHD entries will direct fresh air into the chambers.

Sublevel Stopping With Backfill Unit: Ventilation air from the main entries will flow through the LHD haulageways and discharge into the exhaust entry at the rear of the stopes. Flow through the haulageways will be controlled by regulators situated near the discharge points (Figure 7.4). Ventilation air flowing through the stope floor drilling drifts will discharge into the stopes and ultimately will exhaust on the upper level.

Sublevel Stopping With Full Subsidence Unit: Ventilation of this unit is accomplished in a manner similar to that described for the sublevel stopping with backfill system (Figure 7.4).

Block Caving Unit: The ventilation requirements listed in Table 7.2 for the lower level will be needed if the caving demonstration is continued by drawdown of the block from sublevel stopping drawpoints on the lower level.

7.4 SUBLEVEL VENTILATION

Fresh air for sublevel ventilation will be conducted from the lower level via ramps driven from the lower level. During development one of the ramps will be used as an exhaust air course, discharging into the lower level main entry exhaust system. During demonstration mining, both ramps will carry fresh air which ultimately will discharge through the stopes to the upper level. Air requirements for each segment of sublevel mining are presented in Table 7.3. The quantities listed include assumed losses due to brattice leakage and system inefficiencies.

7.4.1 Development

A general ventilation scheme, applicable to all areas of sublevel development, is shown in Figure 7.5. Sublevel ventilation details are as follows:

LEGEND

- RETURN AIR
- FRESH AIR
- PERMANENT BRATTICE
- TEMPORARY BRATTICE
- REGULATOR
- AIR DOOR
- OVERCAST
- STOPE OUTLINE

CHAMBER AND PILLAR MINING
WITH BACKFILL

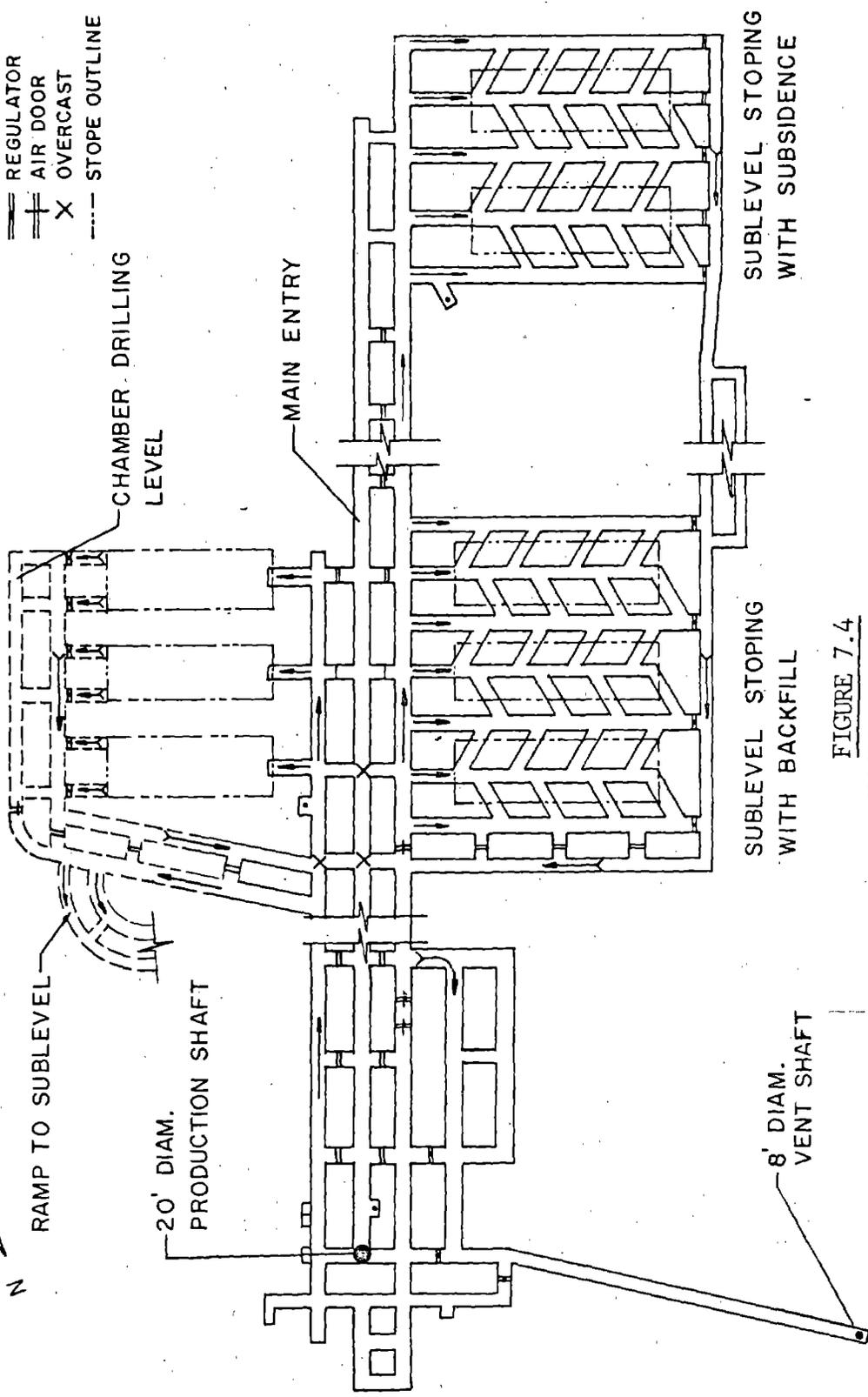
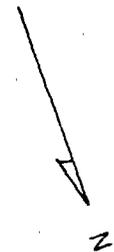


FIGURE 7.4

VENTILATION PLAN

LOWER LEVEL - PRODUCTION PHASE

SCALE: 1"=200'

TABLE 7.3

SUBLEVEL VENTILATION REQUIREMENTS
(ft³/min)

Development:

Ramp and Main Entries	37,500
Sublevel Stopping With Backfill Unit	37,500
Sublevel Stopping With Full Subsidence Unit	37,500

Production:

Sublevel Stopping With Backfill Unit	
2 Stopes	18,000
1 Stope	9,000
Sublevel Stopping With Full Subsidence Unit	13,500



LEGEND

- RETURN AIR
- FRESH AIR
- PERMANENT BRATTICE
- TEMPORARY BRATTICE
- REGULATOR
- AIR DOOR
- OVERCAST
- X
- STOPE OUTLINE

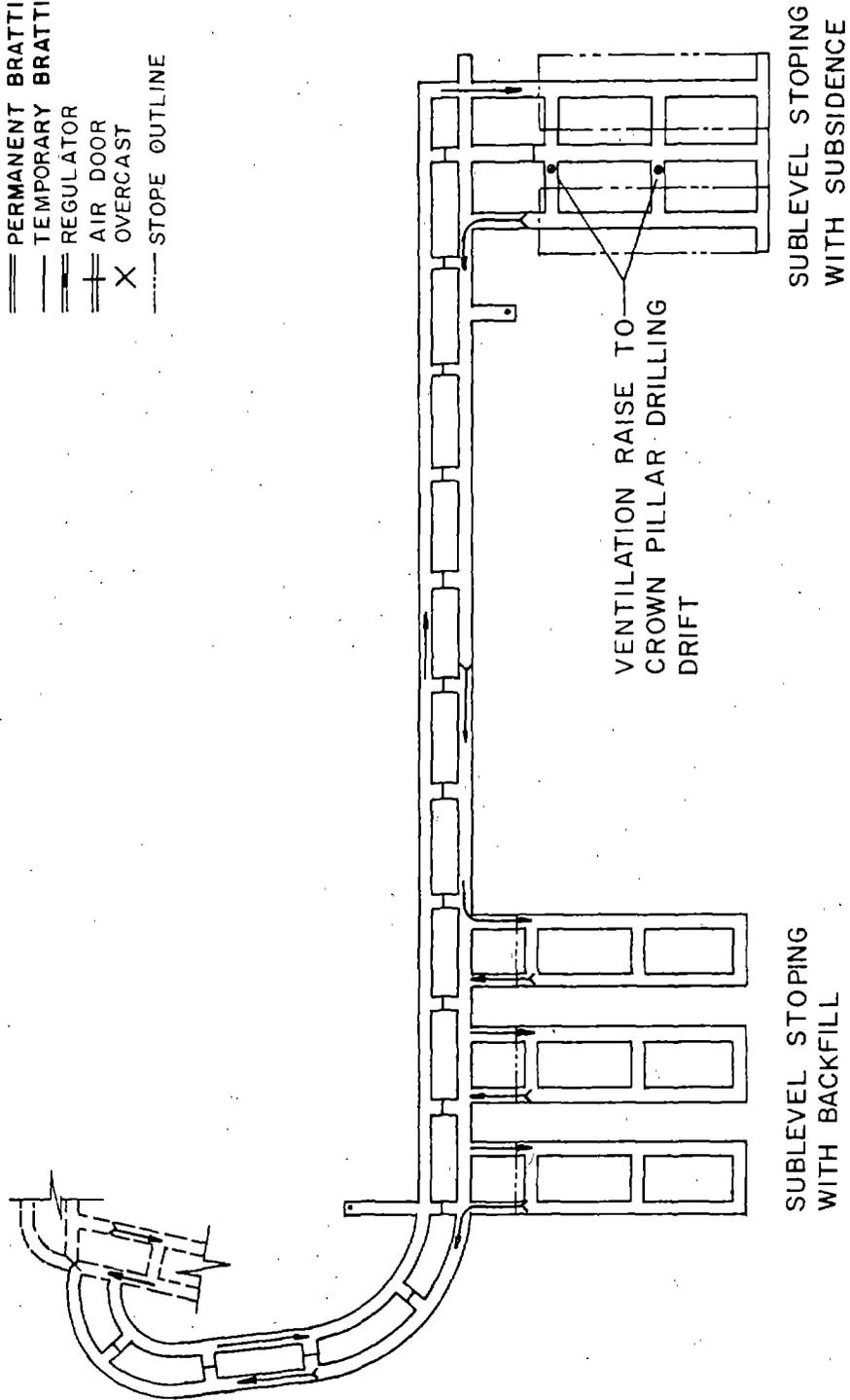


FIGURE 7.5
VENTILATION PLAN
SUBLEVEL - DEVELOPMENT PHASE

SCALE: 1" = 200'

Ramps and Main Entries: All crosscuts between the two drifts, with the exception of the one nearest the working faces, will be blocked with temporary brattices to establish fresh and exhaust airways. Fresh air will be directed into the fresh air course and toward the development headings with a 40-horsepower fan. A similar-sized fan with tubing will be used to exhaust air from the area, overpassing a fresh air stream that ventilates the chamber and pillar drilling level (Figure 7.5).

Sublevel Stopping With Backfill Unit: Sublevel development in the demonstration unit consists of driving two drilling drifts into each of the three planned stopping areas. Crosscuts will be turned to connect the drilling drifts at 100-foot intervals within each stope. Ventilation of the sublevel drilling drifts will be accomplished by heading-to-heading movement of air (Figure 7.5). A 40-horsepower fan will be required to direct air into each of the three stopping areas as the drilling drifts are developed. Within the stopping areas fresh and exhaust air courses will be established by bratticing all but the last crosscuts. Ventilation beyond the last open crosscuts will be directed by small booster fans and tubing.

Sublevel Stopping With Full Subsidence Unit: Three drifts will be driven to develop this unit on the sublevel, one through the center of each stopping area and one within the pillar which will separate the two stopes. The three drifts will be interconnected by crosscuts at required intervals. Ventilation to the development area will be provided by directing fresh air into one of the outside drifts and discharging it from the other (Figure 7.5). Temporary brattices will be erected in crosscuts to establish fresh and exhaust air courses to the working faces. As previously described, small fans with tubing will direct ventilation air beyond the last open crosscut.

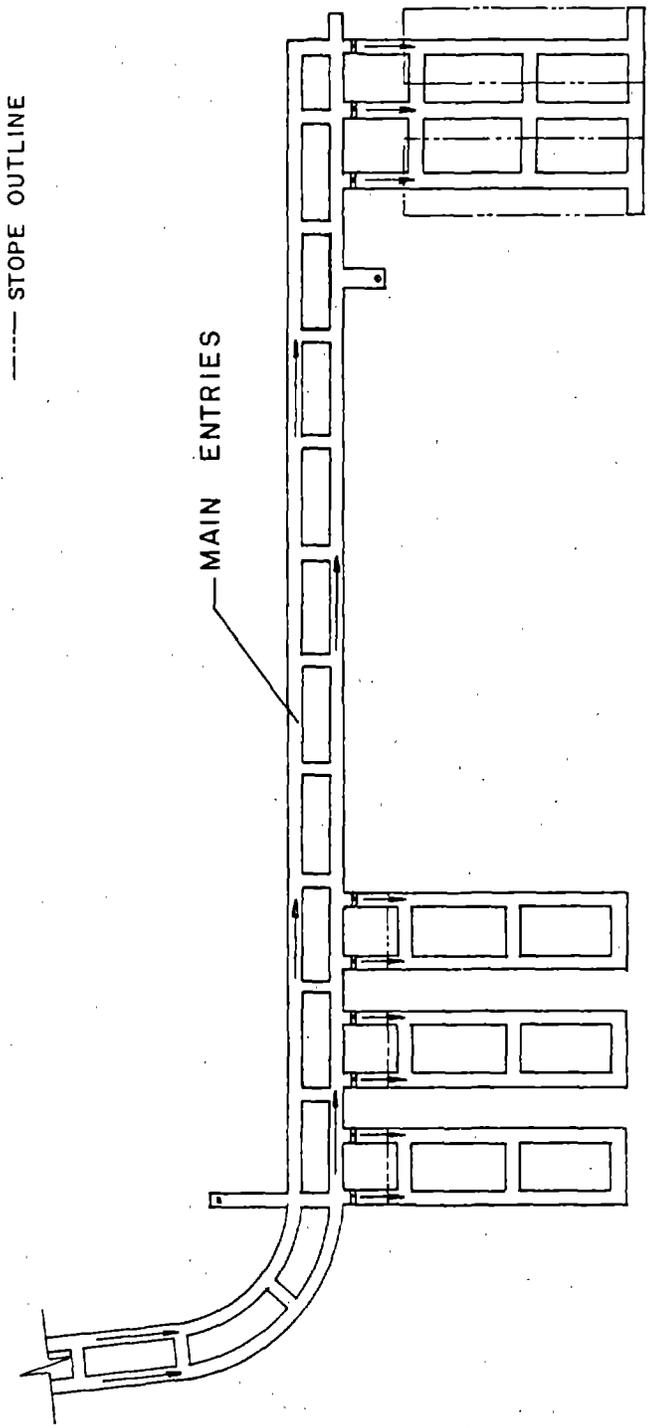
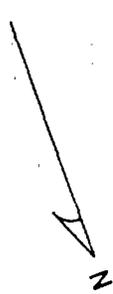
7.4.2 Demonstration Mining

Fresh air will enter the sublevel through ramps connecting the lower level with the sublevel. The airflow pattern on the sublevel is depicted in Figure 7.6. Ventilating air for both sublevel stopping systems will be directed from main entries into drilling drifts and will be discharged into open stopes. From the stopes, exhaust air will flow into a return air course on the upper level. The amount of air entering each drilling drift during demonstration mining will be controlled by regulators. Air quality requirements for the sublevel are shown in Table 7.3.

7.5 UPPER LEVEL VENTILATION

Ventilating air for upper level development and production activities will enter the level from the 20-foot-diameter production shaft, will be conducted to various working areas through established air courses, and will ultimately be discharged to the surface through the 8-foot-diameter exhaust shaft. Air requirements for each segment of mining on the upper level

- RETURN AIR
- FRESH AIR
- PERMANENT BRATTICE
- TEMPORARY BRATTICE
- REGULATOR
- AIR DOOR
- OVERCAST
- STOPE OUTLINE



SUBLEVEL STOPPING
WITH BACKFILL

SUBLEVEL STOPPING
WITH SUBSIDENCE

FIGURE 7.6
VENTILATION PLAN
SUBLEVEL - PRODUCTION PHASE

SCALE: 1" = 200'

are given in Table 7.4. The quantities shown include air losses that are assumed to be 50% of the calculated requirements.

7.5.1 Development

A general ventilation scheme, applicable to all areas of upper level development, is shown in Figure 7.7. Details concerning upper level development ventilation requirements are given in the following paragraphs.

Main Entries: As in other double-entry development situations, all crosscuts between the two entries (except one or two nearest the working faces) will be bratticed to establish isolated fresh and return air courses. When the main entries have been completed, temporary brattices will be replaced with permanent ventilation structures.

Sublevel Stopping With Backfill Unit: Three drilling drifts, one above the center of each stope, will be driven from the upper level mains. The drifts will be interconnected by crosscuts at required intervals. The crosscuts will be blocked with temporary brattices as required to establish positive airflow across the faces. Headings beyond the last open crosscuts will be ventilated by small fans and tubing.

Sublevel Stopping With Full Subsidence Unit: Upper level air will be used for ventilation in this demonstration unit only while access ramps and a portion of the crown pillar drilling drift are driven. When the drilling drift intersects the first of two ventilation raises, bored upward from the rib pillar drilling drift, fresh air will be supplied from the sublevel and will exhaust on the upper level. Prior to intersecting the raise, the two ramps will be used as fresh and return air courses, using temporary brattices and small fans with tubing to sweep headings worked beyond the last crosscut. A large fan and tubing will force air into the fresh air course.

Monitor Drifts: To ventilate the two monitor levels during development, fresh air will be directed from the upper level intake entry to one of two 50° access raises, utilizing an auxiliary fan and tubing. The second access raise will serve as a return airway. On the monitor levels, small fans and tubing will direct air into openings being developed, with temporary brattices installed as needed.

Block Caving Undercut: The area to be caved will be undercut using standard room and pillar mining methods. Fresh and return air courses will be established on either side of the block to direct the air stream across the faces being mined. Temporary brattices will be used to block openings that would allow short-circuiting of the ventilation stream. Small fans and tubing will be used to ventilate headings being worked beyond the last open crosscut.

TABLE 7.4
UPPER LEVEL VENTILATION REQUIREMENTS
 (ft³/min)

Shaft Station	5,000
Development:	
Station Area	37,500
Main Entries	37,500
Sublevel Stoping With Backfill Unit	37,500
Sublevel Stoping With Full Subsidence Unit	25,500
Block Caving Unit:	
Monitor Drifts	36,000
Block Undercut	37,500
Production:	
Sublevel Stoping With Backfill Unit	Exhaust Only
Sublevel Stoping With Full Subsidence Unit	Exhaust Only
Block Caving Unit:	
Monitor Drifts	18,000
Block Undercut	9,000

- LEGEND**
- RETURN AIR
 - FRESH AIR
 - PERMANENT BRATTICE
 - TEMPORARY BRATTICE
 - REGULATOR
 - AIR DOOR
 - OVERCAST
 - STOPE OUTLINE

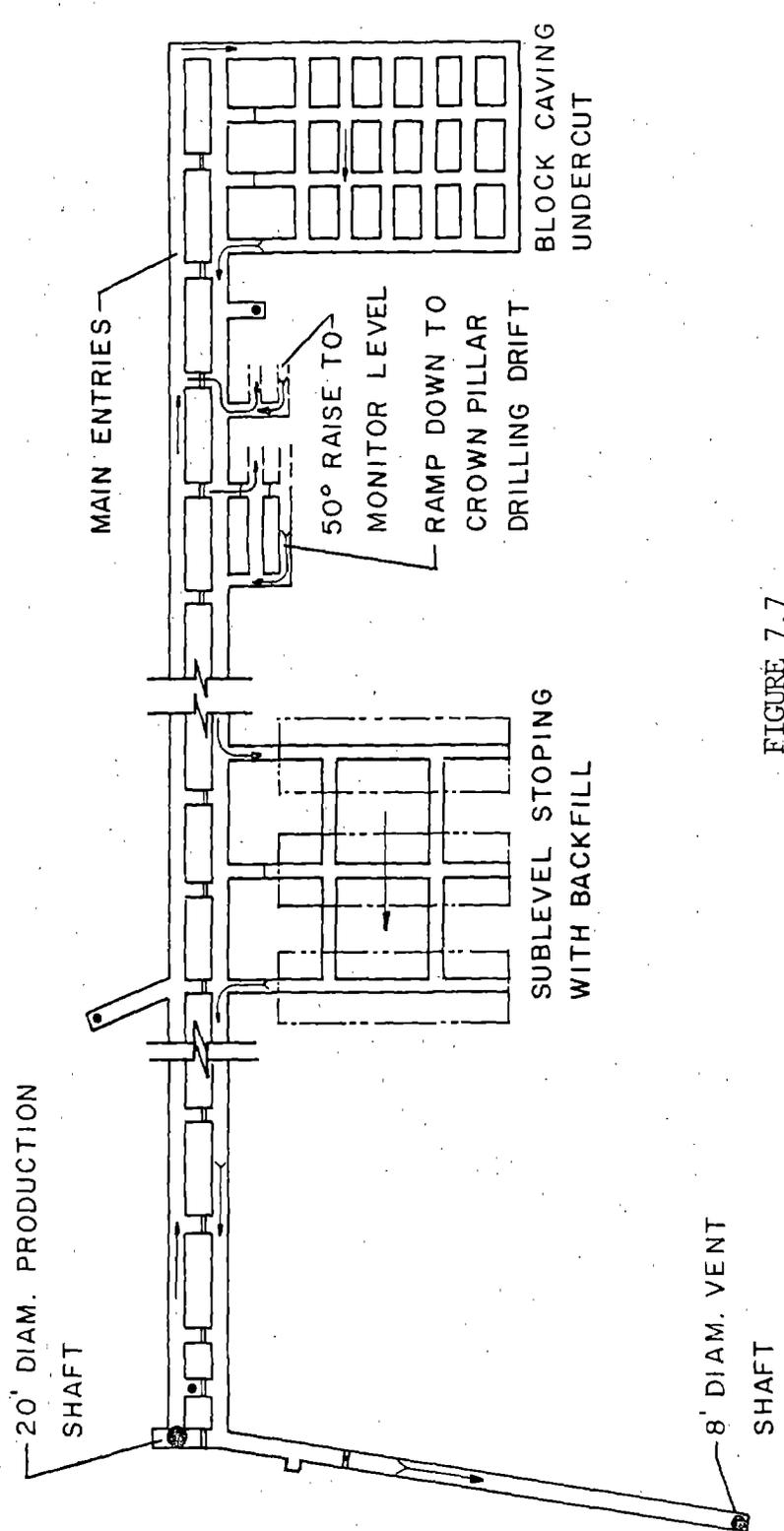


FIGURE 7.7

**VENTILATION PLAN
UPPER LEVEL - DEVELOPMENT PHASE**

SCALE: 1"=200'

7.5.2 Demonstration Mining:

Fresh air requirements on the upper level during demonstration mining will be minimal, limited to the maintenance of adequate circulation of fresh air in monitor drifts to permit observation in the caving zone (Table 7.4). However, the volume of exhaust air flowing in the upper level return airway will be substantially larger. Much of the ventilation flow supplied to the sublevel stoping with backfill and the sublevel stoping with full subsidence systems from the lower level and all of the ventilating air supplied from the sublevel will be discharged through the stopes to the upper level. The airflow pattern through the upper level during demonstration mining is shown in Figure 7.8.

LEGEND

- RETURN AIR
- FRESH AIR
- PERMANENT BRATTICE
- TEMPORARY BRATTICE
- REGULATOR
- AIR DOOR
- OVERCAST
- X
- STOPE OUTLINE

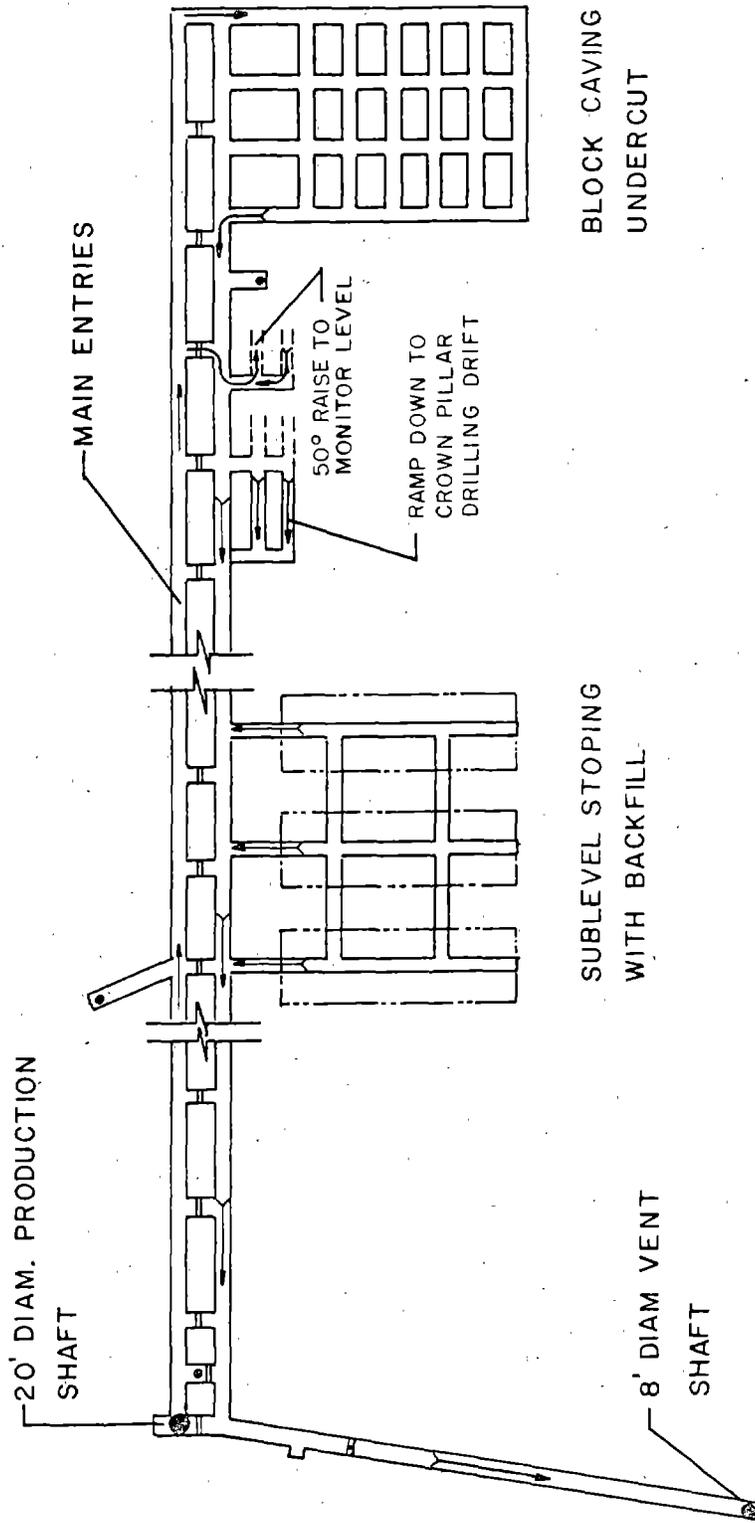


FIGURE 7.8

VENTILATION PLAN
 UPPER LEVEL - PRODUCTION PHASE

SCALE: 1" = 200'

8.0 SUPPORT SYSTEMS

8.1 OIL SHALE HANDLING

Oil shale will be transported by LHD's from the point of extraction to a primary crusher on the lower level. A minus-10-inch product will be discharged from the crusher onto a belt and conveyed to the shaft storage pocket. From the pocket shale will be hoisted to the surface and stockpiled. The handling system is designed to accommodate the maximum anticipated rate of production for the demonstration mine.

8.1.1 Crushing and Conveying

A skid mounted, feeder-breaker-type primary crusher will be provided for reducing run-of-mine oil shale to minus-10-inch material. The feeder-breaker unit will have a nominal capacity of 400 tons per hour. A dust suppression system (and possibly a dust collection system) will be installed to maintain acceptable working conditions in the crusher area.

Initially, the crusher will be located near the shaft to facilitate crushing of development ore excavated from the station area. Upper level station ore will not be crushed, but will be transferred directly through a grizzly-covered orepass to the storage pocket below the lower level. Later, when the lower level main entries have been advanced beyond the shaft station area, a conveyor belt will be installed and the crusher unit will be positioned to serve as the belt feeder. Oil shale from upper level and sublevel development headings will be hauled to the nearest orepass for transfer to the lower level. From the orepass the shale will be carried by LHD's to the feeder-breaker for crushing and conveyance to the shaft. As the demonstration units are developed and mined, the feeder-breaker will be situated near the entrance to the working area.

The minus-10-inch crusher product will be fed onto a 36-inch-wide conveyor belt having a nominal capacity of 400 tons per hour and will be discharged directly into the storage pocket near the production shaft (Figure 8.1). The belt will be located in the central entry of the three lower level mains, and ultimately will extend from the shaft to the sublevel stoping with subsidence demonstration unit. The initial segment of the conveyor will be installed when the main entries have been driven about 450 feet beyond the shaft. Thereafter the belt will be lengthened in increments of approximately 345 feet to match development advance. The conveyor system will be of cable-suspension-type construction and will be equipped with a skid-mounted tail section to expedite periodic extensions. The conveyor will be provided with an electric drive and necessary safety devices.

8.1.2 Hoisting

A two-skip hoisting system is planned. Shale will be transferred from the 500-ton storage pocket into two measuring pockets, from which the

skips will be loaded (Figure 8.1). Skips will be of bottom-dump-type construction and will empty into an ore bin located in the headframe. The two-skip arrangement will ensure a balanced ore hoisting system (Figure 8.2). Based on the results of the 20-foot-diameter shaft study (2), the following specifications pertain to the production shaft ore hoisting potential:

Production	380 tons per hour
Hoisting distance	2,150 feet
Number of loading levels	1
Skip capacity	12 tons
Hoisting speed	1,800 feet/minute

Production from the demonstration mine will average 354 tons per shift in Year 1 while only limited development work is in progress. Peak production averaging 2,385 tons per shift will occur for a brief period in Year 5 when sublevel stopes and the sublevel and upper level main entries are being mined concurrently. Thus, the maximum hoisting capacity of 7,500 tons per day (2,500 tons per shift) for the 20-foot-diameter production shaft specified in the contract will be adequate for development and operation of the demonstration mine.

8.1.3 Surface Storage

Oil shale will be hauled from the surface ore bin (in the headframe) to a surface stockpile area. Normal production from the demonstration mine will require the use of one 35-ton off-highway end-dump truck. During periods of high productivity a second truck will be required.

The stockpile area will be located about 1,600 feet to the west of the production shaft (Figure 8.3) and will be large enough to store approximately 1.3 mm tons of crushed oil shale. Retaining dams will be located above and below the stockpile area. The upstream retention structure will divert surface runoff from adjacent areas around the stockpile and the downstream dam will collect runoff from the pile. The stockpile runoff may require disposal with the saline mine water.

8.2 MINE DEWATERING

Prior to the initiation of caving in the block-caving unit, water inflow is predicted to be less than 200 gallons per minute (gpm). If the subsidence zone should intersect the lower aquifer an inflow of 1,600 gpm is anticipated (4). The mine dewatering system has been designed to handle a steady-state flow of this magnitude, with an actual pumping capacity of 3,200 gpm (short-term).

8.2.1 Mine Drainage

Shale beds in the R-4 mining zone dip approximately 2° in a north-westerly direction. By driving main headings S20°W from the shaft, parallel

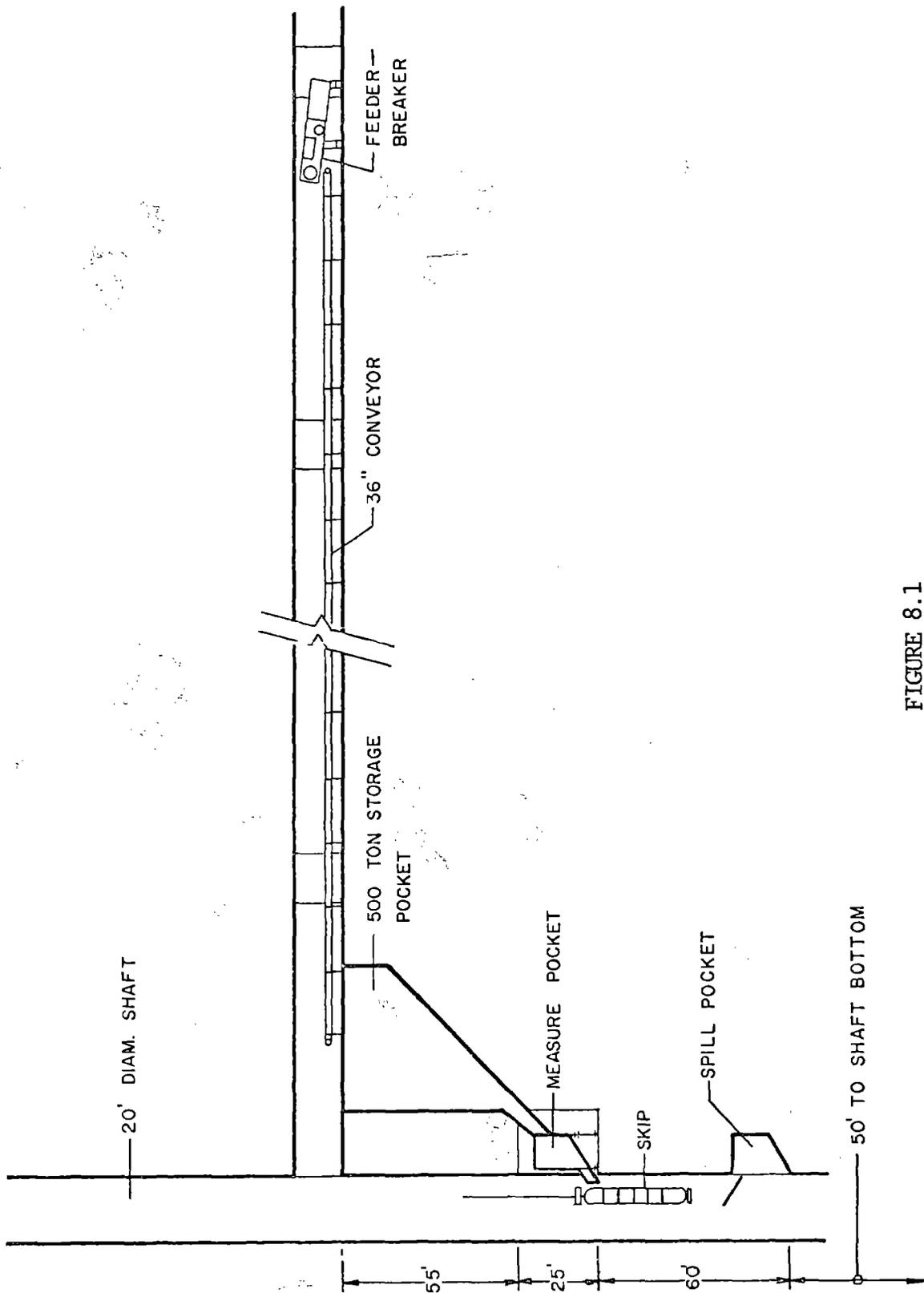


FIGURE 8.1
UNDERGROUND ORE HANDLING

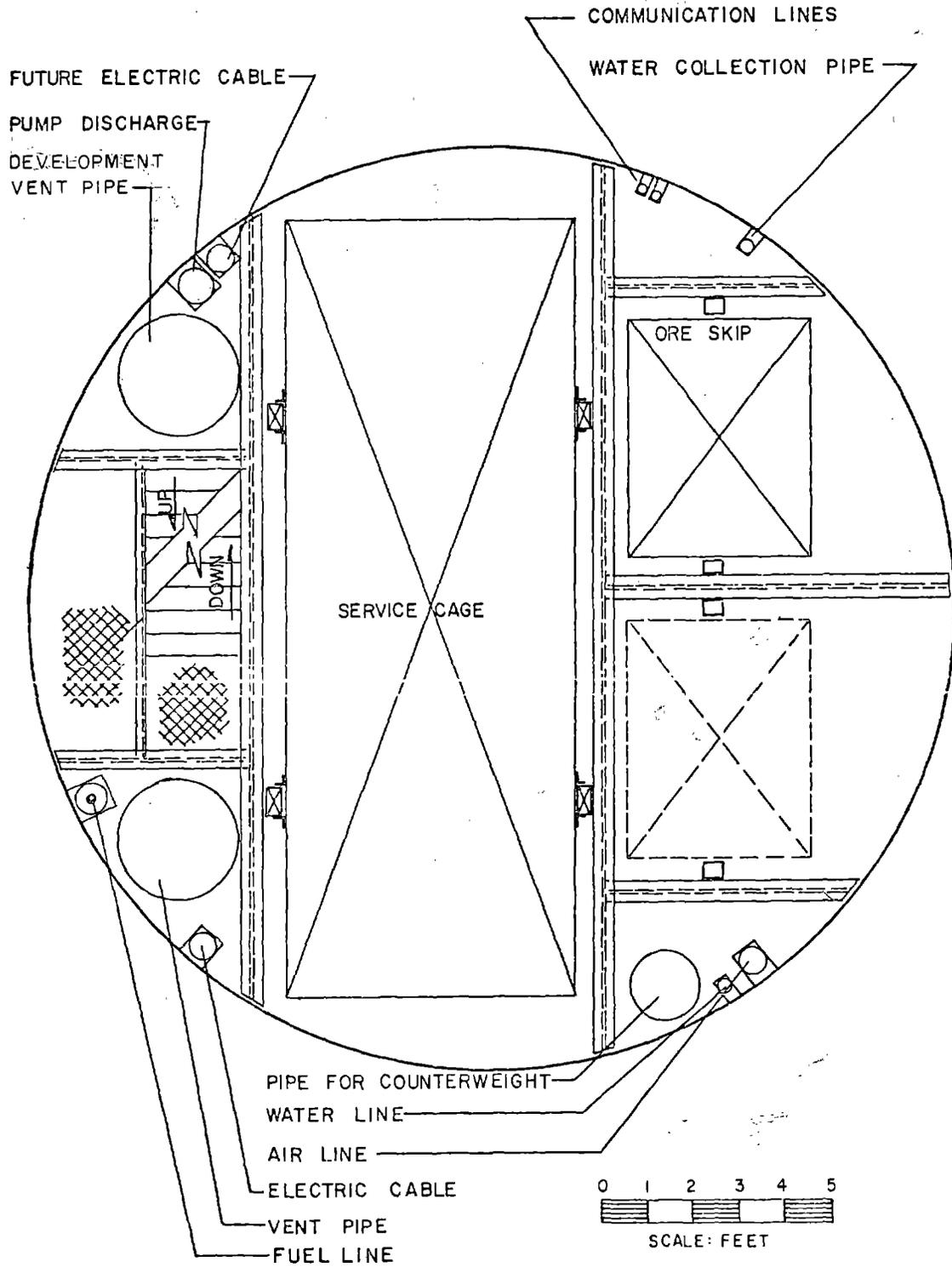


FIGURE 8.2
PRODUCTION SHAFT ARRANGEMENT

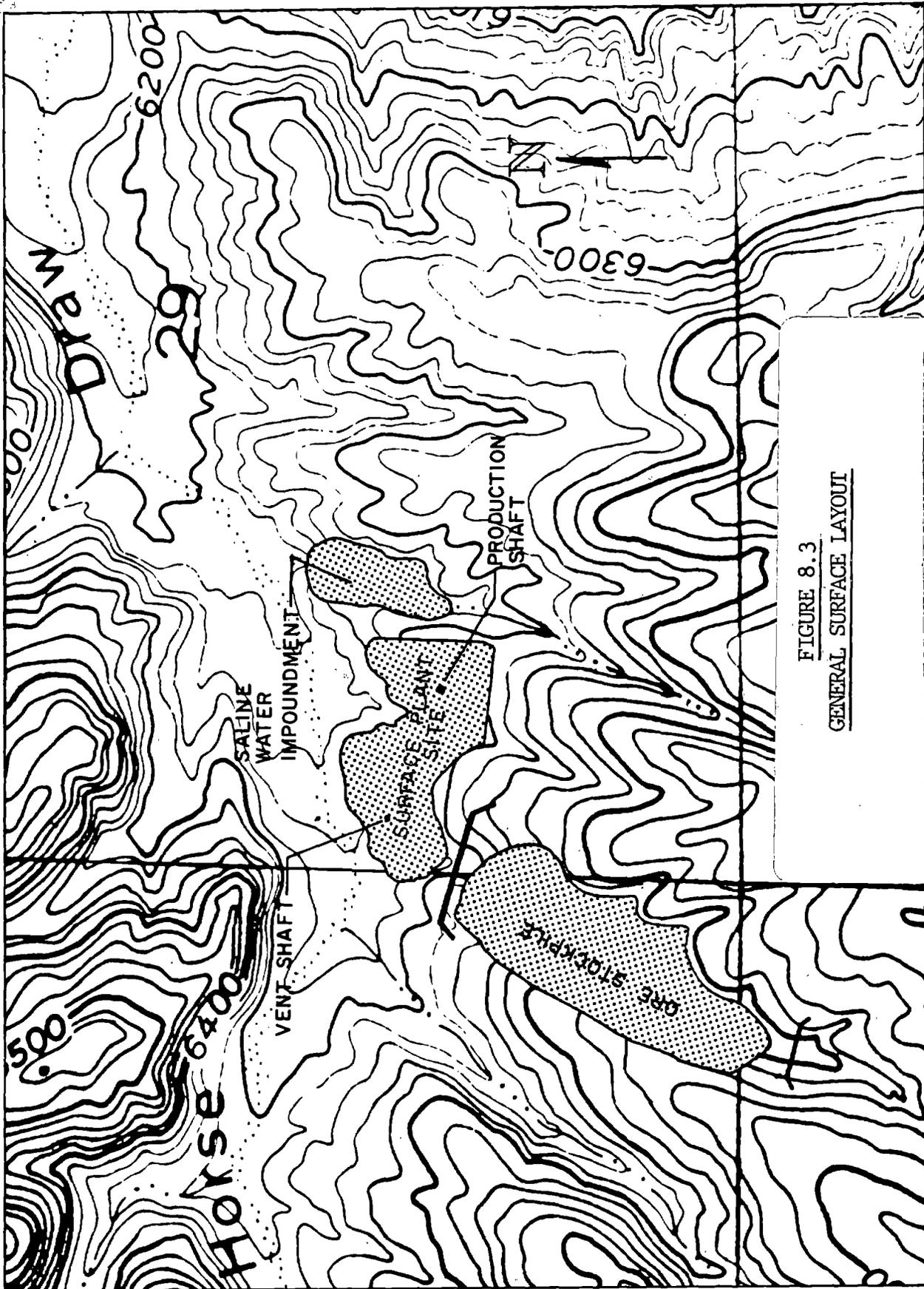


FIGURE 8.3
GENERAL SURFACE LAYOUT

to bedding, the workings will extend slightly updip. Seepage from the faces will drain through ditches to a sump located near the lower level shaft station. Accumulations of water in the production and ventilation shaft bottoms will be pumped to the sump. During the later stages of demonstration mining, when subsidence systems are investigated, it may be necessary to control the anticipated larger inflows by impoundment near the working areas and pumping to the main sump (Section 8.2.4).

8.2.2 Sump Design

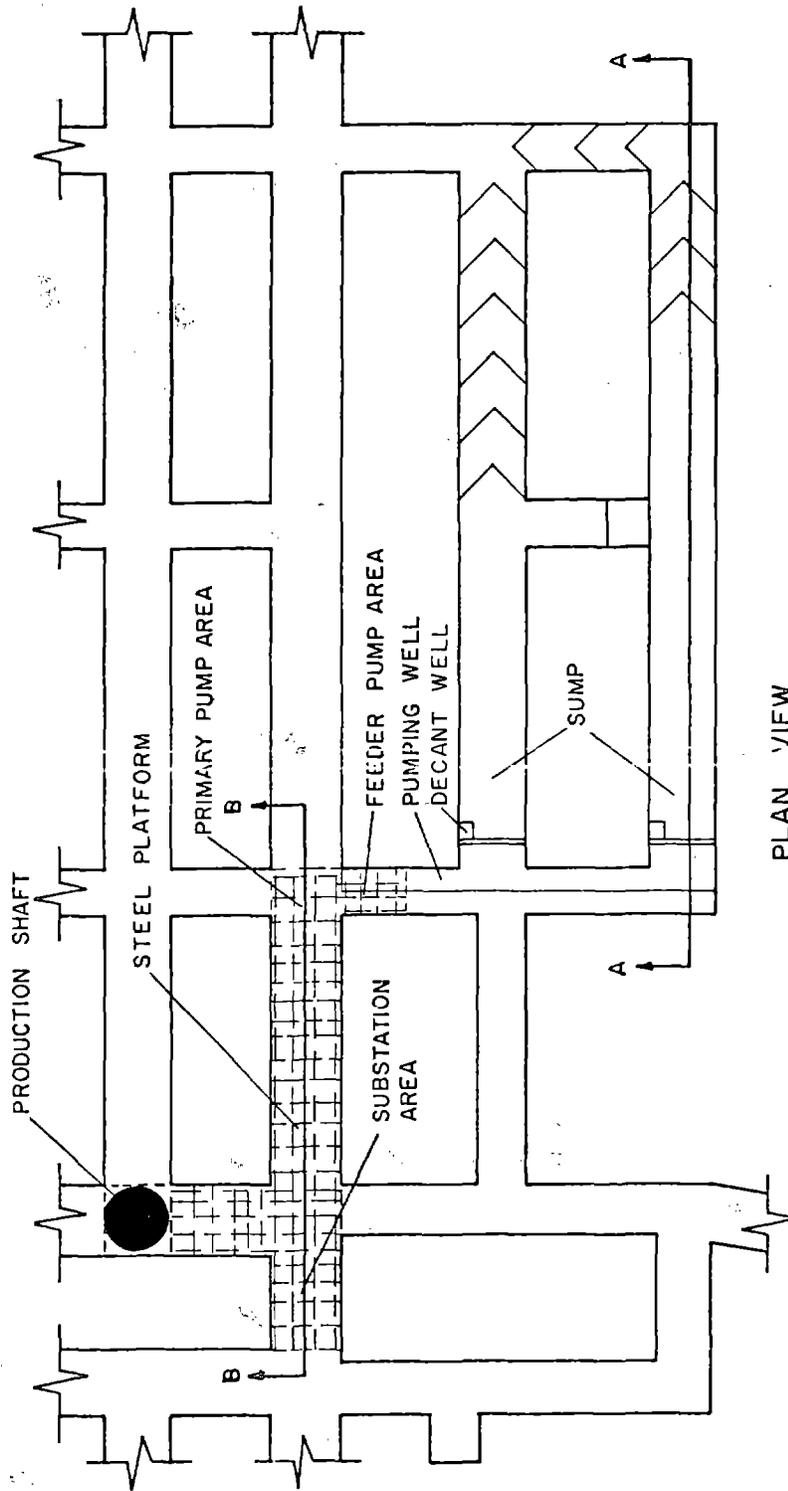
The primary sump will be located in the lower level shaft station area. The facility will consist of two parallel storage areas (Figure 8.4), designed so that either area can be isolated for periodic cleaning of sediment. A concrete bulkhead will separate each storage area from a common pumping well. Outlet pipes will be built into each bulkhead at several elevations to control water flow rate to the pumps, and decanting wells located behind each wall will permit decantation of impoundment areas prior to scheduled cleanout. Total sump capacity (two storage areas) will be approximately 800,000 gallons, a volume sufficient to impound an inflow rate of 1,600 gallons per minute for a period of about eight hours.

8.2.3 Pumping System

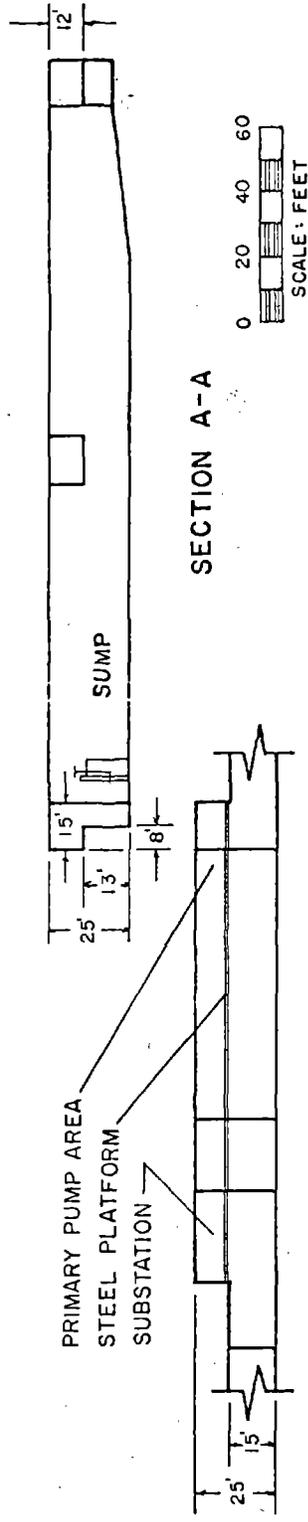
The primary pump room will be located above the roof horizon of the lower mine level (Figure 8.5). The system is designed to permit continued pumping from the mine even if the lower level should completely fill with water. Pump room construction will require roof excavation to a height of 25 feet in connecting drifts between the shaft and sump, and erection of an elevated steel deck for the pumping facility and associated substation.

Five-inch vertical feeder pumps will lift water from the pumping well to multistage horizontal pumps. Water will be moved by the multistage primary pumps through a 14-inch line in the production shaft to a booster station halfway up the shaft. Multistage booster pumps, similar to the primaries, will complete the lift to the surface (Figure 8.5). Each pumping unit (i.e., one feeder, one primary, and one booster pump) will have a capacity of 800 gallons per minute. Two such units (one standby) will be installed for mine startup, and two additional units will be required prior to demonstration mining in the subsidence units. The pumping capacity provided by two 800-gpm units during the early years of the demonstration program will ensure adequate short-term capability to control inflows that may occur as a result of unexpected ground failure.

The mine dewatering system will be fully automated. Motor-driven valves, level probes, pressure sensors, switches, etc. will be installed, together with appropriate warning devices, to achieve this mode of operation. A separate set of warning indicators will be located in the surface security office to permit 24-hour surveillance of the system.



PLAN VIEW



SECTION A-A

SECTION B-B

FIGURE 8.4
PRIMARY SUMP DESIGN

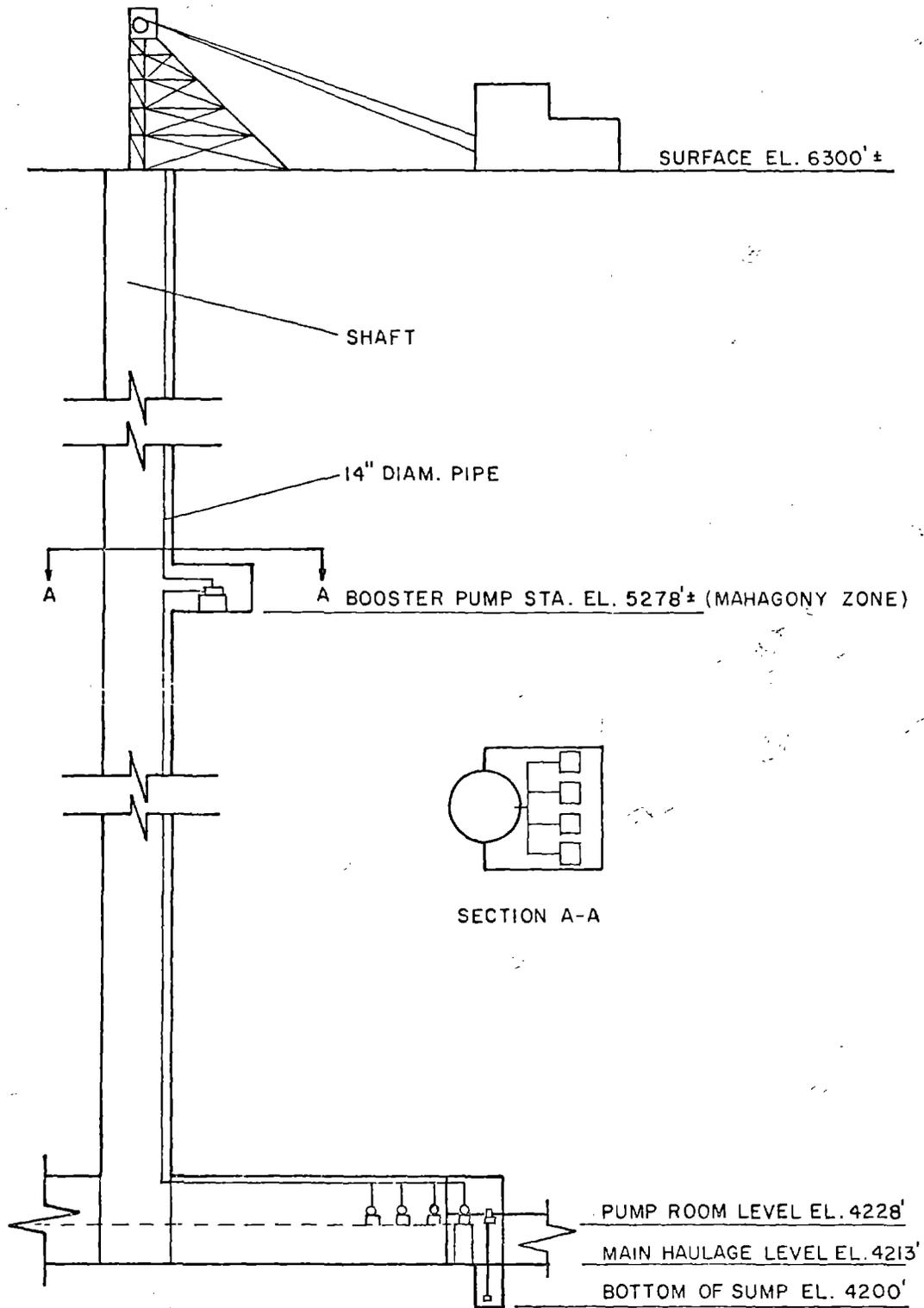


FIGURE 8.5
MINE DEWATERING SYSTEM

8.2.4 Emergency Dams

Dam sites will be prepared in each of the three main entries on the lower level. At each site concrete hitches cast in the entry walls and precut 12- by 12-inch timbers will permit erection of temporary timber dams (Figure 8.6) to aid in controlling sudden, large inflows of water. These dams are designed to retain a maximum 20-foot head of water while permitting limited bypass flow to facilitate controlled drawdown and discharge from the mine. The additional surge capacity created by these structures will provide temporary flood protection for the shaft and related facilities and will extend the time available for inflow control.

8.2.5 Surface Disposal of Mine Water

Water pumped from the mine will be discharged into a three-acre, 20-million-gallon capacity impoundment area. The area will be lined with an impermeable material to minimize seepage of saline mine water into the soil. As discussed in the Water Management Study (4), ultimate disposal of mine water probably will be effected by reinjection into the lower aquifer.

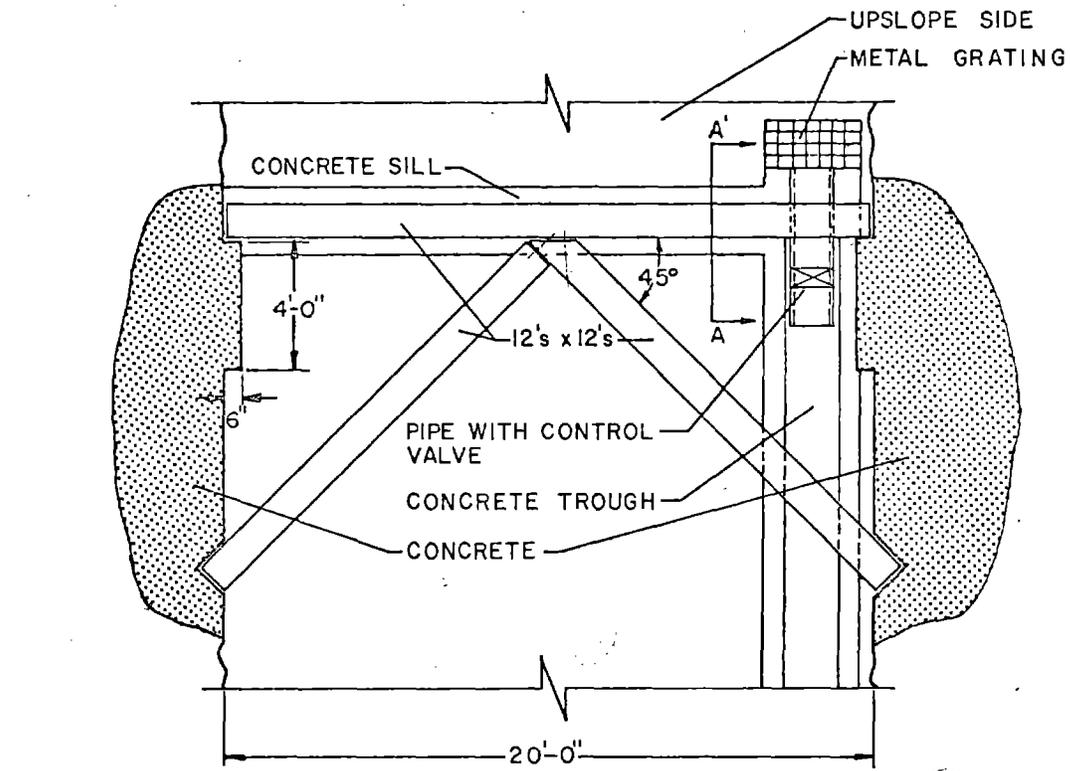
8.3 ANCILLARY OPERATIONS

Ancillary operations include a variety of support functions and activities which are essential for orderly day-to-day mine operation. Such activities include:

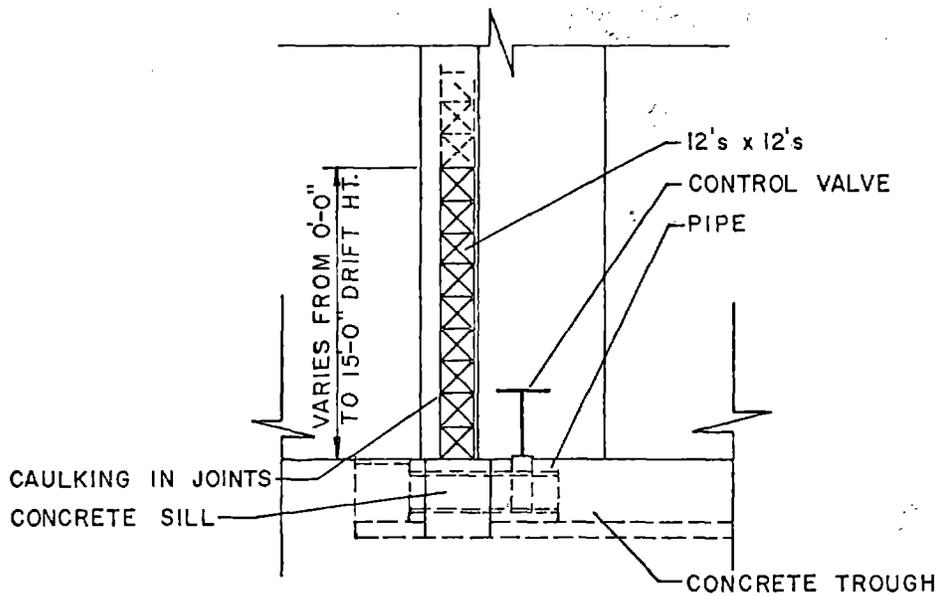
- Power distribution
- Communications
- Maintenance
- Road construction
- Fuel handling
- Supply and debris handling
- Water supply
- Dust suppression

8.3.1 Power Distribution

Primary power will be delivered to a permanent distribution substation located on the surface near the production shaft. Surrounded by a fenced enclosure, the substation will distribute power to the underground mine and to various surface units such as hoists, mine exhaust fan, etc. The exhaust fan will be served by a 4,160-volt line. A 13,800-volt feeder line will be run into the mine and will supply power to all levels. Substations located at the booster-pump level, upper level and lower level station area will supply current at 4,160 volts to mine dewatering pumps and portable power centers. From the power centers, reduced voltages will be supplied for auxiliary ventilation, crushing, lighting, etc.



PLAN



SECTION A-A'

FIGURE 8.6
EMERGENCY WATER CONTROL DAM

A diesel-powered generator, located on the surface, will provide emergency power for the pumping system, exhaust fan, and emergency hoist.

8.3.2 Communications

The communication system for the demonstration mine will be a standard paging telephone network with extensions located at permanent work areas such as shaft stations, crusher area, and maintenance shop. The system will include several phones at critical locations on the surface. A separate communication system will be provided between the cage and hoist house.

New underground mine communication systems currently being developed by the Bureau of Mines can be tested at the demonstration site.

8.3.3 Maintenance

Adequate maintenance facilities and effective procedures are an integral part of a mechanized mining operation when scheduled production depends on maximum availability of equipment. Both preventive maintenance and emergency repair services must be provided underground. Minor breakdowns will be repaired by a mobile maintenance crew. Major repairs, as well as preventive maintenance, will be performed in the shop.

The underground shop openings will be excavated so that the repair bays will be large enough to accommodate the largest units of mobile equipment and provide an adequate laydown area. The floor will be concreted throughout. The walls will be sprayed with a sealer that will minimize air slacking and provide a good light-reflective surface. Fans will maintain a positive airflow and provide adequate fresh air for prompt removal of noxious gases. First-aid supplies, fire-fighting equipment, and sanitation facilities will be located within the immediate vicinity of the shop area.

8.3.4 Road Construction

Underground haul road surfaces will be improved by application of shale fines selectively obtained from the headings and/or crushed aggregate supplied from a source outside the mine.

8.3.5 Fuel Handling

Diesel fuel for mine equipment will be delivered by tanker truck to a surface storage tank. A 2-inch line will deliver fuel underground from the surface tank via the production shaft to fuel stations near the upper and lower shaft stations. To improve safety a separate borehole drilled from surface to isolate the fuel delivery line, may be desirable. All lubricants, hydraulic fluids, etc., will be transported underground in steel drums.

8.3.6 Explosives Handling

Ammonium nitrate and fuel oil (AN/FO) will be the primary explosive agent employed in all underground blasting. AN/FO will be received in truckload shipments, stored in a surface magazine, and transported underground daily in amounts equivalent to the daily consumption. A vehicle, suitably equipped for carrying explosives, will be provided for surface transportation between the magazine and shaft. AN/FO and other explosives will be stored temporarily in underground magazines located in areas set apart from normal travel routes and constructed in accordance with applicable State and Federal Regulations. Underground transportation of blasting agents between the magazine and face will be by a diesel-powered explosives loader or other conveyance suitably equipped for the task. Detonators will be stored on surface and taken underground as needed.

8.3.7 Supply and Debris Handling

Supplies, such as roof bolts, drill steel, pipe, etc., will be stocked in a surface warehouse prior to transport to an underground storage area. A supply vehicle will be used to deliver supplies from the storage area to individual work areas within the mine.

To minimize obstacles and fire hazards, empty crates, packing material, and all other debris will be removed from the mine as it accumulates and will be disposed of properly on the surface.

8.3.8 Water Supply

Potable water will be required at several surface facilities and underground at the shop and lunchrooms. The water will be stored in a tank located on the surface and will be gravity fed into the mine through a waterline located in the production shaft to underground distribution lines. Separate lines will run from the storage tank to surface facilities such as changehouse and office. Raw water for drilling, dust suppression and general mine use will likewise be piped underground via the production shaft and thence to areas of usage. This water will probably be supplied from USBM drill hole 01-A.

8.3.9 Dust Suppression

Sources of dust underground include drilling, blasting, loading, hauling, crushing, and conveying. Dust suppression measures will be required to ensure a safe and comfortable working environment.

Dust generated by drilling will be controlled with water applied to the cutting edges of the drill bits. Water curtains and ample ventilation will be used to remove dust created by blasting. Dust control during loading operations will consist of wetting down the muck pile before loading and intermittently thereafter, if necessary. Surfaces of haul roads will be wetted regularly to minimize dust generation. Dust suppression will be an integral feature of crusher and belt transfer point design. Belt transfer points may be enclosed to aid in controlling dust.

9.0 HEALTH, SAFETY, AND ENVIRONMENTAL CONSIDERATIONS

9.1 HEALTH AND SAFETY

The health and safety of surface and underground personnel were primary concerns throughout the process of demonstration mine layout and design. All aspects of the demonstration program have been designed to ensure that a non-hazardous working environment can be maintained and that the work can be completed safely. Specific health and safety topics discussed below include dust and toxic gases, equipment, underground illumination, fire suppression, first aid and safety training, emergencies, and mine evacuation.

9.1.1 Dust and Toxic Gases

Because dust is generated by almost all mechanized mining activity, suppression must be an integral design consideration. Details regarding specific recommendations for dust suppression measures have been presented in Section 8.3.9. The main objective of these measures will be to provide a relatively dust-free working environment throughout the mine.

Toxic gases such as hydrocarbons, oxides of nitrogen, etc., are associated with the use of diesel equipment underground. In addition to these exhaust gases, methane and hydrogen sulfide occur within oil shale deposits and may be released into the working environment. The ventilation system will dilute and remove toxic gases from the mine and is designed to achieve or exceed all State and Federal mine air quality standards.

Supervisory personnel will carry portable gas detectors to monitor air quality in the headings. The main ventilation fan will be situated on the surface in compliance with MSHA Regulations for fan installations in mines classified as gassy. The fan will be installed in fireproof housing equipped with fireproof air ducts and explosion doors. Controls will be provided for automatic change-over to a standby diesel generator in event of power failure and for reversal of airflow in event of fire.

9.1.2 Equipment

Equipment used beyond the last open crosscut will be classified as permissible (MSHA Schedule 31). All diesel equipment will be provided with built-in fire suppression systems. Individual chemical fire extinguishers will be furnished near stationary equipment. The conveyor will be installed with fire suppression systems and fire-resistant belting. Where possible, the conveyor will be located in a neutral split (between the intake and exhaust airways) to permit isolation in case of fire. Equipment will be muffled as required to achieve acceptable noise levels.

9.1.3 Underground Illumination

Adequate illumination will be provided in the demonstration mine for personnel to work safely. A simple lighting scheme requiring minimal maintenance will be provided. All equipment will be furnished with sufficient lighting to ensure the safety of the operator. All personnel will be required to wear battery cap lamps.

9.1.4 Fire Suppression Devices

Fire suppression hardware will be supplied in both surface and underground work areas. As mentioned in Section 9.1.2, all underground equipment included in the conveyor system will be provided with fire suppression devices. Fire fighting equipment will be available in the underground shop, fuel storage area, magazine, pump station, etc. The practice of general mine cleanliness will be enforced to minimize fire hazards.

9.1.5 First Aid and Safety Training

First aid and safety training, as mandated by MSHA, will be provided for all miners and supervisory personnel. Underground personnel will carry individual self-rescuers and wear required protective apparel. First aid stations, containing stretchers and medical supplies, will be set up in the shaft station areas and other strategic locations within the mine. Supervisory personnel will be called upon to instruct the miners concerning safe work practices.

9.1.6 Emergencies and Mine Evacuation

The mine communication system described in Section 8.3.2 will be useful during emergencies and situations requiring mine evacuation. All mine personnel will be familiarized with the layout of the mine and the escape routes to be used during emergencies. A special rescue team shall be trained for work in mine emergencies. In the event of electrical failure, a standby diesel generator will supply power for the emergency man hoist in the ventilation shaft. Specific procedures for emergencies and mine evacuation will be drawn up and implemented by mine management.

9.2 ENVIRONMENTAL QUALITY

The demonstration mining program has been designed to be performed in a manner which will minimize environmental disturbance. Environmental impacts will be monitored throughout the course of mining. Monitoring will consist of measurements of surface subsidence and disturbance, air contamination, and water quality. Measures will be taken to mitigate both short-term and long-term environmental degradation.

9.2.1 Land

Surface subsidence is one potentially adverse effect demonstration mining may exert on surface land. Subsidence could occur if the upward propagation of caving over the block caving demonstration unit were to extend beyond the intended limits. The consequence of such propagation could be uncontrollable inflows of ground water into the mine as well as surface subsidence. Thus all efforts will be made to avoid such a development.

Propagation of caving will be controlled by regulating the withdrawal of broken ore from the unit. Initial fracturing within the caving zone will be monitored with instrumentation situated in observation drifts driven within the cave block. The caving demonstration will be concluded before subsidence has extended upward to the dissolution surface. The caving chamber and stopes below will be left filled with ore to prevent continuation of caving after mining is concluded.

Instrumentation will also be installed and monitored on the surface to detect the incidence and extent of any surface disturbance which may occur. The subsidence demonstration units have been located beneath a remote area of the site surface to minimize visual impact if subsidence occurs.

The ore stockpile will occupy about 13 acres and the water impoundment dam site will cover about three acres. Other surface facilities such as hoist house, changehouse, and office will occupy a minimal amount of land surface. Exposed soil surface such as downstream faces of dams will be stabilized with grasses or other types of acceptable ground cover.

To control erosion, drainage ditches and permanent culverts will be maintained on haul roads and access roads. Roads will be graded and adequately wetted. Structures such as the power substation and the exhaust fan installation will be enclosed and made inaccessible to wildlife and domestic livestock. These enclosures will not interfere significantly with wildlife migration. Upon completion of the project, all affected surface areas will be reclaimed and revegetated.

9.2.2 Air Quality

Measures to maintain suitable air quality underground have been discussed in Section 7 (Ventilation) and Section 8.3.9 (Dust Suppression). Surface air quality will be affected by surface activities and by air exhausted from the mine. Gas and particulate contaminants from the mine will be diluted sufficiently so that the impact on surface air quality from this source will be negligible.

Potential sources of dust from surface activities are travel on access and haul roads, dumping (into orebin or stockpile), and truck loading. Dust generation at dumping and truck loading points is expected to be minimal.

since the oil shale will have been wetted underground. However, additional dust suppression devices will be provided at these points. Access and haul roads will be wetted periodically to minimize dust generation. Diesel exhaust from off-highway truck(s) and other vehicles will be diluted by surface air and will exert no significant effect on air quality.

9.2.3 Water Quality

Water discharged from the mine and runoff from the oil shale stockpile will be pumped to a saline water storage pond. The base of the impoundment will be covered with an impermeable material to minimize infiltration. The draft report, "Water Management in Oil Shale Mining" (4), states that the only economical means of saline water disposal is by reinjection into an aquifer of equivalent or poorer quality (i.e., the lower aquifer). A reinjection system, assumed to be part of the surface plant designed by others, will dispose of overflow from the saline water storage pond.

Vegetation will be established on exposed soil surfaces to minimize erosion. Suspended sediment in runoff water will be alleviated further by runoff control on roads, and by grading and ditching of sites required for surface facilities.

Fuel storage tanks will be surrounded by impermeable, protective berms of sufficient capacity to isolate fuel spills on the surface. Composting toilets will be provided underground, and on the surface a suitable septic system will be implemented to prevent the occurrence of sewage pollution.

In summary, all project activities will be conducted in a manner intended to conserve and protect the surface and ground-water resources of the area.

10.0 CAPITAL COST DETAIL

10.1 GENERAL ASSUMPTIONS AND CONDITIONS

The cost estimate for the demonstration mining program is broken down into two distinct categories. Labor and consumable supply costs are categorized as operating and maintenance costs and are detailed in Section 11.0. Costs arising from the purchase of equipment or specialized services are classified as capital costs and are presented in this section.

In preparing the capital cost estimate, the authors have relied upon experience in other oil shale operations and/or contacts with manufacturers and vendors to predict equipment requirements and costs. Table 10.1 lists all items whose acquisition is considered to be a capital expense. As can be seen, the list of capital expenditures has been subdivided into several procurement areas each of which will be described briefly in the subsequent discussion. Pursuant to the Scope of Work, some items were assumed to be furnished by others and do not appear in Table 10.1. All such items are listed in Section 10.3.

Mobile Equipment - Underground

Units listed under this heading comprise all underground mobile equipment required to complete the Scope of Work as detailed in the earlier sections of this report. Technical descriptions of the equipment involved have already been given and its procurement schedule is presented in Table 6.9.

Mobile Equipment - Surface

Included in this category are all mobile units required on surface in support of the mining operations. The items listed are self-explanatory and their procurement schedule is also a part of Table 6.9.

Electrical Installations

This procurement area incorporates the purchase of three electrical substations and three portable power centers. One substation will be installed in the lower level shaft station area, one in the upper level station area, and the third will be located at the booster pump level approximately 1,000 feet below the shaft collar. Two of the power centers will be situated on the lower level of the mine - one will be assigned to the shop area while the other will be located near the crusher. The third power center will be utilized on the upper level of the mine. All capital expenses associated with this procurement will be incurred during the first year of operation.

Pumps, Piping, and Controls

The initial purchase of all pumps, pump controls, and required piping to the production shaft are covered under this procurement heading. Included are the four primary pumps, the four booster pumps, and the four feeder pumps,

which together comprise the main pumping system. During Stage I activities only half of the total capacity provided by this system is required; therefore, only two booster pumps, primary pumps, and feeder pumps will be purchased and installed in the first year of operations. The remaining components of the system will be acquired at the initiation of Stage II operations.

Miscellaneous Equipment

Items listed under this category are major capital requirements which do not fall into any of the other classifications. The list includes one raise boring machine, one 1,200-horsepower diesel-powered generator for emergency use, and four 1,500 cfm air compressors to supply air to the underground equipment. The shale crushing and conveying system is also included under this heading. The initial investment in conveyor equipment will provide head and tail pulleys, a take-up unit, a drive unit, miscellaneous controls, and enough belt line to serve the backfilling demonstration units. The second conveyor expenditure will provide for the extension of the belt line to the caving units. Two feeder-breakers will be required; one to support both the initial development and the backfilling units and a second one to be purchased at the outset of Stage II activities to serve the two caving units in conjunction with the conveyor belt extension. The procurement schedule for auxiliary ventilation fans is shown in Table 11.10. The expense listed for the warehouse will provide for stocking of replacement parts required to maintain all mine equipment.

Underground Construction

Expenditures comprising this category are specialized construction projects that normally would not be accomplished by the mining crew. This includes erection of the lower level steel framework and decking associated with the pumping system, pouring of concrete bulkheads in the sump and floors in the shop area, and installation of pumps, substations, and diesel refueling stations. Also covered under this heading is the cost of a backfilling system, including a 30" cased borehole from surface, a 52" inclined raise between the chamber drilling level and the upper level, and other associated excavations.

Health and Safety Equipment

Items listed under this classification are self-explanatory and, for the most part, are purchased during the first year of operation.

10.2 CAPITAL COST DETAIL

Capital expenses anticipated for the project are presented on a yearly basis for both the one-shift-per-day and two-shifts-per-day operating modes. For costing purposes, it has been assumed that all equipment will be purchased outright. No salvage value at the conclusion of the project has been considered. All costs are expressed in 1978 dollars.

Table 10.1 presents the capital expenditure schedule, in detail, for the project operating in a one-shift-per-day mode. Table 10.2 presents scheduled expenditures for a two-shifts-per-day operating mode in a less detailed form, with the understanding that the breakdown shown in Table 10.1 applies equally to both modes. The total capital outlay is identical in either case. The schedule of expenditures vary, however, in that capital requirements for two years of the one-shift-per-day mode will accrue in one year if two-shifts-per-day are worked.

10.3 ITEMS EXCLUDED FROM CAPITAL COST ESTIMATE

Pursuant to the terms of the contract, there are some items necessary for this demonstration project which have not been included in the capital expenditure analysis. Facilities presumed to be furnished by others prior to the initiation of activities described in this report are:

- Twenty-foot-diameter production shaft, completely equipped with hoists, skips, headframe, cage, all ropes, guides, and safety devices. Includes all electrical, air and fuel lines, potable and raw water supply lines, and the sump discharge pipeline.
- Upper and lower shaft stations including approximately 100 feet of drifting on each level.
- Automated skip loading facilities and measuring pocket on lower level, including 500-ton storage pocket and spill pocket.
- Eight-foot-diameter ventilation shaft equipped with main fan, necessary ducting, and controls.
- Emergency hoisting facility situated at ventilation shaft.
- All surface buildings, including hoist house, explosive magazines, changehouse, office, and warehouse/shop.
- All other surface installations, including storage tanks, electrical substation, access roads, stockpile runoff diversion and retention facilities, fresh water supply, and saline water disposal system.

Although the cost of providing these items is assumed to be born by others, their associated cost of operation and maintenance has been included in the analysis of Section 11.0.

TABLE 10.1

CAPITAL EXPENSE SCHEDULE
(One-Shift-Per-Day Operating Mode)

Capital Item	Unit Cost	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total
I. MOBILE EQUIPMENT-UNDERGROUND								
LHD Unit, 5-Cubic-Yard	\$140,000	\$560,000	\$280,000	\$280,000	\$140,000	\$-	\$-	\$1,260,000
Face Jumbo	120,000	360,000	120,000	-	-	-	-	480,000
Ring Drill Jumbo	83,000	-	-	166,000	83,000	-	-	249,000
Low-Heading Roofbolter	90,000	180,000	90,000	-	-	-	-	270,000
Low-Heading Scaler	72,000	144,000	72,000	-	-	-	-	216,000
High-Heading Roofbolter	150,000	-	-	150,000	-	-	-	150,000
High-Heading Scaler	150,000	-	-	150,000	-	-	-	150,000
Powder Truck	60,000	120,000	-	-	-	-	-	120,000
LHD Unit, 1/2-Cubic-Yard	18,000	-	-	-	-	72,000	-	72,000
Utility Vehicle	45,000	45,000	-	-	-	-	-	45,000
Supervisor's Vehicle	11,000	22,000	-	-	22,000	-	-	44,000
Maintenance Vehicle	55,000	110,000	-	-	-	-	-	110,000
Subtotal		1,541,000	562,000	746,000	245,000	72,000	-	3,166,000
II. MOBILE EQUIPMENT-SURFACE								
Dump Truck 35-Ton	175,000	175,000	-	-	175,000	-	-	350,000
Motor Grader	85,000	85,000	-	-	-	-	-	85,000
Water Truck, 1,500 Gallon	40,000	40,000	-	-	-	-	-	40,000
Front End Loader, 4-Cubic-Yard	96,000	96,000	-	-	-	-	-	96,000
Flatbed Truck, 1-Ton	17,000	17,000	-	-	-	-	-	17,000
Pickup Truck, 1/2-Ton	7,000	14,000	-	14,000	-	-	-	28,000
Ambulance	28,000	28,000	-	-	-	-	-	28,000
Superintendent's Vehicle	8,000	8,000	-	8,000	-	8,000	-	24,000
Subtotal		463,000	-	8,000	189,000	8,000	-	668,000
III. ELECTRICAL INSTALLATIONS								
Lower Level Substation	138,000	138,000	-	-	-	-	-	138,000
Rooster Pump Level Substation	75,000	75,000	-	-	-	-	-	75,000
Upper Level Substation	60,000	60,000	-	-	-	-	-	60,000
Power Centers	75,000	150,000	75,000	-	-	-	-	225,000
Subtotal		423,000	75,000	-	-	-	-	498,000
IV. PUMPS, PIPING, & CONTROLS								
Shaft Sump Pump	5,000	10,000	-	-	-	-	-	10,000
Feeder Pump	12,000	24,000	-	-	-	-	24,000	48,000
Primary Pump	60,000	120,000	-	-	-	-	120,000	240,000
Booster Pump	60,000	120,000	-	-	-	-	120,000	240,000
Utility Pump	3,000	6,000	6,000	-	-	-	-	12,000
Subtotal		280,000	6,000	-	-	-	264,000	550,000
Subtotal-This Page		\$2,707,000	\$643,000	\$754,000	\$434,000	\$80,000	\$264,000	\$4,882,000

TABLE 10.1
(Continued)

CAPITAL EXPENSE SCHEDULE
(One-Shift-Per-Day Operating Mode)

Capital Item	Unit Cost	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total
V. MISCELLANEOUS EQUIPMENT								
Air Compressor, 1,500 cfm	\$ 46,000	\$ 92,000	-	\$ 92,000	-	-	-	\$ 184,000
Feeder-Breaker	110,000	110,000	-	-	110,000	-	-	220,000
Conveying System	300,000	196,000	-	-	104,000	-	-	300,000
Raise Boring Machine	235,000	235,000	-	-	-	-	-	235,000
Diesel Generator, 1,200 hp	150,000	150,000	-	-	-	-	-	150,000
Auxiliary Fan, 75 hp	8,300	-	-	8,300	8,300	-	-	16,600
Auxiliary Fan, 40 hp	4,400	4,400	4,400	4,400	-	-	-	13,200
Auxiliary Fan, 10 hp	2,200	17,600	8,800	8,800	-	-	-	35,200
Warehouse Stock	-	160,000	29,000	42,000	23,000	4,000	13,000	271,000
Jackleg Drill	3,500	-	-	-	-	14,000	-	14,000
Subtotal		965,000	42,200	155,500	245,300	18,000	13,000	1,439,000
VI. UNDERGROUND CONSTRUCTION								
Lower Level Station Area	305,000	305,000	-	-	-	-	-	305,000
Upper Level Station Area	55,000	55,000	-	-	-	-	-	55,000
Backfilling System	372,000	-	-	372,000	-	-	-	372,000
Emergency Dams	3,000	-	9,000	-	-	-	-	9,000
Subtotal		360,000	9,000	372,000	-	-	-	741,000
VII. HEALTH & SAFETY EQUIPMENT								
Methane Monitoring System	-	6,000	-	4,000	-	-	-	10,000
CO/H ₂ S Monitoring System	-	12,000	-	4,000	-	-	-	16,000
Hand-Held Gas Detection Equipment	-	2,500	2,500	-	-	-	-	5,000
Dust Monitoring Equipment	-	8,000	8,000	-	-	-	-	16,000
Communication System	-	2,000	-	2,000	-	-	-	4,000
Fire Control Equipment	-	14,000	6,500	-	-	4,500	-	25,000
Cap Lamps	65	5,200	-	-	-	-	-	5,200
Personal Safety Equipment	75	5,000	-	-	-	-	-	5,000
First-Aid Supplies	-	3,000	-	-	-	-	-	3,000
Oxygen-Breathing Apparatus	2,100	16,800	-	-	-	-	-	16,800
Subtotal		74,500	17,000	10,000	-	4,500	-	106,000
Subtotal - This Page		\$1,399,500	\$ 68,200	\$ 537,500	\$245,300	\$ 22,500	\$ 13,000	\$2,286,000
Subtotal - Previous Page		2,707,000	643,000	754,000	434,000	80,000	264,000	4,882,000
Total		\$4,106,500	\$711,200	\$1,291,500	\$679,300	\$102,500	\$277,000	\$7,168,000
Contingency @20%		821,300	142,200	258,300	135,900	20,500	55,400	1,433,600
GRAND TOTAL		\$4,927,800	\$853,400	\$1,549,800	\$815,200	\$123,000	\$332,400	\$8,601,600

TABLE 10.2

CAPITAL EXPENSE SCHEDULE
(Two-Shifts-Per-Day Operating Mode)

<u>Procurement Class</u>	<u>Year 1</u>	<u>Year 2</u>	<u>Year 3</u>	<u>Project Total</u>
Mobile Equipment-U/G	\$2,103,000	\$ 991,000	\$ 72,000	\$3,166,000
Mobile Equipment-Surface	463,000	197,000	8,000	668,000
Electrical Installation	498,000	-	-	498,000
Pumps, Piping, & Controls	286,000	-	264,000	550,000
Miscellaneous Equipment	1,007,200	400,800	31,000	1,439,000
Underground Construction	369,000	372,000	-	741,000
Health & Safety Equipment	<u>91,500</u>	<u>10,000</u>	<u>4,500</u>	<u>106,000</u>
Subtotal	\$4,817,700	1,970,800	379,500	\$7,168,000
Contingency @20%	<u>963,500</u>	<u>394,200</u>	<u>75,900</u>	<u>1,433,600</u>
GRAND TOTAL	\$5,781,200	\$2,365,000	\$455,400	\$8,601,600

11.0 OPERATING AND MAINTENANCE COST DETAIL

11.1 GENERAL ASSUMPTIONS AND CONDITIONS

The operating and maintenance costs detailed herein are the result of actual experience and observations made during Cliffs' participation in a number of mining ventures including oil shale. Cycle times developed for each of the mining functions were based upon this and other extensive mine operating experience. These cycle times were fundamental for subsequent analyses of demonstration mine program tasks. From these analyses, task schedules, as well as production and maintenance manpower, equipment and consumable supply requirements were determined. Costs of demonstrating the four specified mining systems were generated from these data.

Operating and maintenance costs are presented in four subcategories which are operating labor, operating supplies, maintenance labor, and maintenance supplies. Table 11.1 lists wage rates used in computing labor costs. Table 11.2 lists hourly equipment costs (excluding operating labor) used to detail mining costs. Costs are expressed in 1978 dollars.

11.2 DIRECT MINING COSTS - SAMPLE CALCULATION

For the purposes of this study, direct mining costs are defined as those costs arising solely from the six basic mining functions. The functions are drilling, blasting, mucking, heading cleanup, scaling, and roof bolting. Because of the large volume of material involved, it is not practical to detail every computation relating to the costing of the demonstration mine. Instead, the detailed calculations for costing a single task are presented on the following pages as representative of the procedures used to compute the direct mining costs for every case. The task selected for detailed presentation is the excavation of the chamber drilling drifts, an element in the development of the chamber and pillar demonstration unit.

11.2.1 Sample Calculation Summary

Chamber Drilling Drift Development

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 22,600	\$ 12,000	\$ 2,100	\$ 2,600	\$ 39,300
Mucking & Cleanup	6,500	4,800	5,500	4,600	21,400
Scaling & Bolting	8,400	5,200	1,500	1,900	17,000
Total Costs	<u>\$ 37,500</u>	<u>\$ 22,000</u>	<u>\$ 9,100</u>	<u>\$ 9,100</u>	<u>\$ 77,700</u>

TABLE 11.1
HOURLY WAGE SCHEDULE

<u>Job Class</u>	<u>Job Description</u>	<u>Average Number Employees</u>		<u>Hourly Rate</u>	<u>Loaded Shift Rate*</u>
		<u>1 Shift</u>	<u>2 Shifts</u>		
6	Hoistman	1	2	\$7.745	\$83.65
5	Electrician	1	2	7.287	78.70
	Mechanic	5	10		
4	Miner	9	18	6.901	74.53
3	Miner Helper	4	8	6.523	70.45
	Utility Man, Underground	3	6		
	Equipment Operator, Surface	2	4		
2	Skip Tender	1	2	6.231	67.29
1	Utility Man, Surface	<u>2</u>	<u>2</u>	5.847	63.15
	Total	28	54		

* Labor cost for 8-hour shift, including 35% fringe benefits

TABLE 11.2
HOURLY EQUIPMENT COST

<u>Equipment</u>	<u>Operating*</u> <u>Supplies</u>	<u>Maintenance</u> <u>Labor</u>	<u>Maintenance**</u> <u>Supplies</u>
Drill Jumbo	\$ 1.88	\$ 2.66	\$ 1.86
Bolting Jumbo	1.58	2.04	1.56
Charging Equipment	0.62	0.41	0.27
Scaler	2.24	4.80	6.15
LHD	10.39	11.91	10.03

* Operating supplies include fuel, tires, lube, filters, etc.

** Maintenance supplies include replacement parts and prorated shop supplies

11.2.2 Sample Calculation Detail

The following is a detailed description of the operating and maintenance cost calculation methods and results for the chamber drilling drifts. In this example, calculation results have been shown to the nearest dollar. Cost totals have been rounded to the nearest one hundred dollars for presentation in the Task Cost Summaries, as can be seen in Section 11.2.1.

DRILLING COSTS

A. Operating Labor:

Miner:	96.3 manshifts x \$74.53/manshift	\$ 7,177
Helper:	96.3 manshifts x \$70.45/manshift	6,784
	Total	<u>\$13,961</u>

B. Operating Supplies:

Drill jumbo:	512 hours x \$1.88/hour	\$ 963
Drill steel (1''):	16,896 feet drilled/1,500 feet/piece x \$80 each	901
Drill steel (1-1/4''):	21,120 feet drilled/2,000 feet/ piece x \$100 each	1,056
Striker bar:	38,016 feet drilled/1,500 feet/bar x \$50 each	1,267
Couplings:	38,016 feet drilled/1,500 feet/piece x \$14 each	355
Drill bits (1-1/2''):	16,896 feet drilled/1,500 feet/bit x \$20 each	225
Drill bits (2''):	21,120 feet drilled/2,000 feet/bit x \$30 each	317
Miscellaneous:	38,016 feet drilled x \$.006/foot	228
	Total	<u>\$ 5,312</u>

C. Maintenance Labor:

Drill jumbo:	512 hours x \$2.66/hour	\$ 1,362
Drills:	38,016 feet drilled x \$.011/foot	418
Bit sharpening:	38,016 feet drilled x \$.005/foot	190
	Total	<u>\$ 1,970</u>

D. Maintenance Supplies:

Drill jumbo:	512 hours x \$1.86/hour	\$ 952
Drills:	38,016 feet drilled x \$.04/foot	1,521
	Total	<u>\$ 2,473</u>

BLASTING COSTS

A. Operating Labor:

Miners: 115.6 manshifts x \$74.53/manshift		\$ 8,616
	Total	<u>\$ 8,616</u>

B. Operating Supplies:

Charging equipment: 307 hours x \$0.62/hour		\$ 190
AN/FO: 24,960 pounds x \$0.08/pound		1,997
Primers: 1,728 pounds x \$0.60/pound		1,037
Blasting caps: 3,456 units x \$0.80 each		2,765
Connecting Wire: 192 pounds x \$2.70/pound		518
Miscellaneous: 192 rounds x \$1.00/round		192
	Total	<u>\$ 6,699</u>

C. Maintenance Labor:

Charging equipment: 307 hours x \$0.41/hour		\$ 126
	Total	<u>\$ 126</u>

D. Maintenance Supplies:

Charging equipment: 307 hours x \$0.27/hour		\$ 83
	Total	<u>\$ 83</u>

MUCKING COSTS

A. Operating Labor:

Miner: 74.9 manshifts x \$74.53/manshift		\$ 5,582
	Total	<u>\$ 5,582</u>

B. Operating Supplies:

LHD's: 398 hours x \$10.39/hour		\$ 4,135
	Total	<u>\$ 4,135</u>

C. Maintenance Labor:

LHD's: 398 hours x \$11.91/hour		\$ 4,740
	Total	<u>\$ 4,740</u>

D. Maintenance Supplies:

LHD's: 398 hours x \$10.03/hour		\$ 3,992
	Total	<u>\$ 3,992</u>

CLEANUP COSTS

A. Operating Labor:

Miner: 12.0 manshifts x \$74.53/manshift

Total $\$ \frac{894}{894}$

B. Operating Supplies:

LHD's: 64 hours x \$10.39/hour

Total $\$ \frac{665}{665}$

C. Maintenance Labor:

LHD's: 64 hours x \$11.91/hour

Total $\$ \frac{762}{762}$

D. Maintenance Supplies:

LHD's: 64 hours x \$10.03/hour

Total $\$ \frac{642}{642}$

SCALING COSTS

A. Operating Labor:

Miner: 39.8 manshifts x \$74.53/manshift

Total $\$ \frac{2,966}{2,966}$

B. Operating Supplies:

Scaler: 211 hours x \$2.24/hour

Total $\$ \frac{473}{473}$

C. Maintenance Labor:

Scaler: 211 hours x \$4.80/hour

Total $\$ \frac{1,013}{1,013}$

D. Maintenance Supplies:

Scaler: 211 hours x \$6.15/hour

Total $\$ \frac{1,298}{1,298}$

ROOF BOLTING COSTS

A. Operating Labor:

Miner:	37.6 manshifts x \$74.53/manshift	\$ 2,802
Helper:	37.6 manshifts x \$70.45/manshift	2,648
	Total	<u>\$ 5,451</u>

B. Operating Supplies:

Bolting jumbo:	200 hours x \$1.58/hour	\$ 316
Roof bolts:	1,152 bolts x \$3.15/bolt	3,629
Drill steel:	7,488 feet drilled/1,500 feet/piece x \$50 each	250
Striker bar:	7,488 feet drilled/1,500 feet/piece x \$50 each	250
Couplings:	7,488 feet drilled/1,500 feet/piece x \$14 each	70
Drill bits:	7,488 feet drilled/1,500 feet/piece x \$25 each	125
Miscellaneous:	7,488 feet drilled x \$.006/foot	45
	Total	<u>\$ 4,685</u>

C. Maintenance Labor:

Bolting jumbo:	200 hours x \$2.04/hour	\$ 408
Drills:	7,488 feet drilled x \$.011/foot	82
Bit sharpening:	7,488 feet drilled x \$.005/foot	37
	Total	<u>\$ 527</u>

D. Maintenance Supplies:

Bolting jumbo:	200 hours x \$1.56/hour	\$ 312
Drills:	7,488 feet drilled x \$.04/foot	300
	Total	<u>\$ 612</u>

11.2.3 Basic Parameters for the Sample Calculation

Throughout the details of the sample calculation presented in the preceding section, certain parameters appear relating to manpower requirements, machine hours, and supplies consumed. The parameters are determined mainly from estimates of function cycle times as recounted in Section 5.0 and from basic facts about the excavations themselves. This section describes the methods used to derive these elemental cost parameters.

The chamber drilling drifts measure 12 feet wide by 12 feet high. As can be seen in Table 4.2, a total of 192 rounds, each producing approximately 100 tons, will be mined while developing the drilling drifts in the chamber and pillar unit.

Productive manshifts were calculated directly from cycle times. Additions for nonproductive time were determined by bar-chart interference analyses with subsequent proration for each basic function.

DRILLING

A. Operating Labor:

Miner Shifts:

192 rounds x 160 minutes/round ÷ 400 minutes/shift =	76.8
Add nonproductive manshifts	19.5
Total Miner Shifts	<u>96.3</u>
Total Helper Shifts (same as miner shifts)	

B. Total Feet Drilled:

Holes per round = 10 @2" diameter	
8 @1-1/2" diameter	
<u>18 total holes/round</u>	

Average hole length = 11 feet

1-1/2" hole footage: 8 holes x 192 rounds x 11 feet each =	16,896 feet
2" hole footage: 10 holes x 192 rounds x 11 feet each =	<u>21,120 "</u>
Total drilled footage:	<u>38,016 feet</u>

C. Machine Hours:

Drill jumbo: 192 rounds x 160 minutes/round ÷ 60 minutes/hour =	
512 hours	

BLASTING

A. Operating Labor:

Miner Shifts:

192 rounds x 96 minutes/round x 2 men ÷ 400 minutes/shift =	92.2
Add nonproductive manshifts	23.4
Total Miner Shifts	<u>115.6</u>

B. Explosives Supplies:

Primers: 192 round x 18 holes/round x .5 pound/hole = 1,728 pounds
AN/FO: 192 rounds x 130 pounds/round = 24,960 pounds
Blasting caps: 1 unit/hole x 18 holes/round x 192 round = 3,456 units
Connecting Wire: 1 pound/round x 192 rounds = 192 pounds

C. Machine Hours:

192 rounds x 96 minutes/round ÷ 60 minutes/hour = 307 hours

MUCKING

A. Operating Labor:

100 tons/round ÷ 5 tons/mucking cycle = 20 cycles/round
Average haul distance = 800 feet one way
Average haul time = 6.21 minutes/round trip
Miner Shifts:

192 rounds x 20 cycles/round x 6.21 minutes/cycle ÷	
400 minutes/cycle =	59.7
Add nonproductive manshifts	15.2
Total Miner Shifts	<u>74.9</u>

B. Machine Hours:

LHD: 192 rounds x 20 cycles/round x 6.21 minutes/cycle
÷ 60 minutes/hour = 398 hours

HEADING CLEANUP

A. Operating Labor:

Miner Shifts:	
192 rounds x 20 minutes/round ÷ 400 minutes/shift =	9.6
Add nonproductive manshifts	2.4
Total Miner Shifts	<u>12.0</u>

B. Machine Hours:

LHD: 192 rounds x 20 minutes/round ÷ 60 minutes/hour = 64 hours

SCALING

A. Operating Labor:

Miner Shifts;	
192 rounds x 66 minutes/round ÷ 400 minutes/shift =	31.7
Add nonproductive manshifts	8.1
Total Miner Shifts	<u>39.8</u>

B. Machine Hours:

192 rounds x 66 minutes/round ÷ 60 minutes/hour = 211 hours

ROOF BOLTING

A. Operating Labor:

Bolting occurs only every other round:
192 rounds ÷ 2 = 96 bolting cycles

Miner Shifts:	
96 cycles x 125 minutes/cycle ÷ 400 minutes/shift =	30.0
Add nonproductive manshifts	7.6
Total Miner Shifts	<u>37.6</u>
Total Helper Shifts (same as miner shifts)	

B. Bolting Supplies:

Roof Bolts: 12 bolts/round x 96 rounds = 1,152 bolts
Total feet drilled: 1,152 bolts x 6.5 feet average hole
depth = 7,488 feet

C. Machine Hours:

96 rounds x 125 minutes/round ÷ 60 minutes/hour = 200 hours

11.3 DIRECT MINING COSTS BY DEMONSTRATION UNIT

Following the procedures and methods detailed in the sample calculation, direct mining costs were generated for every operational task listed in the project schedule (Figure 6.1). The direct costs are presented here for each of the four demonstration mining units plus a fifth division identified as primary development. These direct costs are summarized in Tables 11.3 and 11.4.

TABLE 11.3

SUMMARY OF DIRECT MINING COSTS BY FUNCTION

<u>Demonstration Unit</u>	<u>Drilling & Blasting*</u>	<u>Mucking & Cleanup**</u>	<u>Scaling & Bolting</u>	<u>Totals</u>
Primary Development	\$ 630,500	\$ 290,200	\$234,200	\$1,154,900
Chamber & Pillar	96,500	504,700	36,800	638,000
Sublevel Stopping With Backfill	312,300	1,326,200	63,200	1,701,700
Sublevel Stopping With Full Subsidence	240,900	133,100	34,100	408,100
Block Caving	<u>146,200</u>	<u>66,400</u>	<u>20,900</u>	<u>233,500</u>
TOTALS	\$1,426,400	\$2,320,600	\$389,200	\$4,136,200

* Includes raise boring

** Includes backfilling with LHD units

TABLE 11.4

SUMMARY OF DIRECT MINING COSTS BY CATEGORY

<u>Demonstration Unit</u>	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Totals</u>
Primary Development	\$ 568,600	\$ 326,300	\$131,600	\$128,400	\$1,154,900
Chamber & Pillar	134,900	406,600	51,100	45,400	638,000
Sublevel Stopping With Backfill	379,800	1,042,300	148,000	131,600	1,701,700
Sublevel Stopping With Full Subsidence	189,100	117,600	52,400	49,000	408,100
Block Caving	<u>127,800</u>	<u>62,100</u>	<u>22,400</u>	<u>21,200</u>	<u>233,500</u>
TOTALS	\$1,400,200	\$1,954,900	\$405,500	\$375,600	\$4,136,200

11.3.1 Primary Development

Section 4.1 describes in detail the tasks considered to be primary development; generally, any excavation made to provide access to the various demonstration units has been categorized as primary development. Work classified as primary development will be performed in two separate stages. Stage I development will consist of excavations requiring completion prior to development and demonstration of the two backfilling units. Stage II development will provide access and ventilation openings to areas reserved for demonstration of the two units involving subsidence or caving. Summaries of direct mining costs for each of the individual tasks classified as primary development are shown in Sections 11.3.1.1 and 11.3.1.2. A summary of direct mining costs for the entire primary development effort is presented in Table 11.5.

11.3.1.1 Stage I Primary Development - Task Cost Summaries

Lower Level Shaft Station Development (Includes Sump)

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 57,300	\$ 27,100	\$ 6,100	\$ 6,100	\$ 96,600
Mucking & Cleanup	10,900	6,700	7,700	6,500	31,800
Scaling & Bolting	16,700	11,300	2,700	3,500	34,200
Total Costs	<u>\$ 84,900</u>	<u>\$ 45,100</u>	<u>\$ 16,500</u>	<u>\$ 16,100</u>	<u>\$162,600</u>

Upper Level Shaft Station Development

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 16,300	\$ 7,200	\$ 1,100	\$ 1,400	\$ 26,000
Mucking & Cleanup	2,700	1,300	1,500	1,300	6,800
Scaling & Bolting	5,400	2,700	800	900	9,800
Total Costs	<u>\$ 24,400</u>	<u>\$ 11,200</u>	<u>\$ 3,400</u>	<u>\$ 3,600</u>	<u>\$ 42,600</u>

Lower Level Main Entries (Includes Two Orepasses)

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 74,300	\$ 43,100	\$ 9,600	\$ 9,400	\$136,400
Mucking & Cleanup	16,000	10,500	12,100	10,200	48,800
Scaling & Bolting	25,200	14,900	4,200	5,200	49,500
Total Costs	<u>\$115,500</u>	<u>\$ 68,500</u>	<u>\$ 25,900</u>	<u>\$ 24,800</u>	<u>\$234,700</u>

TABLE 11.5

DIRECT MINING COST - PRIMARY DEVELOPMENT

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total</u>
<u>STAGE I</u>					
Drilling & Blasting	\$239,600	\$125,600	\$ 24,300	\$ 26,900	\$ 416,400
Mucking & Cleanup	58,000	38,600	44,300	37,400	178,300
Scaling & Bolting	<u>79,100</u>	<u>47,000</u>	<u>13,600</u>	<u>16,800</u>	<u>156,500</u>
Subtotal-Stage I	\$376,700	\$211,200	\$ 82,200	\$ 81,100	\$ 751,200
<u>STAGE II</u>					
Drilling & Blasting	\$117,100	\$ 67,700	\$ 14,400	\$ 14,900	\$ 214,100
Mucking & Cleanup	35,300	24,600	28,200	23,800	111,900
Scaling & Blasting	<u>39,500</u>	<u>22,800</u>	<u>6,800</u>	<u>8,600</u>	<u>77,700</u>
Subtotal-Stage II	\$191,900	\$115,100	\$ 49,400	\$ 47,300	\$ 403,700
TOTALS-STAGES I & II	\$568,600	\$326,300	\$131,600	\$128,400	\$1,154,900

11.3.1.1 Stage I Primary Development - Task Cost Summaries
(Continued)

Upper Level Main Entries

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 43,200	\$ 22,800	\$ 3,900	\$ 4,800	\$ 74,700
Mucking & Cleanup	14,700	10,400	11,900	10,100	47,100
Scaling & Bolting	15,000	8,600	2,700	3,300	29,600
Total Costs	<u>\$ 72,900</u>	<u>\$ 41,800</u>	<u>\$ 18,500</u>	<u>\$ 18,200</u>	<u>\$151,400</u>

Ramp, Chamber Drilling Level Entries

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 20,600	\$ 10,700	\$ 1,300	\$ 2,300	\$ 34,900
Mucking & Cleanup	5,100	3,600	4,100	3,400	16,200
Scaling & Bolting	7,100	4,000	1,500	1,700	14,300
Total Costs	<u>\$ 32,800</u>	<u>\$ 18,300</u>	<u>\$ 6,900</u>	<u>\$ 7,400</u>	<u>\$ 65,400</u>

Ramp, Sublevel Entries

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Costs</u>
Drilling & Blasting	\$ 27,900	\$ 14,700	\$ 2,300	\$ 2,900	\$ 47,800
Mucking & Cleanup	8,600	6,100	7,000	5,900	27,600
Scaling & Bolting	9,700	5,500	1,700	2,200	19,100
Total Costs	<u>\$ 46,200</u>	<u>\$ 26,300</u>	<u>\$ 11,000</u>	<u>\$ 11,000</u>	<u>\$ 94,500</u>

11.3.1.2 Stage II Primary Development - Task Cost Summaries

Lower Level Main Entries (Includes One Orepass)

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 29,000	\$ 19,200	\$ 4,300	\$ 4,300	\$ 56,800
Mucking & Cleanup	5,800	3,900	4,500	3,800	18,000
Scaling & Bolting	9,200	5,500	1,500	1,900	18,100
Total Costs	<u>\$ 44,000</u>	<u>\$ 28,600</u>	<u>\$ 10,300</u>	<u>\$ 10,000</u>	<u>\$ 92,900</u>

11.3.1.2 Stage II Primary Development - Task Cost Summaries
(Continued)

Sublevel Main Entries

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 25,400	\$ 13,400	\$ 2,300	\$ 2,900	\$ 44,000
Mucking & Cleanup	8,400	6,000	6,900	5,800	27,100
Scaling & Bolting	8,800	5,000	1,600	2,000	17,400
Total Costs	<u>\$ 42,600</u>	<u>\$ 24,400</u>	<u>\$ 10,800</u>	<u>\$ 10,700</u>	<u>\$ 88,500</u>

Upper Level Main Entries

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 26,500	\$ 14,200	\$ 2,400	\$ 3,000	\$ 46,100
Mucking & Cleanup	9,300	6,700	7,700	6,500	30,200
Scaling & Bolting	9,200	5,400	1,600	2,100	18,300
Total Costs	<u>\$ 45,000</u>	<u>\$ 26,300</u>	<u>\$ 11,700</u>	<u>\$ 11,600</u>	<u>\$ 94,600</u>

Monitor Level Access

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling	\$ 3,500	\$ 5,000	\$ 2,700	\$ 1,300	\$ 12,500
Mucking	100	100	100	100	400
Total Costs	<u>\$ 3,600</u>	<u>\$ 5,100</u>	<u>\$ 2,800</u>	<u>1,400</u>	<u>\$ 12,900</u>

Crown Pillar Drilling Drift Access

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 12,700	\$ 6,700	\$ 1,100	\$ 1,400	\$ 21,900
Mucking & Cleanup	3,900	2,900	3,300	2,800	12,900
Scaling & Blasting	4,800	2,900	900	1,100	9,700
Total Costs	<u>\$ 21,400</u>	<u>\$ 12,500</u>	<u>\$ 5,300</u>	<u>\$ 5,300</u>	<u>\$ 44,500</u>

11.3.1.2 Stage II Primary Development - Task Cost Summaries
(Continued)

Exhaust Entry Extension

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 20,000	\$ 9,200	\$ 1,600	\$ 2,000	\$ 32,800
Mucking & Cleanup	7,800	5,000	5,700	4,800	23,300
Scaling & Bolting	7,500	4,000	1,200	1,500	14,200
Total Costs	<u>\$ 35,300</u>	<u>\$ 18,200</u>	<u>\$ 8,500</u>	<u>\$ 8,300</u>	<u>\$ 70,300</u>

11.3.2 Chamber and Pillar Demonstration Unit

The direct mining costs for the chamber and pillar demonstration unit are presented in three subdivisions: the cost of developing the unit, the cost of mining the three chambers, and the backfilling costs. Table 11.6 summarizes the total direct costs for the demonstration unit.

11.3.2.1 Unit Development - Task Cost Summaries

Summaries of the costs of the individual tasks comprising the development of the chamber and pillar demonstration unit are presented in this section.

Chamber Drilling Drift Development

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 22,600	\$ 12,000	\$ 2,100	\$ 2,600	\$ 39,300
Mucking & Cleanup	6,500	4,800	5,500	4,600	21,400
Scaling & Bolting	8,400	5,200	1,500	1,900	17,000
Total Costs	<u>\$ 37,500</u>	<u>\$ 22,000</u>	<u>\$ 9,100</u>	<u>\$ 9,100</u>	<u>\$ 77,700</u>

LHD Entry Development (All Chambers)

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 3,300	\$ 1,500	\$ 300	\$ 300	\$ 5,400
Mucking & Cleanup	1,100	600	700	600	3,000
Scaling & Bolting	700	300	100	100	1,200
Total Costs	<u>\$ 5,100</u>	<u>\$ 2,400</u>	<u>\$ 1,100</u>	<u>\$ 1,000</u>	<u>\$ 9,600</u>

TABLE 11.6

DIRECT MINING COST - CHAMBER AND PILLAR DEMONSTRATION UNIT

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 53,600	\$ 31,300	\$ 5,400	\$ 6,200	\$ 96,500
Mucking & Cleanup	37,600	21,200	24,300	20,400	103,500
Scaling & Bolting	18,600	10,900	3,500	3,800	36,800
Backfilling	<u>25,100</u>	<u>343,200</u>	<u>17,900</u>	<u>15,000</u>	<u>401,200</u>
Total	\$134,900	\$406,600	\$ 51,100	\$ 45,400	\$638,000

11.3.2.1 Unit Development - Task Cost Summaries (Continued)

Slot Development (All Chambers)

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting*	\$ 5,500	\$ 3,500	\$ 1,500	\$ 900	\$ 11,400
Mucking & Cleanup	900	500	600	500	2,500
Scaling & Bolting	700	300	100	100	1,200
Total Costs	<u>\$ 7,100</u>	<u>\$ 4,300</u>	<u>\$ 2,200</u>	<u>\$ 1,500</u>	<u>\$ 15,100</u>

* Includes cost of boring slot raises

11.3.2.2 Chamber Mining - Task Cost Summaries

As described in Section 4.2.1, the two outer chambers will be mined and backfilled prior to excavation of the central chamber. This section summarizes the direct costs associated with the mining of all three chambers. Because the two outer chambers are mined concurrently, the number of working places is larger and the nonproductive time involved is not as great as it is in the single case. This condition results in an economy of scale so that the costs of mining two chambers is somewhat less than strictly double the cost of mining one chamber.

Mining Two Outside Chambers

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 13,600	\$ 9,500	\$ 1,000	\$ 1,600	\$ 25,700
Mucking & Cleanup	17,800	10,200	11,700	9,800	49,500
Scaling & Bolting	5,400	3,400	1,200	1,200	11,200
Total Costs	<u>\$ 36,800</u>	<u>\$ 23,100</u>	<u>\$ 13,900</u>	<u>\$ 12,600</u>	<u>\$ 86,400</u>

Mining Central Chamber

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 8,600	\$ 4,800	\$ 500	\$ 800	\$ 14,700
Mucking & Cleanup	11,300	5,100	5,800	4,900	27,100
Scaling & Bolting	3,400	1,700	600	500	6,200
Total Costs	<u>\$ 23,300</u>	<u>\$ 11,600</u>	<u>\$ 6,900</u>	<u>\$ 6,200</u>	<u>\$ 48,000</u>

11.3.2.3 Chamber Backfilling - Task Cost Summary

Chambers will be backfilled with raw shale from the mine supplemented by local alluvial material from the site. The material will be crushed to -8" size and transported to the mining zone via a 30" diameter borehole. Transportation of the material from the lower end of the borehole to the chambers will be accomplished by LHD units.

Provision of fill material will be assigned to a local subcontractor. He will be responsible for assembling the necessary crushing and screening equipment on site, collecting and processing fill material, and delivering fill material to the borehole. The cost of this service has been estimated to be \$3.50 per cubic yard or \$2.60 per ton.

The direct mining costs summarized for this task include the required LHD time as well as the cost of providing the backfill material itself. Since each chamber will accommodate the same amount of fill material, the costs for backfilling each chamber will be identical. Consequently, the following summary depicts the costs of backfilling all three chambers. The cost for one chamber may be obtained by taking one-third of the costs presented below.

Chamber Backfilling (All Chambers)

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Backfilling	\$ 25,100	\$343,200*	\$ 17,900	\$ 15,000	\$401,200

* Includes \$327,600 contractor charge for 126,000 tons of fill material

11.3.3 Sublevel Stopping With Backfilling Demonstration Unit

As with the chamber and pillar unit, the direct mining costs for the sublevel stopping with backfill unit are presented in three subdivisions, which are unit development, stope mining, and stope backfilling. These costs are summarized in Table 11.7.

11.3.3.1 Unit Development - Task Cost Summaries

This unit will require development activities on all three mining levels. Summaries of the individual tasks comprising this development effort are as follows.

TABLE 11.7

DIRECT MINING COSTS - SUBLEVEL STOPING WITH BACKFILL UNIT

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$175,500	\$ 98,700	\$ 18,200	\$ 19,900	\$ 312,300
Mucking & Cleanup	107,600	69,000	79,200	66,700	322,500
Scaling & Bolting	33,600	16,500	5,800	7,300	63,200
Backfilling	<u>63,100</u>	<u>858,100</u>	<u>44,800</u>	<u>37,700</u>	<u>1,003,700</u>
Total Costs	\$379,800	\$1,042,300	\$148,000	\$131,600	\$1,701,700

Lower Level Development

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 65,200	\$ 29,200	\$ 5,100	\$ 6,200	\$105,700
Mucking & Cleanup	15,800	9,700	11,100	9,400	46,000
Scaling & Bolting	20,800	11,600	3,400	4,300	40,100
Total Costs	<u>\$101,800</u>	<u>\$ 50,500</u>	<u>\$ 19,600</u>	<u>\$ 19,900</u>	<u>\$191,800</u>

Sublevel Development

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 30,500	\$ 14,500	\$ 2,500	\$ 3,100	\$ 50,600
Mucking & Cleanup	8,300	5,400	6,200	5,300	25,200
Scaling & Bolting	5,000	1,300	1,300	1,700	9,300
Total Costs	<u>\$ 43,800</u>	<u>\$ 21,200</u>	<u>\$ 10,000</u>	<u>\$ 10,100</u>	<u>\$ 85,100</u>

Upper Level Development

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 20,800	\$ 8,400	\$ 1,500	\$ 1,800	\$ 32,500
Mucking & Cleanup	5,600	3,100	3,600	3,000	15,300
Scaling & Bolting	7,800	3,600	1,100	1,300	13,800
Total Costs	<u>\$ 34,200</u>	<u>\$ 15,100</u>	<u>\$ 6,200</u>	<u>\$ 6,100</u>	<u>\$ 61,600</u>

Slot Development (All Stopes)

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting*	\$ 12,600	\$ 13,400	\$ 5,700	\$ 3,300	\$ 35,000
Mucking	3,300	2,200	2,600	2,100	10,200
Total Costs	<u>\$ 15,900</u>	<u>\$ 15,600</u>	<u>\$ 8,300</u>	<u>\$ 5,400</u>	<u>\$ 45,200</u>

* Includes cost of boring slot raises

11.3.3.2 Stope Mining - Task Cost Summaries

The three stopes in this unit will be mined in the same order in which the three chambers will be excavated; that is, the two outer stopes will be mined and backfilled prior to the excavation of the central stope. As was also the case with chamber mining, the costs of mining two stopes concurrently is less than double the cost of mining a single stope.

Mining Two Outside Stopes

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 29,900	\$ 22,100	\$ 2,300	\$ 3,700	\$ 58,000
Mucking	48,100	32,400	37,100	31,300	148,900
Total Costs	<u>\$ 78,000</u>	<u>\$ 54,500</u>	<u>\$ 39,400</u>	<u>\$ 35,000</u>	<u>\$206,900</u>

Mining Central Stope

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 16,500	\$ 11,100	\$ 1,100	\$ 1,800	\$ 30,500
Mucking	26,500	16,200	18,600	15,600	76,900
Total Costs	<u>\$ 43,000</u>	<u>\$ 27,300</u>	<u>\$ 19,700</u>	<u>\$ 17,400</u>	<u>\$107,400</u>

11.3.3.3 Stope Backfilling - Task Cost Summary

As explained in Section 11.3.2.3, the backfilling costs summarized herein include the cost of the fill material itself as well as the LHD time required to distribute the material to each stope. Since each stope will be filled with the same volume of material, and since the average haulage distance has been assumed to be equal in all cases, the cost for backfilling each stope will also be equal. The costs summarized below depict the cost of backfilling all three stopes. To obtain the cost for a single stope, these figures may be divided by three.

Sublevel Stope Backfilling

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Backfilling	\$ 63,100	\$858,100*	\$ 44,800	\$ 37,700	\$1,003,700

* Includes \$819,000 contractor charge for 315,000 tons of backfill material

11.3.4 Sublevel Stopping With Full Subsidence Demonstration Unit

The presentation of the direct mining costs for this demonstration unit is broken down into the development of the unit and the mining of the stopes and pillars. Table 11.8 summarizes the costs for the unit. It should be noted that the work described herein will not completely demonstrate the system feasibility. To fully simulate a sublevel stopping with full subsidence system, a caved area above the stopes must be created. In the demonstration mine design, the block caving unit has been placed so as to overlay the stopes, thereby providing the caved area required. The cost of establishing the cave, however, is included in the block caving analysis and is not reflected in the costs presented in Table 11.8. It should be understood that the direct costs for demonstrating this system will be accurate only if the block caving system is established as well.

11.3.4.1 Unit Development - Task Cost Summaries

This demonstration unit will require development activities on all three mining levels. Summaries of the direct cost for individual development tasks are listed below.

Lower Level Development

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 39,300	\$ 20,400	\$ 3,500	\$ 4,400	\$ 67,600
Mucking & Cleanup	9,700	6,800	7,800	6,600	30,900
Scaling & Bolting	12,700	7,100	2,400	3,000	25,200
Total Costs	<u>\$ 61,700</u>	<u>\$ 34,300</u>	<u>\$ 13,700</u>	<u>\$ 14,000</u>	<u>\$123,000</u>

Sublevel Development

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 23,100	\$ 8,400	\$ 1,500	\$ 1,800	\$ 34,800
Mucking & Cleanup	6,400	3,300	3,800	3,200	16,700
Scaling & Bolting	4,000	900	800	1,000	6,700
Total Costs	<u>\$ 33,500</u>	<u>\$ 12,600</u>	<u>\$ 6,100</u>	<u>\$ 6,000</u>	<u>\$ 58,200</u>

TABLE 11.8

DIRECT MINING COSTS - SUBLEVEL STOPING WITH FULL
SUBSIDENCE DEMONSTRATION UNIT

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$128,100	\$ 80,400	\$ 15,800	\$ 16,600	\$240,900
Mucking & Cleanup	42,900	28,900	33,200	28,100	133,100
Scaling & Bolting	<u>18,100</u>	<u>8,300</u>	<u>3,400</u>	<u>4,300</u>	<u>34,100</u>
Total Costs	\$189,100	\$117,600	\$ 52,400	\$ 49,000	\$408,100

11.3.4.1 Unit Development - Task Cost Summaries (Continued)

Crown Pillar Drilling Drift Development

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting*	\$ 10,100	\$ 4,600	\$ 3,700	\$ 2,200	\$ 20,600
Mucking & Cleanup	1,900	1,000	1,100	1,000	5,000
Scaling & Bolting	1,400	300	200	300	2,200
Total Costs	<u>\$ 13,400</u>	<u>\$ 5,900</u>	<u>\$ 5,000</u>	<u>\$ 3,500</u>	<u>\$ 27,800</u>

* Includes costs of boring ventilation raises

Slot Development (All Stopes)

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting*	\$ 7,300	\$ 7,600	\$ 3,200	\$ 1,800	\$ 19,900
Mucking & Cleanup	2,400	1,400	1,600	1,400	6,800
Total Costs	<u>\$ 9,700</u>	<u>\$ 9,000</u>	<u>\$ 4,800</u>	<u>\$ 3,200</u>	<u>\$ 26,700</u>

* Includes cost of boring slot raises

11.3.4.2 Stope and Pillar Mining - Task Cost Summaries

The order of production activities in this demonstration unit will be: (1) the concurrent mining of both stopes; (2) the blasting of one crown pillar; and (3) the simultaneous recovery of the second crown pillar and the intervening rib pillar between the two stopes. In an effort to control or at least minimize subsidence as the pillars are mined, not all rumbled material produced will be withdrawn from the stopes. Consequently, the mucking costs presented below are not sufficient for 100% extraction of the broken shale produced.

Stope Mining (Both Stopes)

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 25,100	\$ 20,300	\$ 2,000	\$ 3,200	\$ 50,600
Mucking	22,500	16,400	18,900	15,900	73,700
Total Costs	<u>\$ 47,600</u>	<u>\$ 36,700</u>	<u>\$ 20,900</u>	<u>\$ 19,100</u>	<u>\$124,300</u>

11.3.4.2 Stope and Pillar Mining - Task Cost Summaries
(Continued)

Pillar Mining (Crown and Rib Pillars)

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 23,200	\$ 19,100	\$ 1,900	\$ 3,200	\$ 47,400

11.3.5 Block Caving Demonstration Unit

The final system to be analyzed is the block caving demonstration unit. As explained earlier, this unit - in addition to demonstration block caving techniques - serves as the overcut level for the sublevel stoping with full subsidence demonstration unit. As no ore recovery is planned, all direct mining costs associated with the block caving system will be incurred during the development of the unit. Table 11.9 summarizes the direct mining costs anticipated for this portion of the project.

11.3.5.1 Unit Development - Task Cost Summaries

Development of this unit accounts for all the direct mining costs associated with the block caving system. The tasks comprising unit development are the development of the cave monitoring levels, development of the undercut area (essentially room and pillar mining followed by pillar blasting), and the drilling and blasting of the vertical perimeter cut-off holes. The perimeter cut-off holes and the interior pillars in the undercut areas will be blasted concurrently.

Monitor Drift Development (Both Levels)

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 50,100	\$ 22,200	\$ 3,100	\$ 2,300	\$ 77,700
Mucking & Cleanup	15,500	4,900	5,100	4,900	30,400
Scaling & Bolting	5,800	4,300	300	200	10,600
Total Costs	\$ 71,400	\$ 31,400	\$ 8,500	\$ 7,400	\$118,700

Undercut Development

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 35,400	\$ 18,800	\$ 3,100	\$ 3,800	\$ 61,100
Mucking & Cleanup	11,600	7,800	9,000	7,600	36,000
Scaling & Bolting	5,500	1,400	1,500	1,900	10,300
Total Costs	\$ 52,500	\$ 28,000	\$ 13,600	\$ 13,300	\$107,400

TABLE 11.9

DIRECT MINING COSTS - BLOCK CAVING DEMONSTRATION UNIT

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 89,400	\$ 43,700	\$ 6,500	\$ 6,600	\$146,200
Mucking & Cleanup	27,100	12,700	14,100	12,500	66,400
Scaling & Bolting	<u>11,300</u>	<u>5,700</u>	<u>1,800</u>	<u>2,100</u>	<u>20,900</u>
Total Costs	\$127,800	\$ 62,100	\$ 22,400	\$ 21,200	\$233,500

11.3.5.1 Unit Development - Task Cost Summaries (Continued)

Perimeter Cut-off Holes & Pillar Blasting

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
Drilling & Blasting	\$ 3,900	\$ 2,700	\$ 300	\$ 500	\$ 7,400

11.4 COST OF SUPPORT OPERATIONS

In addition to the direct mining tasks, whose costs were detailed in the preceding section, a large number of activities are required to support the mining operations in an orderly and efficient manner. These support activities have been categorized elsewhere in this report as indirect mining and surface operations. Those operations labeled indirect mining include work performed in and around the mine which is not directly chargeable to some particular development or production activity; examples of such work would be mine ventilation, mine dewatering, shale hoisting, etc. Support activities occurring on the surface would be such functions as shale stockpiling, maintenance of roads, buildings and grounds, supply handling, etc.

The costs of some of these activities (notably shale handling and stockpiling) can be calculated on a tonnage or manpower basis but, for the most part, are chiefly time dependent; that is, the costs are based on a cost-per-day or cost-per-shift approach and, as such, are not greatly affected by changes in mine production. A major determinant in these costs is whether the mine will be operating for one- or two-shifts-per-day. The following subsections are devoted to discussions of the basic parameters used in calculating the anticipated costs of the major support operations.

11.4.1 Mine Ventilation

The mine ventilation system is described in detail in Section 7.0. In general, fresh air will be brought down the production shaft, circulated through the underground workings, directed to the vent shaft, and then exhausted by the main fan located at the vent shaft collar. To control the airflow underground, a number of auxiliary fans and ventilation structures will be required throughout the mine. Airflow requirements are directly proportional to underground activity levels and are presented by time period in Table 6.10.

The ventilation equipment requirements, such as fans, tubing, and flow-control structures, are based on the project schedule. They have been computed on a yearly basis for both operational modes. Ventilation expenses for any given year, as well as for the project as a whole, have been broken down into the four cost centers used previously, namely operating labor, operating supplies, maintenance labor, and maintenance supplies.

Operating labor, as related to ventilation costs, is limited to the construction and installation of the various stoppings, overcasts, and brattices required to effectively control airflow. It has been estimated that an average of six manshifts will be required for the erection of each flow-control structure.

The category "operating supplies" includes the cost of structures and vent tubing as well as the cost of electrical power required to operate all ventilation fans. Fan requirements will consist of a 600 horsepower main fan and three sizes of auxiliary fans. The hourly cost of their operations is presented below.

Main fan:	600 HP	x	.7457 KW/HP	x	\$.06/KWH*	=	\$26.85/hour
Auxiliary fans:	75 HP	x	.7457 KW/HP	x	\$.06/KWH	=	\$ 3.36/hour
	40 HP	x	.7457 KW/HP	x	\$.06/KWH	=	\$ 1.79/hour
	10 HP	x	.7457 KW/HP	x	\$.06/KWH	=	\$ 0.45/hour

* Estimated unit power cost by WREA

It has been assumed that all fans in use at any given time will operate continuously to insure a safe working environment. It has been further assumed that the surface exhaust fan will operate at only two-thirds capacity during start-up (the first two years) and at 100% capacity thereafter. Table 11.10 lists the maximum number of types of auxiliary fans required during specific periods of activity.

Maintenance labor chargeable to ventilation includes regular inspection of all fans, structures and tubing, and replacement or repair of those items damaged or worn out during normal service. One-half manshift per week is contemplated for this activity. Maintenance supplies are restricted to the replacement parts for the ventilation system and have been estimated at 2% per year of the capital cost of the system.

Table 11.11 summarizes the expected yearly operating and maintenance costs for mine ventilation for both the one-shift-per-day and two-shifts-per-day modes of operation. The initial purchase of auxiliary fans is considered to be a capital expense item and, as such, is presented in Section 10.0. It has been assumed that the main exhaust fan will be purchased and installed by others.

11.4.2 Shale Handling

When an LHD unit mucks a heading, it will deliver the broken shale to a semiportable feeder-breaker located in the central entry on the lower level. This unit will crush the shale to an acceptable size for conveyor transport and then feed it onto the mine conveyor belt. The shale will be conveyed to the skip loading facilities at the shaft where it will be hoisted to the surface and discharged into a 35-ton truck for transport to the stockpiling area.

TABLE 11.10

AUXILIARY FAN SCHEDULE

(Figures presented are yearly maximums)

	<u>75</u> <u>Horsepower</u>	<u>40</u> <u>Horsepower</u>	<u>10</u> <u>Horsepower</u>
<u>One-Shift-Per-Day Mode</u>			
Year 1	-	2	8
Year 2	-	4	12
Year 3	1	6	16
Year 4	2	6	16
Year 5	2	6	16
Year 6	2	6	16
Year 7	2	6	16
<u>Two-Shifts-Per-Day Mode</u>			
Year 1	-	4	12
Year 2	2	6	16
Year 3	2	6	16
Year 4	2	6	16

TABLE 11.11

MINE VENTILATION - SUMMARY OF COSTS

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Total Cost</u>
<u>One-Shift-Per-Day-Mode</u>					
Year 1	\$ 8,000	\$ 230,800	\$ 2,000	\$ 3,200	\$ 244,000
Year 2	11,400	268,900	2,000	3,400	285,700
Year 3	12,700	436,200	2,000	4,200	455,100
Year 4	16,500	475,400	2,000	4,300	498,200
Year 5	12,700	474,700	2,000	4,300	493,700
Year 6	18,200	481,700	2,000	4,300	506,200
Year 7	<u>3,000</u>	<u>454,200</u>	<u>2,000</u>	<u>4,200</u>	<u>463,400</u>
Total	\$ 82,500	\$2,821,900	\$ 14,000	\$ 27,900	\$2,946,300
 <u>Two-Shifts-Per-Day-Mode</u>					
Year 1	\$ 19,400	\$ 264,500	\$ 2,000	\$ 3,600	\$ 289,500
Year 2	29,200	483,000	2,000	4,600	518,800
Year 3	30,900	505,700	2,000	4,500	543,100
Year 4	<u>3,000</u>	<u>288,900</u>	<u>1,000</u>	<u>2,000</u>	<u>294,900</u>
Total	\$ 82,500	\$1,542,100	\$ 7,000	\$ 14,700	\$1,646,300

The cost of the LHD time has already been reported as a direct mining cost. This subsection describes the cost of the remainder of the shale handling system. The costs for this system are directly proportional to the tonnage handled for any given year. As before, the costs are presented distributed to the four basic cost centers used throughout this section.

The category "operating labor" in this case covers personnel who actually handle the shale as well as those who regularly service the crushing and conveying system. Included are underground utility men who lubricate the conveyor system, the hoistman, skiptender, surface truck drivers, and an equipment operator at the raw shale stockpile. It has been assumed that 75% of the cost of the hoistman and skiptender will be chargeable to ore hoisting and the remainder of their time will be allotted to hoisting men and supplies. A total of two manshifts per week (one-shift-per-day mode) has been assigned to the function of conveyor and crusher inspection and cleanup. The truck driver and equipment operator's hours are directly calculable on a tonnage basis.

Operating supply costs cover normal replacement of consumable parts for the crusher and conveyor; fuel, tire, and lube costs for surface equipment used in stockpiling activities, and electrical power costs for operating the crusher, conveyor, and hoist. Usually these costs are directly related to the tonnage handled.

Maintenance labor costs include the time required to inspect and service the shaft and hoist on a weekly schedule, normal maintenance of surface equipment, and repair of the crusher and conveyor system. The cost of maintenance supplies covers the anticipated expense of replacement parts over and above consumable parts. The tonnage handled in any given period is the chief determinant of maintenance costs in this case.

Table 11.12 summarizes the projected yearly operating and maintenance costs of the shale handling system for both modes of operation. The original purchase expense of all items in the system (excluding shaft equipment) is accounted for in the capital cost detail of Section 10.0. It has been assumed that the shaft facilities and hoist will be furnished and installed by others.

11.4.3 Waste Water Handling

As described in Section 8.2, mine waste water will be collected in the mine sump, pumped to surface and ultimately reinjected into the ground-water system. The cost of this operation is chiefly time dependent and is based on two distinct cases. The first case assumes a ground-water inflow rate of 100 gpm during the Stage I activities. During the period when the two subsidence systems are investigated, an inflow rate of 1,600 gpm has been assumed.

TABLE 11.12

SHALE HANDLING - SUMMARY OF COSTS

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Totals</u>
<u>One-Shift-Per-Day Mode</u>					
Crushing & Conveying:					
Year 1	\$ 3,700	\$ 32,500	\$ 9,200	\$ 4,700	\$ 50,100
Year 2	3,700	33,500	12,300	6,200	55,700
Year 3	3,700	35,500	18,600	9,500	67,300
Year 4	3,700	35,500	18,600	9,500	67,300
Year 5	3,700	40,400	34,100	17,400	95,600
Year 6	3,700	35,900	19,900	10,200	69,700
Year 7	3,700	35,400	18,300	9,300	66,700
Subtotal	\$ 25,900	\$248,700	\$131,000	\$ 66,800	\$ 472,400
Shale Hoisting:					
Year 1	\$ 28,300	\$ 33,000	\$ 3,400	\$ 1,400	\$ 66,100
Year 2	28,300	40,000	3,400	1,900	73,600
Year 3	28,300	54,900	3,400	2,900	89,500
Year 4	28,300	54,900	3,400	2,900	89,500
Year 5	28,300	90,700	3,400	5,200	127,600
Year 6	28,300	57,900	3,400	3,400	92,700
Year 7	28,300	54,000	3,400	2,800	88,500
Subtotal	\$198,100	\$385,400	\$ 23,800	\$ 20,200	\$ 627,500
Surface Stockpiling:					
Year 1	\$ 12,800	\$ 8,600	\$ 9,700	\$ 7,400	\$ 38,500
Year 2	17,300	11,400	12,900	9,800	51,400
Year 3	22,900	17,300	19,600	15,000	74,800
Year 4	22,900	17,300	19,600	15,000	74,800
Year 5	31,700	30,000	34,000	25,000	120,700
Year 6	22,900	18,300	20,800	15,700	77,700
Year 7	22,900	17,000	19,300	14,800	74,000
Subtotal	\$153,400	\$119,900	\$135,900	\$102,700	\$ 511,900
TOTAL SHALE HANDLING	\$377,400	\$754,000	\$290,700	\$189,700	\$1,611,800

TABLE 11.12
(Continued)

SHALE HANDLING - SUMMARY OF COSTS

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Totals</u>
<u>Two-Shifts-Per-Day Mode</u>					
Crushing & Conveying:					
Year 1	\$ 7,300	\$ 66,000	\$ 21,500	\$ 10,900	\$ 105,700
Year 2	7,300	71,000	37,200	19,000	134,500
Year 3	7,300	76,400	54,000	27,500	165,200
Year 4	3,700	35,400	18,300	9,300	66,700
Subtotal	<u>\$ 25,600</u>	<u>\$248,800</u>	<u>\$131,000</u>	<u>\$ 66,700</u>	<u>\$ 472,100</u>
Shale Hoisting:					
Year 1	\$ 56,600	\$ 72,600	\$ 5,000	\$ 3,300	\$ 137,500
Year 2	56,600	109,300	5,000	5,700	176,600
Year 3	56,600	148,200	5,000	8,300	218,100
Year 4	28,300	53,900	2,500	2,800	87,500
Subtotal	<u>\$198,100</u>	<u>\$384,000</u>	<u>\$ 17,500</u>	<u>\$ 20,100</u>	<u>\$ 619,700</u>
Surface Stockpiling:					
Year 1	\$ 30,200	\$ 19,900	\$ 22,600	\$ 17,200	\$ 89,900
Year 2	45,800	34,600	39,200	29,900	149,500
Year 3	54,600	48,300	54,800	40,700	198,400
Year 4	22,900	17,000	19,300	14,800	74,000
Subtotal	<u>\$153,500</u>	<u>\$119,800</u>	<u>\$135,900</u>	<u>\$102,600</u>	<u>\$ 511,800</u>
TOTAL SHALE HANDLING	\$377,200	\$752,600	\$284,400	\$189,400	\$1,603,600

The pumping system will be largely automated. Therefore, the associated cost of operating labor will be minimal and will be restricted to periodic lubrication, and filter and filter media replacement. In addition to the filter media, filters, and lubricant, the major portion of the cost of operating supplies will be made up of the electrical power necessary to operate the feeder, primary and booster pumps, and the reinjection system. Maintenance costs for the pumping system are expected to be relatively low.

Table 11.13 summarizes the yearly operating and maintenance costs anticipated under both modes of operation. The initial purchase and installation of the mine pumping system is considered a capital expense and is not reflected here. The reinjection system is assumed to be established on the site at project start-up.

11.4.4 Supply Handling:

Supplying the mine with the large volume of materials required to insure orderly and efficient operations will be an on-going task throughout the life of the mine. In general, supplies will be trucked to the site, stored in a surface warehouse (or magazine in the case of explosives) and transported underground in the service cage.

Included in the costs summarized in this section are the operating cost, comprised of 25% of the cost of a hoistman, skiptender, and the partial cost of truck drivers and utility men; the operating supply cost, made up of truck and equipment fuel and related costs as well as the electrical power required to operate the service hoist; and maintenance labor and supply costs associated with supply truck and hoist operation. Supply handling costs for both the one-shift-per-day and two-shifts-per-day operating mode are summarized in Table 11.14.

11.4.5 Roads, Buildings, and Grounds

As stated in the contract, necessary surface facilities such as an office, changehouse, hoist house, and warehouse are assumed to be already on site and available for use at the time of project start-up. The upkeep of these facilities, however, will be a legitimate project expense and has been analyzed as such.

Included in these costs are the costs of operating a motor grader for road maintenance and snow removal, a water truck to be used in roadway wetting for dust suppression, and a loader for various tasks around the grounds. This category also covers the cost of operating the changehouse and supplying heat, light, and janitorial service to all surface buildings. These costs are entirely time-dependent and are constant from year to year (Table 11.15).

TABLE 11.13

WASTE WATER HANDLING - SUMMARY OF COSTS

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Totals</u>
<u>One-Shift-Per-Day Mode</u>					
Year 1	\$ 1,800	\$ 133,600	\$ 4,100	\$ 5,100	\$ 144,600
Year 2	1,800	133,600	4,100	5,100	144,600
Year 3	1,800	133,600	4,100	5,100	144,600
Year 4	1,800	133,600	4,100	5,100	144,600
Year 5	1,800	133,600	4,100	5,100	144,600
Year 6	1,800	434,900	4,100	9,400	450,200
Year 7	1,800	736,200	4,100	9,400	751,500
Total	<u>\$ 12,600</u>	<u>\$1,839,100</u>	<u>\$ 28,700</u>	<u>\$ 44,300</u>	<u>\$1,924,700</u>
<u>Two Shifts-Per-Day Mode</u>					
Year 1	\$ 1,800	\$ 133,600	\$ 4,100	\$ 5,100	\$ 144,600
Year 2	1,800	133,600	4,100	5,100	144,600
Year 3	1,800	284,200	4,100	9,400	299,500
Year 4	900	368,100	2,000	4,700	375,700
Total	<u>\$ 6,300</u>	<u>\$ 919,500</u>	<u>\$ 14,300</u>	<u>\$ 24,300</u>	<u>\$ 964,400</u>

TABLE 11.14

SUPPLY HANDLING - SUMMARY OF COSTS

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Totals</u>
<u>One-Shift-Per-Day Mode</u>					
Year 1	\$ 36,500	\$ 18,300	\$ 5,300	\$ 4,000	\$ 64,100
Year 2	40,700	19,000	5,300	4,100	69,100
Year 3	35,200	20,500	5,300	4,500	65,500
Year 4	35,200	20,500	5,300	4,500	65,500
Year 5	35,200	24,700	5,300	5,200	70,400
Year 6	35,200	20,800	5,300	4,500	65,800
Year 7	35,200	20,500	5,300	4,400	65,400
Total	\$253,200	\$144,300	\$ 37,100	\$ 31,200	\$465,800
<u>Two-Shifts-Per-Day Mode</u>					
Year 1	\$ 55,300	\$ 35,000	\$ 9,600	\$ 7,800	\$107,700
Year 2	48,500	38,700	9,600	8,600	105,400
Year 3	48,500	42,700	9,600	9,500	110,300
Year 4	20,300	30,300	5,000	4,400	60,000
Total	\$172,600	\$146,700	\$ 33,800	\$ 30,300	\$383,400

TABLE 11.15

ROADS, BUILDINGS, AND GROUNDS - SUMMARY OF COSTS

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Totals</u>
<u>One-Shift-Per-Day Mode</u>					
Yearly Total (Years 1-7)	\$ 15,500	\$ 20,600	\$ 4,700	\$ 4,000	\$ 44,800
Project Total	\$108,500	\$144,200	\$ 32,900	\$ 28,000	\$313,600
<u>Two-Shifts-Per-Day Mode</u>					
Yearly Total (Years 1-3)	\$ 23,100	\$ 41,300	\$ 7,400	\$ 7,900	\$ 79,700
Yearly Total (Year 4)	15,500	20,600	3,700	4,000	43,800
Project Total	\$ 84,800	\$144,500	\$ 25,900	\$ 27,700	\$282,900

11.4.6 Salaried Staff

In addition to the hourly personnel, a salaried supervisory staff will be required. These costs are strictly time-dependent and will be constant from year to year. Staffing levels will vary depending on the operational mode. Table 11.16 lists staff personnel and their annual salaries. Table 11.17 summarizes their cost to the project, including associated office supplies and personal transportation for the mine superintendent.

11.4.7 Other Mine Expense

In addition to the distinct cost centers already discussed in this section, there are a number of support activities that, when taken separately, represent only a small portion of the total project cost, but if combined, add significantly to operating expenses. Comprising this category are such diverse items as underground utility distribution, compressor maintenance, potable water supply, safety training, and a general labor pool of utility men. Table 11.18 presents a summary of these costs by year for both operating modes.

11.5 PROJECT OPERATING AND MAINTENANCE COST SUMMARY

Direct mining costs plus the cost of necessary support operations yields the total project expense over and above capital costs. Table 11.19 presents a summary of operating and maintenance costs on a yearly basis for the one-shift-per-day mode. The same information for the two-shifts-per-day mode is listed in Table 11.20. All costs are expressed in 1978 dollars.

11.6 QUANTITIES AND COSTS OF MAJOR SUPPLY ITEMS

The operating and maintenance costs discussed in this chapter are based, in part, on consumable supply items required to perform various tasks associated with the project. In approximating the overall cost of the project, estimates were made as to the usable life or consumption rates of most major items and the total project cost for each of them was then computed. Table 11.21 lists quantities and unit costs of major supply items. All costs are expressed in 1978 dollars.

Since electrical power requirements are directly affected by the operating mode, hourly consumption rates are presented instead of project totals. Hourly figures for the pumping and ventilation systems are the maximums required at full capacity.

11.6.1 Anticipated Escalation Rates

As stated in earlier sections of this report, all cost estimates are expressed in 1978 dollars. Pursuant to the terms of the contract, anticipated annual inflation rates for these costs have been estimated through calendar year 1980. At the time of publication, stated Federal goals call for

wage and price control ceilings averaging a seven percent increase per year. This ceiling coupled with our own long-term forecasting practices, leads to the prediction of an eight percent increase in capital costs and a seven percent increase in the costs of wages and supplies for 1979. For 1980, a seven percent increase in capital expense and a six percent increase in operating and maintenance costs are foreseen.

TABLE 11.16
SALARY SCHEDULE

<u>Title</u>	<u>Number Required Per Operating Mode</u>		<u>Annual Salary</u>	<u>Yearly Cost*</u>
	<u>One-Shift/Day</u>	<u>Two Shifts/Day</u>		
Mine Superintendent	1	1	\$33,400	\$46,760
Mine Captain	1	1	27,300	38,220
Mine Foreman	1	2	21,300	29,820
Engineer	2	2	24,300	34,020
Maintenance Foreman	1	2	21,300	29,820
Master Mechanic	1	1	27,300	38,220
Clerk	1	2	15,200	21,280
Warehouseman	1	1	15,000	21,000
Watchman	<u>4</u>	<u>4</u>	12,000	16,800
Total	13	16		

* Includes 40% fringe benefits

TABLE 11.17

SALARIED PERSONNEL - SUMMARY OF COSTS

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Totals</u>
<u>One-Shift-Per-Day Mode</u>					
Yearly Cost (Years 1-7)	\$ 247,000	\$ 18,800	\$112,600	\$ 7,900	\$ 386,300
Project Cost	\$1,729,000	\$131,600	\$788,200	\$ 55,300	\$2,704,100
<u>Two-Shifts-Per-Day Mode</u>					
Years 1-3	\$ 288,300	\$ 23,600	\$153,000	\$ 10,600	\$ 475,500
Year 4	144,100	11,800	76,500	5,300	237,700
Project Cost	\$1,009,000	\$ 82,600	\$535,500	\$ 37,100	\$1,664,200

TABLE 11.18

SUMMARY OF OTHER MINE OPERATING COSTS

	<u>Operating Labor</u>	<u>Operating Supplies</u>	<u>Maintenance Labor</u>	<u>Maintenance Supplies</u>	<u>Totals</u>
<u>One-Shift-Per-Day Mode</u>					
Year 1	\$ 18,300	\$ 86,600	\$ 8,800	\$ 6,400	\$120,100
Year 2	26,900	90,500	8,800	6,800	133,000
Year 3	31,500	102,200	8,800	7,300	149,800
Year 4	27,700	100,800	8,800	7,000	144,300
Year 5	22,400	104,100	8,800	7,600	142,900
Year 6	23,800	105,400	8,800	6,100	144,100
Year 7	24,100	113,000	8,800	5,500	151,400
Totals	<u>\$174,700</u>	<u>\$702,600</u>	<u>\$ 61,600</u>	<u>\$ 46,700</u>	<u>\$985,600</u>
<u>Two-Shifts-Per-Day Mode</u>					
Year 1	\$ 56,800	\$172,400	\$ 16,100	\$ 12,300	\$257,600
Year 2	70,600	179,200	16,100	13,400	279,300
Year 3	57,500	176,400	16,100	12,800	262,800
Year 4	29,600	90,000	8,000	4,900	132,500
Totals	<u>\$214,500</u>	<u>\$618,000</u>	<u>\$ 56,300</u>	<u>\$ 43,400</u>	<u>\$932,200</u>

TABLE 11.19

OPERATING AND MAINTENANCE COST SUMMARY - ONE SHIFT-PER-DAY MODE

<u>Cost Source</u>	<u>Year 1</u>	<u>Year 2</u>	<u>Year 3</u>	<u>Year 4</u>	<u>Year 5</u>	<u>Year 6</u>	<u>Year 7</u>	<u>Totals</u>
Direct Mining	\$ 255,400	\$ 367,800	\$ 338,100	\$ 905,100	\$ 959,500	\$ 853,700	\$ 406,600	\$ 4,136,200
Mine Ventilation	244,000	285,700	455,100	498,200	493,700	506,200	463,400	2,946,300
Shale Handling	154,700	180,700	231,600	231,600	343,900	240,100	229,200	1,611,800
Waste Water Handling	144,600	144,600	144,600	144,600	144,600	450,200	751,500	1,924,700
Supply Handling	64,100	69,100	65,500	65,500	70,400	65,800	65,400	465,800
Roads, Buildings, Grounds	44,800	44,800	44,800	44,800	44,800	44,800	44,800	313,600
Salaries Personnel	386,300	386,300	386,300	386,300	386,300	386,300	386,300	2,704,100
Other Mine Expense	<u>120,100</u>	<u>133,000</u>	<u>149,800</u>	<u>144,300</u>	<u>142,900</u>	<u>144,100</u>	<u>151,400</u>	<u>985,600</u>
PROJECT TOTALS	\$1,414,000	\$1,612,000	\$1,865,800	\$2,420,400	\$2,586,100	\$2,691,200	\$2,498,600	\$15,088,100

TABLE 11.20

OPERATING AND MAINTENANCE COST SUMMARY - TWO-SHIFTS-PER-DAY MODE

<u>Cost Source</u>	<u>Year 1</u>	<u>Year 2</u>	<u>Year 3</u>	<u>Year 4</u>	<u>Totals</u>
Direct Mining	\$ 623,200	\$1,293,200	\$1,813,200	\$ 406,600	\$ 4,136,200
Mine Ventilation	289,500	518,800	543,100	294,900	1,646,300
Shale Handling	333,100	460,600	581,700	228,200	1,603,600
Waste Water Handling	144,600	144,600	299,500	375,700	964,400
Supply Handling	107,700	105,400	110,300	60,000	383,400
Roads, Buildings, Grounds	79,700	79,700	79,700	43,800	282,900
Salariated Personnel	475,500	475,500	475,500	237,700	1,664,200
Other Mine Expense	<u>257,600</u>	<u>279,300</u>	<u>262,800</u>	<u>132,500</u>	<u>932,200</u>
PROJECT TOTALS	\$2,310,900	\$3,357,100	\$4,165,800	\$1,779,400	\$11,613,200

TABLE 11.21

SUPPLY REQUIREMENTS

<u>Item</u>	<u>Unit Cost</u>	<u>Quantity Required</u>
<u>I. DRILLING SUPPLIES</u>		
Drill steel (1" x 12')	\$ 80/pc	325 pcs
Drill steel (1-1/4" x 12')	\$100/pc	230 pcs
Drill steel (1-3/4" x 6')	\$ 68/pc	295 pcs
Drill steel (1" x 7')	\$ 75/pc	75 pcs
Striker bar (1-1/4")	\$ 47/pc	660 pcs
Striker bar (1-3/4")	\$ 50/pc	120 pcs
Coupling (1-1/4")	\$ 14/pc	660 pcs
Coupling (1-3/4")	\$ 21/pc	365 pcs
Drill bit (1-1/2")	\$ 20/pc	385 pcs
Drill bit (2")	\$ 30/pc	230 pcs
Drill bit (3")	\$ 80/pc	95 pcs
<u>II. BLASTING SUPPLIES</u>		
AN/FO	\$0.08/lb	1,134,000 lbs
High-strength primers	\$0.60/lb	51,325 lbs
Electric blasting caps	\$0.80/unit	97,200 units
Lead wire	\$2.70/lb	5,100 lbs
<u>III. ROOFBOLTING SUPPLIES</u>		
Drill steel (1-1/8" x 8')	\$ 50/pc	110 pcs
Striker bar (1-1/4")	\$ 47/pc	110 pcs
Coupling (1-1/4")	\$ 14/pc	110 pcs
Drill bit (1-3/4")	\$ 25/pc	110 pcs
Roofbolts, plates, & shells	\$3.15/unit	23,550 units
<u>IV. FUEL</u>		
Diesel fuel, underground	\$0.51/gal	162,500 gal
Diesel fuel, surface	\$0.51/gal	130,000 gal
Gasoline, surface	\$0.67/gal	12,000 gal
<u>V. ELECTRIC POWER</u>		
Cage hoist	\$0.06/KWH	448 KW/hr
Skip hoist	\$0.06/KWH	1,492 KW/hr
Crusher & conveyor	\$0.06/KWH	206 KW/hr
Mine pumps	\$0.06/KWH	1,223 KW/hr
Reinjection pump	\$0.06/KWH	150 KW/hr
Fresh water pump	\$0.06/KWH	38 KW/hr
Mine compressors	\$0.06/KWH	224 KW/hr
Mine ventilation-main fan	\$0.06/KWH	448 KW/hr
Mine ventilation-auxiliary fans	\$0.06/KWH	411 KW/hr
Miscellaneous utilities	\$0.06/KWH	1,200 KWH/shift

12.0 CONCLUSIONS AND RECOMMENDATIONS

12.1 TECHNICAL FEASIBILITY

The objective of the demonstration mining program is to evaluate, by using conventional mining methods for conceptual mining systems, the feasibility of extracting oil shale and associated saline minerals from deep, rich oil shale zones which occur within the central portion of the Piceance Creek Basin. Geologic and hydrologic data obtained from core holes intersecting the zone of interest indicate that relatively high rock temperatures, gassy mining conditions, and a moderate to high rate of ground-water inflow may be encountered. Physical properties tests of core samples from the projected mining zone suggest that the rock mass within the zone is competent, but manifests low unconfined compressive strength (5,000 to 8,000 psi per TCMRC) and significant creep characteristics. In the absence of specific knowledge of mining conditions gained through operating experience, the inferred conditions are considered to be a realistic basis for mine design.

The layout and design of the proposed oil shale demonstration mine incorporate appropriate provisions to cope with all anticipated adverse mining conditions. Optimum mine orientation was determined after review of all surface and subsurface geologic and geotechnical data available from the site. A gassy mining environment and high rock temperatures were assumed throughout mine development and ventilation planning. The mine dewatering system was designed to accommodate the worst conceivable inflow conditions, which are anticipated during demonstration of subsidence mining systems. Limiting dimensions of roof spans and pillars were calculated from rock strength data for the mining interval. Provisions have been made to eliminate potential hazards to the health and safety of mine personnel and to mitigate any potential impacts to the environment which may develop as a consequence of demonstration mining. From a technical standpoint, the demonstration mining program is feasible and practical.

12.2 ECONOMIC FEASIBILITY

Costs summarized in Tables 12.1 and 12.2 represent capital and operating costs for demonstrating the four specified mining systems. Anticipated costs (expressed in 1978 dollars) are presented for two alternate modes of mine operation. Cost estimates are based on performance of the work in the sequence depicted in Figure 6.1 (mine operating schedule). The operating schedule represents an optimum combination of concurrent and sequential performance of mining tasks, arranged to achieve maximum manpower and equipment utilization. From these tables it is evident that the two-shifts-per-day operating mode is the more cost-effective method of mine operation.

Several features have been incorporated in the demonstration mine layout to achieve significant reductions in overall project costs. Commercial designs of the four mining systems were scaled down and modified to minimize mining

TABLE 12.1

PROJECT COST SUMMARY
(One-Shift-Per-Day Operation)

(1978 Dollars)

<u>Year</u>	<u>Capital Expense</u>	<u>Operating & Maintenance Costs</u>	<u>Total</u>
1	\$4,927,800	\$ 1,414,000	\$ 6,341,800
2	853,400	1,612,000	2,465,400
3	1,549,800	1,865,800	3,415,600
4	815,200	2,420,400	3,235,600
5	123,000	2,586,100	2,709,100
6	332,400	2,691,200	3,023,600
7	<u>-</u>	<u>2,498,600</u>	<u>2,498,600</u>
TOTAL	\$8,601,600	\$15,088,100	\$23,689,700

TABLE 12.2
PROJECT COST SUMMARY
 (Two-Shifts-Per-Day Operation)
 (1978 Dollars)

<u>Year</u>	<u>Capital Expense</u>	<u>Operating & Maintenance Costs</u>	<u>Total</u>
1	\$5,781,200	\$ 2,310,900	\$ 8,092,100
2	2,365,000	3,357,100	5,722,100
3	455,500	4,165,800	4,621,200
4	<u>-</u>	<u>1,779,400</u>	<u>1,779,400</u>
TOTAL	\$8,601,600	\$11,613,200	\$20,214,800

costs. A common interval was selected for demonstration of all mining systems as opposed to a multiple-level layout. Development drifts were sized to minimize excavating costs. The block caving unit was situated directly above the sublevel stoping with full subsidence unit to combine analogous elements of the two systems, to reduce the cost of demonstrating the two systems, and to minimize environmental disturbance as a result of subsidence above the two units.

It is estimated that a significant portion (about 29%) of projected direct manpower requirements will be nonproductive. Typically, manpower utilization is a variable in mining which is sensitive to the "economies of scale." Because of the limited number of headings and/or production areas that can be worked concurrently during a demonstration-scale program, inefficient manpower utilization is unavoidable. This condition obviously exerts a negative impact on operating costs.

Several major capital items are to be designed and costed by others and were excluded from the Scope of Work (USEM Contract No. J0265020). These items include construction and equipping of a large-diameter vertical production/service shaft, a small-diameter ventilation shaft, and all necessary surface buildings and facilities. Costs for these structures and facilities have not been included in economic evaluations performed as a part of the present study. Hence, costs listed in Tables 12.1 and 12.2 do not represent total project costs for the demonstration mining program.

12.3 CONCLUSIONS

The oil shale deposits of the Piceance Creek Basin in northwestern Colorado represent a significant portion of the nation's known remaining domestic petroleum reserves. An estimated 70% of these reserves exist in deep, rich zones found at depths greater than 1,000 feet beneath the surface of the Basin. Despite intensified efforts in recent years to develop commercial technologies for the production of oil from shale, opinion remains divided regarding the most efficient means of resource exploitation. Schemes for in situ retorting appear to be attractive for environmental reasons, and may yield shale oil at a lower overall cost per barrel than comparable plants for surface retorting. However, conventional mining and surface retorting of oil shale offers the potential for recovery of a higher percentage of the total resource.

A demonstration mining program will provide essential information concerning the technical and economic feasibility of extracting oil shale and associated saline minerals from deep, rich zones. In view of the program's potential as a source of direction for the oil shale industry and its implications in areas of national energy planning, it is concluded that the oil-shale demonstration mining program should be funded and performed.

12.4 RECOMMENDATIONS FOR ADDITIONAL STUDY

One of the most significant benefits to industry afforded by the demonstration mining program will be the opportunity to monitor and evaluate rock

mass responses to the excavation, stabilization, and caving of prototype production openings. During design of the demonstration mine it was assumed that a carefully conceived and organized rock mechanics instrumentation plan will be implemented in conjunction with the mining program. Detailed observations of stresses, deformation, and creep characteristics of roof spans, rib pillars, and the caving block will permit an ongoing evaluation of mine structural performance during the demonstration program, and will greatly enhance the program's long-term value as a case study for commercial-scale mine design. Development of a rock mechanics instrumentation plan was beyond the Scope of Work of the present design study. However, it is recommended that such a plan be included as an integral feature of the demonstration mining program.

13.0 REFERENCES

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